



NI 43-101 TECHNICAL REPORT

ON THE

LOFDAL HEAVY RARE EARTHS PROJECT 2B-4 PRELIMINARY ECONOMIC ASSESSMENT (PEA) NAMIBIA

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Namibia Critical Metals Inc.

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1 SUMMARY

1.1 Introduction

The Lofdal property comprises exclusive prospecting license (EPL) 3400 and Mining License (ML) 200, located approximately 25 km NW of the town of Khorixas in the Kunene Region of north-western Namibia. The EPL and ML are held by Namibia Rare Earths (Pty) Ltd. (NRE (Pty)), a wholly owned subsidiary of Namibia Critical Metals Inc. (NMI). EPL 3400 was renewed by the Government of Namibia in 2020 and is in good standing until 27 September 2023. ML 200 was granted on 15 July 2021 and is valid until 10 May 2046.

Namibia Critical Metals (NMI or the Company) is a Canadian company listed on the TSX Venture Exchange which holds a diversified portfolio of projects within the Republic of Namibia. The company was formally known as Namibia Rare Earths Inc. The subject of this technical report is the Lofdal Heavy Rare Earth Project “2B-4” (Lofdal). Lofdal is developed under a Joint Venture Agreement with Japan Oil, Gas and Metals National Corporation (JOGMEC). The Company’s registered corporate office is Suite 802, Sun Tower, 1550 Bedford Highway, Halifax, Nova Scotia, NS B4A 1E6 Canada.

This preliminary economic assessment technical report (PEA) for the Lofdal Project “2B-4” is submitted herewith as an independent qualified person’s (QP) review and according to the National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

1.2 Accessibility, Climate, Local Resources, Infrastructures, and Physiography

The Project site is located on the farm Lofdal, in the Kunene region of Central Northwest Namibia. The (WGS84) coordinates of the Lofdal mine site are 20°21'S 14°45'E.

Northern Namibia is a semi-arid environment. The property is characterised by gently rolling topography and is lightly forested. There is good road access to the property, and the town of Khorixas is connected to the national telecommunications and electricity infrastructure.

There is no existing project infrastructure besides an exploration camp with PV power supply, a network of gravel roads and several water boreholes.

1.3 Geology and Mineralization

The Lofdal property is underlain by Paleoproterozoic metamorphic rocks of the Huab Metamorphic Complex, which outcrop as an inlier of the Congo Craton surrounded by stratified rocks of the Damaran Orogen. The metamorphic basement was intruded at ca 760 Ma by alkaline silicate rocks and carbonatites of the Lofdal Carbonatite Complex. The complex comprises an early silicate intrusive assemblage of dominantly nepheline syenite, and a later carbonatite intrusive assemblage ranging from calcitic through dolomitic and ankeritic carbonatites.

The Lofdal Carbonatite Complex comprises a central intrusive core characterized by several plugs of nepheline syenite and carbonatite with associated diatreme breccias, surrounded by a wide area of dyke intrusion and associated hydrothermal alteration. The phonolite and carbonatite dykes have exploited pre-existing structures in the basement that were re-activated during Neoproterozoic tectonism.

Rare earth element mineralization in the Lofdal Carbonatite Complex is closely associated with the carbonatite dykes and related hydrothermal alteration. These occur within an area of more than 200 km².

The lithogeochemical database demonstrates that many of the dykes are geochemically anomalous in rare earth elements (REE, which includes yttrium as a heavy rare earth) with a significant number being of economic interest. Of particular significance is the frequent enrichment of heavy rare earths in the dykes and in structurally controlled hydrothermal alteration zones, which trend predominantly in NE - SW and NNE - SSW directions.

The REE are subdivided into heavy rare earth elements (HREE) and light rare earth elements (LREE). Lanthanum, cerium, praseodymium, neodymium, promethium, and samarium are the LREE. Yttrium, europium, gadolinium, terbium, dysprosium, holmium, erbium, thulium, ytterbium, and lutetium are the HREE. Although yttrium is lighter than the light rare earth elements, it is included in the heavy rare earth group because of its chemical and physical associations with heavy rare earths in natural deposits.

The REE mineralization in the Lofdal Carbonatite Complex is variable and includes both LREE and HREE enriched varieties that appear to have been introduced in separate mineralizing events.

Petrographic evidence suggests that the HREE-rich mineralization resulted from a dominantly hydrothermal event that occurred relatively late in the history of carbonatite emplacement. Mineralogically, the heavy rare earth-enriched mineralization is dominated by xenotime (Y-REE phosphate), which is commonly associated with zircon, rutile, apatite and/or thorite. The mineralized hydrothermal alteration systems are continuous both along strike and at depth and produce clear geological, geochemical and radiometric signatures that are easily recognized, particularly in drillhole core.

1.4 Exploration Status

Two areas have been evaluated by the recent drilling, these being known as Area 4 and Area 2B. The first Mineral Resource estimate in accordance with NI 43-101 was reported in 2012 for Area 4 based on geochemical analyses and density measurements of core samples obtained from 93 diamond drillholes completed by NRE in 2011 and 2012. An additional 17 holes were drilled in 2012 and 2013 and a further 56 were drilled in 2020 with the purpose of extending the Mineral Resource along strike to the west and at depth. For Area 2B, 17 holes were drilled in 2010 and 2011 with an additional 29 drillholes completed in 2020. The purpose of the 2020 campaign for Area 2B was to demonstrate the continuity of the mineralization within and adjacent to the pre-existing drilling.

Drilling was orientated in a north to north-northwest direction with inclinations from 60° to 68° for Area 2B and from 55° to 71° for Area 4. Drillhole spacing for Area 4 is variable, with holes drilled in 2011 to 2012 positioned as close as 25 m apart. The 2020 campaign extended the Mineral Resource westwards with a 50 m spaced grid near surface, widening to 100 m down-dip. At Area 2B, drillhole spacing is predominantly 50 m, widening to 100 m in the northeast strike extension. Drilling has demonstrated that the mineralization continues down to a vertical depth of at least 300 m for Area 4 and 150 m for Area 2B.

1.5 Mineral Processing and Metallurgical Testing

Two main rounds of testing were conducted on samples originating from the Lofdal property (first on sorted products from trench samples and second on the low grade run of mine samples of fresh material from the starter pit) and provided to SGS Canada, located in Lakefield, Ontario. Samples received for each of these rounds of testing were subjected to a scoping and optimization beneficiation test program followed by hydrometallurgical testing to produce a mixed rare earth oxide product. Each beneficiation program demonstrated that upgrading via flotation could upgrade the received samples 13-25 times with a recovery of 67-70%.

The first flotation concentrate, assayed at 7.6% TREO, was subjected to various hydrometallurgical flowsheets including a caustic-crack with acid leach process, a gangue pre-leach process, and direct acid baking with sulphuric acid. The sulphuric acid bake process proved most effective and was further optimized to solubilize 98% of the contained yttrium, culminating in a leach liquor with a composition of 14.2 g/L TREE alongside impurities. Impurity removal steps consisted of pH adjustment with magnesium carbonate (removing Fe, Th, and some Al), crude REE precipitation and redissolution (removing additional Fe, Th, and Al) before the re-leach liquor was subjected to ion exchange and solvent extraction testing to remove uranium and thorium, respectively. Oxalic acid precipitation of rare earth elements followed by calcination to convert to rare earth oxides generated a product assaying at 98.1% TREO representing 94% of the rare earth elements found in the flotation concentrate.

The second flotation concentrate, assayed at 5.6% TREO, was used to confirm operating conditions determined in earlier hydrometallurgical testing with some acid bake testwork also considering adjusted reagent additions to verify optimum conditions.

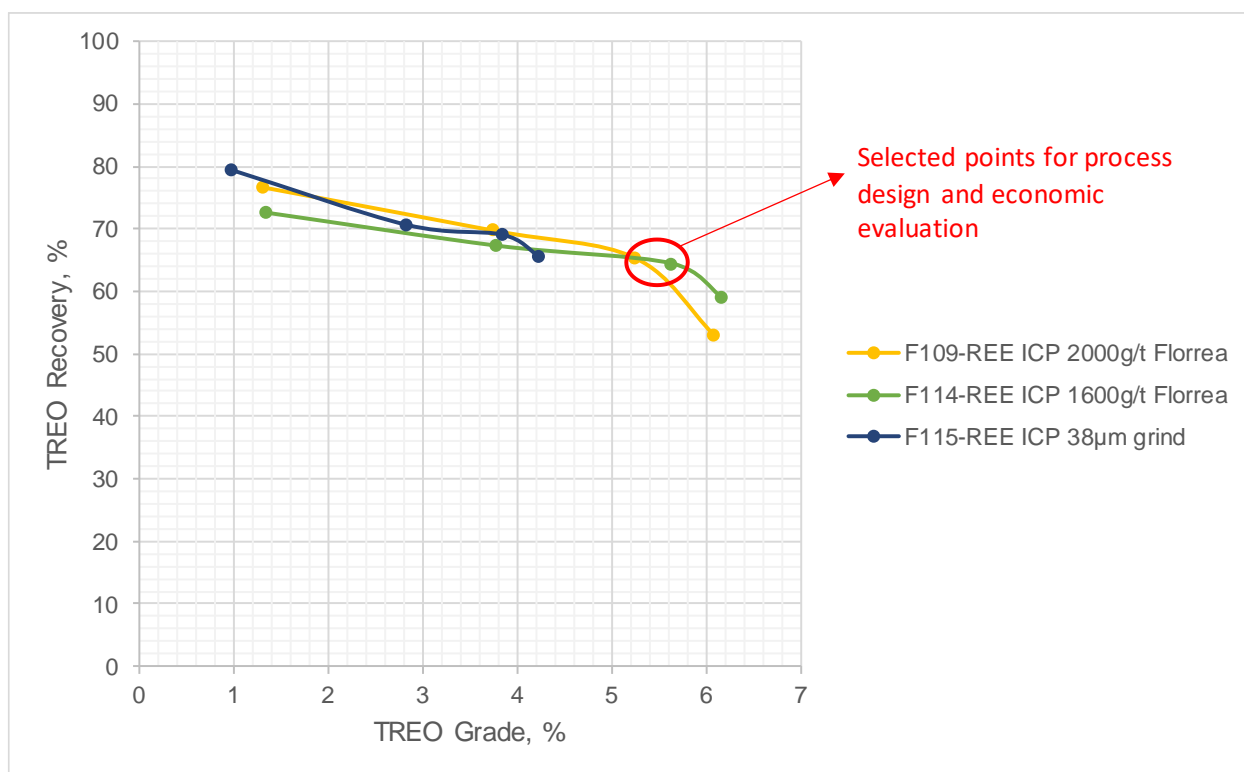


Figure 1-1 Flotation Grade / Recovery Curves for Process Design and Economic Evaluation

Only part of the hydrometallurgical flowsheet could be confirmed with the available flotation concentrate due to mass restrictions – acid baking with water leaching, impurity removal with magnesium carbonate, and crude REE precipitation. These confirmatory tests demonstrated that 96% of the yttrium was solubilized and that the impurity removal and crude REE precipitation steps performed similarly.

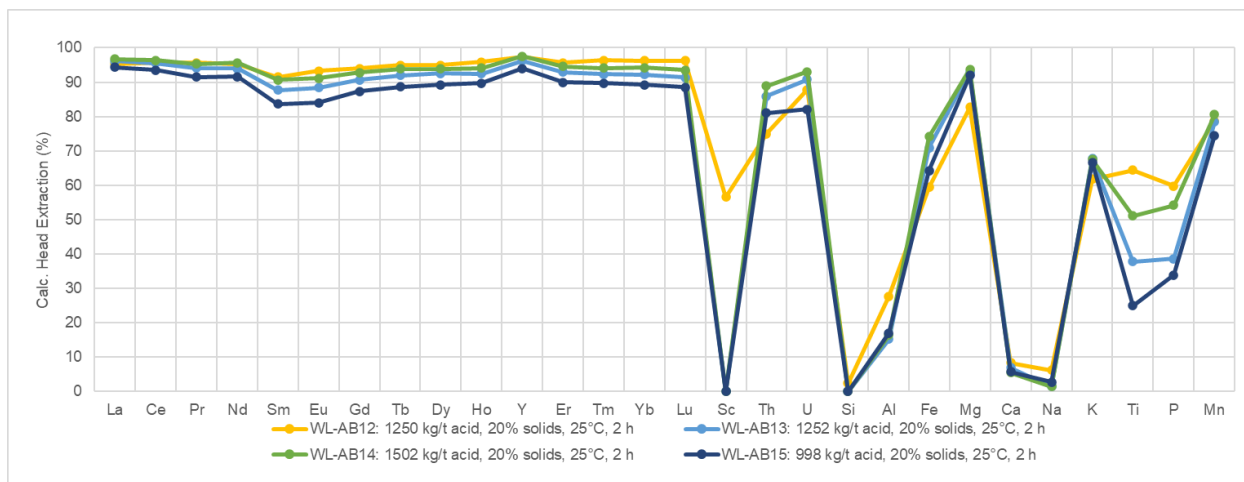


Figure 1-2 Acid Bake and Water Leach Extraction

While the results of the confirmatory tests are positive, there remains room to optimize each process in the flowsheet regarding OPEX and CAPEX as well as recoveries in proposed continuous pilot plant testing.

The selected flowsheet for the PEA basis of design is shown in Figure 1-1.

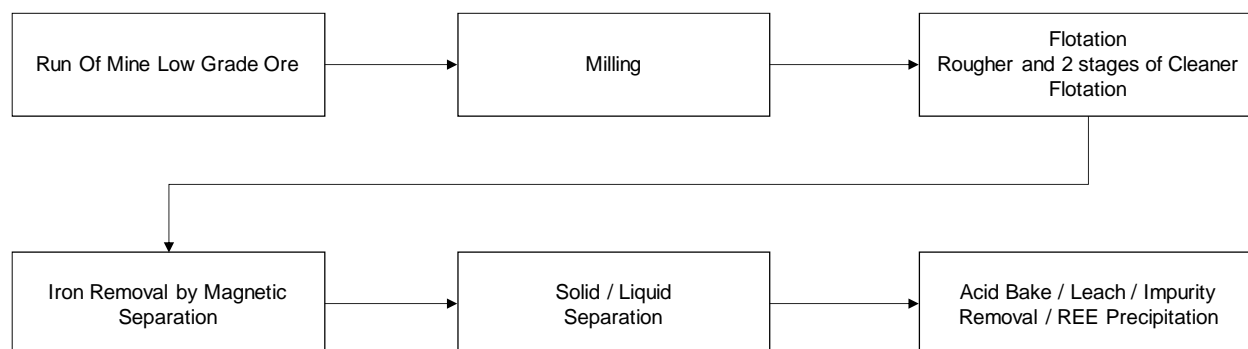


Figure 1-3 Process Block Flow Diagram

High level process design criteria for the PEA minerals processing plant includes:

- ROM tonnage 2160 ktpa
- ROM TREO grade 0.187% TREO (as per LOM mine plan)
- Mill & Flotation Feed Tonnage 2160 ktpa
- Flotation plant mass pull 2.9%
- Flotation plant recovery 64.4 % TREO (56.2% LREO and 68.1% HREO)
- Flotation product grade 5.6% TREO
- Hydrometallurgical recovery 93.5 % TREO
- Overall recovery 60% TREO (53% LREO and 68% HREO)

1.6 Mineral Resource Estimate

Lofdal was visited by Jeremy Witley, who is a Principal Mineral Resource Consultant with MSA and the Qualified Person for this Mineral Resource Estimate, from October 28 to 30, 2020 and again on 10 November 2022. The occurrences and setting of the REE mineralization were observed in the field as well as the drilling in progress at the time. The mineralization was examined in a selection of drillhole cores from the 2020 and previous drilling programs. The QP was satisfied that the procedures and protocols used in drilling are consistent with CIM Exploration Best Practices Guidelines. On the second site visit three of the drillholes completed since the previous site visit were inspected as well as the bulk sampling pit from which the metallurgical samples were extracted.

The assay results received from the primary laboratory (Actlabs in Ancaster, Ontario, Canada) were subjected to a quality assurance and quality control program and the assays have been confirmed by check assays completed by a second laboratory (ALS Minerals, North Vancouver, Canada). The drilling, logging, sampling and assay data are contained in a well organised drillhole database that the QP considers to be suitable for the purposes of mineral resource estimation.

The Mineral Resource Estimate was based on sample assays and density measurements obtained from the cores of diamond drillholes completed in two phases of drilling; 2011 to 2012 and 2020.

For the purposes of creating a framework for mineral resource estimation, fourteen mineralised zones were modelled for Area 4 and seven for Area 2B using a statistically defined cut-off of 10 ppm Dy_2O_3 and 12 ppm Dy_2O_3 respectively. The resultant vein-like bodies within each deposit tend to be orientated parallel to one another, some of which coalesce in places at depth and along strike.

Ordinary kriging was used to estimate the individual rare earth element grades into a three dimensional block model. Density was estimated into the same block model using inverse distance weighting. The Mineral Resource for Area 4 extends up to 1,600 m along strike near surface and attains a maximum depth of approximately 400 m. For Area 2B, the Mineral Resource extends for 700 m near surface and attains a maximum depth of approximately 250 m.

The Mineral Resource Estimate was completed by Mr R. Goncalves (BSc Hons) under the supervision of Mr J.C. Witley (BSc Hons, MSc (Eng.)).

The Mineral Resource was estimated using The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Mineral Resources and Mineral Reserves Best Practice Guidelines (2019) and is reported in accordance with the 2014 CIM Definition Standards, which have been incorporated by reference into National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101). The Mineral Resource is classified into the Measured, Indicated and Inferred categories for Area 4 (Table 1-1) and into the Indicated and Inferred categories for Area 2B (Table 1-2).

The Mineral Resource is reported from a Whittle optimised pit shell at a base case total rare earth oxide (TREO) grade of 0.10%, which the QP considers will satisfy reasonable prospects for eventual economic extraction. The Whittle optimised pit shell used the following assumed parameters.

Mining will be by open-pit methods:

- 45° slope angle in the partially weathered rock and 55° slope angle in the fresh rock
- 5% mining dilution
- 5% mining loss
- 10 m bench height
- Ore production rate of 1.68 million tonnes per annum.
- 75% final metallurgical recovery of TREO

Costs were assumed as follows:

- Mining cost for drill and blast: USD 2.75 / tonne mined.
- Processing costs: USD 30.6 / tonne milled
- G&A cost: USD 7.36 / tonne milled
- NMI price USD 66.20 per Kg TREO+Y₂O₃ (based on a discounted price deck from ARGUS Rare Earths Analytics and the estimated TREO proportion in concentrate.
- Offshore treatment cost and shipment priced in discounted basket price.

Table 1-1 Area 4, Measured, Indicated and Inferred Mineral Resource Estimates above 0.1% TREO Cut-Off Grade – 12 May 2021

Category	Tonnes (Mt)	TREO* %	HREO** %	LREO*** %	Dy ₂ O ₃ ppm	TREO (kt)
Measured	5.93	0.21	0.14	0.07	138	12.71
Indicated	36.63	0.16	0.08	0.08	82	59.97
Measured & Indicated	42.57	0.17	0.09	0.08	90	72.68
Inferred	6.09	0.17	0.07	0.09	72	10.12

Notes:

- (1) All tabulated data have been rounded and as a result minor computational errors may occur.
- (2) Mineral Resources, which are not Mineral Reserves, have no demonstrated economic viability.
- (3) Quantities reported are the total quantities for the project regardless of ownership.
- (4) *TREO = Total Rare Earth Oxides and includes Y₂O₃
- (5) **HREO = Heavy Rare Earth Oxides and includes Y₂O₃
- (6) ***LREO = Light Rare Earth Oxides
- (7) Mt = Million tonnes, kt = Thousand tonnes.

Table 1-2 Area 2B, Indicated and Inferred Mineral Resource Estimates above 0.1% TREO Cut-Off Grade – 12 May 2021

Category	Tonnes (Mt)	TREO* %	HREO** %	LREO*** %	Dy ₂ O ₃ ppm	TREO (kt)
Indicated	2.20	0.19	0.10	0.09	104	4.27
Inferred	2.58	0.19	0.09	0.09	92	4.80

Notes:

- (1) All tabulated data have been rounded and as a result minor computational errors may occur.
- (2) Mineral Resources, which are not Mineral Reserves, have no demonstrated economic viability.
- (3) Quantities reported are the total quantities for the project regardless of ownership.
- (4) *TREO = Total Rare Earth Oxides and includes Y₂O₃
- (5) **HREO = Heavy Rare Earth Oxides and includes Y₂O₃
- (6) ***LREO = Light Rare Earth Oxides
- (7) Mt = Million tonnes, kt = Thousand tonnes.

1.7 Mineral Reserve Estimate

No mineral reserves have been estimated for the property.

1.8 Mining Methods

The proposed mining method is conventional open pit mining. Mineralized rock and waste would be drilled, blasted, loaded by hydraulic shovels and hydraulic excavators into off-highway dump trucks, and hauled to the processing plant.

The basis for the pit design work was the mineral resource block model that was developed by MSA as part of a NI 43-101-compliant mineral resource estimate (refer to Section 14).

There are two primary Lofdal deposits currently under consideration. Due to the nature of the deposit, the resultant pits are narrow and deep. Currently no backfilling is contemplated.

The proposed mining method is the development of a slot in front of the mineralized zone at each level. The centralized slot will enable waste mining on one side while mining mineralized material on the other side. This methodology will also facilitate separating mineralized material from waste material.

The target ROM feed to the processing plant is 2 000 000 tonnes/annum. The plant feed is mineralized material with a TREO $\geq 0.1\%$. In the current schedule the production ramps up to this target over 4 years. Mineralized material with TREO $\geq 0.05\%$ and $< 0.1\%$ is sent to a stockpile so it could be processed at a later stage if economics allow.

The combined Life of Mine of the two pits is 16 years.

The initial geo model was in Datamine and the mining model was created in Surpac. Merging the Datamine domain values into the Surpac models resulted in blocks containing a combination of mineralised material and waste. Attributes were added for mining and processing cost adjustment factors used by Whittle, and for REO prices and the calculated revenue for HREO and LREO.

The processing cost adjustment factor (PCAF) was set at 1, that is all mineralized blocks incur the same processing cost. The mining cost adjustment factor (MCAF) increased with depth. The models were then exported to Whittle using the revenue values as grade values.

The Whittle pit shells selected as the templates for the pit designs for Area 4 and Area 2B are those with a revenue factor of 1, corresponding to shell 36 in each case, with resultant tonnages. As per standard practice in mine design, the Whittle pit shells are used as templates to guide the pit design process.

The initial step was to design a pit shell without ramps to determine how closely the design could be matched to the Whittle shell while applying batter angles and berm widths. Whittle adds blocks to the pit until the maximum value is reached without consideration for the practicality of mining the resultant pit. This results in drop-cuts of single blocks or small groups of blocks into the pit floor. Consequently, in a narrow deposit such as Lofdal it is not practical to design a pit as deep as the Whittle shell as the pit bottom becomes too small to deploy equipment. Removing these drop-cuts results in a more practical layout.

These initial designs were then reviewed to determine the number and location of ramps to ensure access for all operating benches. With the inclusion of the ramp system the overall highwall slope in Pit 4 is approximately 55° .

At this stage of the mine design (PEA) no optimization of the relevant waste dumps or topsoil stockpile dumps has been attempted. Preliminary dump designs were produced for both Area 4 and Area 2B. The nominal lift height assumed is 15m.

Due to the arid nature of the region, no serious dewatering issues are expected. However, minimal pumping capacity has been allowed for. Water ingress due to rainfall will be managed with berms and cut-off drains. Separate sets of mining equipment are envisaged for waste mining and ROM production. The production schedule was developed using Xpac scheduling software.

A 20m x 20m grid was created for each area such that the block orientation is approximately aligned with the general strike of the deposit. This grid was then used to reserve the mining model within the design pit to create input files for Xpac.

Haulage reference points were defined at the entrance to the ramp on each level, and where the ramp reaches surface. In-pit haul distances, both on bench and on ramp were calculated in the Xpac database with reference to these points. The on-bench distances were calculated for each block as the sum of the Northing and Easting distance to the applicable reference point. The on-ramp distance is the vertical height from the reference point to the ramp exit multiplied by the ramp gradient of 1:10.

Resultant schedules were then exported as csv files and used as input to the economic model.

1.9 Recovery Methods

Ore from the A4 and A2B deposit will be processed from low grade run-of-mine ore into concentrate using the flowsheet developed from the testwork described in Section 17. The main sections of the concentrate flowsheet consist of crushing and screening, grinding, flotation, magnetic separation (iron removal), water leach and acid bake, impurity removal, U IX and precipitation, REE precipitation, re-leach, silicon removal, thorium solvent extraction and REE oxalate precipitation/calcination and final neutralization. A mixed rare earth oxide product is produced.

1.10 Project Infrastructure

Project infrastructure will exist in the vicinity of the two open pits: A4 and A2B general site and the Mill Process Plant site.

The Lofdal general site includes A4 and A2B open pit, waste dumps, mill complex, tailings storage facility, electrical distribution, offices, warehouse, maintenance, and effluent treatment.

The concentrate (mixed rare earth oxide final product) will be produced from mill processing.

1.11 Rare Earth Pricing

Current market conditions indicate low pricing for lanthanum and cerium products; thus, no recovered value was assigned to these products.

Other prices used for this PEA are derived from recent market analysis and other published NI 43-101 resource reports and presented in the following Table 1-3.

Table 1-3 Current Market Prices used for the Current PEA in USD/Kg

Pricing Forecast for REE	Pricing Used for Analysis	Distribution
La ₂ O ₃	\$-	9.2%
Ce ₂ O ₃	\$-	16.0%
Pr ₂ O ₃	\$201.00	1.7%
Nd ₂ O ₃	\$212.00	6.3%
Sm ₂ O ₃	\$5.00	2.2%
Eu ₂ O ₃	\$36.00	1.1%
Gd ₂ O ₃	\$109.00	4.3%
Tb ₂ O ₃	\$2,493.00	0.9%
Dy ₂ O ₃	\$587.00	6.2%
Ho ₂ O ₃	\$290.00	1.3%
Er ₂ O ₃	\$64.00	3.8%
Yb ₂ O ₃	\$20.00	3.5%
Lu ₂ O ₃	\$947.00	0.5%
Y ₂ O ₃	\$17.00	42.4%
Tm ₂ O ₃	\$500.00	0.6%
Average Basket Value	\$103.64	
Realized Basket Price after Treatment Charges	\$91.64	

1.12 Socio-Economic and Environmental Impact

Collection of environmental baseline data for the Lofdal Project has been ongoing since 2016. The baseline studies were designed and implemented to support requirements for future planning and permitting purposes. NMI has taken on active role in communicating and consulting with the local communities.

1.13 Legal and Statutory

For the Project to proceed successfully, a number of legislative requirements will need to be fulfilled according to the Namibian legislation and possible international legislation and guidelines. NMI will be responsible for ensuring that the welfare of the local population is not significantly impacted upon due to the mining activities. In addition, NMI must ensure that adequate rehabilitation and closure of the mine takes place following the conclusion of the proposed mine.

To ensure that the legislative requirements are met, as well as best practices are implemented, environmental degradation and pollution must be prevented and, where unavoidable, mitigated, and managed. The predominant impacts associated with the mining activities are due to groundwater quantity, groundwater and dust contamination and the potential side effects of thorium. Other social ills may result from the project due to the influx of job seekers causing an increase in the population of Khorixas.

1.14 Capital Cost Estimate

The total capital costs for the Project are estimated at USD \$207.08 M and include direct capital costs for mill site process plant, tailings storage facility; sustaining capital for the mill site process plant, TSF closure costs; indirect costs and contingency. Indirect costs, including EPCM, first fills, spares and a camp allowance have been estimated at 30% of Direct and Indirect plant capital costs. TSF construction has a contingency of 15% as much of the construction material will be sourced from the pit stripping. The summary of the initial direct capital and total capital costs are presented in Table 1-4.

Table 1-4 Total Capital Cost Summary

Description	Total Cost (USD)
Direct Mill Site Process Plant	\$117.58 M
Direct Tailings Storage Facility	\$13.63 M
Subtotal Initial Direct Capital Costs	\$131.21 M
Sustaining Capital Processing	\$6.01 M
Sustaining Capital Tailings Storage Facility	\$5.43 M
Mine Closure Costs	\$5.00 M
Indirect Costs	\$18.56 M
Contingency	\$40.87 M
Total Capital Cost Estimate	\$207.08 M

1.15 Operating Cost Estimate

The overall operating costs for the Mill Site process plant, mining operation and G&A are in Table 1-5. This table shows the overall operating costs per tonne of ROM material mined and per kg of TREO produced.

Table 1-5 Total Operating Cost Summary

Description	Cost per Tonne Processed (USD/t)	Cost Per kg TREO Recovered (USD/t)
Mining	\$16.25	\$14.32
Mill Site Process Plant	\$32.00	\$28.21
General and Administration	\$1.41	\$1.25
Royalties	\$5.20	\$4.58
Total Operating Cost	\$54.86	\$48.36

1.16 Economic Analysis

An engineering economic model was prepared for the Project to estimate annual cash flows and assess sensitivities to certain economic parameters. The economic results of this report are based upon the services performed by SGS and NMI. The Project includes two open pits, surface infrastructure to support the mine operations (maintenance and office facilities), water management features, a run-of-mine stockpiling area, processing facility, and a tailings storage facility.

The economic analysis assumes that the Project will be 100% equity financed and uses parameters relevant as of September 2022, under conditions likely to be applicable to project development and operation and analyses the sensitivity of the Project to changes in the key Project parameters. All costs have been presented in United States Dollars (US\$) and wherever applicable conversion from South African Rand (ZAR) has utilized an exchange ratio (ZAR/US\$) of 16.07. Mining and treatment data, capital

cost estimates and operating cost estimates have been put into a base case financial model to calculate the IRR and NPV based on calculated Project after tax cash flows. The scope of the financial model has been restricted to the Project level and as such, the effects of interest charges and financing have been excluded. For the purposes of the PEA, the evaluation is based on 100% of the Project cash flows before distribution of profits to the equity owners. Both pre-tax and after-tax cash flows have taken 5% royalty payments into account.

Based on the extraction of 2,160,000 t/a of ROM feed from the mine, the project is anticipated to yield a pre-tax IRR of 34% with a pre-tax NPV, at a discount rate of 5% of USD \$632.7M, and an after-tax IRR of 28% with an after-tax NPV, at a discount rate of 5%, of USD \$391 M. Cumulative cash flows are USD \$1,110 M pre-tax and USD \$698.7 M after-tax over the sixteen-year LOM.

The project is expected to pay back initial capital in 3.2 years after production starts.

Table 1-6 Summary Financial Results

Pre-Tax NPV @5%	\$632.7 M
Pre-Tax IRR	34%
Pre-Tax Net Cash Flow	\$1110.4 M
After Tax NPV @5%	\$391.0 M
After Tax IRR @5%	28%
After Tax Net Cash Flow	\$698.7 M

1.17 Conclusions

This PEA demonstrate that the Lofdal Heavy Rare Earth Project has the potential to be technically and economically viable as a producer of rare earth elements concentrate.

1.18 Recommendations

- Undertake an infill drilling programme in the portions of Inferred Mineral Resource falling within the Mineral Resource pit-shells to ensure that all Mineral Resources that have the potential to be converted to Mineral Reserves are in the Indicated category.
- The project area has potential to increase the Mineral Resources. Should additional Mineral Resources be required to support the project, further exploration would be warranted.
- On the mill processing area, there are future opportunities as well as possible trade off studies for crushing and comminution circuit optimization. The future testwork can introduce circuit optimization, variability, and clarification testwork regimes including further flotation optimization. These will provide further clarity to the circuit design. Some of these suggested campaigns are listed in Section 17 (Future Testwork).
- No geotechnical study has yet been undertaken. This should be done if the project advances to PFS as this will have a material impact on the stripping ratio. A preliminary hydrological study should be commissioned to validate the assumption that there is no water related issues at depth.
- Knight Piésold had completed the PEA level study and report for the Tailings Storage Facility (TSF). Based on the TSF report, it was recommended to complete at pre-feasibility level an alternative assessment for the TSF location in consideration of overall risk to the operation, waste, and water management. The cost reduction opportunity for the TSF can unfold through revised deposition

strategy, staging and scheduling. Conducting field and laboratory testwork including foundation and tailings physical and geochemical characterization is required to advance the design and confirm lining requirements and the sourcing of aggregate material on site. Develop site water balance, freshwater requirements, and overall site wide water management. Finally develop a detailed dam breach analysis for dam design to assess risks to the downstream users.

- Carry out a six-month PFS to further develop the engineering design of the plant and recognise value engineering where possible.
- Revisit the capital cost estimates in general for possible savings due to optimising the cost estimates from $\pm 50\%$ to $\pm 10\%$ (PFS Level).
- To advance the Lofdal heavy rare earth project 2b-4 towards the next stage of Engineering, the proposed budget is estimated at \$USD 3.6 million, involves a major upgrade drilling program, water supply studies, geotechnical/hydrogeological, TSF optimization studies, continued environmental and community liaison, mineral processing (metallurgy and hydrometallurgy), mine access study and an engineering study.

The detailed recommendations list is provided in Section 26.

2 INTRODUCTION

This Report was prepared and compiled by the QPs under employment with SGS and Consultants at the request of Namibia Critical Metals Inc. (“Namibia Critical Metals” or the “Company” or “NMI”). The purpose of this Report is to provide a Preliminary Economic Assessment (“PEA”) for the Lofdal Heavy Rare Earth Project “2B-4” (“Lofdal” or the “Project”) in Namibia. This PEA aims at a significantly larger annual run-of-mine and plant throughput of 2 million tonnes per year and longer mine life than the historical PEA of 2014 by mining from two sub-deposits namely “Pit 2B” and “Pit 4”. Further, the processing flow sheet was simplified to a direct flotation of the run-of-mine material and expanded to include a hydrometallurgical unit producing a >98% mixed rare earth oxide as final product instead of xenotime concentrate.

This Technical Report has been prepared to comply with disclosure and reporting requirements set forth in the Toronto Venture Exchange (TSX-V) Corporate Finance Manual, Canadian National Instrument 43-101, Companion Policy 43-101CP, Form 43-101F1, the ‘Standards of Disclosure for Mineral Projects’ of January 2011 (the Instrument) and the Mineral Resource and Reserve classifications as defined in the CIM Definition Standards 2014 document.

Namibia Critical Metals (NMI) is a Canadian company listed on the TSX Venture Exchange which holds a diversified portfolio of projects within the Republic of Namibia. The company was formally known as Namibia Rare Earths Inc. The subject of this technical report is the Lofdal Heavy Rare Earth Project 2B-4 (Lofdal) which is held in a Joint Venture Agreement with Japan Oil, Gas and Metals National Corporation (JOGMEC). The Company’s registered corporate office is Suite 802, Sun Tower, 1550 Bedford Highway, Halifax, Nova Scotia, NS B4A 1E6 Canada.

2.1 Purpose of Report

The purpose of this Report is to publish a Technical Report on the Lofdal Heavy Rare Earth 2B-4 Project summarizing:

- the land tenures, exploration history, and drilling;
- the mineral resource estimates at “Pit 2B” and “Pit 4”;
- a conceptual mine plan at a level to support a Preliminary Economic Assessment;
- the supporting infrastructure including, power, buildings, tailings management facility,
- processing plant, etc. to support the conceptual mine plan;
- the environmental permitting requirements;
- capital expenditure and operating expenditure estimates;
- a financial model and perform an economic analysis and;
- provide recommendations and additional work.

2.2 Terms of Reference

NMI engaged the services of SGS and authors on January 22, 2021, to write an independent NI 43-101 Technical Report on the Lofdal Heavy Rare Earth Project 2B-4 Property in Namibia. This Report was prepared in accordance with NI 43-101 and Form NI 43-101F1 and Companion Policy 43 101CP.

2.3 Qualifications of Consultants

SGS and Consultants preparing this technical report are specialists in the fields of geology, exploration, mineral resource estimation, open pit mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, civil, mechanical, electrical, capital and operating cost estimation, and mineral economics.

None of the Consultants or any associates employed in the preparation of this report has any beneficial interest in NMI. The Consultants are not insiders, associates, or affiliates of Namibia Critical Metal Inc. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between NMI, SGS and the Consultants. The Consultants are being paid a fee for their work in accordance with normal professional consulting practice

2.4 Report Responsibility and Qualified Persons

The following individuals, by virtue of their education, experience, and professional association, are considered Qualified Persons (QP) as defined in the NI 43-101 standard, for this report, and are members in good standing of appropriate professional institutions:

- Swinden Geoscience Consultants under the supervision of Scott Swinden, (Ph.D, P.Geo) :
Sections of the Report dealing with Property Description and Location (Item 4), Accessibility, Climate, Local Resources, Infrastructure and Physiography (Item 5), History (Item 6), Geological Setting and Mineralisation (Item 7), Deposit Types (Item 8), Exploration (Item 9), Drilling (Item 10), Sample Preparation, Analyses and Security (Item 11).
- MSA Group under the supervision of Jeremy Witley, (BSc Hons, MSc (Eng.)):
Sections of the Report dealing with Data Verification (Item 12) and Mineral Resource Estimate (Item 14).
- SGS Canada under the supervision of Johnny Canosa (B.Sc., P.Eng.):
Sections of the Report dealing with Project Infrastructure (Item 18 except for Items 18.9, 18.11, 18.12).
- SGS Canada Inc. under the supervision of Micheal Archer (Sr. Metallurgist):
Section of the Report dealing with the Mineral Processing and Metallurgical Testing (Item 13).
- SGS Canada Inc. under the supervision of Daniel Millar (Pr.Eng.):
Section of the Report dealing with the Recovery Methods (Item 17).

- SGS Canada under the supervision of William van Breugel (B.Sc. Hons, P.Eng.):
Sections of the Report dealing with Market Studies and Contracts (Item 19), Environmental Studies, Permitting and social or community impact (Item 20), Capital and Operating Costs (Item 21), and Economic Analysis (Item 22).
- BBMC RSA under the supervision of Johann Hager (BEng Hons, MEng Mining, P.Eng.):
Sections of the Report dealing with Mining Methods (Item 16).
- Knight Piésold Consulting (Pty) Ltd under the supervision of Veronique Daigle (Pr.Eng.):
Sections of the Report dealing with Tailings Storage Facility (TSF) and associated structures (Item 18.9), Return Water Dam (RWD) pond (Item 18.11), and conceptual Tailings Storage Facility (TSF) (Item 18.12).

A NI 43-101 compliant report entitled Preliminary Economic Assessment on the Lofdal Heavy Rare Earths 2B-4 Project, Namibia (“the Report”)

The preceding QPs have contributed to the writing of this Report and have provided QP certificates, included at the end of this Report. The information contained in the certificates outlines the sections in this Report for which each QP is responsible. Each QP has also contributed figures, tables, and portions of Sections 1 (Summary), 2, (Introduction), 3 (Reliance on other Experts), 25 (Interpretation and Conclusions), 26 (Recommendations), and 27 (References). Table 2-1 outlines the responsibilities for the various sections of the Report and the name of the corresponding Qualified Person.

2.5 Site Visit

Personal inspections made by the Qualified Persons and their items of responsibility for this report are shown in Table 2-1.

Table 2-1 Details of Site Visits and Responsibilities of the Qualified Persons

Qualified Person	Personal Inspection Dates	Site Items Responsible for	Items Co-Responsible for
Jeremy Witley	October 28 to 30, 2020; November 10, 2022	1.6, 12, 14, 25.1	2, 24, 26 and 27
Scott Swinden	April 25, 2018; July 31, 2017; May 28 to July 24, 2014	1.3, 4 to 11	1, 2, 24 to 27

2.6 Currency, Units, Abbreviations and Definitions

All units of measurement used in this technical report are International System of Units (SI) or metric, except for Imperial units that are commonly used in industry (e.g., ounces (oz.) and pounds (lb.) for the mass of precious and base metals). Every effort has been made to clearly display the appropriate units being used throughout the Report. All currency is in US dollars (USD or \$), unless otherwise noted. The locations of all

maps are referenced to WGS 84, UTM Zone 33S, unless otherwise stated. Frequently used abbreviations and acronyms can be found in Table 2-2.

This Report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs consider them immaterial.

Table 2-2 List of Abbreviations

%	Percent sign
°	Degree
°C	Degree Celsius
cm	Centimetre
g	Grams
g/t	Grams per metric tonne
Ga	Billion years
GPS	Global Positioning System
ha	Hectare
ICP-MS	Inductively coupled plasma mass spectrometry
ICP-OES	Inductively coupled plasma optical emission spectrometry
kg	Kilograms
km	Kilometres
m	Metres
m ³	Cubic metres
Ma	Million years
mm	Millimetre
MRE	Mineral Resource Estimate
Mt	Million tonnes
N, S, E, W	North, South, East, West
ppm	Parts per million
QA	Quality Assurance
QC	Quality Control
QP	Qualified Person
SG	Specific Gravity
SGS	SGS Canada Inc. Geological Services
SGS Lakefield	SGS Minerals Services Lakefield Facility
tonnes or t	Metric tonnes
µm	Micrometers
US\$	US Dollar
UTM	Universal Transverse Mercator

2.7 Effective Date

The effective date of this technical report is October 3, 2022.

As of the effective date of this Report, the authors are not aware of any material fact or material change with respect to the subject matter of this Technical Report that is not presented herein, or which the omission to disclose could make this Report misleading.

2.8 Previous Technical Reports

A Preliminary Economic Assessment (PEA) on the Project was completed by MDM Engineering (MDM, 2014) on October 1, 2014, titled “Preliminary Economic Assessment on the Lofdal Rare Earths Project, Namibia”. Information considered by the QPs to be both current and relevant was sourced from this document. The 2021 Mineral Resource Estimate reported in this current Technical Report is substantially different to that on which the 2014 PEA was completed and therefore the results of the 2014 PEA are not considered current and are no longer relevant.

The sources of information as referenced throughout this report are as follows:

- Data supplied by NMI;
- Technical report by MSA on mineral resource estimate and associated geological background information;
- Information Provided for Section 16 Mining Method provided by BBMC;
- Information provided by SLR on Environmental Management Plan for the Lofdal Mine (SLR Project No.: 734.14013.00005, Report No.:5 Dated June 2016 and Summary of the Environmental Impact Assessment Report and Specialist Studies for the Lofdal Mining Project;
- Information provided by Knight Piésold Consulting on Tailings Storage Facility Conceptual Design Report Dated October 4, 2022. (RI301-00928/01-A).

3 RELIANCE ON OTHER EXPERTS

MSA/SGS has not independently verified, nor is it qualified to verify, the legal status of the Lofdal property. The report has been prepared on the assumption that the tenements will prove lawfully accessible for evaluation.

Neither MSA/SGS nor the authors of this report are qualified to provide extensive comment on legal issues associated with the Project. MSA/SGS has relied on a legal opinion by Lorentz Angula Inc., Attorneys, Notaries and Conveyancers, in Windhoek, dated April 14, 2011, that NMI is the legal holder of 95% of the interest in EPL 3400 and ML200, which is valid as of the date of the opinion.

The QPs have reviewed and analyzed data and reports provided by NMI, together with publicly available data, drawing its own conclusions augmented by direct field examination.

The QP who prepared this report relied on information provided by experts who are not QPs. The QP believes that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designations, and relevant experience on matters relevant to the technical report.

SGS has relied upon Sharon Meyer- SLR (Project Manager) matters pertaining to the Summary of Environmental Impact Assessment Report and special Studies for the Lofdal Mining Project dated March 2021 as disclosed in Section 20.

Johnny Canosa, P. Eng. (SGS) has relied upon Veronique Daigle (Knight Piésold), Daniel Millar and Johann Hager, who completed an independent analysis on the TSF, Process Plant and Mining respectively for the infrastructures summarized in Section 18.

William van Breugel, P. Eng. (SGS) has relied upon NMI who supplied pricing forecast for this PEA and derived from recent market analysis and other published NI 43-101 complaint resource reports on selling prices for REE, as summarized in Section 19.

William van Breugel, P. Eng. (SGS) has relied upon Johann Hager, who completed an independent analysis on the Mining and Scheduling for the data used in the Capital and Operating Expenses estimate, and Economic Analysis as summarized in Section 21.

William van Breugel, P. Eng. (SGS) has relied upon Veronique Daigle, who completed an independent analysis on the Tailings Storage Facility for the data used in the Capital and Operating Expenses estimate as summarized in Sections 21.

William van Breugel, P. Eng. (SGS) has relied upon Daniel Millar and SGS Bateman, who completed an independent analysis on the Process Plant quantities and costs for the data used in the Capital and Operating Expenses estimate, and Economic Analysis as summarized in Sections 21 and 22.

The QPs have assumed, and relied on the fact, that all the information and existing technical documents listed in the References Section 27 of this report are accurate and complete in all material aspects. While the QPs reviewed all the available information presented, we cannot guarantee its accuracy and completeness. The QPs reserve the right, but will not be obligated, to revise the report and conclusions, if additional information becomes known subsequent to the date of this report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Lofdal property comprises Exclusive Prospecting License (EPL) 3400 and Mining License (ML) 200 and is located approximately 25 km northwest of the town of Khorixas in the Kunene Region of northwestern Namibia (Figure 4-1). Khorixas is approximately 325 km in a straight line and 450 km by paved road northwest of the capital Windhoek.



Source: Modified from Nations Online (2012)

Figure 4-1 Location of the Lofdal Property (Red Square NW of Khorixas)

4.2 Property Description

4.2.1 Exclusive Prospecting Licenses in Namibia

Namibia has a well-organised, structured and reasonable framework for mineral tenure. EPLs are governed by the provisions of the Minerals Act, 1992, Part X. The Act is interpreted in the context of the Minerals Policy of Namibia which recognises the contribution of the minerals sector to the economy and expresses support for the sustainable development of the country's mineral resources.

The holder of an EPL is entitled to carry out prospecting operations related to the mineral or group of minerals specified in the license and to remove any mineral or group of minerals for any purpose other than sale or disposal, from any place where it was found or incidentally won in the course of prospecting. EPLs are subject to renewal after the first three years, and subsequently every two years. At the first renewal, the EPL may have to be reduced by 25% and in subsequent renewals by 50%. Any subsequent renewals after the first two renewals are subject to Ministerial discretion, and experience in Namibia is that they are typically granted provided the holder has fulfilled the terms of the existing license in good faith.

4.2.2 ML 200 and EPL 3400 Lofdal

EPL 3400 (Lofdal) originally comprised 99,900 ha and allowed the company the exclusive right to prospect for “base and rare metals” and “precious metals” the former including REE. The history of tenure is given in Table 4-1. The coordinates of the current EPL are given in Table 4-2 and its boundaries are plotted on Figure 4-2.

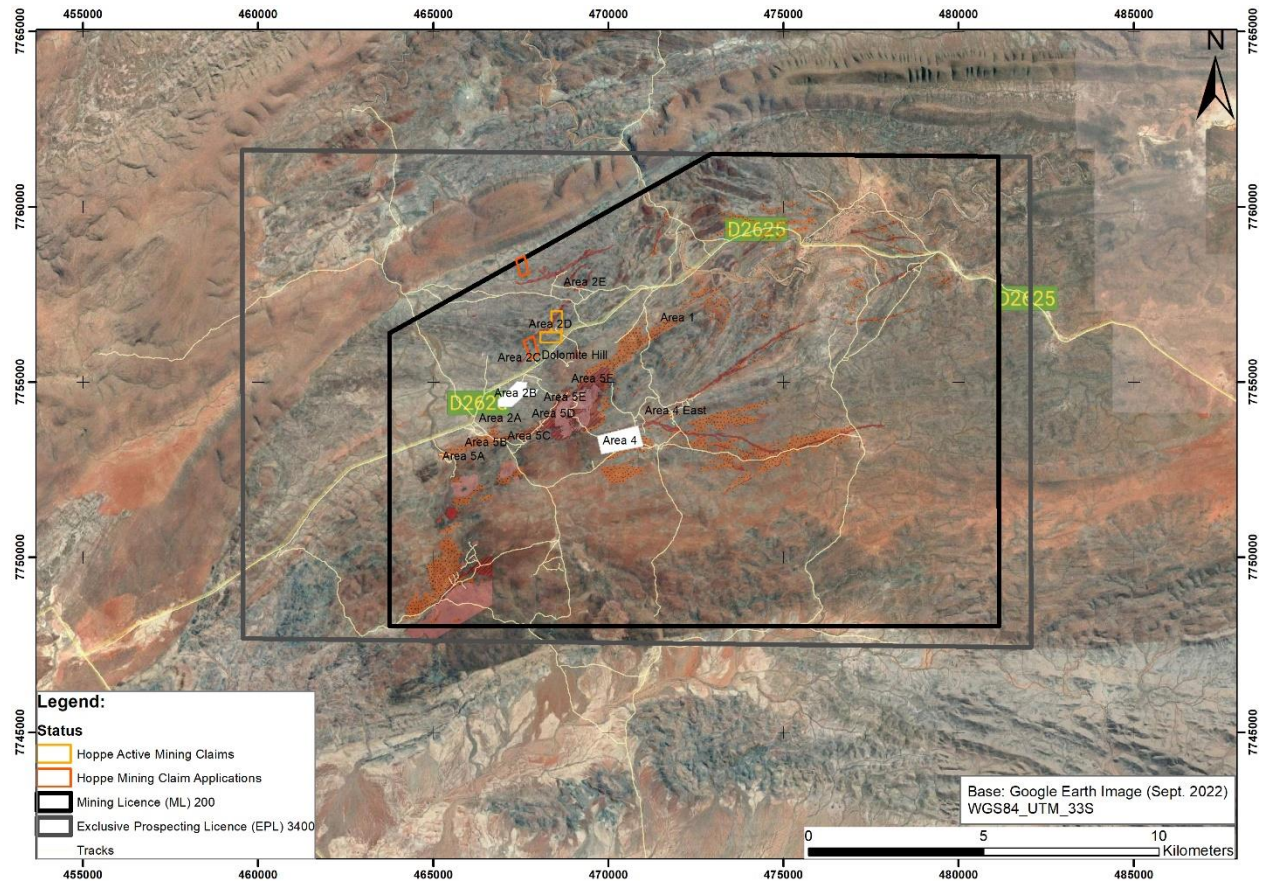
The EPL was most recently renewed by the Department of Mines and Energy (DME) on September 28, 2021 for a two-year period and the renewed EPL is in good standing until September 27, 2023.

Table 4-1 History of Tenure of EPL 3400 (Lofdal)

Renewal Year	Date Applied	Date Granted	Date of Expiry	EPL Size Ha	Proposed Program Cost \$NA
	27-Jun-05	01-Dec-05	14-Nov-2008	99,900.00	\$4,320,000
2008	15-Aug-08	19-Jun-09	14-Nov-2010	99,900.00	\$4,300,000
2010	16-Aug-10	15-Dec-10	14-Nov-2012	74,000.00	\$9,100,000
2012	12-Jul-12	16-Oct-12	14-Nov-2014	57,315.90	\$35,000,000
2014	30-Jul-14	01-Dec-14	16-Nov-2016	57,315.90	\$45,456,100
2016	11-Aug-16	14-Feb-17	16-Nov-2018	42,016.04	\$34,088,000
2018	14-Aug-18	15-May-19	14-May-2021	31,420.99	\$21,133,000
2021	12-Feb-21	28-Sept-21	27-Sept-2023	31,420.99	\$23,273,000

The EPL is for mineral rights only while the surface rights in the area of the Lofdal project are communally held and such land is vested in the State by the Constitution. The State is under a duty to administer communal lands in trust for the benefit of the traditional communities residing on these lands and for the purpose of promoting the economic and social development of the people of Namibia. Communal lands are administered through the office of the Regional Governor and the office of Rural Development.

The Lofdal EPL lies entirely within the Huab Conservancy, one of a network of 86 Namibian Conservancies. Namibian Conservancies are self-governing, democratic entities, run by their members. Communal conservancies are obliged to have wildlife management plans, to conduct annual general meetings and to prepare financial reports. They are managed under committees elected by their members.



Base is 2000 Landsat Geocover Mosaic. UTM WGS84, Zone 33S.

Source: Base map published by NASA. Compiled by NRI.

Figure 4-2 Location of EPL 3400 and ML 200 Showing Current Boundaries, Roads, and Location of the Hoppe Mineral Claims and Mining License Applications

Table 4-2 Coordinates of the Current EPL 3400 as Valid until September 27, 2023

Corner Point	Longitude ^{o*}	Latitude ^{o*}
1 (NW)	14.61261075	-20.24265805
2 (NE)	14.82802805	-20.24473262
3 (SE)	14.82820882	-20.37124294
4 (SW)	14.61262433	-20.36854456
Corner Point	X Coordinate UTM WGS84 Zone 33S (m)	Y Coordinate UTM WGS84 Zone 33S (m)
1 (NW)	459542.6392	7761615.797
2 (NE)	482041.8235	7761424.214
3 (SE)	482075.2341	7747424.250
4 (SW)	459576.7412	7747684.660

Note: *The boundaries of the EPL are established by reference to latitude and longitude coordinates in reference to the Bessel 1841 Spheroid, Central Meridian 17 degrees East

4.2.3 Mining Licence Application

A mining licence application was lodged by Namibia Rare Earths (Pty) Ltd on November 16, 2016. Notice was received on December 22, 2020 that the Minister of Mines and Energy was prepared to grant the application for a mining licence. ML200 was granted on May 11, 2021 for a period of 25 years (expiring on 10 May, 2046) in respect of “Base and Rare Metals of Minerals” subject to certain terms and conditions (Ministry of Mines and Energy, Republic of Namibia, 2020), which are as follows:

“Part 1 - General

- 1. The mining licence shall endure for a period of twenty five years (25) reckoned from the date of acceptance (hereinafter “the date of issue”) of the terms and conditions referred to in this notice unless it is abandoned in terms of section 54 of the Minerals (Prospecting and Mining) Act, 1993 (hereinafter “the Act”) or cancelled in terms of section 55 of the Act or an application made to the Minister in terms of Section 96 of the Act, it is renewed by the Minister for any further period or periods.*
- 2. In consideration of the rights hereby granted, the holder of the mining licence shall pay to the Commissioner for the benefit of the State Revenue Fund, such licence fee as may from time to time be prescribed in terms of section 123 of the Act, it being recorded that the annual licence fee prescribed in relation to the licence at the time of its issue shall be N\$5000.00 payable annually on or before each anniversary date of the date of issue of the licence.*

Part 2 – Work Program and Obligations

- 3. The holder of the licence shall:*

3.1 commence with, and thereafter continue without undue interruption or delay, mining operations within one month of the date of issue of the licence in substantial conformity with the proposed work program, schedule and budget which accompanied the original application for the licence and which served as motivation of the granting thereof;

3.2 where any material deviation of such work program, schedule and budget is in the opinion of the holder of the licence, necessitated by the nature of the results of mining operations (but specifically excluding any circumstances of Vis Major provided for in terms of section 56 of the Act), apply in writing to the Minister for approval of the revision of such work program, schedule and budget in terms of section 99 of the Act; and

3.3 execute such additional work program and expend such additional expenditure within a specified period of time as may be imposed by the Minister from time to time.

3.4 The Minister may, in the interest of the reasonable development of the mining operations, impose from time to time such additional terms and conditions as may seem fit.

3.5A all funds raised anywhere in respect of this licence shall be committed to this licence and shall be banked at a Financial Institution in Namibia.

Part 3 – Environment

4. The holder of the mining licence shall observe any requirements, limitations or prohibitions on his or her prospecting operations as may in the interest of the environmental protection, be imposed by the Minister.
5. The holder of the Exclusive Prospecting Licence shall adhere to the terms and conditions upon which the Environmental Clearance Certificate was issued by the Ministry of Forestry Environment and Tourism.

Part 5 – Additional Conditions

6. Within 30 days of the grant of a new Mining Licence, the applicant shall submit to the Minister a declaration signed by a duly authorised director of the applicant to the effect and including:

6.1 Proof that there is a minimum 20% representation of historically disadvantaged Namibians in the management structure (including the board) of the applicant; and

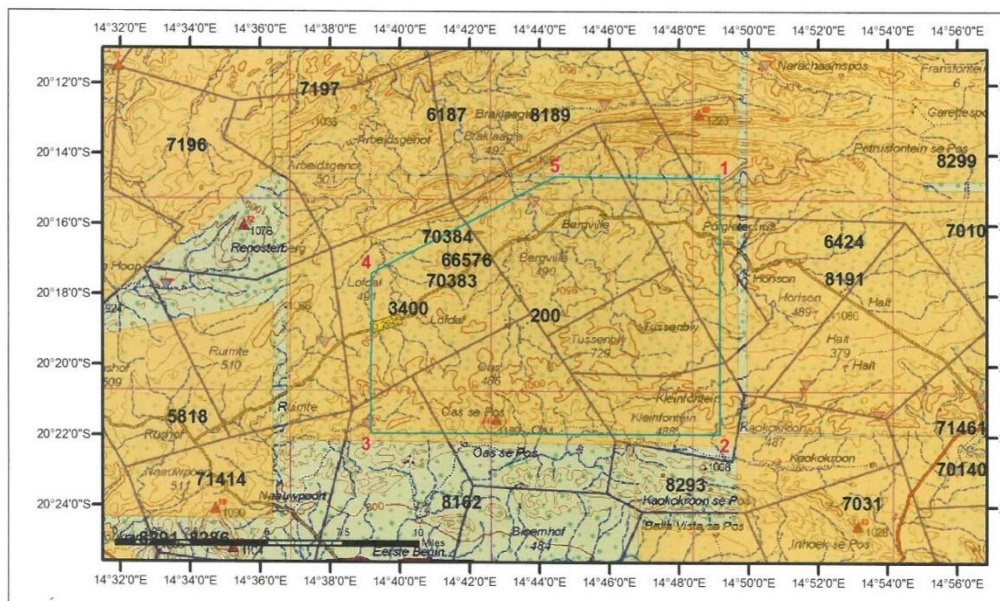
6.2 Proof that at least 5% (five percent) of the principal voting shares in the applicant or at least 5% (five percent) of the holding of the Mining Licence, as the case may be, is held by historically disadvantaged Namibians. For the purposes of this condition, the term “held” includes a holding of such principal voting shares directly or indirectly through a company, close corporation, trust, traditional authority, or other similar association, and includes ownership by entities representing Government or in which Government holds a meaningful stake.

6.3 The applicant's strategy for addressing the Government's objective of poverty eradication, including benefitting the Namibian youth and women from disadvantaged groups and the poorest of the poor.

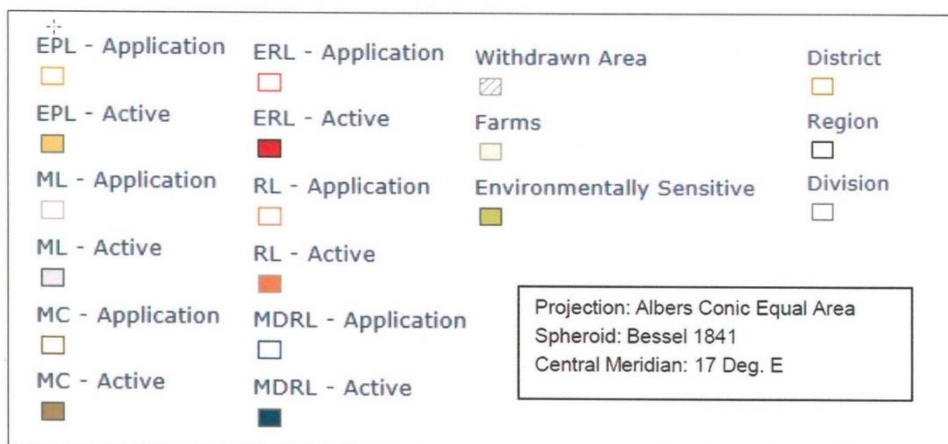
7. If the applicant has been misleading in relation to declarations made under condition 13, the Minister may cancel the licence under section 55(1)(a) of the Act and the remaining provisions of section 55 will apply.
8. For the purposes of these conditions, the term “historically disadvantaged Namibians” shall mean Namibian citizens falling within the category of designated groups as defined in the (Affirmative Action (Employment) Act, 1998).”

The Mining Licence area is shown in Figure 4-2 and Figure 4-3, and the coordinates of the boundary corners are given in Table 4-3.

Issued in favour of: Namibia Rare Earths (Pty) Ltd



Latitude and Longitude lines refer to the Bessel 1841 Spheroid



AREA: 21034.6274 Hectares

Source: Ministry of Mines and Energy, Republic of Namibia, 2020

Figure 4-3 Location of Mining Licence 200 (blue line). Corner Numbers in Red are same as Table 4-3.

Table 4-3 Coordinates of Mining Licence 200

Corner Point	Longitude (Degrees, Minutes, Seconds)	Latitude (Degrees, Minutes, Seconds)
1 (NE)	14 49 9.99 E	-20 14 40.83 S
2 (SE)	14 49 9.92 E	-20 21 56.49 S
3 (SW)	14 39 10.06 E	-20 21 55.59 S
4 (NW)	14 39 10.68 E	-20 17 23.87 S
5. (top)	14 44 26.94 E	-20 14 38.08 S

Note: The boundaries of the EPL are established by reference to latitude and longitude coordinates in reference to the Bessel 1841 Spheroid, Central Meridian 17 degrees East

Source: Ministry of Mines and Energy, Republic of Namibia, 2020

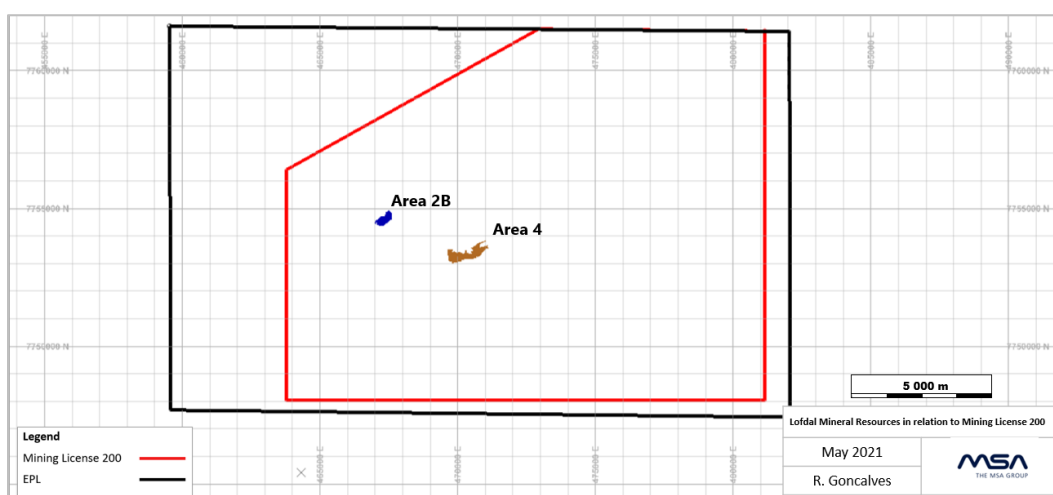


Figure 4-4 Extent of Mining Licence 200 in Relation to the Mineral Resources of Area 4 and Area 2B

MSA has examined the documentation regarding Mining Licence 200 as supplied by NMI for review. Although MSA is not qualified to provide legal opinion, it has no reason to doubt the authenticity of the information provided.

4.2.4 General Provisions

NMI is in receipt of a legal opinion by Lorentz Angula Inc., Attorneys, Notaries and Conveyancers, in Windhoek, dated April 14, 2011, that NMI is the legal holder of 100% of the interest in EPL 3400, which is valid as of the date of the opinion. There have been no changes to legislation or policy in the intervening time that would give reason to question the continuing validity of this opinion.

Under the Minerals Act, 1992, and as declared in Government Gazette 45 of 2009, REE are subject to a royalty of three percent of the fair market value of minerals produced in Namibia. The property is also subject to a two percent Net Smelter Royalty (NSR) to Alberto Lobo-Guerrero Sanz, who introduced NMI to the project.

Neither the applications by NMI to acquire or renew the EPL, nor the environmental contract that was agreed to by NMI and the Government of Namibia (Environmental Contract), identify any pre-existing environmental liabilities on the property and none are known to exist.

Under the provisions of the Environmental Contract, NMI is required to submit bi-annual environmental reports detailing work and potential impacts. NMI has fully complied with this provision and copies of these reports are filed in company files which are complete and up to date. The most recent application for renewal of the Environmental Clearance Certificate for EPL 3400 was submitted to the Environmental Commissioner on July 1, 2022 and awaits granting.

Notifications of trenching and drilling programs are required to be filed with the Mining Commissioner, Department of Mines and Energy. Notification of drilling for all holes in the 2020 drilling campaign were filed by forms dated February 26, March 13, July 8, August 12, and September 24, 2020. The authors are not aware of any other permits that are required to conduct the planned work.

4.2.5 Adjacent and Overlapping EPLs

The Lofdal Carbonatite Complex is entirely contained within EPL 3400. As far as is known, there are no similar intrusions or potential for similar mineralization outside the EPL and there is no active exploration for similar targets on nearby EPLs.

The area of the former Lofdal copper mine is held under mining claims by a Mr. Hoppe. These claims predate the EPL and take precedence over it and are indicated by the orange rectangles polygon in Figure 4-2. The claims expired on August 27, 2019 but are still active pending renewal or expiry.

Mr. Hoppe has also applied for two mining licenses overlapping the northern part of EPL 3400 totalling approximately 28,4 Ha (red rectangles on Figure 4-2). As at the effective date of this report, these applications had not been granted.

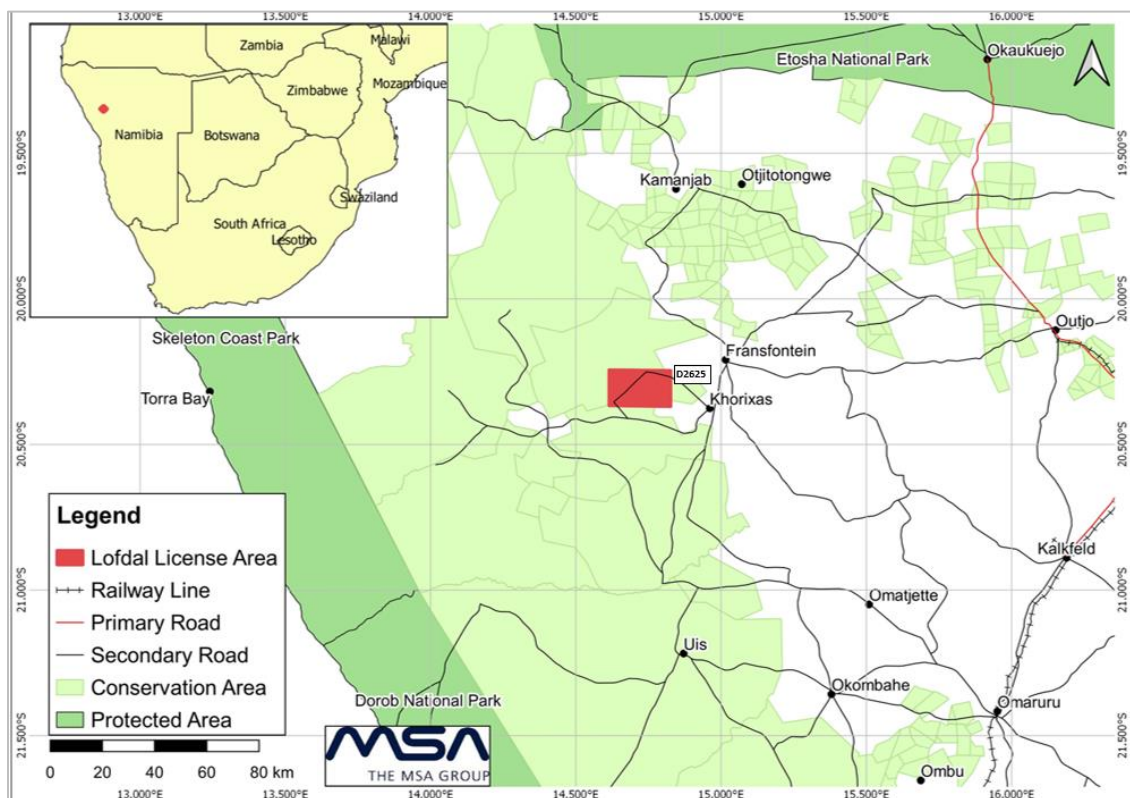
There are no other factors or risks known to the authors that might affect NMI's right or ability to perform work on EPL 3400.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The town of Khorixas is connected to the capital city of Windhoek by approximately 450 km of paved road via Otjiwarongo and Outjo. Windhoek is the country's commercial and administrative centre and has international and regional airports with scheduled services to regional centres in southern Africa and Europe. Driving time from Windhoek to Khorixas is approximately 4.5 hours.

From Khorixas, the Lofdal project area can be reached via 25 km of secondary all weather gravel road (national road D2625; Figure 5-1). Bush tracks provide good access to most parts of the project area and are generally negotiable by two-wheel drive (2WD) vehicles, although four-wheel drive (4WD) is occasionally required to cross gullies or wet areas during the rainy season.



Note: Coordinate system latitude and longitude coordinates in reference to the Bessel 1841 Spheroid, Central Meridian 17 degrees East
Source: MSA, (2021)

Figure 5-1 Location and Road Access to the Lofdal Project Area

5.2 Climate

North-western Namibia is an arid to semi-arid region. Rainfall is largely confined to the summer months (November to April) and averages 150 mm to 200 mm per annum. Average daytime high temperatures range from less than 25°C in June/July to more than 35°C from October to April and locally exceed 40°C during hot spells. Night-time lows reach 5°C in winter rising to about 20°C in summer. Sunshine averages more than 11 hours per day during winter and eight to nine hours during the summer rainy season when there is frequent cloud cover. The climate is conducive to year-round operations.

The property lies within the catchment area of the Huab River and has little perennial surface water. The hydrological map of Namibia (van Wyck et al., undated) indicates that the project area is characterised by moderate to low water availability in the bedrock. Information from water boreholes in the area suggests that the water table is about 25 m below surface and comprises mainly fracture permeability in the crystalline basement rocks. Experience to date indicates that wells can supply sufficient water for the needs of exploration without compromising the requirements of local communities.

5.3 Local Resources and Infrastructure

Khorixas is a regional administrative centre. Local services include two fuel stations, hardware and general stores, a small supermarket and several convenience stores, a bank and facilities for basic vehicle servicing, welding and other trades. There is a small dirt airstrip but no scheduled air services. There are three tourist lodges in and near Khorixas offering accommodation, camping, and restaurants. NMI rents a 330 m² warehouse and a 7,600 m² surrounding yard supplied with municipal water and electricity that serves for equipment storage, local office, and drillhole core processing and storage (Figure 5-2). Core is processed at covered core logging areas at the tented field camp nearer the project area (Figure 5-3).



Source: Swinden (2012) – top left, Witley (2020) – top right and bottom

Figure 5-2 Facilities in Khorixas for Equipment Storage, Core Logging and Storage



Source: Witley (2020)

Figure 5-3 Core Processing Facilities at the Lofdal Field Camp

The nearest large centres are Outjo, a town with a population of more than 6,000 that lies 120 km east of Khorixas; and Otjiwarongo, a town of approximately 20,000 people that lies 200 km east of Khorixas. Otjiwarongo is a regional commercial and service centre for, among other things, the former Okoruso fluorspar and Okanjande graphite mines.

There are local farms in and around the project area that raise cattle, sheep, goats, donkeys and horses. These farms have wells that can supply adequate water for exploration needs and together with Khorixas, provide a stable pool of workers that can be tapped for exploration requirements.

Khorixas is connected to Namibia's land telecommunications grid and has a Telecom Namibia office. Cellular telephone services are provided by MTC and internet data services are readily available through MTC as well.

Khorixas is connected to the national power grid via a 132 kV transmission line that runs to the east of Khorixas and northwards to the town of Kamanjab.

5.4 Physiography

The project area is characterised by low, gently rolling and sparsely vegetated hills with peaks ranging up to an altitude of approximately 1,030 m (Figure 5-4). There is an overall relief throughout the project area of slightly more than 100 m.



Source: Swinden (2012)

Figure 5-4 Physiography of the Project Area showing Typical Low Rolling Hills and Sparse Vegetation

There is little soil, with much of the ground covered by residual gravel that closely reflects the composition of the underlying bedrock. This residuum is typically less than one metre thick on the high ground but thickens in the dry valleys. Outcrop is widespread throughout the area.

Vegetative cover includes a ubiquitous cover of native grass after the rainy season and numerous arid-adapted low shrubs. Wildlife is relatively sparse but includes springbok, kudu and gemsbok as well as baboons, elephants, zebras, leopards and various small mammals, lizards and snakes.

6 HISTORY

The definitive geological mapping of the area was carried out by Frets (1969), who set out the general stratigraphic and tectonic framework. He identified and described the basement metamorphic rocks and the overlying sedimentary and volcanic rocks of the Damara Orogen and documented the alkaline intrusive plugs in the Lofdal area. Frets (1969) recognized and described the Oas quartz syenite and the silica-undersaturated syenitic rocks of the Lofdal complex. However, he did not recognize the associated carbonatites.

Other published accounts of carbonatites in the Lofdal area are found in Diehl (1990, 1992), Verwoerd (1993), and Woolley (2001) and accounts of the geological setting of carbonatites in the Lofdal area in Niku-Paavola et al. (2001) and Wall et al. (2008), which were summarised in Swinden and Siegfried (2011).

The current published geological map of the area is the 1:250,000 Fransfontein Sheet compiled in 2006 by the Geological Survey of Namibia (GSN) but it is too large a scale to be a useful base map at the detailed map scales needed for mineral exploration.

Historically, mineral exploration activities in the area have focused on copper, gold and tantalite associated with quartz veins and/or pegmatites hosted in the metasedimentary and metavolcanic gneisses of the Huab Complex. The copper and gold mineralization was generally interpreted to be related to faults and shears (GSN, 1992).

Small scale mining by way of shallow adits is evident in at least two locations within the EPL immediately north of the project area (Figure 6-1). The adits were opened in the 1950's and one is reported to have yielded ore grading ten percent copper (GSN, 1992).



Source: Swinden (2012)

Figure 6-1 Adit at the Former Lofdal Copper Mine with Copper Staining Around the Portal

Exploration for copper and gold in this area was conducted by Messina (Tvl) Development Co. Ltd. From 1974 to 1976 (Davidson, 1977). The GSN (1992) reported that diamond drilling in 1974 intersected cupriferous silicified zones, the best of which assayed 1.02% and 1.51% Cu over 1.95 m and 1.2 m respectively. Davidson (1977) interpreted the deposit to possibly represent the root zone of a largely eroded deposit. Tsumeb Corporation Ltd explored for base and precious metals between 1981 and 1986.

The area was prospected for gold by Anglo American Prospecting Services Namibia (Pty) Ltd. between 1987 and 1989. Reconnaissance bulk stream sediment samples were processed and analysed for gold content. Twenty-seven anomalous areas were selected for follow-up by detailed stream sediment sampling, soil sampling, rock sampling and geological mapping. This work did not yield any significant concentrations of gold and the project was terminated in 1989 (Marsh et. Al., 1989). The presence of copper mineralization, coupled with the presence of REE-bearing carbonatite dykes, abundant iron oxide mineralization, and magnetite-cemented diatreme breccias, led Lobo-Guerrero (2005) to suggest a potential for iron oxide copper gold (IOCG) type deposits in the area, which was influential in attracting the interest of NMI to the area. NMI explored for copper and gold in the area from 2005 to 2007 but did not delineate any IOCG targets from its regional exploration work. In 2008, NMI switched its focus to the potential for REE mineralization associated with the carbonatites.

Although not extensively described in the literature, carbonatite dykes have been reported in the area of Lofdal and Bergville farms at least since the early 1980's and were the focus of an exploration program for yttrium (Y) and rare earth elements (REE) by Rouna (Pty) Ltd. (Rouna) between 1981 and 1983. Following a reconnaissance radiometric survey and some rock sampling in 1981 and 1982 (Figure 6-2), attention was focused on anomalous responses in the area of farms Lofdal 491 and Bergville 491. The preliminary work by Rouna identified the presence of yttrium hosted in the mineral xenotime (YPO_4) and demonstrated that radiometrics is an effective prospecting tool because of an association of rare earth elements with thorium. Detailed sampling in 1983 yielded ThO_2 values ranging from 0.17% to 14.4% and yttrium from 207 ppm to 6,300 ppm with one analysis of 1.01% Y. One carbonatite dyke sample yielded anomalous contents of rare earths i.e., 0.82% Ce, 1.5% La and 0.74% Nd (Barbour, 1982). There are no other analyses of REE available from this phase of exploration.



Source: Swinden (2012)

Figure 6-2 Pit Sampling of a Carbonatite Dyke from Rouna's Exploration

More recently, the Namibia Small Miners Assistance Centre held EPL 2821 over portions of Lofdal and Bergville farms for precious stones, semi-precious stones, precious metals, and base and rare metals from 2002 to 2004.

Mr. P. Siegfried investigated the greater Lofdal area in 1999 for Norsk Hydro ASA and in 2001 extensive sampling of the carbonatite dykes was carried out together with Dr. T. Mariano for the Canadian REE company Advanced Metals Research (AMR). REE mineralization and highly anomalous HREE were identified.

Geological investigations in the area by the GSN have been ongoing since V. Niku- Paavola began a Ph.D. research project on the carbonatites in the Lofdal area in 2004 at the Camborne School of Mines in the United Kingdom. This project was completed in 2014 (Do Cabo, 2014). Dr. R. Ellmies, previously of the GSN, has maintained an active research interest in this area and facilitated research particularly by students at the University of Namibia. Several B.Sc. theses describing aspects of the carbonatites and rare earth mineralization have been facilitated by the GSN (Ndalulilwa, 2009; Mutilifa, 2010; Shikongo, 2010). GSN's work in this area has provided significant new information on the unusual HREE enrichment at Lofdal and has yielded much detailed information about the mineralogy of the carbonatites and the related rare earth mineralization (Niku-Paavola et al., 2001; Wall et al., 2008; do Cabo and Ellmies, 2010).

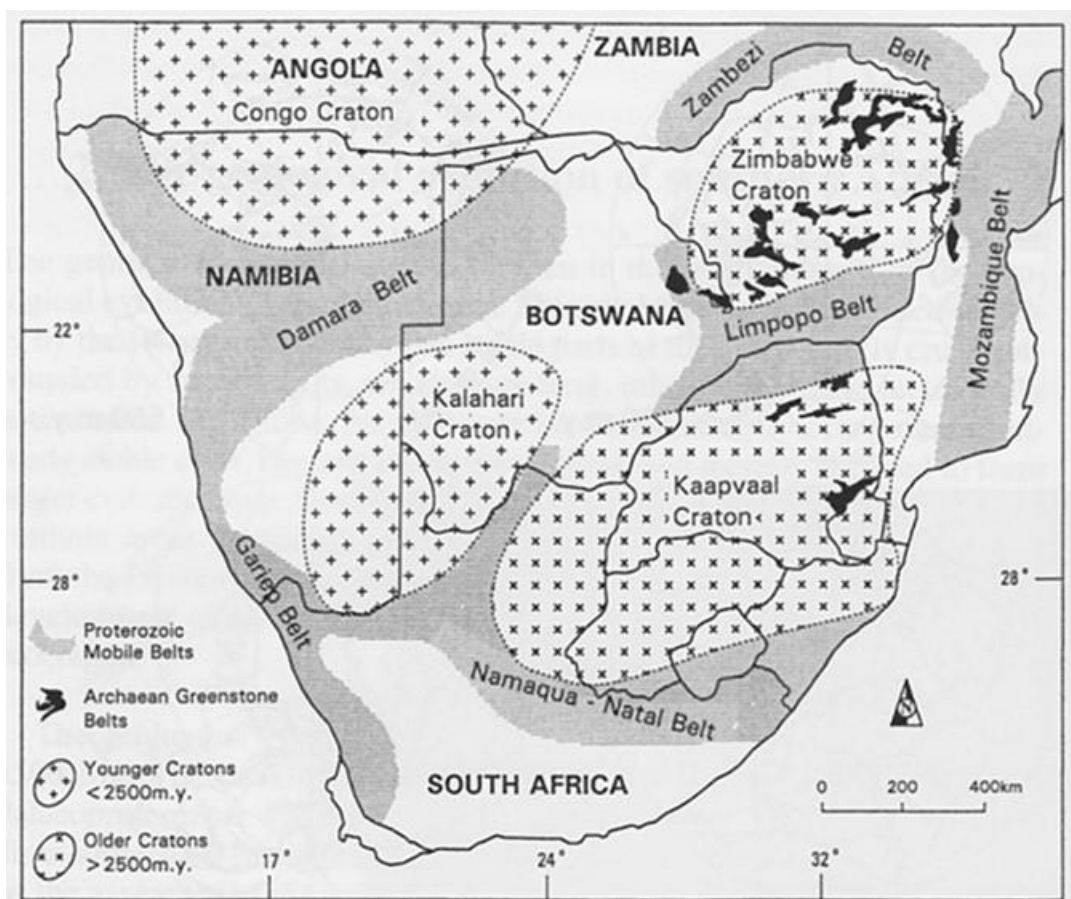
In addition to the work at the University of Namibia, NMI has sponsored student work at a number of other universities. These include four B.Sc. (Honours) theses, respectively at Acadia University (Canada) (Kaul 2010), Dalhousie University (Canada) (O'Connor, 2011; Gaudet, 2012) and Stellenbosch University South Africa) (Kruger, 2012). Two M.Sc. theses have been completed through the Camborne School of Mines

(UK) (Loye, 2012, 2014). Two M.Sc. theses have been carried out on the petrology of the carbonatite intrusions, one at McGill University (Canada) (Bodeving, 2015) and a second at The University of St. Andrews (UK) (Robinson, 2020). A PhD study of the mineralized alteration zones and dykes is in progress at McGill University and preliminary results have been reported by Wollenberg et. al. (2016).

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The regional bedrock geology of north-western Namibia is defined by Archaean to Palaeoproterozoic cratons to the north and south, the Congo and Kalahari cratons respectively, separated by a Neoproterozoic orogenic belt of Pan African affinity termed the Damara Fold Belt or Damara Orogen (Figure 7-1).



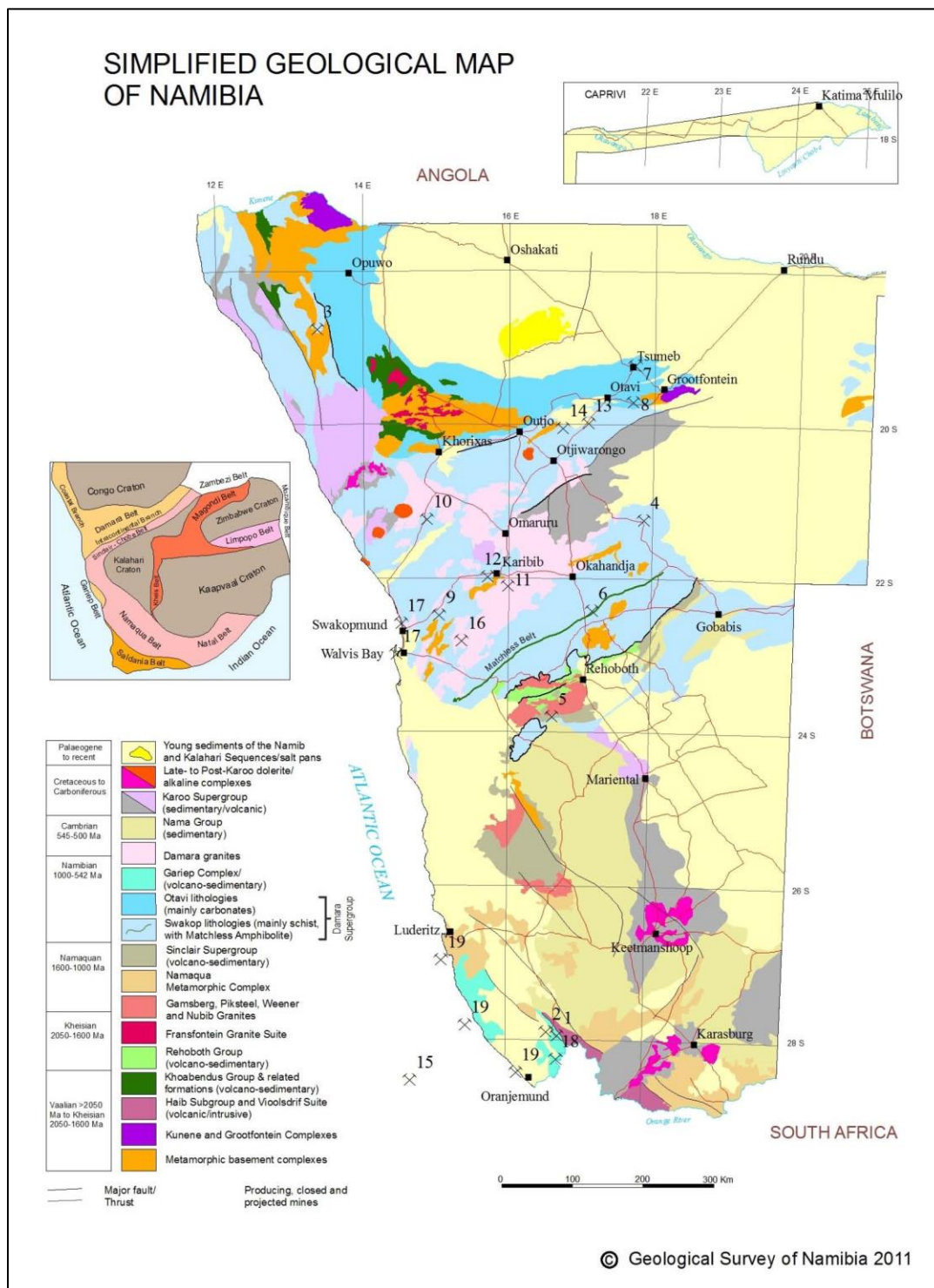
Note: Coordinate system is latitude and longitude in reference to the Bessel 1841 Spheroid, Central Meridian 17 degrees East

Source: Schneider, 2008

Figure 7-1 Cratons and Orogenic Belts in Southern Africa

The southern edge of the Congo Craton is exposed in three inliers in the Khorixas – Kamanjab area, termed respectively the Kamanjab, Braklaagte and Welwitschia inliers (Figure 7-2), separated from each other by belts of younger volcanic and sedimentary rocks of the Damara Orogen.

The basement rocks in the Welwitschia Inlier were intruded post-tectonically by the Oas Syenite and the related Lofdal Carbonatite Complex, comprising syenite, nepheline syenite, phonolite and carbonatite.

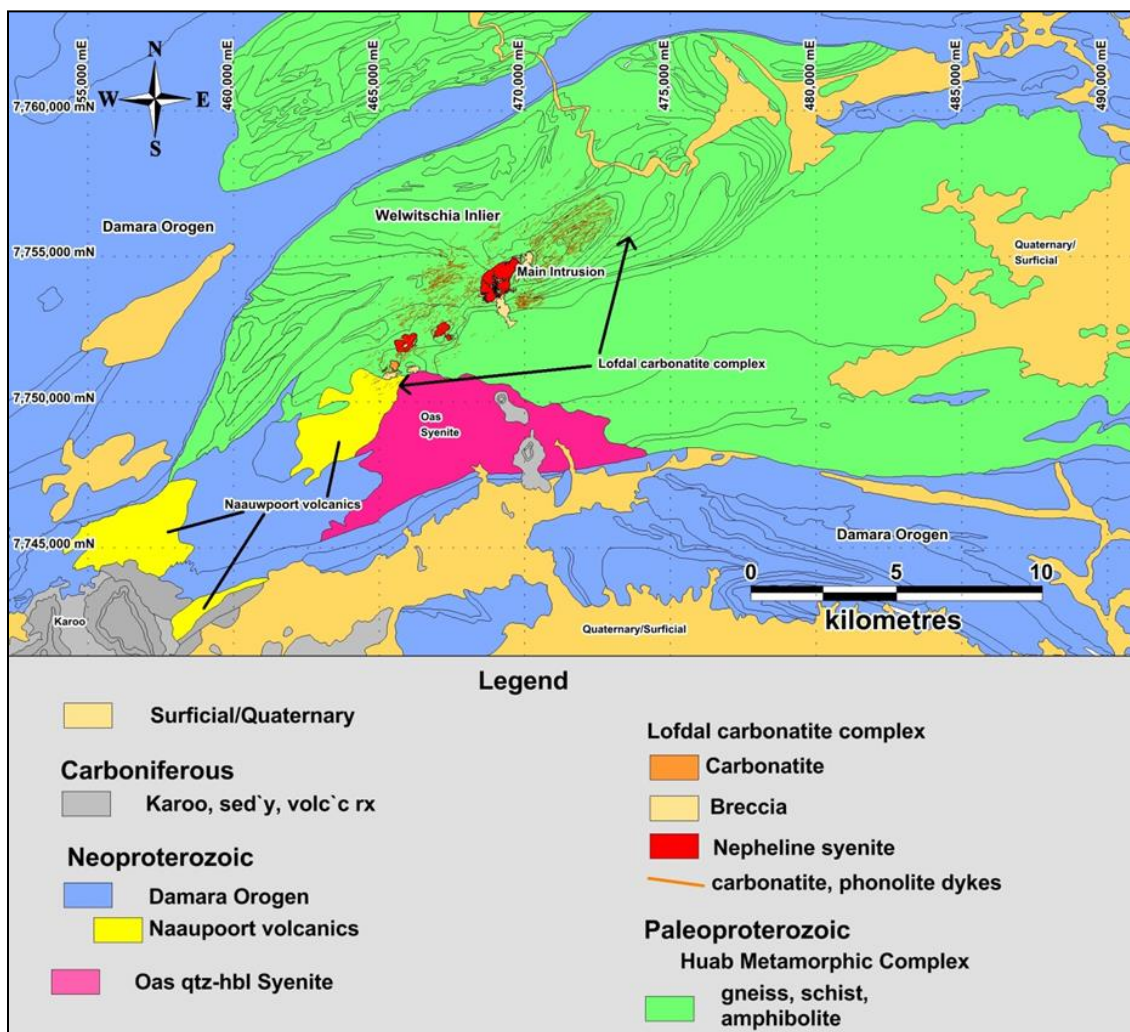


Note: Huab Metamorphic Complex Inliers (brown) are labelled: K : Kamanjab; B : Braklaage; W : Welwitschia. Coordinate system is latitude and longitude in reference to the Bessel 1841 Spheroid, Central Meridian 17 degrees East
Source: (GSN, 2002)

Figure 7-2 General Geology of Namibia

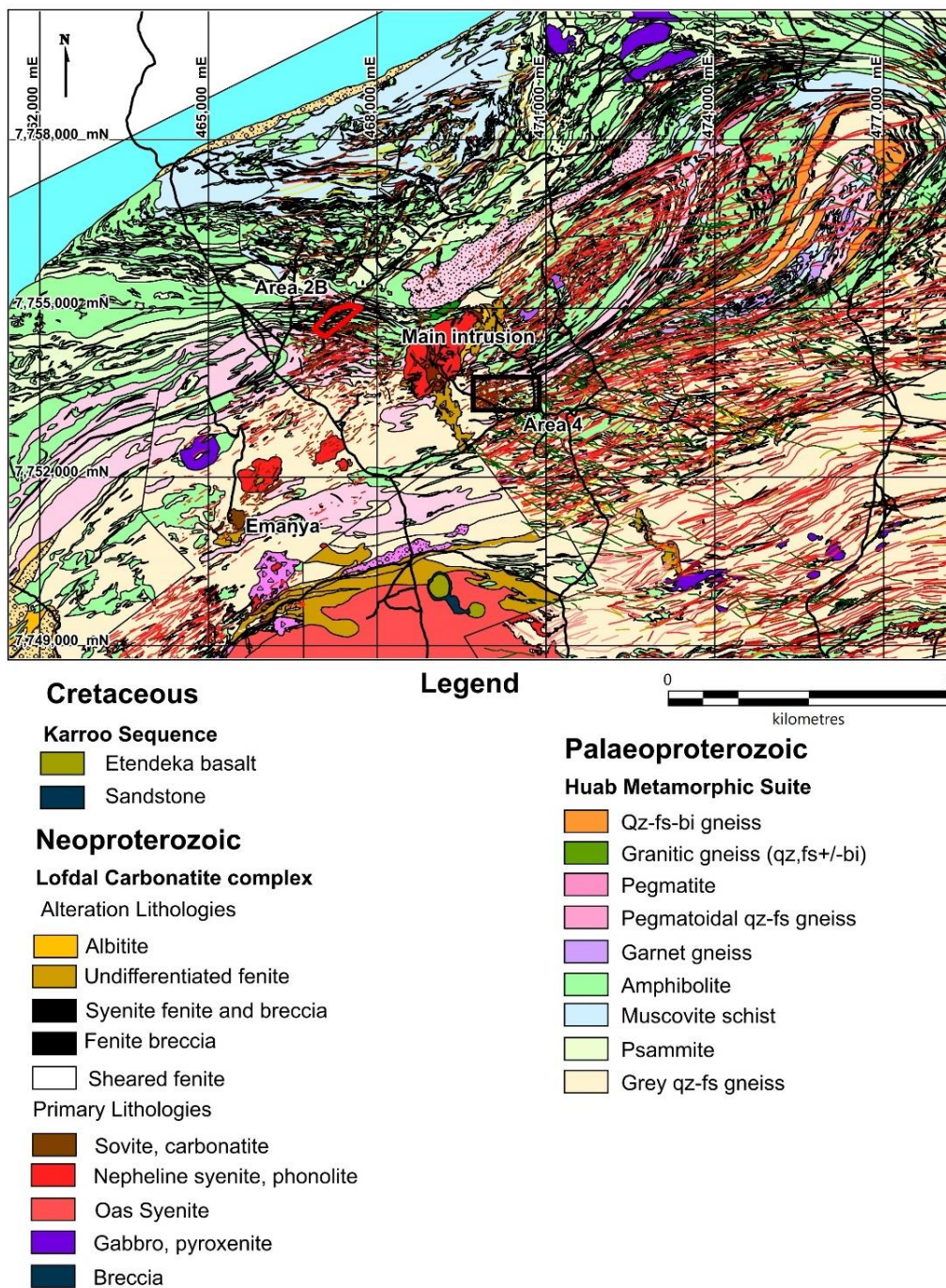
7.2 Local Geology

The general geology of the Welwitschia Inlier, which hosts the Lofdal Carbonatite Complex, is shown in Figure 7-3. A more detailed geology map of the Lofdal area from recent geological mapping is shown in Figure 7-4. The principal geological units are briefly described below.



Source: (GSN, 2008)

Figure 7-3 General Geology in the Area of the Welwitschia Inlier



Note: Detailed geology of Area 4 (black polygon) is shown in Figure 7-14 and detailed geology of Area 2B (red polygon) is shown in Figure 7-16.

Source: Swinden, 2014

Figure 7-4 Detailed Geology of the Area of the Lofdal Carbonatite Complex

7.2.1 The Huab Metamorphic Complex

The oldest rocks in the inlier are leucocratic granitic gneiss, banded paragneiss and quartzite, amphibolite, and mica/chlorite schist assigned to the Huab Metamorphic Complex (Frets, 1969) (Figure 7-5). Locally, mafic sills, dykes and stocks cut the sequence. The Huab Metamorphic Complex is polydeformed, affected by at least one phase of high-temperature isoclinal folding and is locally migmatized. The Huab Metamorphic Complex has not been directly dated but is considered to be about 2.0 billion years (Ga) old as it is intruded post-tectonically by the Fransfontein Granite, which has been imprecisely dated by U/Pb with two discordia lines giving ages of 1871 ± 30 million years (Ma) and 1730 ± 30 Ma (Burger et al., 1976).



Looking north across the Huab Metamorphic Complex in the Huab Welwitschia Inlier. High ground in distance is underlain by Damara Orogen sedimentary rocks.



Banded grey leucocratic granitic gneiss of the Huab Metamorphic Complex.

Source: Swinden, 2012

Figure 7-5 General Topography and Outcrop Appearance of the Huab Metamorphic Complex

7.2.2 The Damara Orogen

The Welwitschia Inlier is overlain to the north and south by sedimentary and volcanic rocks of the Damara Orogen. To the north it is in fault contact with clastic sedimentary rocks of the Mulden Group. To the south it is unconformably and/or structurally overlain by volcanic and sedimentary rocks of the Nosib and Swakop groups.

The Damara sequences represent a Pan African orogenic belt between the Congo and Kalahari Cratons. The basal successions (Nosib Group) comprise quartzite, arkose, conglomerate and subordinate calc-silicate and limestone that were laid down in, or marginal to, the intra-continental rifts. Locally, alkaline ignimbrite and associated subvolcanic intrusions are present (Naauwpoort Formation) the basal part of which has been dated by U-Pb single zircon as 752 ± 7 Ma (de Kock et al., 2000). Early rift sedimentation was followed by widespread carbonate shelf and slope deposition, which grades laterally into deep water clastic sediments with local accumulations of within-plate basic volcanic rocks (i.e., the Swakop Group). Subsequent subduction and continental collision resulted in widespread deposition of molasse (Mulden Group).

7.2.3 Early Damaran Alkaline / Carbonatitic Intrusions

At about 760 Ma, more or less contemporaneously with eruption of the early Damaran alkalic Naauwpoort Formation volcanic rocks, a suite of alkali silicate rocks and carbonatites were emplaced in the Huab Metamorphic Complex.

7.2.3.1 The Oas Syenite

The largest of these bodies is the Oas Syenite, first described by Frets (1969). It underlies approximately 20 km² immediately south of the Lofdal project area, and comprises a dominantly coarse grained, alkali feldspar, sodium plagioclase, hornblende and quartz syenite (Figure 7-6). Apatite and sphene are important accessory minerals (Frets, 1969).

The Oas Syenite intrudes the basement gneisses and the basal sedimentary rocks of the Naauwpoort Formation (Frets, 1969) but is apparently overlain by Damaran limestones from higher in the sequence. The Oas Syenite has been dated by U/Pb in zircon as 756 ± 2 Ma (Hoffman et al., 1996) and by U/Pb in titanite as 758 ± 4 Ma (Jung et al., 2007) and is therefore approximately coeval with the Naauwpoort Formation volcanic rocks.



Source: Swinden, 2012

Figure 7-6 Coarse Grained Oas Syenite (Alkali Feldspar, Amphibole and Mica)

7.2.3.2 The Lofdal Carbonatite Complex

Frets (1969) mapped a body of nepheline syenite in the southern part of farm Lofdal 491, intruding the gneiss complex. He noted a number of smaller satellite plugs, consisting dominantly of medium to coarse-grained leucocratic nepheline syenite as well as the prevalence of calcite and siderite in cracks and marginal facies of the intrusive.

Recent work has shown that the intrusive complex at Lofdal is more complicated than envisaged by Frets (1969) and comprises an assemblage of nepheline syenite and carbonatite intrusive plugs, dykes and hydrothermal alteration with related phonolite dykes and breccias defining an intrusive complex that appears to underlie an area of more than 200 km².

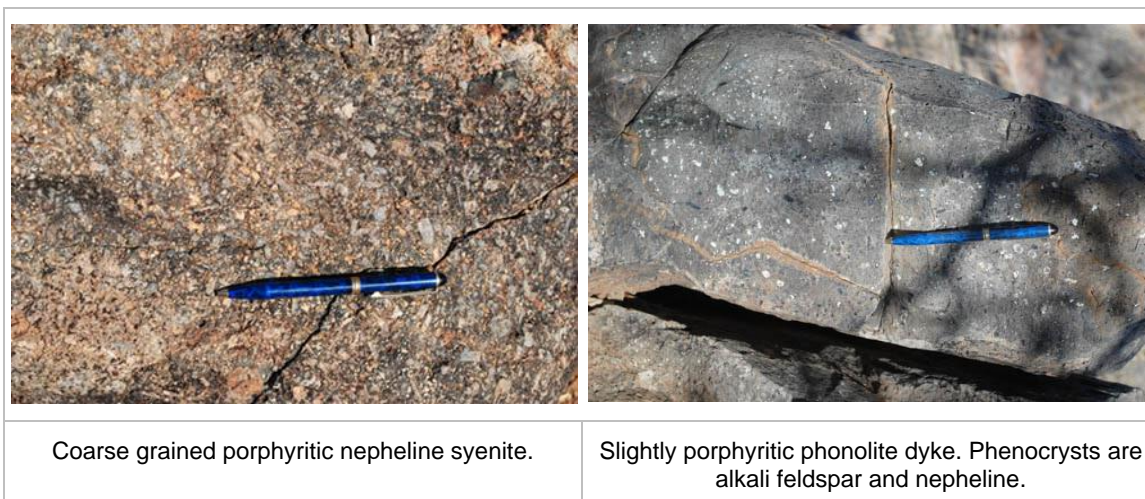
The regional setting of the Lofdal Carbonatite Complex is shown in Figure 7-3. The most important primary lithologies are nepheline syenite, phonolite, breccias and carbonatites as described below.

Nepheline Syenite

Nepheline syenites in the Lofdal area are medium to coarse grained, locally porphyritic syenites dominated by alkali feldspar, nepheline, sericite and biotite (O'Connor, 2011) (Figure 7-7). The original syenite intrusion mapped by Frets (1969) is, in fact, composite, comprising a carapace of syenite, which is intruded from below by carbonatite (the Main intrusion, see below). This results in a surface map pattern dominated by syenite. Syenite occurs in a number of satellite intrusions where it displays a wide variety of textures including very coarse-grained pegmatitic syenite phases. The syenite intrusions are locally cut by phonolite and carbonatite dykes, and fragments of the syenite are incorporated in the Lofdal breccias. The syenites are typically undeformed but locally exhibit mild shearing, characterised by development of a cleavage and alignment of feldspar phenocrysts. The Lofdal nepheline syenite has been dated by U/Pb in magmatic titanite as 754 ± 8 Ma (Jung et al., 2007) and it is therefore coeval with both the Oas Syenite and the Naauwpoort Formation volcanic rocks.

Phonolite

Phonolite (nepheline, alkali feldspar) dykes are widespread in the Lofdal area. They are dominantly northeast striking with fine-grained to moderately porphyritic with locally a trachytic texture (Figure 7-7). Where phenocrysts are present, they are dominantly alkali feldspar lathes up to two mm long with lesser nepheline. Phonolite dykes have been observed to cut syenite and breccia and they are typically closely associated with carbonatite dykes throughout the area. A comparative petrographic and mineralogical study of the syenites and phonolites led O'Connor (2011) to conclude that they are likely co-magmatic.



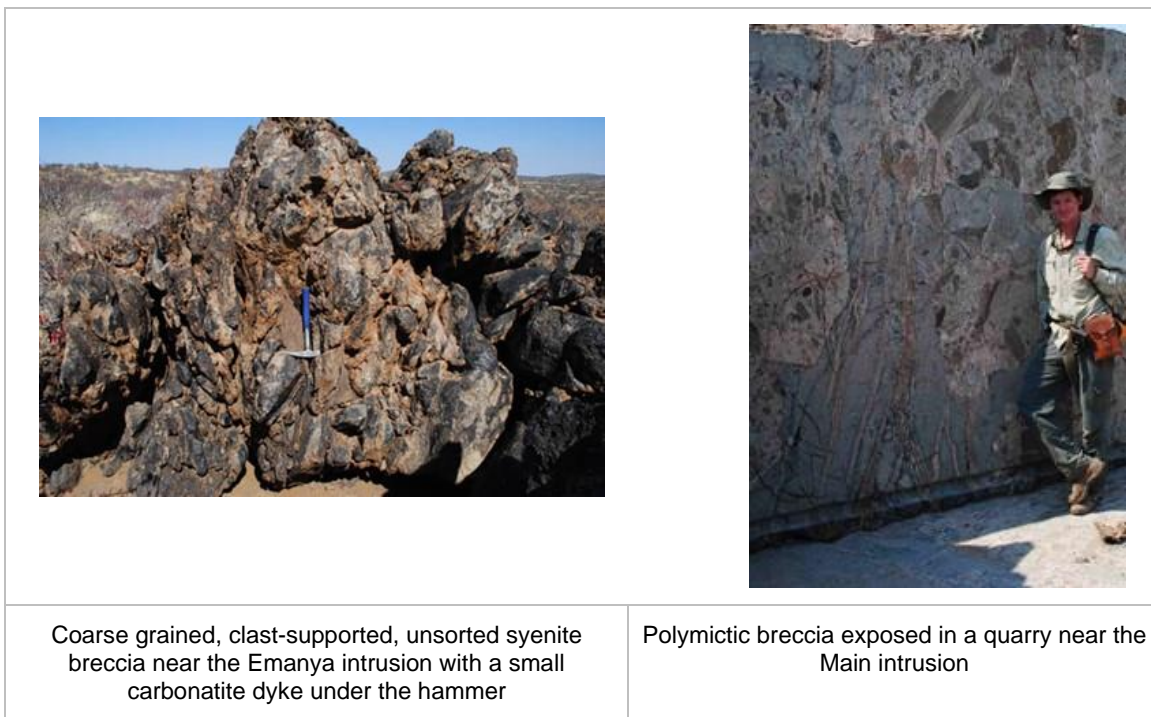
Source: Swinden, 2012

Figure 7-7 Examples of Nepheline Syenite and Phonolite Dyke

Lofdal Breccias

Frets (1969) first noted the presence of a very coarse breccia associated with the nepheline syenites, which he described as “closely packed angular fragments of gneiss which vary in size between one cm and 50 cm, embedded in a fine-grained, contaminated facies of the syenite”. He suggested that it indicated a forceful intrusion of the syenite.

Geological mapping in the area has demonstrated that these breccias are widespread and are associated with virtually all the syenite intrusions identified. Although locally dominated by country rock fragments as described by Frets (1969), in other areas they are dominated by syenite fragments. The breccias range from poly lithic (basement clasts) to monolithic (syenite clasts) and are typically unsorted, angular, and chaotic (Figure 7-8). Locally, the breccias are intruded by carbonatite and phonolite. They clearly post-date the intrusion of the syenite plugs, as the syenites form fragments in them, but they must be closely related in time as phonolite dykes are locally observed to cut the breccias. There are no carbonatite fragments in the breccias and they apparently predate the carbonatite intrusion.

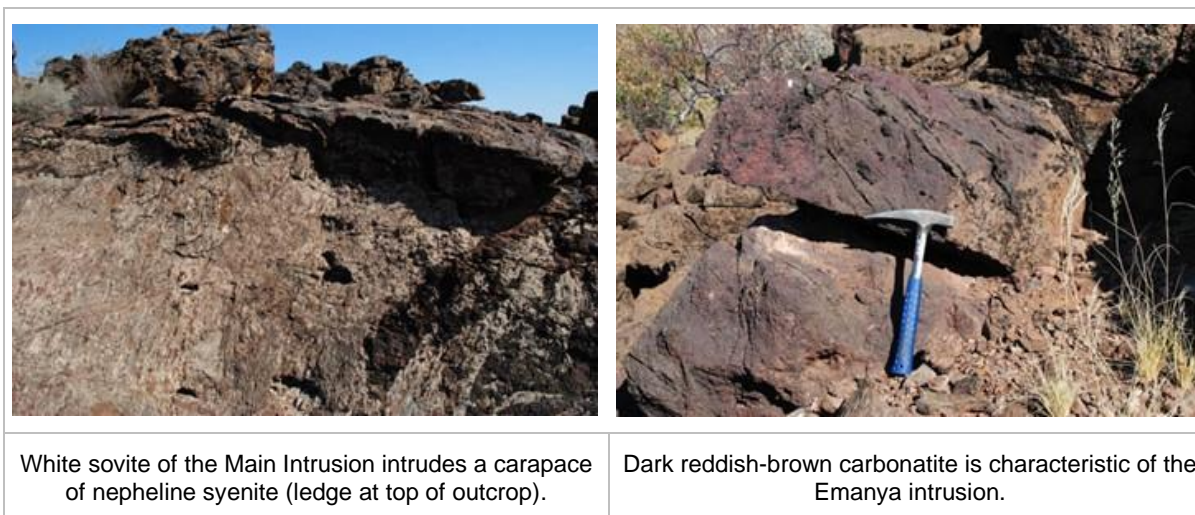


Source: Swinden, 2012

Figure 7-8 Examples of Lofdal Breccias

Carbonatites

Carbonatite dykes have been reported in the Lofdal area for a considerable time (Barbour, 1982; Verwoerd, 1993; Miller, 2008). However, the recognition of larger, plug-like carbonatite intrusions in the area is relatively recent. The composite syenite- carbonatite plug that is now referred to as the “Main” intrusion appears on a map by Barbour (1982) but its full extent and significance was only first recognized by the GSN geologists (V. Do Cabo, pers. comm.). A second, smaller intrusion about 4.5 km to the southwest, now referred to as the “Emanyia” intrusion or plug, was only discovered in 2008 (Figure 7-9). Regional geological investigations between 2008 and 2010 have shown that there are literally hundreds of carbonatite dykes and fentitised-carbonatised alteration structures over an area of about 200 km². Wall, et al., (2008) dated xenotime in the carbonatites by U-Pb and obtained an age of 765 ±16 Ma, indicating that they are approximately coeval with the Oas Syenite and the Lofdal nepheline syenites.



Source: Swinden, 2012

Figure 7-9 Examples of Carbonatite from the Main and Emanyra Intrusions

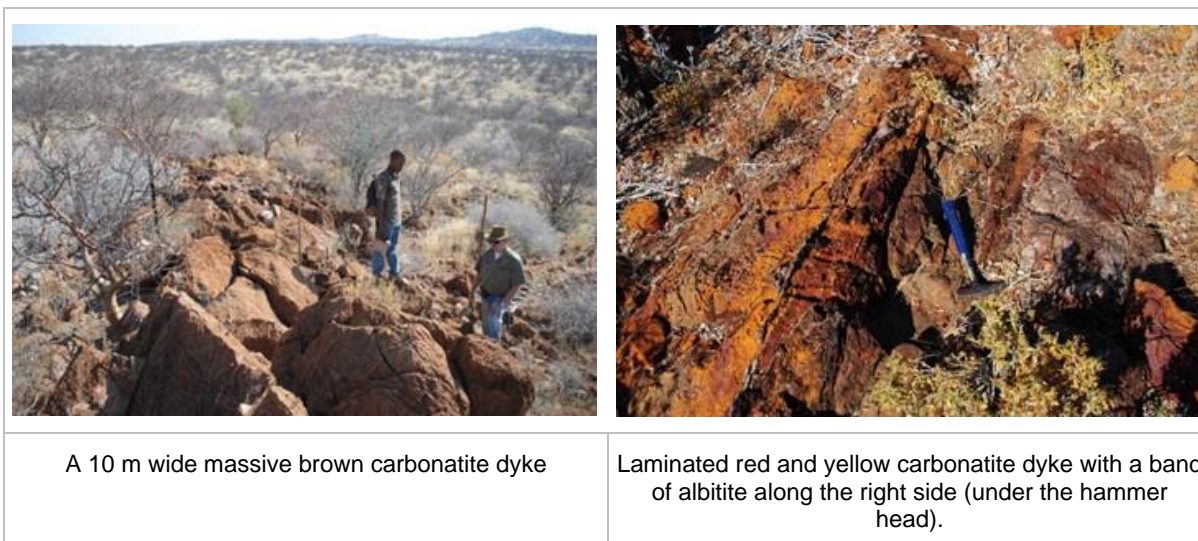
The Main intrusion is the largest carbonatite body found to date in the complex. Its outcrop area forms an ovoid with long axis of about two km and an area of about 1.5 km² (Figure 7-4). The Main intrusion carbonatites are dominantly coarse-grained white sovite consisting mainly of calcite with lesser aegerine, apatite and magnetite, and trace feldspars, sulphide minerals and pyrochlore (Gaudet, 2013; Bodeving, 2015). The carbonatite intrudes nepheline syenite and Lofdal breccias, which form an outcrop carapace on top of the carbonatite at the present level of exposure (Figure 7-9). Because of this, syenite, and to a lesser extent breccia, dominate the outcrop pattern in the area of the intrusion and it is not surprising that previous workers mapped this body as dominantly syenite (e.g., Frets, 1969). The Main intrusion is relatively uniform geochemically, with low iron contents and exhibits a LREE-enriched rare earth element distribution that is typical of carbonatite magmas but too low in absolute REE concentration to be of economic interest.

The Emanyra intrusion is located about 3.8 km southwest of the Main intrusion (Figure 7-4). It comprises a main body, roughly circular in outcrop with a diameter of approximately 350 m, as well as several smaller satellite bodies within a 450 m radius. The carbonatites in this intrusion are calcitic but contrast with the Main intrusion in that they are finer grained and dominantly brown to reddish-brown on outcrop surfaces with abundant iron oxide throughout (Kruger, 2012) (Figure 7-9). Fluorite is locally present in veinlets. On average, the Emanyra carbonatites contain approximately 8.6 times more LREE and 3.6 times more HREE than the Main intrusion. The REE in Emanyra are fractionated in favour of the LREE compared to the Main intrusion.

Carbonatite “dykes” have been mapped over an area of more than 200 km² throughout the Lofdal Carbonatite Complex. Although typically mapped and referred to as “dykes”, these appear to be dominantly hydrothermal and/or carbothermal vein systems, resulting from fluid expulsion from the magmas during crystallisation. They typically follow the structural grain of the country rocks, striking in a northeasterly direction and dipping steeply to the south. They exhibit a wide range of lithological and alteration characteristics, ranging from <10 cm to several 10's of metres wide and have a wide range of colour variation on weathered surfaces, from white and grey, through shades of brown, red and yellow (Figure 7-10). They are closely associated with phonolite dykes throughout the area, often occurring together in the same structure (with the ‘dykes’ always the later phase) or in closely spaced parallel dyke swarms.

There is considerable variability in the internal structure of the “dykes”. Many are uniform in character, exhibiting little internal banding or colour variation. Most however, exhibit some internal structure, commonly colour- and/or compositional-banding on a scale ranging from millimetres to centimetres.

The alteration in these “dyke” systems is variable but locally intense, characterised by an early pervasive albitisation followed by brittle fracturing and infusion of carbonate minerals and micas (see also Section 7.4.2). HREE mineralization is associated with the later stages of alteration. In some areas the alteration systems are 10's of metres wide.



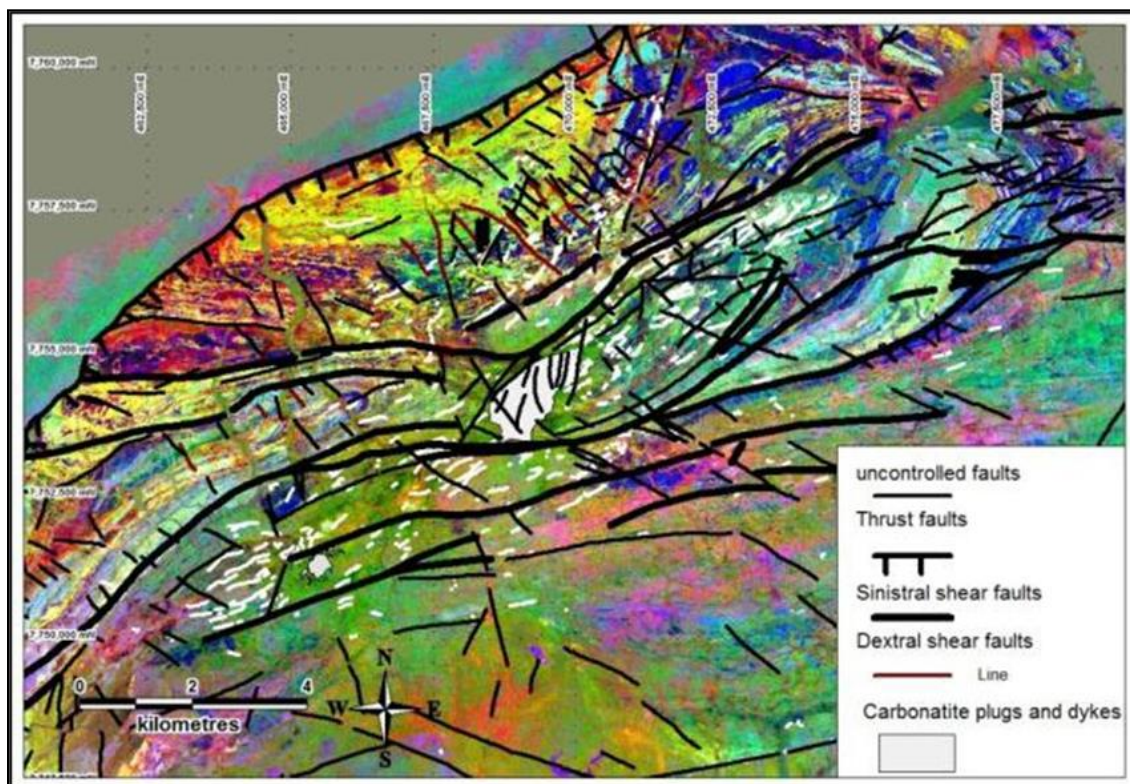
Source: Swinden, 2012

Figure 7-10 Examples of Brown and Red to Yellow Carbonatite Dykes

7.3 Structural Setting

Rocks of the Huab Metamorphic Complex were polydeformed and metamorphosed prior to intrusion of the Fransfontein Granite at about 1.7 Ga. They were subsequently affected by extensional tectonics during the rifting event that initiated the Damara Orogen at 850 Ma to 750 Ma and transpression during the orogenic events that accompanied the Damara Orogeny from 580 Ma to 500 Ma. The effects of both Neoproterozoic rifting and early Paleozoic transpression are recorded in the Lofdal carbonatites.

A comprehensive structural interpretation of the area was undertaken by NPA Fugro (2010) on behalf of NMI, integrating hyperspectral and Landsat data with all available airborne geophysical surveys as well as NMI geological and geochemical databases to produce a high level interpretation of structural features in the Welwitschia Inlier (Figure 7-11). This interpretation identifies regional structures in the basement that probably reflect the pre-Damara history of these rocks. It also identifies a series of sinuous NE-SW striking major fault structures that systematically offset the basement structures in a sinistral sense. These structures are locally offset by a series of NNE-SSW striking dextral structures and NW-SE striking structures.



Note: Base is hyperspectral image. Main and Emanya intrusions and carbonatite dykes are shown in white.
Source: NPA-Fugro, 2010

Figure 7-11 Structural Elements of the Lofdal Area, Interpreted from Landsat and Hyperspectral Data

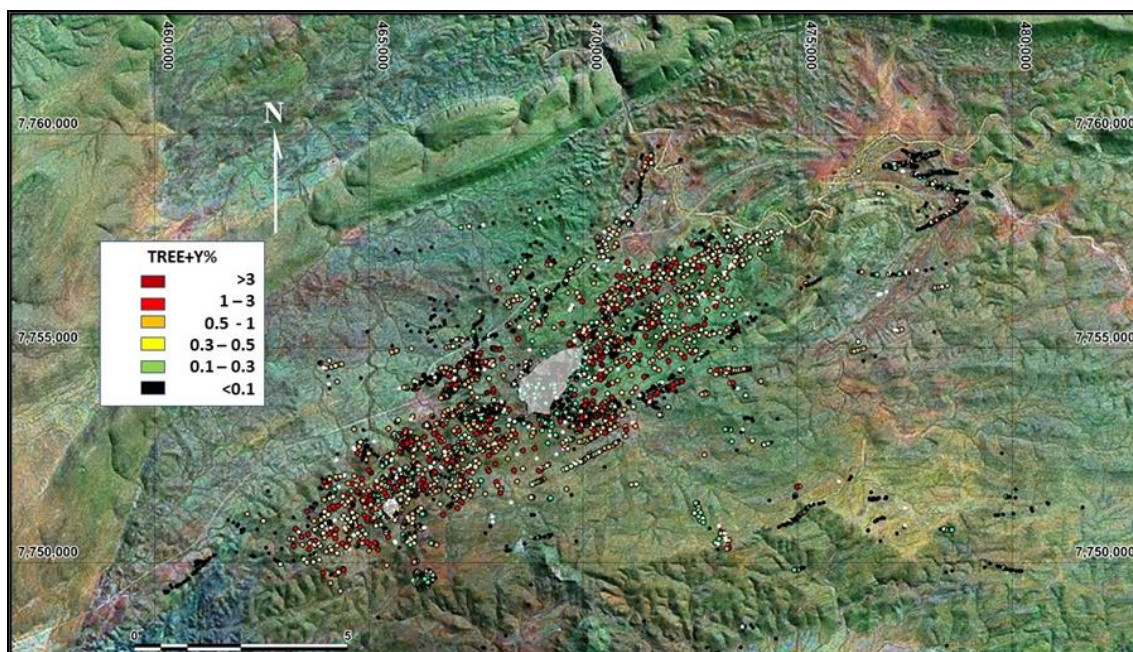
The intrusive complex at Lofdal shows a close relationship to the structural grain of the basement. The Main and Emanya intrusions are located between large sinistral structures and the phonolite and carbonatite dykes and veins in the complex are typically structurally aligned with these ENE- and NNE-trending structures (Figure 7-11). The intrusive complex seems, therefore, to have exploited regional structures during emplacement and subsequent hydrothermal alteration activity.

Structures within the carbonatites appear to record both extensional and transpressional events. The repeated injection of first phonolite and then several phases of carbonatite, and hydrothermal fluids into many of the structures may be the result of repeated opening of pre-existing structures during the extensional regime that prevailed at about 750 Ma. However, there is also evidence of transpressional deformation within the dykes, with minor structures (folds, shear bands) indicating transpression with the same sense of shear as is interpreted for the major structures. It may be that at least some of the deformation recorded in the dykes is Damara in age. It seems likely that the intrusions and the associated hydrothermal alteration were focused by extension faults that determined the location and orientation of the intrusions and provided pathways for the escaping hydrothermal fluids. These faults and fractures may have been reactivated during the Damara Orogeny, producing the observed structures.

7.4 REE Mineralization

7.4.1 Regional Setting

Exploration at Lofdal has demonstrated that there is widespread REE mineralization related to intrusion of the Lofdal Carbonatite Complex and that many occurrences are specifically enriched in HREE. The regional setting of REE mineralization was evaluated through an extensive regional surface grab sampling program (documented in Swinden and Siegfried, 2011), geological mapping with locally detailed lithogeochemical sampling and core drilling. Figure 7-12 illustrates the distribution of anomalous concentrations of TREE+Y in 3,764 outcrop grab samples collected between 2008 and 2011. The REE mineralization at Lofdal occurs mainly within a NE-SW trending corridor approximately 20 km long and 5 km wide, the axis of which is occupied by the Main and Emanyā intrusions. Mineralization occurs at a district scale over an area of at least 200 km².



Note: Color-coded for total REE + Y. Main intrusion is large grey body. Emanyā plug (small grey body) to the southwest
Source: Landsat Geocover Mosaic, 2000

Figure 7-12 Distribution of Lithogeochemical Grab Samples in the Lofdal Area

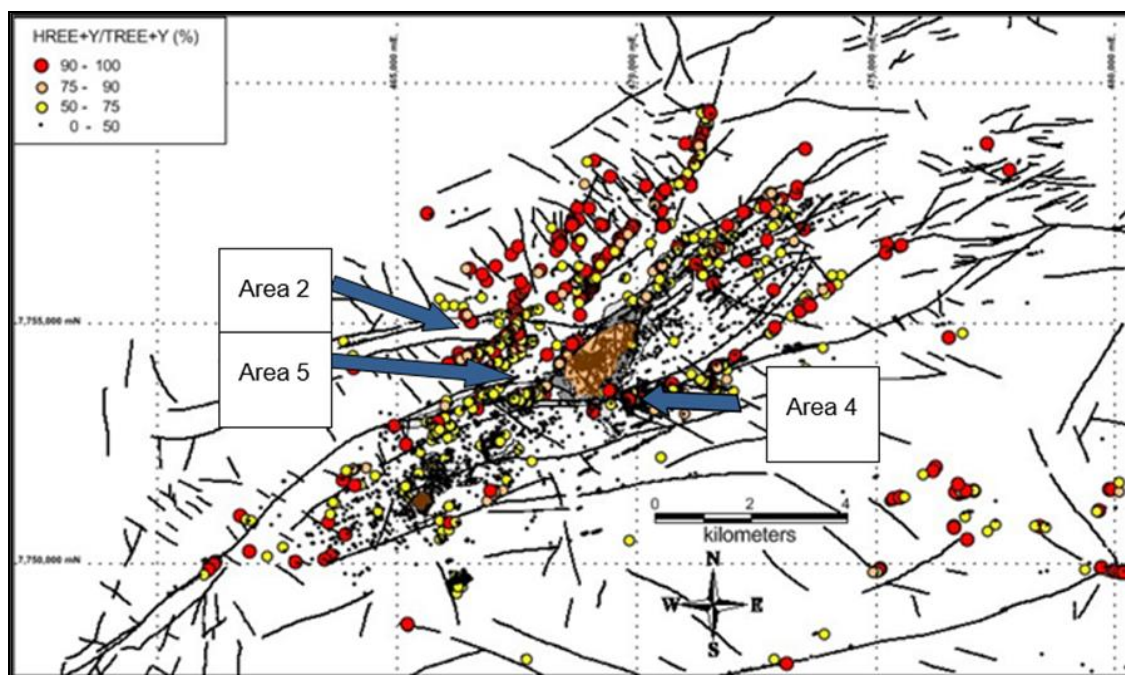
There is considerable variation in both the absolute concentrations of REE, and in the relative proportions of LREE versus HREE in mineralised samples. As a general rule, the Main intrusion shows a typical carbonatite REE profile of LREE enrichment but has very low overall concentrations of all REE.

The Emanyā intrusion shows a trend towards higher TREE+Y, which is mainly a result of LREE-enrichment, with little enrichment in HREE. The dykes and related alteration lithologies show a wide variation with much stronger enrichment trends in both LREE and HREE (Swinden and Siegfried, 2011).

Studies of both surface outcrops and drillhole cores have demonstrated that the HREE-rich mineralization is not principally hosted by carbonatites, but typically occurs in hydrothermal and/or carbothermal alteration zones that are localized in key basement structures irrespective of host lithologies (do Cabo et al., 2011; Swinden and Burton, 2012; Wollenberg et al. 2016). The delineation core drilling for mineral resource

estimation in Areas 2B and 4 shows that the HREE-rich mineralization is locally continuous in three dimensions over significant strike and dip extents.

Despite the complexity of the REE distribution overall, there is a regularity to the distribution of the most HREE-enriched samples. Figure 7-13 shows the enrichment of heavy rare earths as $(\text{HREE}+\text{Y})/(\text{TREE}+\text{Y})$ expressed as percentage, irrespective of grade.



Note: Most HREE-enriched samples plot along linear trends that are interpreted to reflect structures that provided fluid pathways during the hydrothermal event. Main and Emanyia intrusions are shown in light brown; alteration intensity in shades of grey (see also Figure 7-15 and Figure 7-17). Black lines are structures interpreted from remote sensing and geophysical data.

Source: Swinden, 2011

Figure 7-13 Lithogeochemical Grab Samples Plotted on the Basis of $(\text{HREE}+\text{Y})/(\text{TREE}+\text{Y})$

Figure 7-13 demonstrates a number of important distribution characteristics of the HREE-enriched mineralization:

1. The most HREE-enriched samples tend to be concentrated in linear belts, which very often coincide with the traces of fault structures interpreted from remote sensing data. These are interpreted to be the structures that provided fluid pathways or conduits for the HREE-rich hydrothermal fluids.
2. Even where overall grades are low in these structures, the HREE enrichment remains high, emphasizing the potential of these zones for concentrations of HREE-rich mineralization.
3. The drill targets in Areas 2B, 4 and 5 stand out as zones of HREE-enrichment.

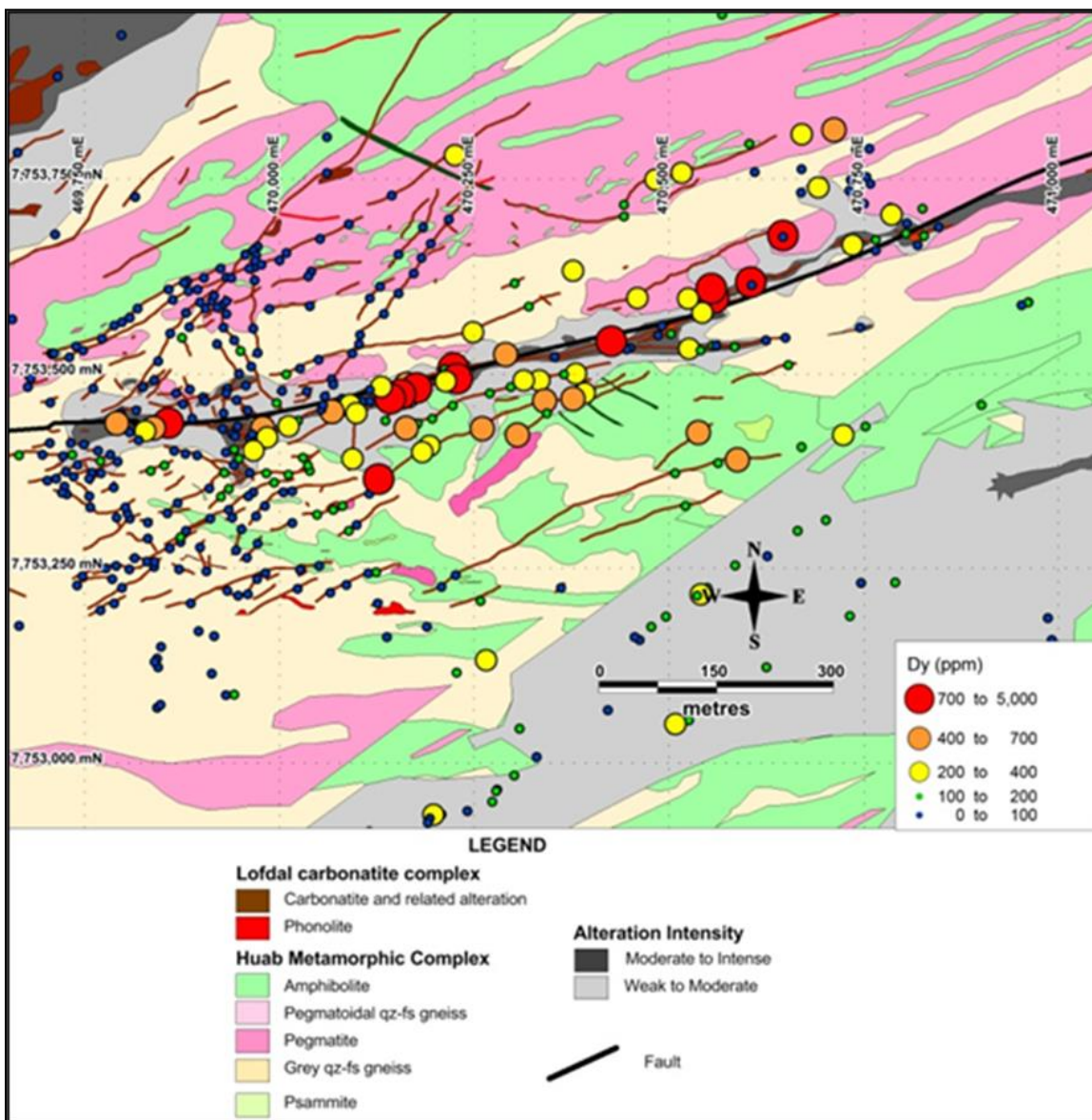
7.4.2 Mineralization in Area 4

The location of Area 4 in the Lofdal Carbonatite Complex is shown in Figure 7-4 and a detailed geological map of Area 4 is shown in Figure 7-14 and Figure 7-15.

The Huab Metamorphic Complex in this area is dominated by quartzo-feldspathic gneiss and metasedimentary grey gneiss with lesser amphibolite and pegmatite. The gneisses strike approximately ENE-WSW and generally dip steeply southwards.

The geological element of principal economic interest in Area 4 is a major fault, first interpreted from remote sensing data (NPA-Fugro, 2010), that strikes ENE-WSW and dips to the south, bisecting the area. The fault can be traced for several kilometres east and west of Area 4 and the offset of geological elements interpreted from hyperspectral data indicates that it has a sinistral sense of movement. At surface, the fault system is marked by carbonate veining and extensive hydrothermal alteration, dominantly albitisation and carbonatization accompanied by biotite \pm phlogopite and iron oxides. Mapping the alteration intensity associated with the structure shows that there is a core of intense alteration, within which rocks have been completely converted to albitite and are cut by carbonatite and highly carbonic alteration. In this intense alteration zone, all original textures have been destroyed and crackle breccias with altered, albitised clasts set in a matrix of carbonatite and/or iron oxides are common. Surrounding this intensely altered core is a halo of less intense alteration, in which the rocks are bleached and albitised, but retain some original textures.

The outline of the alteration zone is highly irregular at map scale (Figure 7-14). The intense alteration in the core is typically between 15 m and 30 m wide on the surface (not true width). The less intense alteration halo exhibits gradational and diffuse contacts with the wall rocks and is typically on the order of 50 m to 60 m wide at surface but can range to more than 100 m wide. At the eastern side of Area 4, the fault zone bifurcates, with the main system branching in a slightly more northerly direction and a splay of the fault continuing in an ENE direction. There is alteration and mineralization associated with both splays, but the southern splay appears to decrease in intensity along strike and alteration appears to die out within a few hundred metres.



Note: The mineralised structure reports high values in both the HREE (represented by Dy in ppm) and in percentage of HREE in TREE

Source: Swinden, 2021

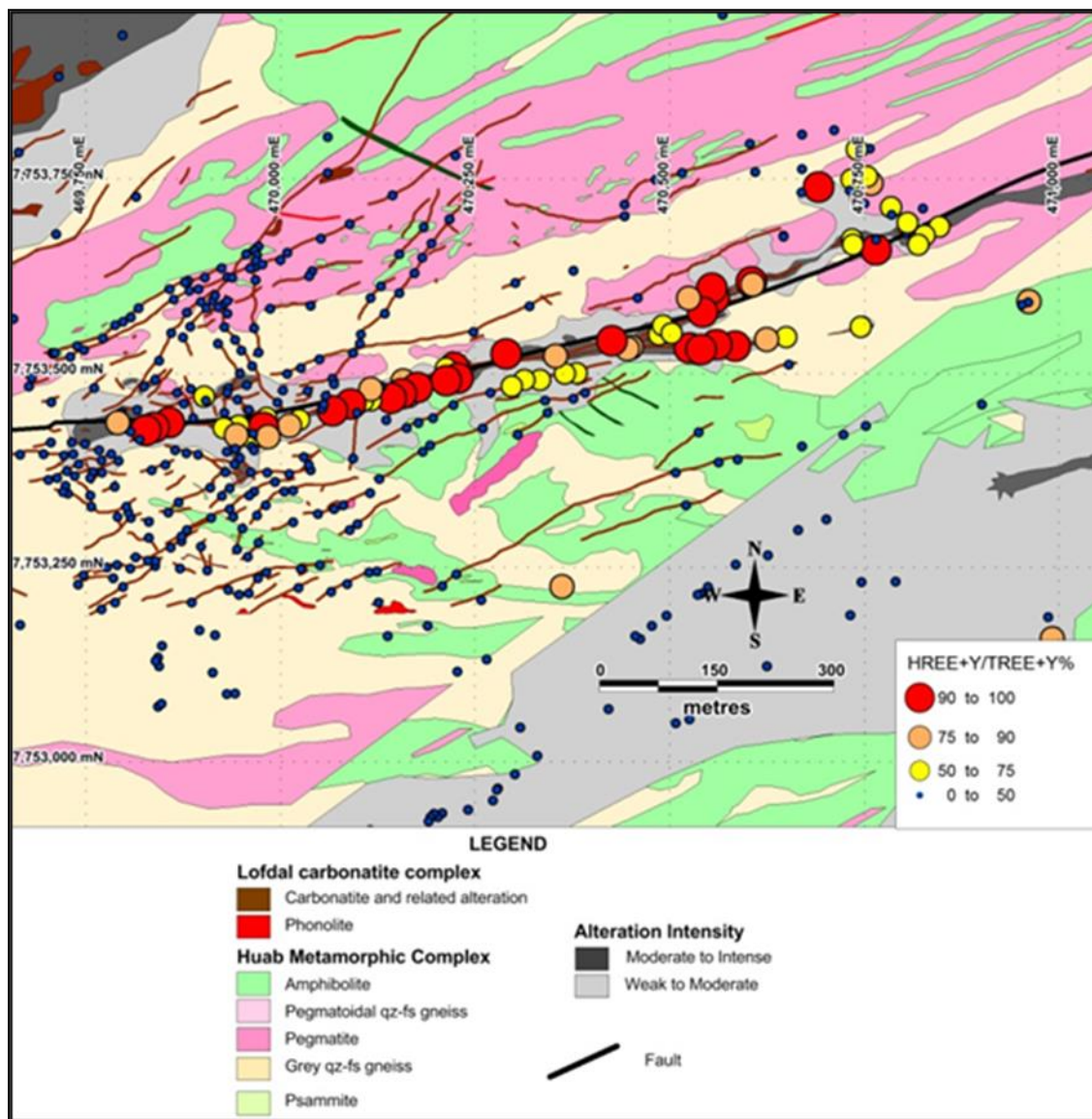
Figure 7-14 Geology of Area 4 with Dysprosium (Dy) Grade in Surface Grab Samples

A large number of carbonatite veins and carbonatic alteration zones have been mapped in Area 4. Although strike directions are dominantly NE-SW and NNE-SSW following the dominant structural grain of the basement, other directions are locally seen. Outside the central alteration zone, carbonatite veins are thin, (<1 m wide) and do not exhibit significant alteration beyond their margins. However, within the alteration zone, they are more continuous, and alteration is ubiquitous.

A more or less continuous zone of albite-carbonate alteration with significant grades of REE has been traced by mapping, trenching and drilling for more than 1,100 m along strike and regional geological mapping to the east and west indicates that it continues for several kilometres beyond Area 4. Within this

zone, the intense alteration typically thickens and thins, and locally forms lensoid bodies that can range on surface up to about 100 m long and 10 m wide.

The mineralization in Area 4 occupies a structurally-controlled, linear alteration zone. The Area 4 alteration zone is the largest and best mineralised and is clearly manifested and easily mappable in surface outcrops by variably intense albitisation and brown carbonization with locally abundant phlogopite. Grab samples from outcrops typically return highly anomalous values of HREE (Figure 7-14) and also have a very high HREE/TREE ratio (Figure 7-15).



Note: The mineralised structure returns high values in both the HREE and in percentage of HREE

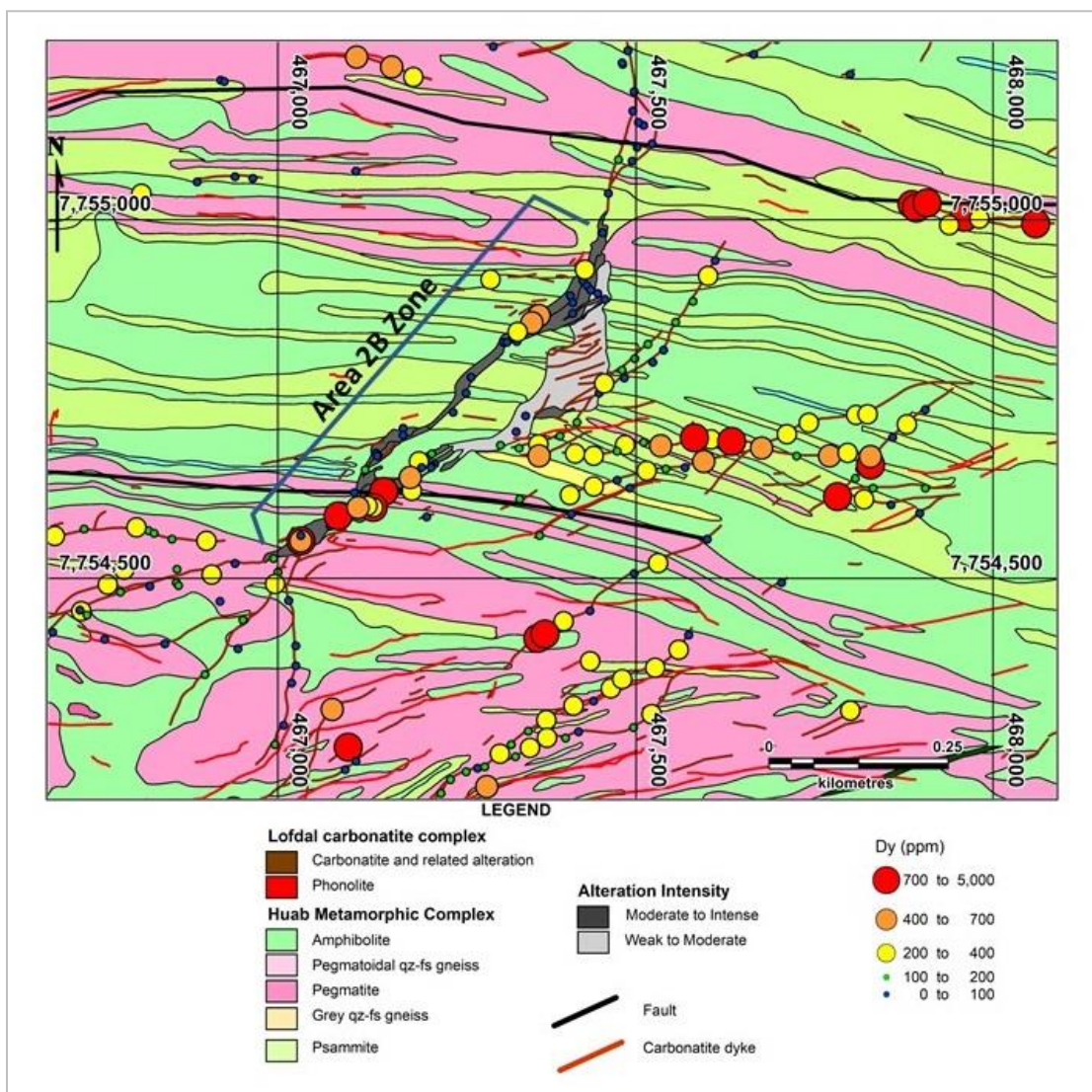
Source: Swinden, 2021

Figure 7-15 (HREE+Y)/(TREE+Y) % in Surface Grab Samples in Area 4

7.4.3 Mineralization in Area 2B

The location of Area 2B in the Lofdal Carbonatite Complex is shown in Figure 7-4 and a detailed geological map of this area is shown in Figure 7-16 and Figure 7-17.

The Huab Metamorphic Complex in Area 2 is dominated by amphibolitic schist interbanded at outcrop scale with leucocratic quartzo-feldspathic paragneiss and muscovite schist and locally intruded by coarse grained granitic pegmatites. The rocks are complexly folded on a fine scale.



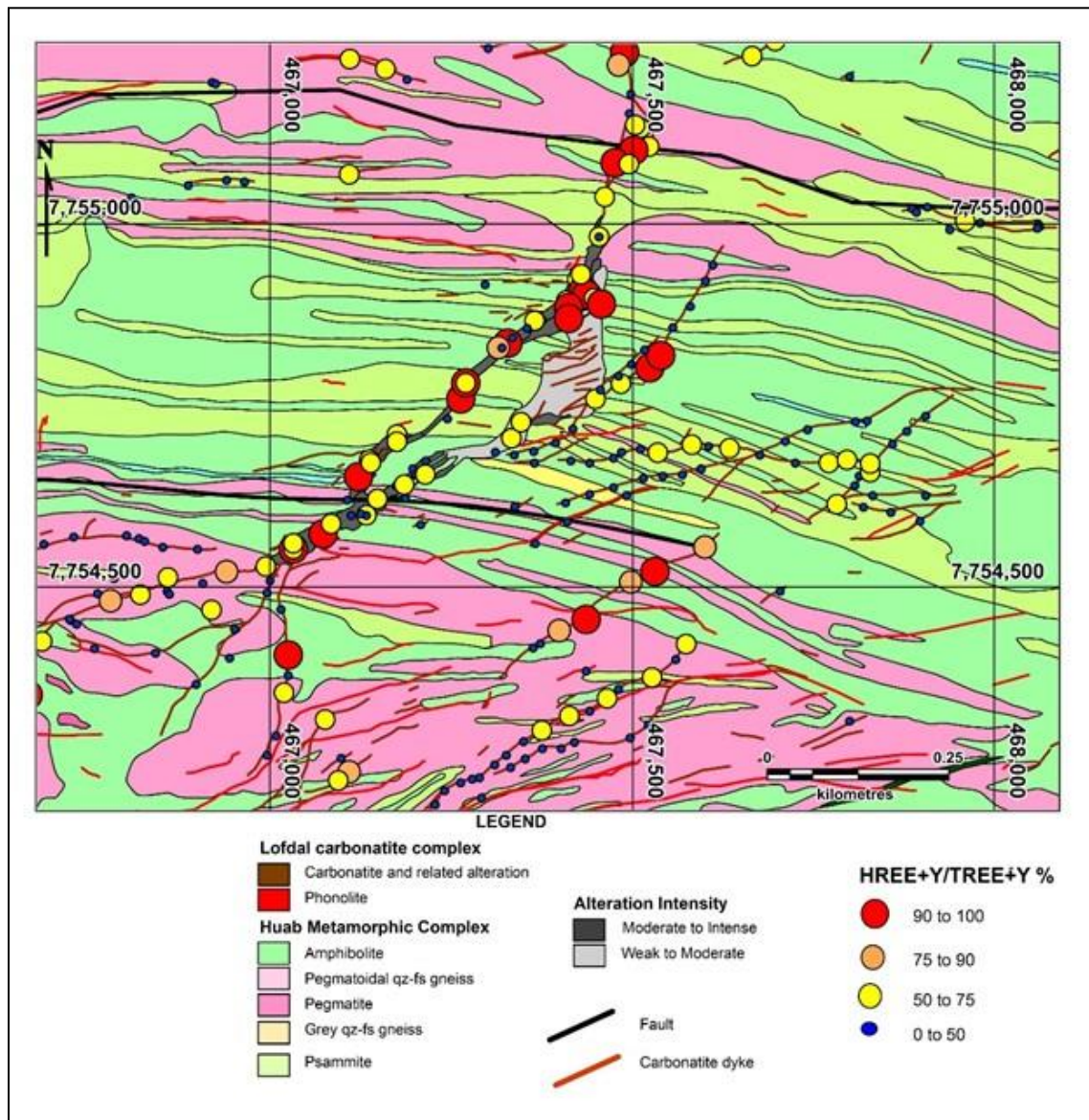
Note: Alteration related to the Area 2B mineralization is shown in shades of grey

Source: Swinden, 2021

Figure 7-16 Geology of Area 2B with Dysprosium (Dy) Grade in Surface Grab Samples

Phonolite and carbonatite dykes and related alteration zones of variable orientation and thickness are common in the area. Carbonatite dykes average a few cm in width but carbonatitic and albititic alteration zones can range up to more than 10 metres in width in outcrop.

Dykes and alteration zones in this area dominantly trend from NE-SW to NNE-SSW and generally are at a considerable angle to the structural grain of the basement, which in this area trends from E-W to ESE-WNW. The area is bounded to the north and south by major sinistral faults interpreted from remote sensing data (Fugro, 2010) and it may be that the dominantly NE-trends of dykes and alteration zones in Area 2B reflect fracture systems related to these linked faults.



Source: Swinden, 2021

Figure 7-17 (HREE+Y)/(TREE+Y) % in Surface Grab Samples in Area 2

The principal mineralization in Area 2 is the 2B zone (Figure 7-16), a wide zone of hydrothermal alteration and carbonatite intrusion. Like Area 4, the mineralization is characteristically enriched in HREE and samples throughout the alteration zone show a very high ratio of HREE to TREE (Figure 7-17). The mineralized zone has been traced in outcrop along a strike length of more than 600 m and remote sensing information and regional sampling results suggest that the zone may ultimately have a strike length of more

than 3 km. The width of the zone in outcrop is variable. At its southern end, the width of the zone of alteration and carbonatization ranges from about 20 m to 35 m but thins to less than 10 m in the central section where it bifurcates into two separate zones. At the northern end, where the zone of alteration and carbonatization is again amalgamated, it is more than 60 m wide. In outcrop, it comprises a zone of massive carbonatite dykes, within a complex envelope of hydrothermal alteration and brecciation. The southern part of the zone is dominantly brown carbonatite and related alteration. Alteration zone lithologies include massive albitite, localized zones of green phlogopitic fenite, brown stained albitite breccias infused with carbonatite, as well as altered mafic schist that has been carbonatized, variably albitized, and intruded by carbonate veinlets. Although massive and continuous across strike at its southern end, the zone bifurcates at surface in its central section with a hanging wall zone of dominantly carbonatite and related alteration and a footwall zone that includes considerable altered, carbonatized schist and carbonatized stockwork.

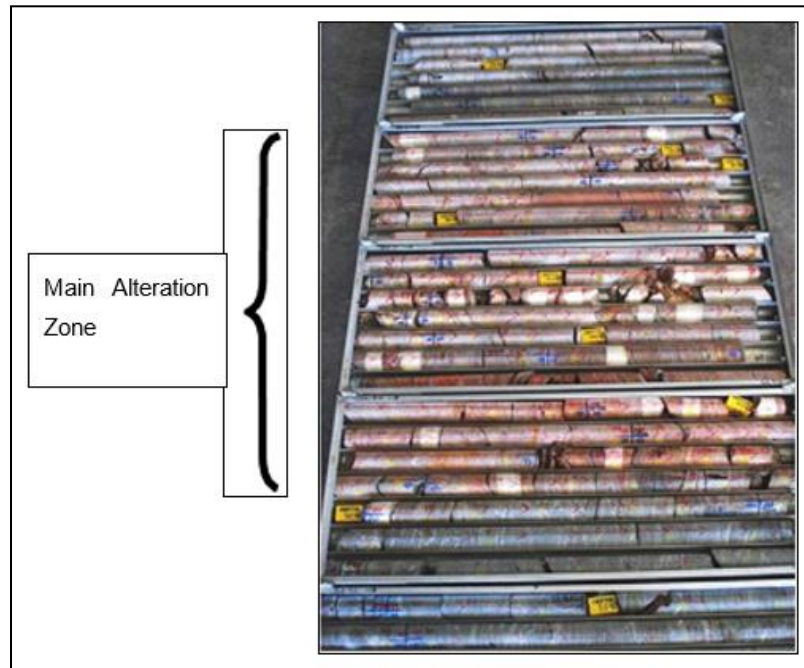
The REE mineralization in the Area 2B zone is restricted to the zone of alteration and carbonatization. In drillhole core, the zone is seen to consist of a zone of intense brecciation and hydrothermal alteration. The best assay values are related to late veining and alteration that cuts most of the pre-existing alteration lithologies.

Shearing is very common in the alteration zones. In most sections, there is a prominent shear zone at or near the footwall, which is itself variably albitized and carbonatized. The shear zones range from centimetres to as much as five metres wide and the shear fabrics are cut by both albitite and carbonatite suggesting that they represent structures that pre-date the mineralization. There is a main footwall shear zone in most sections, which may be the controlling structure for much of the alteration and mineralization.

7.4.4 Nature of the Alteration

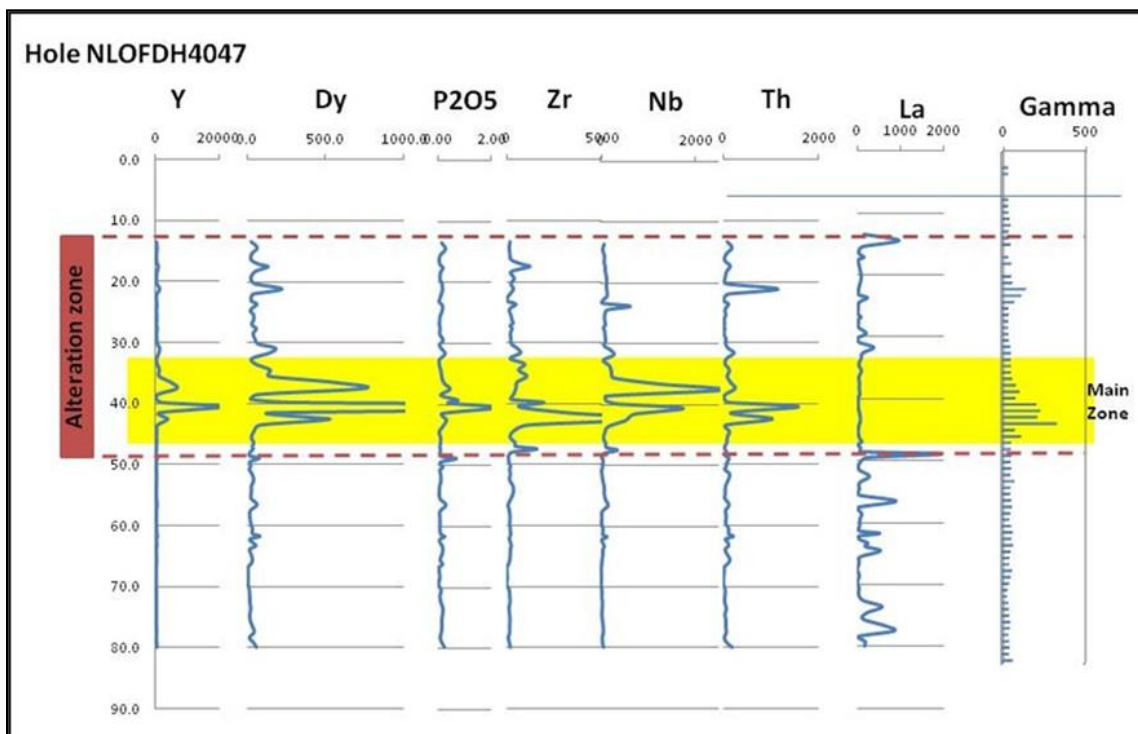
The alteration in Areas 4 and 2B is geologically and mineralogically very similar. In drillhole core, the zones are characterised visually by the bleaching and reddening that accompanies the alteration (Figure 7-18). The boundaries of the alteration zones are locally sheared and/or intensely broken, clearly indicating the structural nature of the zones. The structure is internally complex with multiple internal shears that are typically micaceous, characterised by the development of black biotite, phlogopite and chlorite as well as calcite.

The alteration and contained mineralization are characterised by both radiometric and geochemical anomalies. The presence of Th (see Section 7.4.6) results in a generally elevated radiometric signature in the alteration zones, although the Th is not always spatially associated with the REE. Geochemically, the alteration zones are characterised by elevated concentrations of the HREE, Y and P₂O₅ as well as Nb and Zr, although, as with Th, the Nb and Zr concentrations do not closely correlate with the HREE on a sample-by-sample basis (Figure 7-19). There are no correlations between the HREE and the LREE. Visual radiometric and geochemical characteristics allow the alteration zone to be readily traced between drillholes both down-section and along strike (Figure 7-20).



Source: Swinden, 2012

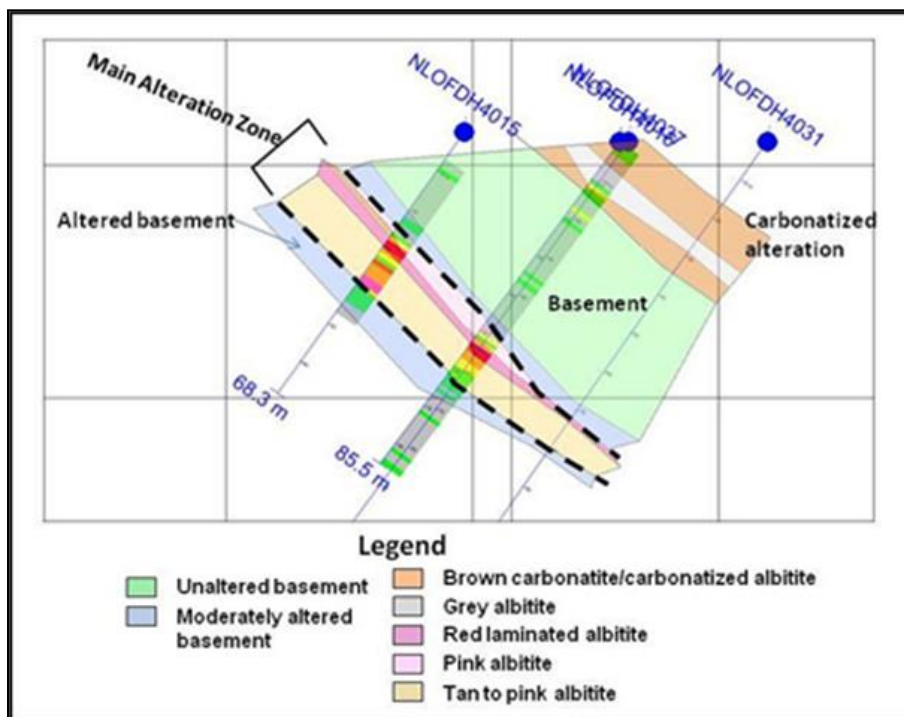
Figure 7-18 Colour Anomaly in Drillhole Core Associated with the Main Zone Alteration in Area 4



Note: Element concentrations in ppm except P_2O_5 in weight %. Depth in metres (vertical axis). Alteration zone is defined visually in drillhole core, Main zone mineralization by assays correlated with visual alteration.

Source: Swinden, 2014

Figure 7-19 Schematic Illustration of Geochemical and Radiometric Anomalies Associated with the Area 4 Alteration Zone in Drillhole NLOFDH4047



Note: Overlapping drillholes are NLOFDH4016 and 4037 which were twinned. Colour bars along drillholes represent concentrations of Y ranging from highest (red) through orange, pink, yellow and green to lowest (grey). Heavy dashed lines are structural boundaries of the Main zone.

Source: Swinden, 2012

Figure 7-20 Schematic Cross Section of the Upper Part of the Area 4 Main Alteration Zone from Drillhole Data

Rocks within the alteration zones are typically completely replaced by an assemblage containing variable proportions of albite, quartz, biotite/phlogopite, chlorite, calcite and iron oxides. The alteration is characterised by a pervasive background albitisation which has converted the entire rock mass to fine grained albitite. The early albitite has been brecciated and overprinted by a second generation of albite, calcite, brown calcite and dolomite, which locally form thin brown carbonatite veins, red iron oxides, and green to black biotite/phlogopite and chlorite (Figure 7-21). The alteration is typically texture-destructive and pervasive.



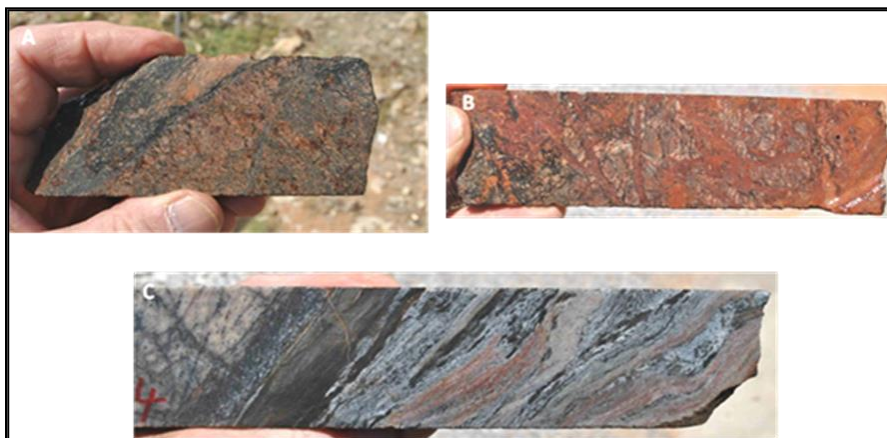
Note:

1. Early albite (white) brecciated and infilled by hydrothermal carbonate (brown)
2. Alteration in drillhole core. White albitization overprinted by brown carbonate/iron oxides
3. Narrow highly mineralized iron oxide-rich vein cutting albitized, mica rich alteration

Source: Swinden, 2010

Figure 7-21 Typical Alteration and Mineralization in Area 2B

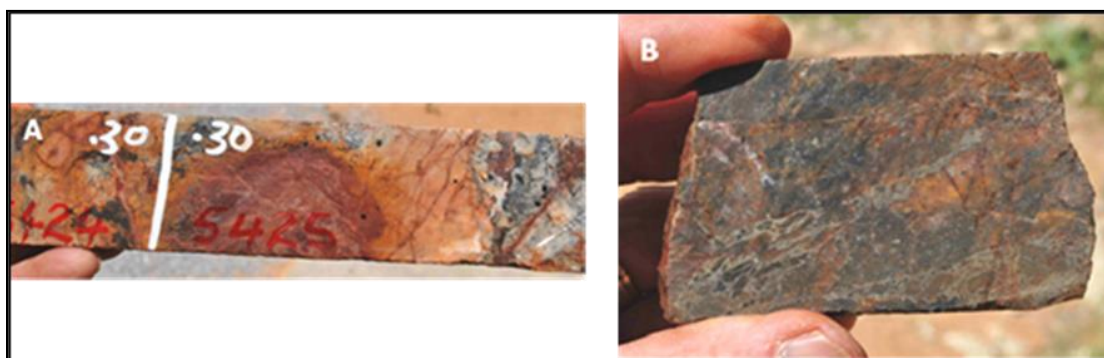
The alteration halo is typically broader than the mineralised intervals. Alteration relationships are complex, but there are regularities and consistencies to the distribution of mineralization. Mineralization is variously found in red to pink albite-rich veins and patches, black mica- and chlorite-rich alteration veins and shear zones, tan and silver-grey variegated albitite, white to grey dolomite, white, grey or yellow albitite, or brown carbonatite (Figure 7-22). Typically, several styles of mineralised alteration occur within the same drillhole and where the alteration is most complex; there are clear overprinting relationships between different generations of alteration. The main unifying geological characteristic of the higher-grade mineralization is the presence of complex overprinting relationships, reflecting multiple alteration events. Almost invariably, when paragenetic relationships between different generations of alteration are observed, the alteration associated with highest-grade mineralization is the latest event.



Note: A: Dark grey albite-calcite-biotite/phlogopite matrix with tan xenotime-rich alteration; B: Red vein network rich in HREE cuts a brecciated white albite; C: HREE-rich red alteration cross-cuts shear fabric in a mica-rich shear zone
Source: Swinden, 2012

Figure 7-22 Examples of Area 4 Alteration in Drillhole Core

In areas where significant grades and widths of HREE mineralization are intersected by drilling, the mineralised zones typically exhibit an increase in the background HREE content of more than threefold over the background in unmineralised rocks. These intervals generally contain local areas of high-grade mineralization that raise the overall grade from anomalous to economically- significant. High-grade mineralization may take various forms, the most common of which are veinlets, vein networks, alteration patches and micro-breccia veins (Figure 7-23). The higher-grade mineralization is characterised by extreme HREE enrichment ($[\text{HREE}+\text{Y}]/[\text{TREE}+\text{Y}] > 90\%$) and this enrichment typically extends beyond the boundaries of the higher grades into adjacent rocks where the overall grades are lower.



Note: A : Red alteration patch in centre contains $>10\%$ Y; B : micro-breccia vein trends from lower left to upper right and matrix is rich in HREE
Source: Swinden, 2012

Figure 7-23 High Grade Mineralization in Area 4 Alteration in Drillhole Core

7.4.5 Mineralogy

Petrographic and scanning electron microscope (SEM) observations provide detailed information on the mineralogy of the mineralized zones. Most of the detailed work was carried out on Area 4, but regional

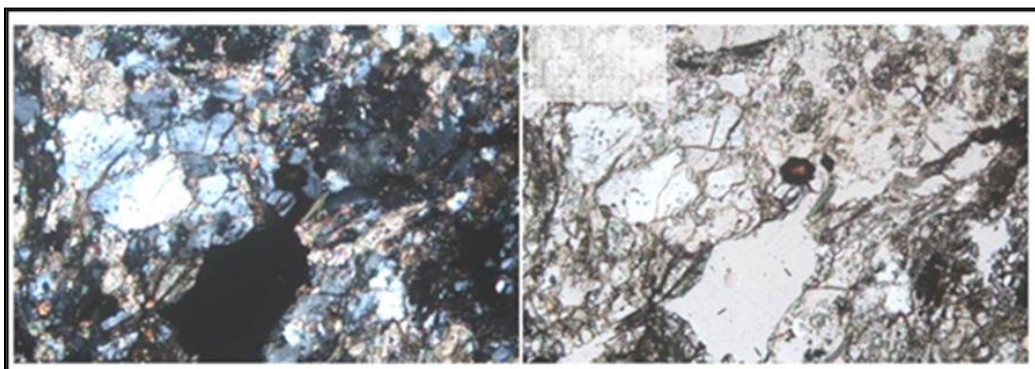
studies in other alteration zones indicate that the mineralogical characteristics are similar throughout the property.

Detailed mineralogical studies of the Lofdal carbonatites and associated mineralization were first carried out by Mariano (2001) who studied a mineralised sample from a carbonatite dyke. His petrographic and cathodoluminescence studies showed the principal REE-bearing mineral to be xenotime, with associated monazite, parisite, apatite and thorite.

A petrographic study of 18 variably mineralised samples from outcrops in the Lofdal area was carried out by Schandl (2010). The principal HREE mineral identified was xenotime with minor aeschynite (Y). Minor amounts of LREE minerals were identified including bastnaesite, parisite, synchysite, monazite and a single occurrence of allanite. Schandl (2010) noted that the low Th content of most REE minerals may signal a hydrothermal origin and identified secondary albite, riebeckite and aegerine which were interpreted as evidence of sodic fenitisation.

Detailed mineralogical studies on a suite of outcrop samples by V. do Cabo of the GSN, including whole rock geochemistry and scanning electron microprobe studies, confirmed the dominance of xenotime in mineralised samples and identified a suite of accessory minerals that include zircon, monazite-(Ce), synchysite-(Ce), thorite, apatite and rutile in a calcite-albite-quartz-chlorite-Fe-oxide gangue (Wall et al., 2008).

Detailed mineralogical studies of mineralised drillhole core from Area 4 were undertaken by Dr. James Clark of Applied Petrographics (Clark, 2012). Petrographic and SEM studies show that the gangue to the mineralization comprises mainly albite, phlogopite/biotite and chlorite, calcite iron oxides and quartz. In most mineralised sections, the background lithology is albitite, dominated by coarse to fine crystals and crystal fragments of albite. The textures in the albitite indicate that it has been granulated resulting in remnant coarse albite crystals and crystal aggregates within a comminuted matrix of rock flour flooded by calcite and iron oxide (Figure 7-24).



Note: Aggregates of xenotime and zircon (high relief, examples highlighted), in association with calcite, distributed along micaceous stringers and intergranular to granulated albite. The dark brown phase near photo centre is niobian rutile. FOV=1.35 mm. Cross polars on left, plane light on right.

Source: Clark, 2012

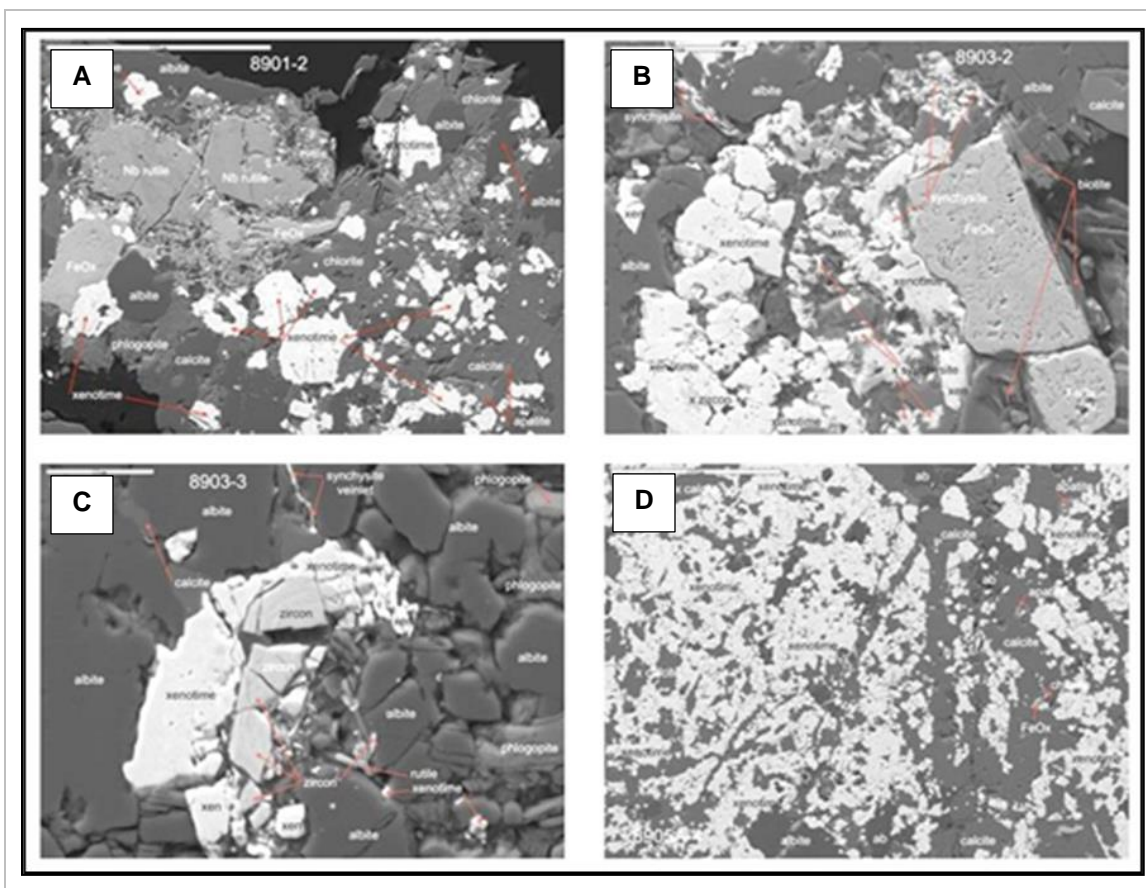
Figure 7-24 Albitite with Aggregates of Xenotime and Zircon

Accessory minerals, including REE-bearing phases, occur in the matrix, in hairline fractures and along shear fabrics defined by biotite/phlogopite. The working hypothesis is that pervasive albitisation was an early alteration event, and that the albitites were affected by further movement on the host structures, resulting in brecciation and the introduction of new hydrothermal fluids which resulted in overprinting alteration and mineralization.

The most abundant accessory minerals are Nb-rutile and rutile, zircon, thorite and apatite. Minor amounts of pyrite, ilmenite and galena were also identified.

The most abundant REE-bearing mineral is xenotime, which occurs in more than 80% of the samples examined from Area 4. Synchysite-(Ce) and Synchysite-(Y) are common, although minor, phases and minor amounts of monazite-(Ce), bastnaesite-(Ce), parisite-(Ce) and aeschynite-(Y) are locally present as well as a number of REE-bearing phases that have not yet been identified. SEM spectra suggest that some REE may be present in zircon and thorite.

SEM backscatter images show that the mineralization is typically fine grained. Individual grains and grain aggregates are locally >100 µm but are typically less than 50 µm. Xenotime is locally intergrown with zircon, rutile and thorite on a fine scale (Figure 7-25).



Note:

A : Xenotime crystals and aggregates in a lens of calcite and phlogopite/chlorite. Rutile and niobian rutile are present, along with minor accessory apatite and iron oxide after sulphide. Scale bar is 200 µm;

B : Xenotime aggregates locally envelop earlier-crystallising zircon. Minor synchysite is intergranular to albite and biotite, and locally in edge contact with xenotime. Scale bar is 50 µm;

C : Xenotime deposition along a rounded edge of the zircon crystal at photo centre. Xenotime appears to nucleate on the earlier zircon. Phlogopite and granulated albite are the gangue phases. Scale bar is 20 µm;

D : Monophase xenotime aggregates. Scale bar is 200 µm.

Source: Clark, 2012

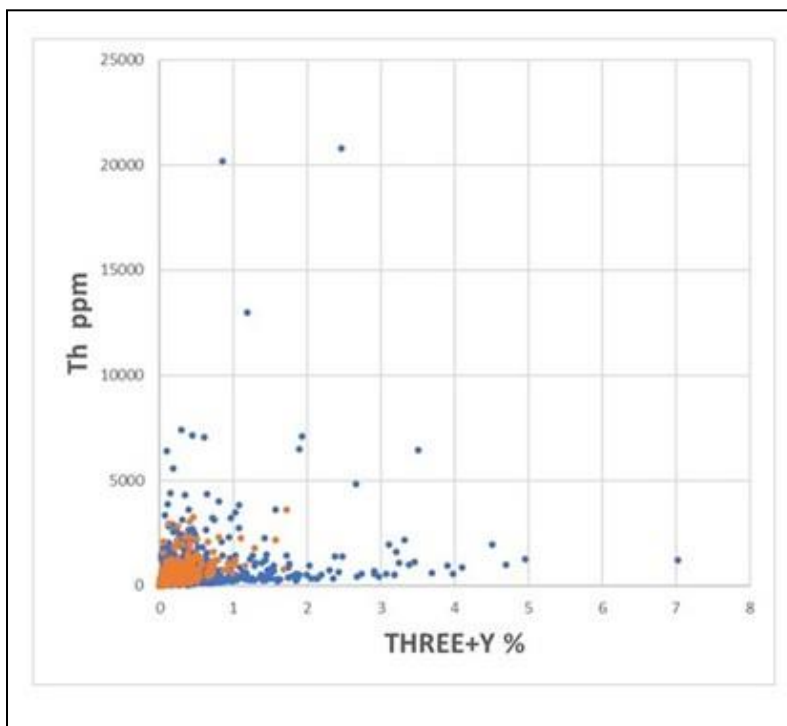
Figure 7-25 Backscatter Images of Area 4 Mineralization

Loye (2014) carried out a detailed study of the Area 4 alteration and mineralization using observations of drillhole core, geochemistry, cathodoluminescence, SEM and microprobe data. He recognized six different modes of occurrence of xenotime and ascribed these to an extended process of HREE mineralization and remobilisation spanning the late magmatic and hydrothermal phases of the intrusions. His model involves early ground preparation of key structures by alkalic fluids expelled during crystallisation of the nepheline syenites, which resulted in pervasive and widespread albitisation. Continued movement on these structures resulted in brecciation of the brittle albitites. Fluids exsolved from carbonatite magmas utilised the fluid pathways created by the brecciation, overprinted the albitites with a complex alteration assemblage that included dolomite and ankerite, biotite/phlogopite, iron oxides and pyrite, and a variety of accessory phases, and introduced HREE rich mineralization. Early alteration was dominated by calcite and dolomite, and late alteration by ankerite. Late alteration fluids re-worked the early alteration assemblages remobilizing and redistributing previously present REE.

7.4.6 Thorium

The presence of thorium (Th) can potentially be problematic in carbonatite-associated REE deposits because of its radioactive nature. Alteration associated with REE mineralization in the Lofdal Carbonatite Complex is variably anomalous in Th, and this largely contributes to the regional Th airborne radiometric anomaly that defines the area of interest. It also provides a convenient and important prospecting and evaluation tool on the ground, as most carbonatites and their associated alteration have elevated radiometric signatures.

The alteration zones in Areas 4 and 2B typically carry anomalous concentrations of Th (approximately 2% of drill samples returned >1,000 ppm Th), and the zones give a low-level radiometric response which is generally a good guide to mineralised alteration zones. However, overall, there is not a close geochemical relationship between the HREE and Th. Figure 7-26 shows the total HREE versus Th results for 5,940 drillhole core and trench samples from Area 4 and Area 2B.



Note: Although Th is locally present in mineralised samples and radiometrics often provide a good guide to mineralization, there is no clear geochemical correlation between Th and the HREE.

Blue dots- Area 4; orange dots-Area 2B

Source: Swinden, 2021

Figure 7-26 THREE versus Th in Trench and Drillhole Core Samples

7.4.7 Mineralization Summary

Mineralization in Areas 4 and 2B is structurally controlled and hydrothermal in origin. The host structures are first- and perhaps second-order basement structures that were apparently reactivated more than once during the mineralizing event. Repeated movement promoted the introduction of several generations of hydrothermal fluids, which resulted in a complex series of overprinting alteration events. The mineralization is dominantly present in xenotime and is interpreted to be related to the waning stages of hydrothermal alteration related to carbonatite intrusion. The highest-grade mineralization does not occupy a consistent position within the structural zones. It is interpreted to occupy structures within the zone that were still open during the last phases of hydrothermal alteration. The mineralised structures can be traced from hole to hole and are variably mineralised.

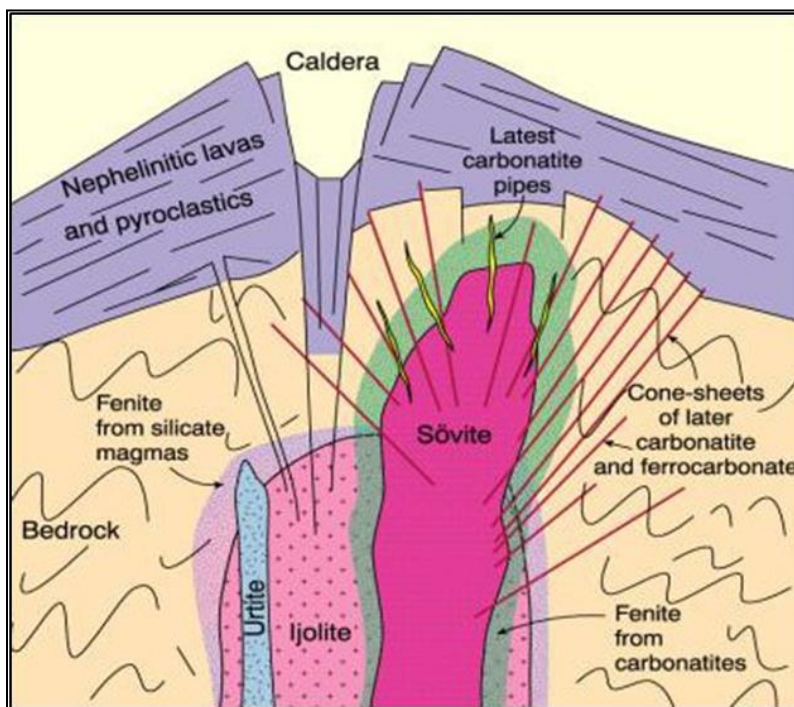
8 DEPOSIT TYPES

8.1 General Models for REE Mineralization in Carbonatites

Carbonatites and related, often undersaturated, silicate rocks originate in the earth's mantle through very low degrees of partial melting. They typically display geochemical enrichments in Ba, Nb, P, Fe, Ti, REE, F, Sr, Ta, Th, U and Zr. Carbonatites are important for a variety of economic mineral deposit types including REE (e.g., Bayan Obo, Mountain Pass), Nb (e.g., Araxa, Oka), Ti (e.g., Tapira) P (e.g., Araxa, Palabora), vermiculite, and fluorite (e.g., Okoruso). Carbonatite-associated deposits, including the giant Bayan Obo deposit in China, are the principal source of REE. Carbonatites are the focus of much current exploration for REE throughout the world.

A widely cited general model for the intrusion of a carbonatite complex (Le Bas, 1987) is shown in Figure 8-1. Although most complexes differ from each other in detail, this model provides a useful framework for description of observed mineralization at Lofdal. At Lofdal, the early silicate intrusions are dominantly syenite and nepheline syenite, rather than ijolite and urtite. Like the model, they are succeeded by a sovite intrusion (the Main intrusion) and by one and potentially more, subsidiary intrusive plugs, represented by the Emany intrusion.

Similar to the model, Lofdal has abundant later stage dykes of both silicate (phonolite) and carbonatite, ranging in composition from calcitic through dolomitic and ankeritic phases. There is abundant fentisation related to hydrothermal alteration around the margins of the intrusions and in basement structures that have served as pathways for both phonolite and carbonatite magmas and later hydrothermal fluids.



Note: Early silicate intrusions are intruded by carbonatite intrusions, which are cut by later carbonatitic dyke complexes.

Figure 8-1 General Cross-Sectional Model for an Alkali Silicate-Carbonate Intrusive Complex (after Le Bas, 1987)

REE, particularly the LREE, are typically significantly enriched in carbonatites over normal crustal abundances – a result of their partial melting history in the mantle, subsequent concentration through fractional crystallisation in the crust and sub-solidus hydrothermal activity accompanying the intrusion. Locally, the enrichment in REE produces deposits of economic proportions. Mariano (1989) identified three types of mineralization that might be expected in a carbonatite complex:

- Magmatic primary crystallisation of REE minerals;
- Hydrothermal concentration; and
- Supergene concentration.

The rocks at Lofdal are not deeply weathered and there is a very low likelihood of extensive supergene enrichment. However, there is significant potential for both magmatic and hydrothermal deposits.

There appear to be at least two styles of mineralization on the Lofdal property:

- Early LREE-enrichment in magmatic carbonatites, particularly the Main and Emanyia intrusions: Absolute abundances in the Main intrusion do not appear to attain economically interesting grades but overall grades are higher in the Emanyia intrusion. Associated sovite dykes are also significantly LREE-enriched; and
- Late hydrothermal mineralization characterised by extreme HREE enrichment: This mineralization is dominantly structurally controlled and occupies hydrothermal alteration zones within major structures. This mineralization is characteristic of Area 4 and Area 2B where diamond drilling has outlined mineral resources.

8.2 Magmatic Mineralization

It is rare to find a REE deposit that has formed through primary crystallisation from carbonatite magma. The best-known example is the Mountain Pass deposit in California where a 1.4 Ga intrusive complex consisting of a total of eight plugs, ranging in composition from shonkinite to carbonatite, intrude Precambrian basement metamorphic rocks (Castor, 2008). The deposit reportedly has “current reserves” of more than 20 million tonnes of ore with an average grade of 8.9% rare-earth oxides (Castor, 2008). The ore typically contains 10% to 15% bastnäsite-(Ce) with subsidiary monazite and apatite, and is mostly composed of calcite, dolomite and barite, with generally minor amounts of other minerals. Texture and mineral paragenesis shows that bastnaesite and parisite are primary magmatic minerals.

The Mountain Pass carbonatite plug provides an analogue for potential LREE targets at Lofdal. There is widespread LREE-rich mineralization in both intrusive plugs and carbonatite dykes, particularly in the central intrusive core of the Lofdal complex. This mineralization has not been extensively explored to date, with the exception of seven drillholes testing the Emanyia intrusion. The carbonatite intrusion at Mountain Pass is comparable in size to the Emanyia intrusion and there is potential for the discovery of additional plugs of similar size at Lofdal. There continues to be significant potential for LREE-rich mineralization associated with the intrusion of the Lofdal carbonatites but to date, exploration has not focused on the LREE targets.

8.3 Hydrothermal Mineralization

There is abundant evidence, both observational and theoretical, that REE minerals are precipitated from hydrothermal solutions (Williams-Jones et al., 2013) and, according to Mariano (1989), this is the origin of REE minerals in most carbonatites. REE mineralization in carbonatites may result from the breakdown of REE-bearing primary minerals such as calcite, dolomite, apatite or sulphides. The solutions become increasingly enriched in Ba, F, SO, Sr, REE and Th, and precipitate REE phosphates if phosphate is

available, or carbonates if phosphate is not sufficiently abundant. The REE mineralization in these environments tends to mirror that of the original carbonatite, i.e., fractionated in favour of the LREE.

The Emanyia intrusion has abundant iron oxide veining and locally fluorite, indicating that some hydrothermal alteration has occurred and its REE concentrations are significantly enriched over those in the Main intrusion, although it is still LREE-dominated. Like the Emanyia intrusion, many of the carbonatite dykes are significantly enriched in REE and fractionated in favour of the LREE.

At Lofdal, in addition to the LREE-enriched carbonatite-hosted mineralization, there is a late stage, structurally-controlled hydrothermal alteration that has resulted in HREE-rich mineralization in dynamic basement structures (see Figure 7-11 and Figure 7-13). These structural zones apparently acted as fluid pathways during mineralization and late-stage alteration in these structures introduced a HREE-rich mineral assemblage dominated by xenotime, and accompanied by zircon, rutile, apatite, fluorite and thorite. This is economically significant because it is the HREE that are the most valuable of the REE.

The current working hypothesis is that this HREE-rich hydrothermal alteration resulted from some combination of extended fractional crystallisation of the carbonatite magma and/or differential transport of the REE in exsolved hydrothermal fluids. Crystallisation of LREE-rich minerals early in the fractionation history could have resulted in a HREE-rich residual fluid phase, which escaped into selected structures during the later stages of crystallisation, resulting in the HREE-rich mineralization (Swinden and Burton, 2012). Depending on the chemistry of the hydrothermal fluids, fractionation of the HREE from LREE may also have occurred as a result of hydrothermal activity.

To date, the highly anomalous HREE enrichment at Lofdal has only been observed in selected structures associated with a complex series of alteration lithologies. The origin of the HREE enrichment is still uncertain. If it represents a late-stage fluid that evolved from extended fractionation of carbonatite magma, then the possibility exists that a plug of similar enrichment may be present in the subsurface. Alternatively, this HREE event may record expulsion of late hydrothermal fluids that re-distributed HREE already present in the rocks, in which case the primary targets will continue to be in the structures.

9 EXPLORATION

NMI began exploration on EPL 3400 in 2006. Exploration to date includes:

- Regional and detailed exploration between 2006 and 2008 for copper and gold, targeting an IOCG model.
- Regional assessment of the REE potential of the Lofdal Carbonatite Complex beginning in 2008 and continuing to the present.

9.1 Copper – Gold Exploration: 2006 – 2008

NMI was originally attracted to the Lofdal area as an IOCG target (Lobo-Guerrero, 2005). The model was predicated on the presence of copper sulphides associated with magnetite in the matrix of hydrothermal breccias and sulfidation accompanying magnetite and hematite in quartz veins possibly related to REE, Th, U and P bearing carbonatite dykes.

The IOCG exploration program consisted of:

- A comprehensive structural and satellite mapping exercise over an area of more than 10,000 km². The work was carried out under contract by the NPA Group of consultants, London, UK. Additional Landsat images were obtained and the Landsat thematic mapper TM images with two spectral bands (bands 5 and 7) in the Short-Wave Infrared (SWIR) were used to identify targets/outcrops of hydrothermal alteration and to delineate major structural features.
- Targets generated by the NPA study as well as other areas of interest were systematically sampled by NMI personnel who collected a total of 2,371 rock grab samples. During the latter part of this phase of exploration in 2008, the first 255 samples were collected from the Lofdal Carbonatite Complex.
- Areas returning anomalous Au values were sampled further, and a program of detailed geological mapping and trenching was undertaken to assess the most prospective targets. Three of these targets were tested by reverse circulation drilling in 2008 but no significant Cu-Au mineralization was found.
- An orientation stream sediment geochemical program was undertaken by NMI personnel in the area of the former Lofdal Copper Mine but did not generate any significant targets.

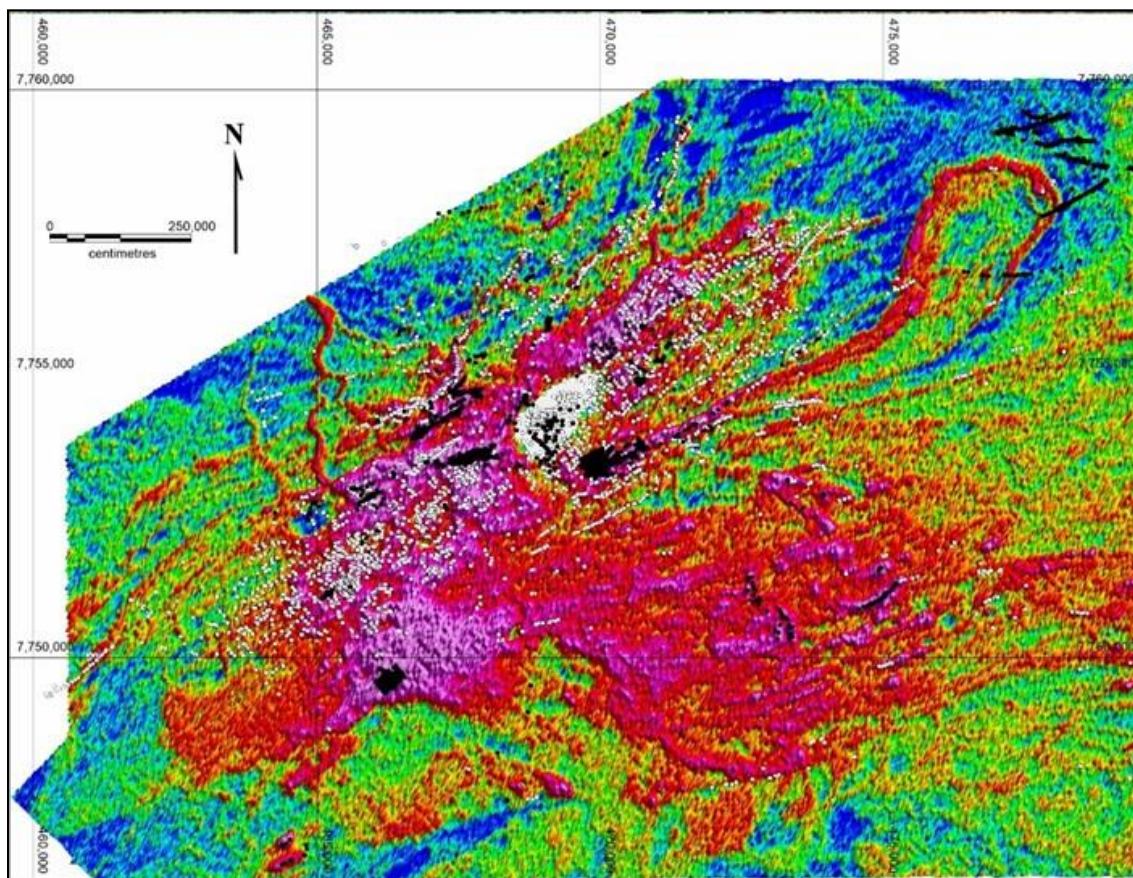
Virtually all of this exploration was outside of the current area of interest for REE mineralization and few of the data are directly applicable or relevant to the current project. Where the 2006 to 2008 data are relevant to the REE exploration, they are noted in the following sections.

9.2 Regional Assessment of Rare Earth Element Potential

9.2.1 Geological and Lithogeochemical Survey

Lobo-Guerrero (2005) recognized and recommended to NMI the potential for REE mineralization associated with carbonatite dykes in the area. This potential was reinforced by the results of investigations by the GSN on the carbonatite. The company initiated an exploration campaign to test this potential towards the middle of 2008.

The initial regional field surveys of the Lofdal Carbonatite Complex were carried out by NMI personnel in two field campaigns from 2008 to February 2010. The aim was to systematically map and sample REE mineralization within the Lofdal Carbonatite Complex. The area of interest was defined by an airborne radiometric high (dominantly thorium) that NMI personnel interpreted to potentially define the extent of the intrusive complex. Sampling during these two campaigns covered roughly the northwestern half of the thorium anomaly, including the areas of known REE mineralization (Figure 9-1).



Note: Samples taken before early 2010 are indicated by white dots, samples taken after late 2010 are indicated by black dots; Background map shows Th radiometric counts from the 2010 airborne survey; Red and purple colours represent high values and define the extent of the Th anomaly associated with the Lofdal Carbonatite Complex. Intrusion (Carbonatite) shown in grey.

Source: Base map from New Resolution Geophysics, 2010

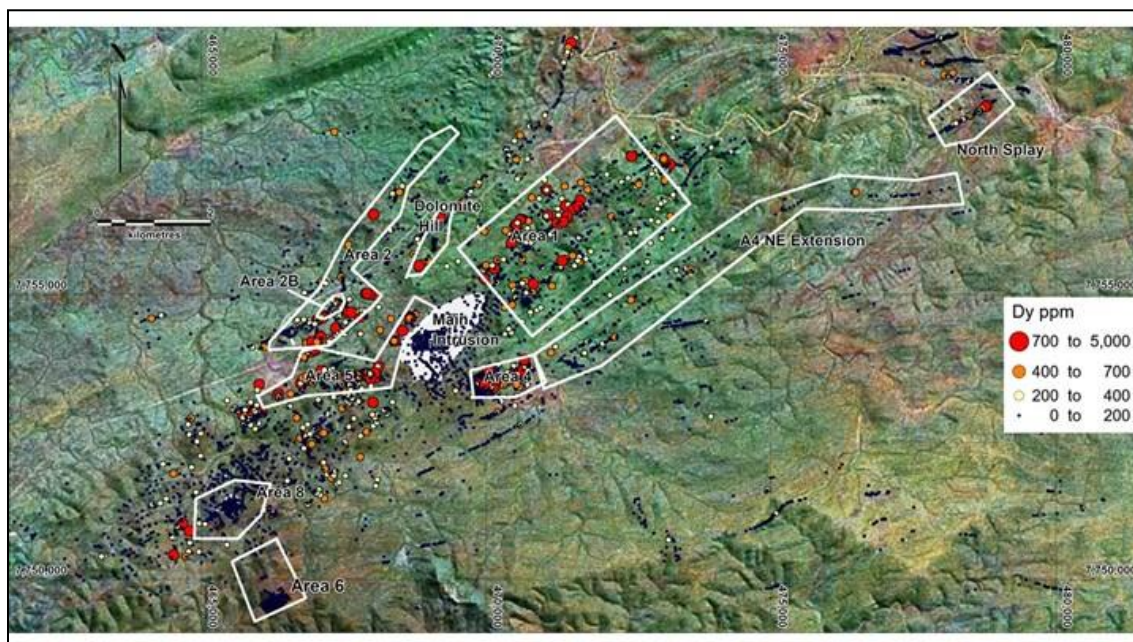
Figure 9-1 Distribution of Regional Lithogeochemical Samples in the Lofdal Area

In addition to the regional traversing, detailed sampling on approximately 50 m intervals was carried out over the Emania intrusion and portions of the Main intrusion with 217 samples collected from the Emania intrusion and 171 from the Main intrusion. A total of 3,680 grab samples were taken during these campaigns and the results were discussed in detail by Swinden and Siegfried (2011).

Surface grab samples continued to be a principal exploration tool on the property between 2011 and 2013. Regional geological mapping, coupled with regional airborne geophysics, continued to identify new alteration zones and carbonatite dykes which were systematically sampled to test their mineralizing potential and define drill targets. Systematic surface grab sampling was carried out to better define the mineralized systems throughout the area of interest (Figure 9-1). This sampling totalled approximately 1,900 additional samples. All surface grab samples were analysed at Actlabs in Ancaster, Ontario. The grab samples were analysed for the same suite of trace elements and were subjected to the same QAQC protocols as the drillhole core samples from Area 2B (detailed in Swinden and Siegfried, 2011) and Area 4 (detailed by Siegfried and Hall, 2012).

The lithogeochemical surveys outlined the distribution of the REE mineralization at a district scale. They showed that there is considerable variation in the mineralization of the alteration systems – some being

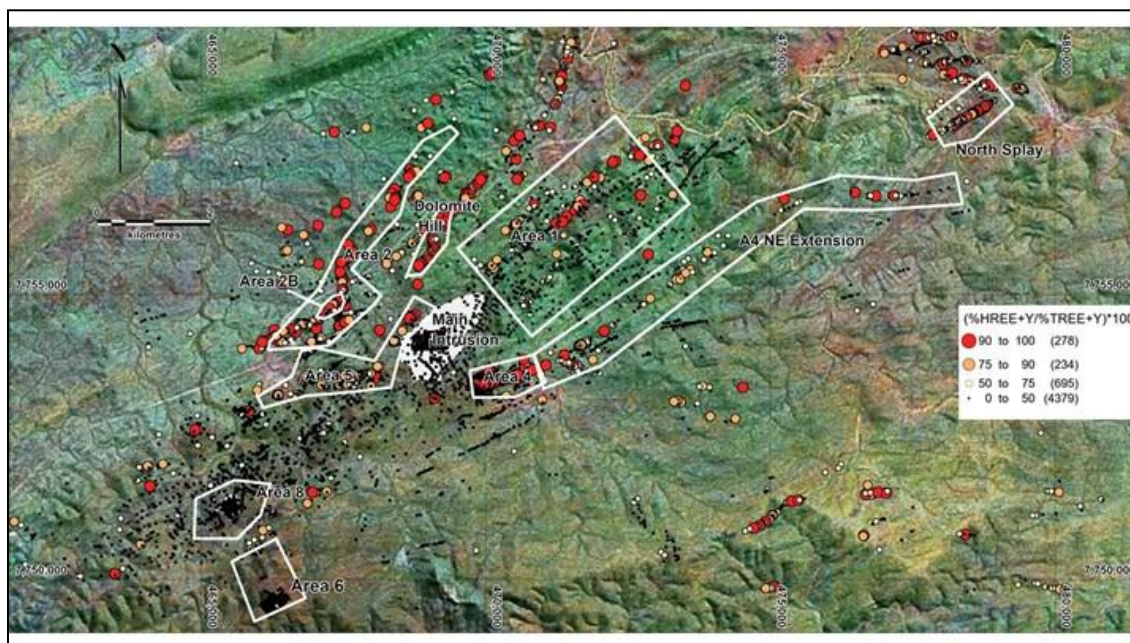
well mineralized and others containing no REE. Within the REE-mineralized zones, there is considerable variation in the grade of REE mineralization at both large and small scales. Mineralization is dominantly hosted by basement structures which have been altered and mineralized by hydrothermal fluids and intruded by carbonatite dykes. The linear spatial distribution of anomalous grab samples reflects the favourable structures, provides further evidence that only certain structures on the property contain significant HREE mineralization (Figure 9-2) and allowed these structures to be traced for considerable distances along strike. The regional geochemical surveys showed that the most favourable structures are notably enriched in the HREE and integration of surface lithogeochemistry with regional geophysical and geological studies resulted in the definition of nine priority exploration areas on the property (Figure 9-2 and Figure 9-3) and were a primary tool in identifying drilling targets within these areas. Priority target areas have all been tested by drilling (see Item 10).



Source: Base map sourced from Landsat Geocover mosaic, 2011

Note: Main intrusion shown in grey for reference.

Figure 9-2 Priority Exploration Areas defined by Dy in Surface Lithogeochemistry Samples



Source: Base map sourced from Landsat Geocover mosaic, 2011

Note: Main intrusion shown in grey for reference.

Figure 9-3 Priority Exploration Areas defined by HREE/TREE Ratio in Surface Lithogeochemistry Samples

9.2.2 Remote Sensing and Regional Geophysics

The results of the regional remote sensing and geophysical surveys were previously reported by Swinden and Siegfried (2011) and are summarized in Table 9-1. NMI has made extensive use of remote sensing data in interpreting the geological relationships on the property and identifying priority exploration targets.

Table 9-1 Summary of Remote Sensing and Regional Geophysical Surveys and Interpretations

Method	Contractor	Objectives	Results
HyMap data analysis – 126 bands from 0.4 to 2.5µm t 4.6m resolution	NamibGeoVista (2010)	Identify hyperspectral signatures characteristic of carbonatite dykes and plugs.	Successfully imaged larger carbonatite and phonolite dykes.
ASTER	NPA Fugro (2010)	Interpret geology, delineate carbonatite bodies.	Data reflect mineralogical characteristics of some basement lithologies. Not effective in targeting individual dykes or intrusions.
Structural interpretation using Landsat/HyMap	NPA Fugro (2010)	Identify major structural features.	Identified and mapped major first order shear zones and other second order structures, some of which are mineralized; provides a structural context for the basement and the Lofdal Carbonatite Complex intrusions.
Integration of remote sensing data with geophysics and geochemistry	NPA Fugro (2010)	Identify priority target areas.	First pass at regional target definition Primary targets identified using magnetic, calcite/iron spectral signatures, structural control, Th radiometric signature and HREE in proximity. Secondary targets as above but lacking strong magnetic signature.
High resolution airborne radiometric and magnetometer survey; helicopter-borne, 75 m line spacing, 3760 line km at 315°Az	NPA Fugro (2010)	Achieve better resolution of radiometric and magnetic features.	Confirmed the radiometric signatures of the carbonatite intrusions and dykes. Confirmed that individual dykes can be traced. Confirmed interpreted structural trends.
Regional ground radiometric, magnetic, gravity profiles (5 lines)	Greg Symons Geophysics (2010)	Test geophysical signatures of intrusions and airborne geophysical targets, test whether gravity identifies known intrusions and can identify buried bodies.	Radiometric and magnetic data respond to individual dykes, consistent with airborne results. Gravity was inconclusive regarding response of carbonatite intrusions or presence of additional carbonatite bodies at depth.

The 2010 high-resolution airborne survey provides high-resolution information that correlates well with existing geological, geophysical and lithogeochemical data for the area. In particular:

- It confirmed the contrasting radiometric signatures of the Main and Emanyia intrusions.
- The resolution was sufficiently high to confirm and enhance the mapping of REE-bearing structures.

- It confirmed interpreted structural trends and allowed more detailed mapping of major structures across the area, particularly where these structures appear to have served as conduits for carbonatite intrusion and mineralization.

In summary, the regional exploration identified multiple, high quality REE target areas and demonstrated significant potential for discovery of deposits of REE associated with the Lofdal Carbonatite Complex. Key results of the regional exploration include:

- Recognition of the district scale thorium anomaly which provides a first order regional target for REE exploration in the Lofdal area.
- Dramatic expansion of the number and extent of known carbonatite dykes and related alteration zones and documentation of their geological characteristics and associated REE.
- Recognition that carbonatites and the associated rocks are extensively hydrothermally altered and variably mineralised with REE.
- Recognition that the HREE-rich mineralization is structurally controlled and that certain structures are preferentially enriched in HREE mineralization.
- Geological and geophysical characterisation of two intrusive plugs in the centre of the complex.
- High resolution geophysical characterisation of the area, interpretation of the regional structural setting of the complex, and recognition of hyperspectral and geophysical signatures that characterize carbonatite dykes and plugs.
- Identification of a number of high-priority target areas for detailed exploration with new targets being generated as field work and compilations continue.

9.2.3 Regional Geological Mapping

Published geological maps for the Lofdal area are at a scale of 1:250,000 (GSN, 2006). This is too broad a scale to be useful for property-scale investigations. Accordingly, detailed mapping of the core of the Lofdal Carbonatite Complex was initiated in 2010 and continued through the latter part of 2013. The mapping was carried out by geologists of South Africa-based Remote Exploration Services Ltd. (RES) on 100 m spaced traverse lines. Extensive use was made of hyperspectral data, which was recognized to closely reflect basement lithologies and areas of hydrothermal alteration. This allowed accurate extrapolation of rock units and alteration zones between and beyond traverse lines. The mapping began with detailed mapping in Areas 2 and 4 to support the planned trenching and drilling in 2010 and then expanded to include the area of the intrusive core of the Lofdal Carbonatite Complex. The mapping continued in 2012 and 2013 in outlying areas of the property and by the end of 2013, the entire extent of the thorium anomaly that defines the exploration area of interest had been geologically mapped. The geological map of the property is shown in Figure 7-4.

Detailed mapping has contributed to the understanding of the geology and therefore the exploration at Lofdal in several important ways:

- Clarified the nature and distribution of basement lithologies.
- Clarified the distribution of intrusive lithologies related to the Lofdal Carbonatite Complex. It demonstrated that the Main intrusion is dominantly syenite at the current level of exposure, but, close to the contact with the underlying white calcite carbonatite, it identified and mapped the distribution of two additional nepheline syenite plugs to the southwest of the Main intrusion and mapped the extent of the Emanyā carbonatite intrusion.
- Mapped the distribution and intensity of phonolite and carbonatite dykes related to the complex.

- Mapped the distribution of Lofdal breccias, showing that they are widespread along the axis of the Lofdal intrusions. The intrusive axis of the Lofdal Carbonatite Complex, as defined by nepheline syenite and carbonatite intrusions and related breccia, occupies a strike length of more than 6 km.
- Showed that most intrusions have a halo of fenitisation and that hydrothermal alteration in the form of albitisation can be mapped out along some major structures for several kilometres along strike. The combination of alteration, anomalous HREE geochemistry and radiometric anomalies related to Th have been important in identifying priority exploration targets in the complex.
- Traced hydrothermal alteration zones along strike well beyond their previously known extent and identified a number of new albitite-carbonate alteration systems, particularly in the northeast and southeastern parts of the property, that comprise exploration targets for further concentrations of HREE.

9.3 Target Exploration in Area 2B

The regional assessment of the REE potential of the Lofdal Complex led to an initial selection of Areas 2 and 4 for more detailed exploration (Figure 9-2). The areas were chosen on the basis of the presence of carbonatite dykes and albitic/carbonatitic alteration zones with significant widths in outcrop, a high relative proportion of samples with anomalous REE values (particularly high HREE/TREE ratios), and geophysical / hyperspectral signatures potentially indicating the presence of more extensive zones of carbonatite. The initial focus of this work was the prominent carbonatitic and albitic alteration zone in Area 2B. The exploration of this zone comprised detailed geological mapping, lithogeochemistry, and trenching and was largely completed during 2010. This work was described in detail by Swinden and Seigfried (2011) and is summarized in the following sections:

9.3.1 Geological Mapping and Lithogeochemistry

Area 2B contains a segment of a regionally significant, ± 3 km long, carbonatite dyke – alteration system in which almost 50% of surface samples contain more than 0.5% TREE and more than 25% of samples contain more than 0.1% HREE. Regional mapping defined the overall setting of the zone and detailed mapping in Area 2B defined a mineralized alteration zone that is exposed over significant widths (up to 70 m at surface) along a strike length of more than 650 m. The detailed mapping showed that the Area 2B alteration/carbonatite zone strikes approximately 060° and dips steeply to the SE, cutting the structural grain of the basement at a high angle. The basement comprises mainly interlayered amphibolite and quartzo-feldspathic gneiss, locally cut by coarse grained pegmatite. Abundant carbonatitic dykes in this area vary widely in orientation but there is a dominant northeasterly strike similar to the mineralized alteration zone.

Detailed mapping showed that the alteration is dominantly expressed as an early pervasive albitization, which has been brecciated and overprinted by an albite-carbonatite-mica assemblage. There are abundant breccias in which angular albitite fragments are surrounded by carbonatite, indicating that the alteration zone was a brittle fracture zone throughout its active history.

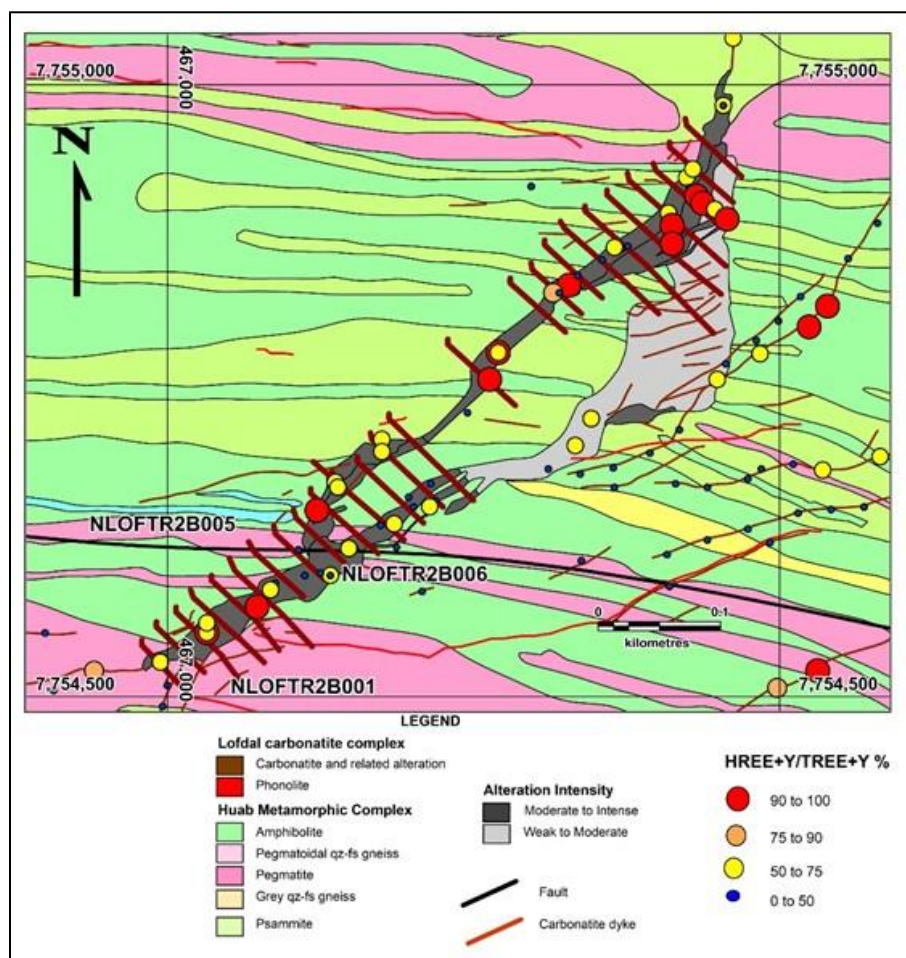
The zone was initially targeted by the results of regional grab sampling. A second round of more detailed surface sampling resulted in a surface suite of approximately 50 grab samples. These samples were intended to be broadly representative of the mineralized zone but were not taken on a systematic grid pattern so do not provide a complete picture of the grade distribution in the surface exposure of the alteration zone. The second round of sampling confirmed the anomalous nature of the alteration system overall and the fact that the REE are very unevenly distributed within the alteration. The five highest-grade analyses from these samples are presented in Table 9-2.

Table 9-2 Analyses of the 5 Highest-Grade Surface Samples in Area 2B

Sample	La ppm	Ce ppm	Pr ppm	Nd ppm	Sm ppm	Eu ppm	Gd ppm	Tb ppm	Dy ppm	Ho ppm	Er ppm	Tm ppm	Yb ppm	Lu ppm	Y ppm	HREE %	TREE %
NLOFR1636	1630	2560	243	1060	765	320	954	182	994	217	544	82.3	417	61.5	4840	0.86	1.48
NLOFR1653	1035	1640	149	526	301	174	648	163	945	201	469	67.1	319	43.9	4840	0.78	1.15
NLOFR1652	1640	2430	240	754	237	142	542	138	874	166	400	44.2	205	24.6	4220	0.67	1.2
ERN15349	1470	2060	186	574	254	120	392	81.2	529	110	321	44.4	265	35.5	3070	0.49	0.95
NLOFR1618	862	1385	163	805	764	246	730	99.3	456	74.9	211	30	198	27.8	2390	0.44	0.84

9.3.1.1 Trenching

25 trenches (Figure 9-4, Table 9-3) were dug across the Area 2B alteration zone to determine the distribution and geological setting of the REE in two dimensions. Trenching was carried out using a Bell 315J backhoe/loader (Figure 9-5). Bedrock is typically within 0.5 m of surface, although in parts of some trenches, up to 1.5 m of soil or calcrete cover was encountered. The trenches were then cleaned by hand in preparation for mapping and sampling (Figure 9-5) and metre waypoints were measured and marked by aluminium tags.



Source: Swinden, 2020

Figure 9-4 Trenches on the Area 2B Zone (Heavy Brown Lines) – Trenches Illustrated in Table 9-4 are labelled

Table 9-3 Locational Information for Trenches on the 2B Zone – WGS84; UTM Zone 33S

Trench ID	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Length (m)
NLOFTR2B001	467013.9	7754568.9	957.5	135	77
NLOFTR2B002	467032.8	7754582.8	957.3	135	76
NLOFTR2B003	467046.5	7754596.0	956.6	135	79
NLOFTR2B004	467063.3	7754602.5	956.4	135	87
NLOFTR2B005	467077.7	7754619.1	955.3	135	53
NLOFTR2B006	467102.3	7754628.1	955.1	135	53
NLOFTR2B007	467111.1	7754648.6	953.7	135	60
NLOFTR2B008	467121.7	7754667.5	952.4	135	74
NLOFTR2B009	467126.0	7754694.3	949.9	135	100
NLOFTR2B010	467154.2	7754698.8	949.4	135	81
NLOFTR2B011	467174.4	7754710.6	947.9	135	78
NLOFTR2B012	467189.9	7754725.0	947.2	135	92
NLOFTR2B013	467235.2	7754788.1	945.1	135	73
NLOFTR2B014	467300.3	7754827.7	949.5	135	31
NLOFTR2B015	467307.0	7754852.8	950.4	135	55
NLOFTR2B016	467317.9	7754865.5	951.4	135	58
NLOFTR2B017	467332.3	7754885.3	951.6	135	104
NLOFTR2B018	467346.3	7754895.3	952.9	135	145
NLOFTR2B019	467364.2	7754910.1	953.3	135	122
NLOFTR2B020	467382.9	7754923.9	953.7	135	105
NLOFTR2B021	467403.6	7754932.7	953.1	135	75
NLOFTR2B022	467415.3	7754953.9	951.1	135	72
NLOFTR2B023	467431.9	7754966.7	949.4	135	60
NLOFTR2B024	466982.1	7754542.0	956.7	135	53
NLOFTR2B025	466996.2	7754561.2	957.0	135	60



Source: Swinden, 2010

Figure 9-5 A – Digging Trenches on Area 2B with a Backhoe, B – Cleaning Trenches in Preparation for Sampling and Mapping

All trenches were geologically mapped prior to sampling and total average count radiometric readings were taken for each sample interval using a RadEye personal radiation detector (PRD). Total count K, U, and Th concentrations with a hand-held spectrometer and magnetic susceptibility readings were taken at 1 m intervals.

The trenches were sampled using a hand-held diamond saw to make two parallel cuts approximately four cm apart. Calcrete coatings were removed by hammer prior to sampling. The trenches were then continuously sampled with hammer and chisel by removing the rock between the saw cuts.

The results of trench mapping confirm the fine structure of the carbonatite/alteration system. The mineralized zone is characterised by carbonatite dykes of variable width associated with variably altered schists and gneisses. The alteration is characteristic of the mineralized zones and consists of variable, locally intense, albitization, carbonatization and local concentrations of phlogopite. Channel samples were taken throughout the entire length of the trenches to test whether mineralization is restricted to carbonatites or is related to hydrothermal alteration and occurs in other lithologies as well. The favourable zones of carbonatization, alteration, and mineralization are well outlined by both carbonatite intensity and by total count radiometric signature.

The results of channel sampling and analysis of trench samples were presented in detail by Swinden and Seigfried (2011) and the best intersections are summarized in Table 9-4. The alteration zone is generally characterized by significantly elevated REE contents as well as MnO and P₂O₅, Th, U, Ba, Nb and Zr. Similar patterns of enrichment are seen in the alteration zone in other trenches.

The alteration system and its associated radioactive and geochemical anomalies was encountered to varying degrees in all trenches in Area 2B. However, the assays show that the zone is not consistently mineralized along its length. The best mineralized intersections in terms of both grade and width were encountered in Trenches NLOFTR2B001, 005 and 006. The assay results for these mineralized zones are illustrated in Table 9-4. Quoted intervals are horizontal, not true widths. In all three trenches, REE mineralization is HREE-enriched and developed over widths of several metres.

This trench sample information contributed to the mineral resource model by providing a position to which the mineralised zone can be extended to surface from the shallowest drillholes. The grades of the samples were not used in the block model grade estimation, however, due to biases considered to be the result of very near surface enrichment with very limited vertical extent.

Table 9-4 Assays from Best Trench Intersections -NLOFTR001, NLOFTR005 and NLOFTR006, Area 2B

Trench	From	To	Width (m)	La ppm	Ce ppm	Pr ppm	Nd ppm	Sm ppm	Eu ppm	Gd ppm	Tb ppm	Dy ppm	Ho ppm	Er ppm	Tm ppm	Yb ppm	Lu ppm	Y ppm	HREE %	TREE %
Trench 1	26	28	2	2338	3455	363	1758	1287	472	1574	238	1134	182	382	44	244	32	4479	0.88	1.8
	39	42	2	22	40	5	29	45	26	103	23	155	34	98	14	85	12	1177	0.17	0.19
Trench 5	33	44	11	817	1230	117	483	268	130	502	94	545	97	245	32	184	25	2676	0.45	0.74
	36	40	4	1129	1669	159	675	451	233	919	167	937	159	384	47	260	35	4295	0.74	1.15
	11	12	1	281	459	45	172	51	25	86	17	107	19	47	6	36	5	511	0.09	0.19
Trench 6	29	45	16	648	961	93	407	236	96	355	59	329	57	144	19	113	16	1595	0.28	0.51
	32	36	4	561	887	92	419	313	132	495	81	434	72	175	23	133	19	1947	0.35	0.58
	39	45	6	1236	1802	170	736	382	152	561	96	536	92	234	30	164	22	2662	0.45	0.89

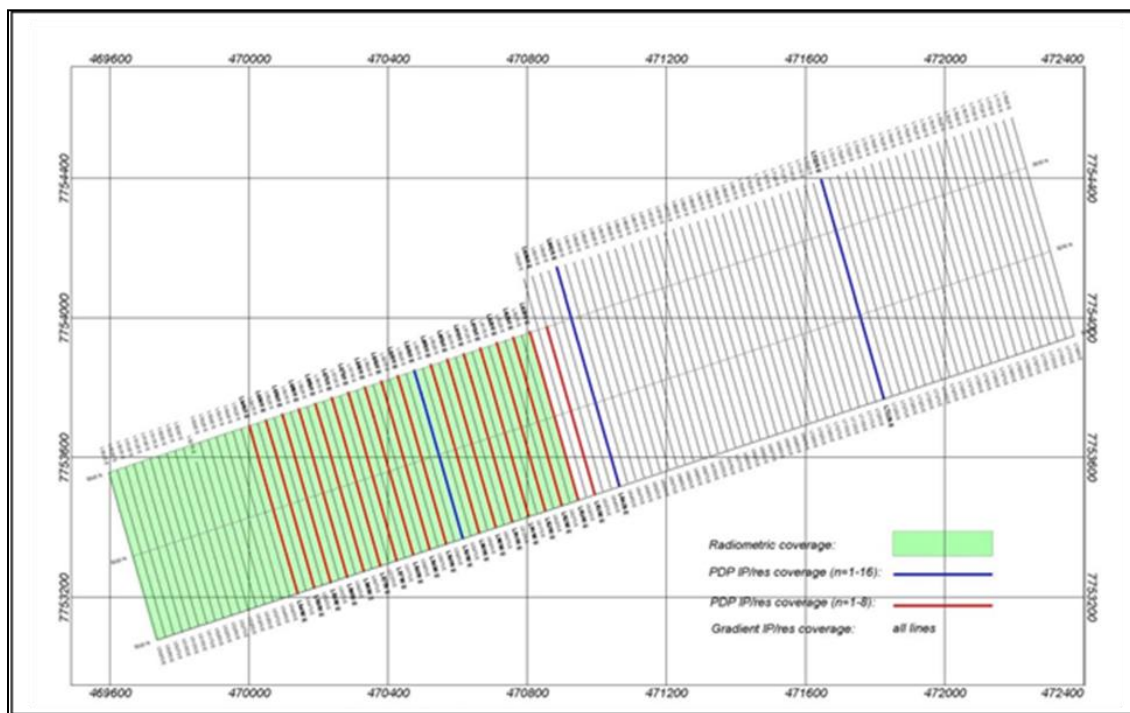
9.4 Target exploration in Area 4

9.4.1 Geological Mapping and Surface Sampling

Geological mapping in Area 4 identified a prominent zone of carbonatization and REE mineralization trending approximately east to west across the area which is particularly enriched in HREE (Figure 7-14 and Figure 7-15). The alteration zone is readily mapped as a zone in which both carbonatites and alteration lithologies are coloured dark brown by the weathering of iron oxides. The zone has both significant width and strike length and is associated with a prominent Th radiometric anomaly. Detailed sampling reported by Swinden and Siegfried (2011) showed that a large proportion of samples from the core of the alteration zone returned very strong HREE-enrichment, and also identified an area in the southwest quadrant of the area which returned a large number of LREE-enriched samples. These samples were intended to be broadly representative of the mineralized zone but were not taken on a systematic grid pattern so do not provide a complete picture of the grade distribution in the surface exposure of the alteration zone. Assays from selected grab samples confirmed that the mineralization is unevenly distributed in the alteration zone. Samples from these two areas returned up to 4.69% TREO with 96.3% HREE enrichment (%HREE/%TREE) and 5.82% TREO with 2.8% HREE-enrichment, respectively.

9.4.2 Ground Geophysics

Ground radiometric and gradient induced polarization/resistivity surveys were carried out across the Area 4 alteration zone on 25 m spaced lines. The radiometric survey was carried out by Greg Symons Geophysics in 2010. The gradient survey was carried out by Remote Exploration Services in 2011 and extended by Greg Symons Geophysics in 2012. The results of this work were described in detail by Siegfried and Hall (2012) and are summarized in Table 9-5 and illustrated in Figure 9-6.



Note: North is upwards, grid line spacing is 400 m for scale
Source: RES, 2011; Symons Geophysics, 2012

Figure 9-6 Area 4 Grid showing Ground Geophysical Coverage

Table 9-5 Summary of Geophysical Surveys in Area 4

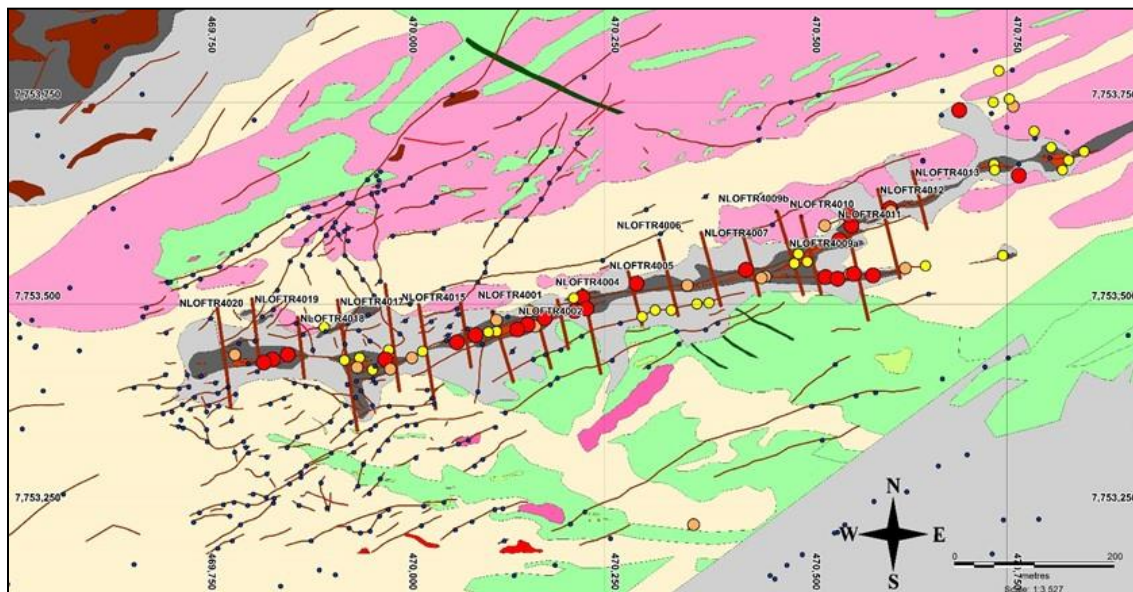
Radiometric Survey	Continuous walking mode = station spacing ~2m	Alteration zone follows a prominent ENE-striking Th anomaly, follows 1 st order sinistral shear
Gradient array induced polarization/resistivity surveys	RES- 25 m line spacing, 25m station spacing and a-spacing; GSG – 50 m linespacing, select lines of pole-dipole array with a=25m	Good agreement between resistivity and inverted 3D PDP. Central low resistivity belts coincide with alteration zone

Geophysical surveys along the Area 4 trend identified a number of priority targets for further investigation both on the central trend that is interpreted to represent the NE extension of the Area 4 fault zone, as well as off the corridor trend. The on-axis targets reflect combinations of low resistivity (suggesting fault structures, particularly cross-cutting or splay structures), strong alteration (mapped) and moderate to strong HREE enrichment in surface samples, and correlation with high Th in ground and airborne geophysics.

9.4.3 Trenching

Twenty trenches were dug across the Main zone in Area 4 to locate and sample the mineralised zone precisely at surface and examine its alteration and geological relationships to the surrounding rocks (Figure

9-7 and Table 9-6). This information contributed to the mineral resource model by providing a position to which the mineralised zone can be extended to surface from the shallowest drillholes. The grades of the samples were not used in the block model grade estimation, however, due to biases considered to be the result of very near surface enrichment with very limited vertical extent.



Note: Trenches are shown with brown lines. The Area 4 alteration zone is indicated by dark grey (intense alteration) and light grey (moderate alteration). Samples are keyed to the ratio HREE/TREE. Geological and sample legends same as Figure 9-4.

Source: Swinden and RES, 2012

Figure 9-7 Location of Trenches in Area 4

The trenches were dug using a JCB backhoe and the width of the trenches was determined by the excavator bucket (Figure 9-8), which was approximately 1 m. Trench endpoints and midpoints were located by GPS. Bedrock is typically within 50 cm from surface although in parts of some trenches up to 1.5 m of soil or calcrete cover was encountered. The trenches were cleaned by hand in preparation for mapping and sampling and metre waypoints were measured and marked by aluminium tags.

All trenches were geologically mapped prior to sampling and total average count radiometric readings were taken for each sample interval using a RadEye personal radiation detector (PRD). Total count K, U and Th concentration readings were taken at 1 m intervals using a hand-held spectrometer and magnetic susceptibility.



Source: Hall, 2012

Figure 9-8 Examples of Trenches in Area 4

Table 9-6 Locational Information for Trenches in Area 4. WGS84, UTM Zone 33S

Trench ID	East_Start (m)	North_Start (m)	East_End (m)	North_End (m)	Length (m)
NLOFTR4001	470136	7753404	470111	7753489	88
NLOFTR4002	470183	7753421	470164	7753494	99
NLOFTR4003	470204	7753445	470192	7753506	75
NLOFTR4004	470245	7753415	470223	7753516	62
NLOFTR4005	470297	7753474	470280	7753539	104
NLOFTR4006	470342	7753486	470319	7753578	67
NLOFTR4007	470395	7753497	470371	7753589	95
NLOFTR4008a	470442	7753510	470425	7753577	95
NLOFTR4008b	470422	7753585	470421	7753592	70
NLOFTR4009a	470496	7753513	470476	7753576	7
NLOFTR4009b	470473	7753585	470465	7753614	66
NLOFTR4010	470520	7753512	470494	7753608	30
NLOFTR4011	470578	7753480	470548	7753612	135
NLOFTR4012	470618	7753532	470589	7753642	113
NLOFTR4013	470651	7753593	470633	7753664	74
NLOFTR4014	470084	7753422	470072	7753502	80
NLOFTR4015	470040	7753370	470020	7753500	131
NLOFTR4016	469996	7753391	469979	7753524	134
NLOFTR4017a	469943	7753344	469935	7753398	54
NLOFTR4017b	469934	7753412	469918	7753504	94
NLOFTR4018	469877	7753408	469869	7753483	75
NLOFTR4019	469821	7753420	469812	7753507	86
NLOFTR4020a	469785	7753373	469783	7753393	21
NLOFTR4020b	469781	7753402	469781	7753415	13
NLOFTR4020c	469778	7753426	469769	7753495	72

The trenches were sampled using a hand-held diamond saw to make two parallel cuts approximately four cm apart. Calcrete coatings were removed by hammer prior to sampling. The trenches were then continuously sampled with hammer and chisel by removing the rock between the saw cuts.

All trenches crossed the Main zone of REE mineralization and alteration, and this zone is marked in each trench by anomalous radioactivity, visible evidence of alteration and geochemical anomalies in the HREE and Y, HREE+Y/TREE+Y, and P₂O₅. In almost all cases, the geochemical and radiometric anomalies coincide closely with the mapped extent of the alteration and mineralization. Representative assays are given in Table 9-7. The detailed sampling from the trenches confirms the preliminary observations of significant grades (%TREE+Y) accompanied by very high levels of HREE-enrichment (66% to >90%) over a continuous strike length of up to 650 m.

Table 9-7 Representative Analyses from Trench Samples, Area 4

Trench	From	To	Width (m)	La ppm	Ce ppm	Pr ppm	Nd ppm	Sm ppm	Eu ppm	Gd ppm	Tb ppm	Dy ppm	Ho ppm	Er ppm	Tm ppm	Yb ppm	Lu ppm	Y ppm	HREE %	TREE %
NLOFTR4001	52	54	2	67	159	22	110	67	30	110	22	141	28	77	11	64	9	939	0.14	0.19
NLOFTR4004	78	94	16	46	82	11	48	67	41	196	46	308	67	190	28	162	23	1993	0.31	0.33
	88	90	2	76	155	19	91	161	107	520	122	866	187	533	77	447	64	5573	0.85	0.90
NLOFTR4005	45	57	12	26	47	8	41	78	52	300	80	568	128	376	53	298	43	4459	0.64	0.66
	49	52	3	20	37	9	60	152	110	683	190	1379	317	938	132	738	106	11084	1.57	1.60
NLOFTR4006	39	66	27	132	232	26	101	52	31	170	47	325	76	220	33	191	28	2483	0.36	0.41
	48	49	1	295	518	58	230	183	129	714	199	1360	304	843	123	701	100	8563	1.30	1.43
	58	59	1	42	93	14	86	169	121	729	211	1520	353	1030	153	884	127	12230	1.74	1.78
NLOFTR4011	112	130	18	154	298	33	127	74	46	264	73	547	127	379	61	366	55	3765	0.57	0.64
	113	114	1	162	305	33	132	110	74	465	125	979	227	659	103	600	87	7342	1.07	1.14
	118	119	1	43	87	11	46	67	58	360	104	828	194	570	90	541	79	5594	0.84	0.87
	126	129	3	301	565	63	244	180	127	801	238	1783	420	1295	213	1279	193	12483	1.88	2.02
NLOFTR4013	35	43	8	304	508	51	176	53	24	119	30	209	47	140	22	139	21	1314	0.21	0.32
	37	38	1	180	318	36	150	99	61	376	105	765	176	513	76	468	68	5137	0.77	0.85

10 DRILLING

Several campaigns of diamond drillhole drilling were undertaken from 2010 to 2020, including detailed mineral resource drilling in Areas 2B and 4, and exploration drilling on a wide variety of targets throughout the EPL. Drilling procedures and results for the earlier campaigns have previously been reported in detail (Swinden and Siegfried, 2011; Siegfried and Hall, 2012; Dodd et al., 2014).

The drilling is discussed in several sections:

- 2010 and 2011 – exploration drilling in Area 2B, which eventually contributed to the mineral resource estimate in this area;
- 2011 and 2012 – mineral resource drilling in Area 4, which led to the definition of the first mineral resource on the property;
- 2020 – mineral resource drilling in Areas 2B and 4 which defined and expanded the existing resource in Area 4 and led to the definition of the Area 2B mineral resource; and
- 2010 to 2020 – exploration drilling which has tested multiple targets but has not led to definition of further mineral resources.

Table 10-1 Lofdal Drilling Summary

Area	2008-2016		JOGMEC		TOTAL PROJECT	
	# Holes	Metres	# Holes	Metres	# Holes	Metres
2 (excl. 2B)	13	1,265			13	1,265
2B	17	2,134	29	4,400	46	6,534
4 Resource	93	10,761	56	10,162	149	20,923
4 Metallurgy	8	1,022			8	1,022
4 NE Extension	17	1,873			17	1,873
5	57	5,787			57	5,787
6	24	4,495			24	4,495
7 (Main Intrusion)	1	240			1	240
8	7	1,021			7	1,021
Northern Splay			10	1,276	10	1,276
Dolomite Hill			4	377	4	377
Total Drilling	237	28,598	99	16,215	336	44,813

10.1 Area 2B, 2010 and 2011 Drilling

The 2010 drilling (Holes NLOFDDH2B001 – 013) was previously reported in detail by Swinden and Siegfried (2011) and is summarized below. The initial drilling was carried out in October and November 2010 and managed by GeoAfrica Prospecting Services. Drilling procedures were governed by a Standard Operating Procedures (SOP) Manual developed for the project by GeoAfrica Prospecting Services and approved by Namibia Rare Earths Ltd. Diamond core drilling on the Area 2B target in 2010 comprised an exploratory phase of 13 drillholes totalling 154.5 m.

Table 10-2 presents a summary of information and procedures with respect to this drilling campaign. Table 10-3 shows the locational and orientation information for these drillholes and Figure 10-1 shows the location of the drillhole collars.

Table 10-2 Summary of Drilling Procedures for the 2010 Drilling Campaign in Area 2B

Purpose of Drilling	Investigate the lateral and vertical extent, geology and grade of the Area 2B alteration zone.
Drill Equipment	Two Atlas Copco Christianson CS14 diamond drill rigs.
Core diameter	NQ (47.6 mm).
Hole Characteristics	All holes drilled at 323° to 330° azimuth and at 50° to 60° degrees dip. See Table 10.1.2 for locational and directional information. The alteration zone was found to dip at ~50° to 60° and intersections were at depths of 25 m to 50 m.
Rig Set-up	Checked by geologist prior to drilling. Holes collared with HQ to about 6 m depth.
Casing	All holes cased to bedrock and casing left in holes.
Site rehabilitation	Area of holes was rehabilitated and a concrete plinth constructed showing UTM coordinates, drillhole number, hole depth, month and year. Top of casing sealed with a closed galvanized iron tube riveted to casing.
Drillhole orientation	All holes were downhole surveyed with a Reflex EZ-TRC system to determine drillhole azimuth and inclination.
Collar Locations	Collars of completed holes were DGPS surveyed by a professional surveyor (WGS84/UTM 33S).
RQD	Run by run recoveries monitored and recorded on a standard rock quality designation (RQD) sheet. RQD and recoveries were captured in the field.
Core Marking	Metre marking of core was done on site. Depth corrections were done by identifying drilling breaks and rejoining/remasuring the core. Zones of core loss were recorded. High points of contacts/layering were used for orienting and marking the rejoined core. An orientation line was marked on the entire length of the core with arrows added frequently to indicate downhole direction
Core Handling	Core was secured in boxes and transported securely by vehicle from the drill site to the core yard in Khorixas.
Core Logging	All holes were logged and sampled by the same qualified geologist. Logging was carried out on standard company logging forms and imported into spreadsheets and a Microsoft Access database. Several holes were radiometrically logged using a RadEye PRD gamma scintillometer. All holes were photographed when wet and then marked with indelible pens for sampling.
Downhole logging contracted to Terratec Geoservices (Namibia)	Three arm caliper/gamma as an initial test for the drillhole integrity.
	Dual compensated density/gamma side walled tool which provided a measure of the density of the material surrounding the drillhole
	Magnetic susceptibility.
	Acoustic televiwer provided a 360° acoustic image of the drillhole with directional information (dip and azimuth) of fractures and layers.
	Down hole spectrometric record of the U, Th and K concentration of the rock.

Table 10-3 Locational and Orientation Information for 2010 and 2011 Drillholes in Area 2B. WGS84, UTM Zone 33S

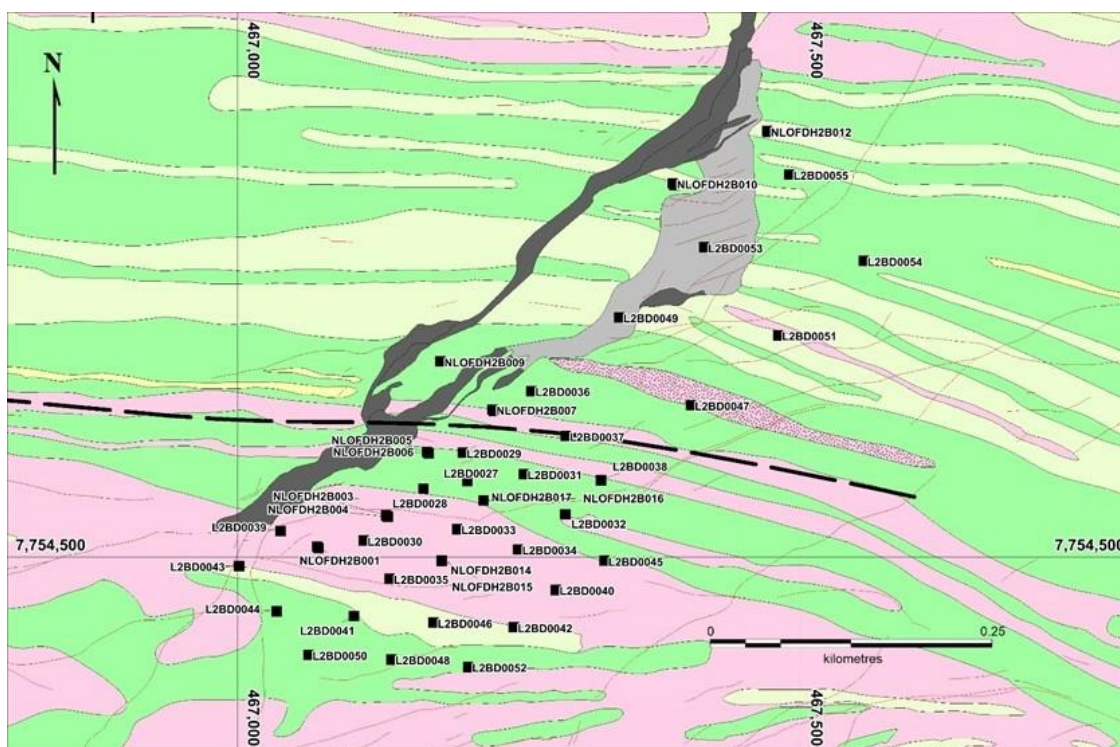
Hole ID	Easting (m)	Northing (m)	Elevation (m)	Azimuth (m)	Dip (°)	Length (m)	Year
NLOFDH2B001	467070	7754509	962.08	316	-50	145.89	2010
NLOFDH2B002	467071	7754508	962.25	320	-65	120.40	2010
NLOFDH2B003	467132	7754537	959.82	318	-50	188.15	2010
NLOFDH2B004	467133	7754536	959.90	318	-65	120.39	2010
NLOFDH2B005	467168	7754594	954.89	318	-50	98.20	2010
NLOFDH2B006	467170	7754592	954.91	318	-65	110.42	2010
NLOFDH2B007	467226	7754631	950.34	311	-50	110.23	2010
NLOFDH2B008	467228	7754630	950.46	311	-65	110.30	2010
NLOFDH2B009	467180	7754674	949.97	315	-50	69.93	2010
NLOFDH2B010	467387	7754832	958.05	311	-50	110.20	2010
NLOFDH2B011	467388	7754831	958.08	314	-65	125.43	2010
NLOFDH2B012	467470	7754879	952.12	311	-50	110.35	2010
NLOFDH2B013	467471	7754878	952.14	311	-65	125.65	2010
NLOFDH2B014	467181	7754497	960.94	318	-55	125.20	2011
NLOFDH2B015	467181	7754497	960.88	318	-80	149.20	2011
NLOFDH2B016	467218	7754550	955.27	318	-55	137.20	2011
NLOFDH2B017	467218	7754550	955.27	318	-70	176.50	2011

Four additional diamond drillholes were drilled in Area 2B in August 2011 (Holes NLOFDDH2B014 – 017). These holes were drilled as part of the 2011 drilling campaign that included the resource drilling in Area 4. Drilling procedures were identical to the Area 4 drilling, governed by the same SOPs, and QAQC for these holes was integrated in and accounted for by the Area 4 QAQC described by Siegfried and Hall (2012). The intention of these holes was to test a part of Area 2B that had returned the best values in the previous year. The results were not deemed to be sufficiently encouraging to continue at that time.

Additional infill sampling was undertaken in 2020 on intervals from five drillholes completed during 2010 and 2011. These samples were assayed as part of the 2020 campaign, and sampling, sampling preparation and assaying were carried out in accordance with the SOP in effect in 2020. QAQC was accounted for as part of the 2020 drilling campaign. Table 10-4 details the intervals sampled.

Table 10-4 Infill Samples from 2010-2011 Drillholes Sampled and Assayed during 2020

Hole ID	From (m)	To (m)	Number of samples
NLOFDH2B007	50.00	55.00	5
NLOFDH2B008	60.30	66.00	5
NLOFDH2B012	17.35	22.70	6
	57.00	64.00	7
NLOFDH2B013	13.00	22.00	9
	67.00	73.00	6
NLOFDH2B015	66.00	73.00	7
	145.00	149.20	5



Note: Geological legend as for Figure 7-4

Source: Swinden, 2014

Figure 10-1 Location of Drillhole Collars in Area 2B

10.2 Area 4 Mineral Resource Drilling, 2011 and 2012

Drilling on Area 4 started in June 2011 and continued until the end of April, 2012. The drilling was managed on-site by Remote Exploration Services (RES) of Cape Town, South Africa, under Standard Operating Procedures dated May 11, 2011 and developed for this project by RES. A total of 101 diamond drillholes

were completed, of which 93 comprised a systematic grid-based assessment of alteration and mineralization in the Main zone and eight were drilled down dip on the Main zone with larger HQ diameter core to obtain material for metallurgical testwork (Figure 10-2). Drilling in this campaign totalled 11,783.6 metres.

The 2011 and 2012 drilling were previously reported in detail by Siegfried and Hall (2012) and Dodd et al. (2014) and are summarized in Table 10-5.

Table 10-5 Summary of Drilling Procedures for the 2011-2012 Drilling Campaign in Area 4

Purpose of Drilling	Investigate the lateral and vertical extent, geology and grade of the Area 4 alteration zone. Resource drilling and metallurgical sampling.
Drilling Contractor	JGM Drilling and Exploration Namibia.
Drill Equipment	Two CF90 platform-mounted diamond drill rigs.
Core diameter	NQ (47.6 mm). Drilling utilized standard and triple tube 4 9/16" HQ and NQ core barrels.
Hole Characteristics	All exploration and resource holes drilled at approximately 345° azimuth and at -55° dip, metallurgical holes at -40° dip. See Table 10.1.2 for locational and directional information. The alteration zone dips southerly at between 45° and 60°. The systematic grid drilling intersected mineralization to vertical depths of between 150 m and 200 m. Four deep holes intersected the zone at 250 m and 300 m vertical depth.
Rig Set-up	Location and inclination checked by geologist prior to drilling. Holes collared with HQ to about 6 m depth.
Core Recovery	>90%; Drillholes with inadequate recovery were re-drilled by the contractor with suffix "B" added to the original drillhole number.
Casing	All holes cased to bedrock and casing left in holes.
Site rehabilitation	Area of holes was rehabilitated and a concrete plinth constructed showing UTM coordinates, drillhole number, hole depth, month and year. Top of casing sealed with a closed galvanized iron tube riveted to casing.
Drillhole orientation	All holes were downhole surveyed with a Reflex EZ-Shot system to determine drillhole azimuth and inclination.
Collar Locations	The collar positions and elevation were surveyed by a professional surveyor with a real-time kinematic ("RTK") GPS. (WGS84/UTM 33S).
RQD	Run by run recoveries monitored and recorded on a standard rock quality designation (RQD) sheet. RQD and recoveries were captured in the field.
Core Marking	Metre marking of core was done on site. Depth corrections were done by identifying drilling breaks and rejoining/remasuring the core. Zones of core loss were recorded. High points of contacts/layering were used for orientating and marking the rejoined core. An orientation line was marked on the entire length of the core with arrows added frequently to indicate downhole direction.
Core Handling	Core was transported securely from the drill site to the exploration camp on a daily basis.
Core Logging	Geological and geophysical logging was carried out by RES and NMI geologists and followed a comprehensive protocol. All holes were radiometrically logged using a RadEye PRD gamma scintillometer and magnetically logged using a hand-held magnetic susceptibility meter. All holes were photographed when wet and then marked with indelible pens for sampling.
Downhole logging contracted to Terratec Geoservices (Namibia)	Three arm caliper/gamma as an initial test for the drillhole integrity.
	Dual compensated density/gamma side walled tool which provided a measure of the density of the material surrounding the drillhole.
	Magnetic susceptibility
	Down hole spectrometric record of the U, Th and K concentration of the rock.

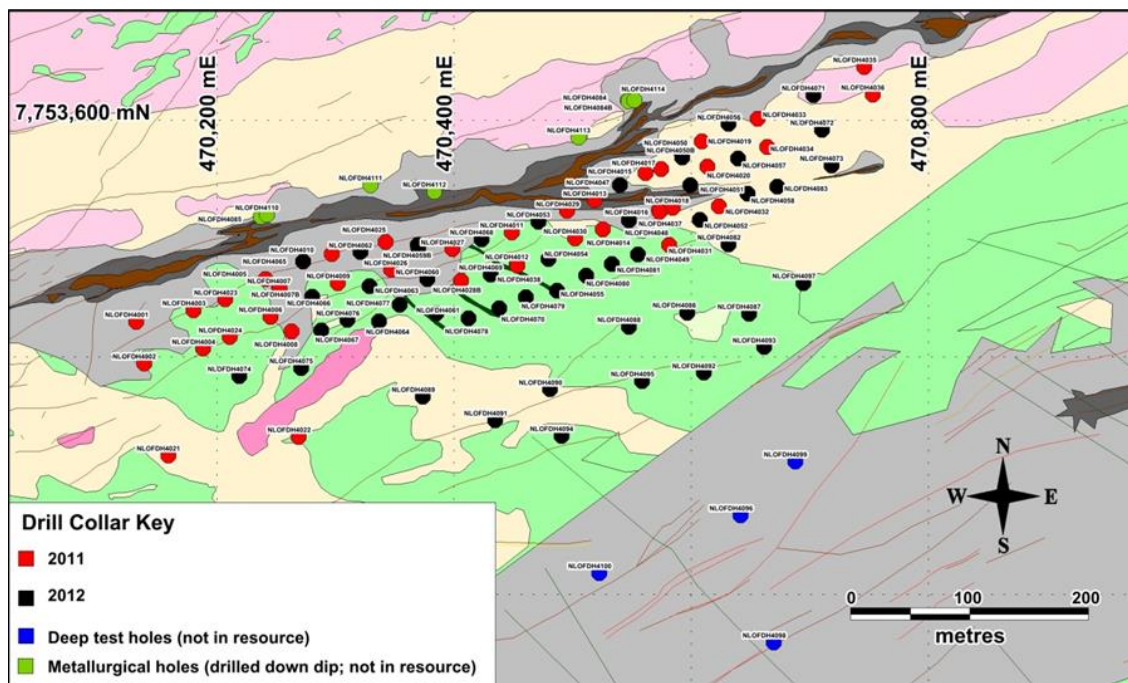
**Table 10-6 Location and Orientation Data for 101 Diamond Drillholes in Area 4. WGS84
UTM Zone 33S**

Drillhole ID	Easting (m)	Northing (m)	Elevation (masl)	Azimuth (°)	Dip (°)	Depth of hole (m)	Comment
NLOFDH4001	470131.93	7753429.83	969.06	345	-55	50.3	
NLOFDH4002	470138.51	7753394.70	965.52	345	-55	80.3	
NLOFDH4003	470180.24	7753439.65	969.98	345	-55	50.4	
NLOFDH4004	470188.37	7753407.26	967.11	345	-55	72.0	
NLOFDH4005	470240.91	7753465.91	971.55	345	-55	53.4	
NLOFDH4006	470245.05	7753434.02	969.03	345	-55	74.3	
NLOFDH4007	470252.29	7753459.97	971.17	345	-55	50.3	
NLOFDH4007B	470252.68	7753458.79	971.09	345	-55	62.4	
NLOFDH4008	470263.05	7753421.77	967.97	345	-55	113.2	
NLOFDH4009	470301.96	7753462.67	970.04	345	-55	83.4	
NLOFDH4010	470296.88	7753486.91	971.57	345	-55	51.7	
NLOFDH4011	470448.4	7753505.22	963.76	345	-55	55.4	
NLOFDH4012	470452.69	7753478.89	962.81	345	-55	80.2	
NLOFDH4013	470518.42	7753532.36	959.19	345	-55	56.2	
NLOFDH4014	470525.09	7753507.85	958.79	345	-55	83.2	
NLOFDH4015	470561.49	7753555.36	957.05	345	-55	68.3	
NLOFDH4016	470572.61	7753524.59	955.07	345	-55	85.5	
NLOFDH4017	470574.31	7753558.69	957.63	345	-55	62.3	
NLOFDH4018	470584.18	7753526.26	955.43	345	-55	80.	
NLOFDH4019	470608.69	7753582.03	959.12	345	-55	41.20	
NLOFDH4020	470613.46	7753560.89	957.67	345	-55	74.4	
NLOFDH4021	470158.81	7753317.04	962.43	345	-60	143.1	
NLOFDH4022	470268.94	7753332.88	962.5	345	-55	150.0	
NLOFDH4023	470207.04	7753448.83	970.42	345	-55	53.2	
NLOFDH4024	470210.76	7753417.00	966.77	345	-55	82.6	
NLOFDH4025	470342.03	7753497.28	969.03	345	-55	56.2	
NLOFDH4026	470346.81	7753473.85	967.49	345	-55	86.4	
NLOFDH4027	470398.48	7753491.20	965.92	345	-55	56.2	
NLOFDH4028B	470399.00	7753457.00	967.81	345	-55	89.4	
NLOFDH4029	470495.24	7753523.40	960.77	345	-55	56.6	
NLOFDH4030	470501.97	7753500.40	960.20	345	-55	107.2	
NLOFDH4031	470581.31	7753494.82	955.05	345	-55	200.3	

Drillhole ID	Easting (m)	Northing (m)	Elevation (masl)	Azimuth (°)	Dip (°)	Depth of hole (m)	Comment
NLOFDH4032	470623.19	7753527.25	955.72	345	-55	83.6	
NLOFDH4033	470656.06	7753601.05	957.98	345	-55	56.4	
NLOFDH4034	470664.00	7753577.46	956.27	345	-55	77.4	
NLOFDH4035	470745.78	7753644.75	955.36	345	-55	62.4	
NLOFDH4036	470752.88	7753621.35	954.24	345	-55	125.4	
NLOFDH4037	470573.05	7753522.83	954.97	345	-55	149.4	
NLOFDH4038	470453.01	7753477.42	962.66	345	-55	80.3	
NLOFDH4047	470539.59	7753545.19	957.33	345	-55	80.2	
NLOFDH4048	470547.31	7753515.55	957.31	345	-55	130.3	
NLOFDH4049	470554.76	7753486.78	956.47	345	-55	176.3	
NLOFDH4050	470591.03	7753573.41	958.19	345	-55	38.4	
NLOFDH4050B	470592.21	7753568.60	958.08	345	-55	83.4	
NLOFDH4051	470599.08	7753545.09	956.63	345	-55	131.5	
NLOFDH4052	470607.08	7753516.12	954.95	345	-55	170.5	
NLOFDH4053	470470.94	7753514.33	962.21	345	-55	80.3	
NLOFDH4054	470479.49	7753482.91	961.2	345	-55	131.4	
NLOFDH4055	470486.85	7753456.06	960.52	345	-55	170.4	
NLOFDH4056	470631.38	7753596.69	959.03	345	-55	81.0	
NLOFDH4057	470639.3	7753567.56	957.00	345	-55	134.1	
NLOFDH4058	470647.17	7753538.09	955.44	345	-55	170.3	
NLOFDH4059	470369.17	7753494.60	967.32	345	-55	43.5	
NLOFDH4059B	470369.69	7753492.49	967.18	345	-55	65.3	
NLOFDH4060	470377.19	7753465.87	965.46	345	-55	86.3	
NLOFDH4061	470385.13	7753437.01	963.66	345	-55	110.2	
NLOFDH4062	470320.96	7753488.63	970.03	345	-55	62.3	
NLOFDH4063	470328.56	7753459.77	968.09	345	-55	86.3	
NLOFDH4064	470336.48	7753430.35	966.76	345	-55	110.4	
NLOFDH4065	470272.73	7753480.70	971.98	345	-55	62.3	
NLOFDH4066	470280.32	7753451.04	969.95	345	-55	86.2	
NLOFDH4067	470287.81	7753422.68	967.91	345	-55	110.3	
NLOFDH4068	470423.14	7753499.34	964.78	345	-55	62.2	
NLOFDH4069	470430.66	7753469.91	963.26	345	-55	92.2	
NLOFDH4070	470437.87	7753441.16	961.91	345	-55	113.4	
NLOFDH4071	470702.72	7753621.17	955.21	345	-55	62.4	

Drillhole ID	Easting (m)	Northing (m)	Elevation (masl)	Azimuth (°)	Dip (°)	Depth of hole (m)	Comment
NLOFDH4072	470710.01	7753591.89	952.91	345	-55	86.3	
NLOFDH4073	470717.95	7753561.82	952.24	345	-55	110.8	
NLOFDH4074	470218.78	7753384.25	964.32	345	-55	122.3	
NLOFDH4075	470270.89	7753391.02	966.11	345	-55	113.3	
NLOFDH4076	470309.98	7753431.40	967.92	345	-55	110.5	
NLOFDH4077	470354.23	7753444.21	966.34	345	-55	110.3	
NLOFDH4078	470412.07	7753432.93	962.69	345	-55	110.4	
NLOFDH4079	470460.34	7753450.54	961.38	345	-55	110.3	
NLOFDH4080	470511.11	7753469.03	959.44	345	-55	113.4	
NLOFDH4081	470532.76	7753478.72	957.99	345	-55	109.7	
NLOFDH4082	470631.42	7753495.11	953.82	345	-55	116.3	
NLOFDH4083	470672.16	7753543.85	954.52	345	-55	110.3	
NLOFDH4084	470546.38	7753616.67	962.00	165	-40	155.3	Holes drilled HQ for metallurgical testwork
NLOFDH4084B	470546.86	7753615.86	961.91	165	-40	20.1	
NLOFDH4085	470237.01	7753518.56	975.16	165	-40	137.3	
NLOFDH4086	470596.64	7753437.50	954.79	345	-55	260.2	
NLOFDH4087	470648.48	7753436.72	952.20	345	-55	212.3	
NLOFDH4088	470547.14	7753425.67	956.20	345	-55	152.3	
NLOFDH4089	470373.62	7753366.69	962.16	345	-55	150.3	
NLOFDH4090	470480.85	7753372.90	959.12	345	-55	161.2	
NLOFDH4091	470434.65	7753346.81	960.27	345	-55	179.3	
NLOFDH4092	470610.46	7753386.84	955.00	345	-75	227.4	
NLOFDH4093	470661.31	7753408.50	954.07	345	-65	239.4	
NLOFDH4094	470490.55	7753333.83	958.62	345	-65	233.4	
NLOFDH4095	470558.52	7753379.89	956.74	345	-65	227.4	
NLOFDH4096	470641.47	7753266.68	958.31	345	-65	308.4	
NLOFDH4097	470694.46	7753462.47	952.96	345	-55	161.3	
NLOFDH4098	470669.51	7753159.52	959.75	345	-70	401.5	
NLOFDH4099	470687.65	7753312.03	958.03	345	-70	308.5	
NLOFDH4100	470522.29	7753218.11	959.13	345	-70	338.4	
NLOFDH4110	470242.05	7753520.35	975.14	165	-40	149.4	Holes drilled for metallurgical testwork
NLOFDH4111	470329.31	7753545.34	972.56	165	-40	151.8	
NLOFDH4112	470383.12	7753539.79	969.17	165	-45	131.2	
NLOFDH4113	470504.91	7753585.55	959.82	165	-45	152.2	

Drillhole ID	Easting (m)	Northing (m)	Elevation (masl)	Azimuth (°)	Dip (°)	Depth of hole (m)	Comment
NLOFDH4114	470551.79	7753616.79	961.78	165	-40	124.6	
Total metres						11783.6	



Note: Geological legend as in Figure 7-4

Source: Swinden, 2014

Figure 10-2 Plan showing Collars of the 101 Drillholes drilled in 2011 and 2012 in Area 4

10.3 Areas 4 and 2B Mineral Resource Drilling, 2020

A campaign of drilling was undertaken on the Lofdal EPL during 2020 with the objective of defining a mineral resource at Area 2B and expanding the existing mineral resource at Area 4. Gecko Exploration (Pty) Ltd was contracted to oversee and manage the drill program.

Drilling was undertaken in Area 4 between late February, 2020 and early December, 2020. The objective of this work was to extend the mineral resource in this area, particularly along strike to the west and to vertical depths of greater than 200 m. A total of 56 diamond drillholes were completed totalling 10,162.1 metres. Locational and orientation information is given in Table 10-7 and collar locations are shown in Figure 10-3.

Table 10-7 Locational Information for 2020 Drillholes in Area 4. WGS84 UTM Zone 33S

Hole ID	Easting (m)	Northing (m)	Elevation (masl)	Azimuth (m)	Dip (°)	Length (m)
L4D0115	470317	7753312	961	343.0	63	191.80
L4D0116	470346	7753215	959	340.0	65	272.88
L4D0117	470439	7753256	957	338.5	65	245.34
L4D0118	470538	7753267	957	339.0	65	284.88
L4D0119	470564	7753168	961	338.0	67	377.83
L4D0120	470677	7753518	953	342.0	58	177.08
L4D0121	470708	7753414	956	341.0	63	200.08
L4D0122	470731	7753316	958	340.0	65	266.88
L4D0123	470776	7753536	952	342.0	58	224.73
L4D0124	470800	7753645	952	343.5	55	47.78
L4D0125	470851	7753652	948	343.5	55	50.73
L4D0126	470874	7753563	952	343.0	58	131.73
L4D0127	470802	7753441	955	341.0	63	209.75
L4D0128	470827	7753341	956	340.0	65	278.98
L4D0129	470101	7753353	964	343.0	58	125.78
L4D0130	470098	7753411	967	343.5	55	74.78
L4D0131	470126	7753263	966	342.0	63	191.88
L4D0132	470227	7753283	960	342.0	63	194.88
L4D0133	470247	7753196	962	340.0	65	257.88
L4D0134	470150	7753165	968	336.0	65	287.88
L4D0135	470757	7753220	959	340.0	67	368.83
L4D0136	470469	7753144	962	340.0	67	356.88
L4D0137	470032	7753392	967	343.5	55	77.08
L4D0138	470051	7753345	965	342.0	60	110.88
L4D0139	470032	7753392	966	343.5	60	64.38
L4D0140	469949	7753367	969	343.0	58	101.63
L4D0141	469990	7753387	967	343.0	55	89.08
L4D0142	469925	7753416	969	343.5	55	37.06
L4D0143	470001	7753341	966	343.0	58	98.18
L4D0144	469954	7753320	971	340.0	62	143.18
L4D0145	469882	7753411	969	343.5	55	59.18
L4D0146	469906	7753306	976	342.0	63	147.93
L4D0147	469894	7753356	973	342.0	62	112.52

Hole ID	Easting (m)	Northing (m)	Elevation (masl)	Azimuth (m)	Dip (°)	Length (m)
L4D0148	470027	7753233	967	340.0	67	212.88
L4D0149	469833	7753393	971	343.0	57	76.68
L4D0150	469847	7753342	995	341.0	63	115.53
L4D0151	469860	7753296	984	340.0	64	160.88
L4D0152	470054	7753138	955	337.0	70	316.8
L4D0153	469793	7753380	974	343.0	57	73.63
L4D0154	469797	7753333	979	342.0	60	129.73
L4D0155	469933	7753211	982	339.0	70	266.98
L4D0156	469808	7753282	1015	342.0	60	157.78
L4D0157	469836	7753182	992	339.0	69	266.38
L4D0158	469958	7753110	981	336.0	71	353.88
L4D0159	469762	7753270	989	339.0	65	190.83
L4D0160	469742	7753367	983	342.0	60	109.68
L4D0161	470322	7753391	968	342.0	60	134.88
L4D0162	470366	7753421	956	342.0	60	116.13
L4D0163	470415	7753403	980	342.0	60	134.88
L4D0164	470447	7753417	904	342.0	60	128.08
L4D0165	470575	7753399	953	342.0	60	185.88
L4D0166	470501	7753418	953	342.0	60	134.08
L4D0167	470625	7753391	965	342.0	60	200.78
L4D0168	470517	7753357	961	342.0	60	197.08
L4D0169	469862	7753087	965	336.0	71	374.88
L4D0170	470558	7753304	958	339.0	68	263.38

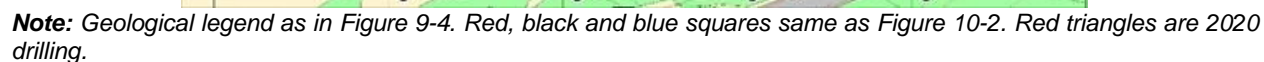


Figure 10-3 Plan showing Drillhole Collars in Area 4

SGS

Table 10-8 Locational Information for 2020 Drillholes in Area 2B, WGS84 UTM Zone 33S

Hole ID	Easting (m)	Northing (m)	Elevation (masl)	Azimuth (m)	Dip (°)	Length (m)
L2BD0027	467204	7754568	952	315	-60	122.78
L2BD0028	467165	7754561	956	315	-60	116.78
L2BD0029	467200	7754593	950	315	-60	107.63
L2BD0030	467112	7754515	958	315	-60	95.88
L2BD0031	467254	7754574	950	315	-60	182.88
L2BD0032	467291	7754538	1040	315	-60	218.88
L2BD0033	467195	7754525	957	313	-60	176.18
L2BD0034	467249	7754507	947	313	-60	188.88
L2BD0035	467135	7754481	976	313	-60	115.88
L2BD0036	467261	7754648	948	315	-60	119.88
L2BD0037	467291	7754608	948	313	-60	179.83
L2BD0038	467323	7754568	948	313	-60	199.98
L2BD0039	467038	7754524	961	315	-60	41.75
L2BD0040	467283	7754471	954	313	-60	218.13
L2BD0041	467104	7754448	967	313	-60	122.78
L2BD0042	467245	7754438	959	312	64	203.93
L2BD0043	467001	7754492	961	315	-60	52.78
L2BD0044	467035	7754452	966	315	-62	100.98
L2BD0045	467326	7754497	952	313	-64	253.63
L2BD0046	467174	7754442	965	313	-62	161.08
L2BD0047	467403	7754635	953	313	-62	190.88
L2BD0048	467137	7754409	969	313	-64	173.83
L2BD0049	467339	7754713	950	313	-62	97.98
L2BD0050	467063	7754413	970	313	-62	137.83
L2BD0051	46748	7754697	954	311	-64	178.88
L2BD0052	467205	7754403	965	311	-64	200.93
L2BD0053	467414	7754775	954	313	-68	104.38
L2BD0054	467556	7754763	951	311	-64	209.88
L2BD0055	467490	7754840	951	313	-68	125.28

10.3.1 Area 4 and 2B Diamond Drilling Procedures

Diamond drilling during 2020 was governed by Standard Operating Procedures developed for Namibia Critical Metals Inc. by Gecko Exploration (Pty) Ltd. and approved by MSA. Diamond core drilling was undertaken by Günzel Drilling of Namibia with two diamond drill rigs; an Atlas Copco CS14 and an Atlas Copco CS1000. The first three to six metres of each hole were drilled with HQ diameter (63.5 mm) effectively collaring the hole and allowing casing to be inserted. The remainder of each hole was usually completed at NQ diameter core (47.6 mm). All holes in Area 2B were drilled towards an azimuth of 310° to 315° at dips between 60° and 68°. Those in Area 4 were drilled towards an azimuth of approximately 340° at dips between 55° and 71°. The collar position and length of each drillhole, together with the azimuth and inclination, are presented in Table 10-7 and Table 10-8.

All of the 2020 diamond drillhole cores were orientated and an orientation line was marked on the core to guide structural measurements.

10.3.2 Core Recovery

The drill advance was marked by a Günzel Drilling technician on depth blocks after each drill run. Metre marking of the core as well as rock quality designation (RQD) and core recovery measurements were undertaken at the drill site by Gecko technicians. Orientation lines were drawn on the core with arrows indicating the down-dip direction (Figure 10-4). Core recovery was generally very good (>95 %). Core boxes were transported by vehicle daily from the drill site to the logging facility at the Lofdal base camp. Core was carefully loaded and ratchet strapped for transportation.

At the Lofdal camp, the core was logged radiometrically using a RadEye PRD gamma scintillometer and visibly altered sections were checked with an Olympus Delta 50 portable XRF to ensure that all mineralized sections were identified and sampled. Following geological logging, the core was sampled for assay.



Source: Ellmies, 2020

Figure 10-4 Drillers' Metre Marks, measured Metre Marks and Orientation Lines on Uncut Core

10.3.3 Collar and Downhole Surveys

The drillhole collar positions were pegged by a geologist using a handheld global positioning system (GPS) set within WGS84, UTM Zone 33S coordinate system. The senior geologist then verified the correct

orientation and inclination of the rig derrick prior to drilling. After the completion of each hole, Günzel Drilling carried out downhole surveys using a Reflex EZ-Trac survey tool determining the downhole orientation i.e., the dip and azimuth. The collar positions and elevation were surveyed by Greg Symonds Geophysics with a real-time kinematic (RTK) GPS. The drillhole collars were marked with a concrete beacon recording the relevant details of each hole on a metal plate (Figure 10-5).



Source: Witley, 2021

Figure 10-5 Concrete Plinth over Capped Drillhole L4D017

All sites were rehabilitated by Günzel Drilling according to the site Environmental Management Plan, with foreign material removed and sumps filled and smoothed.

10.4 Interpretation of Drilling Results

10.4.1 Area 4

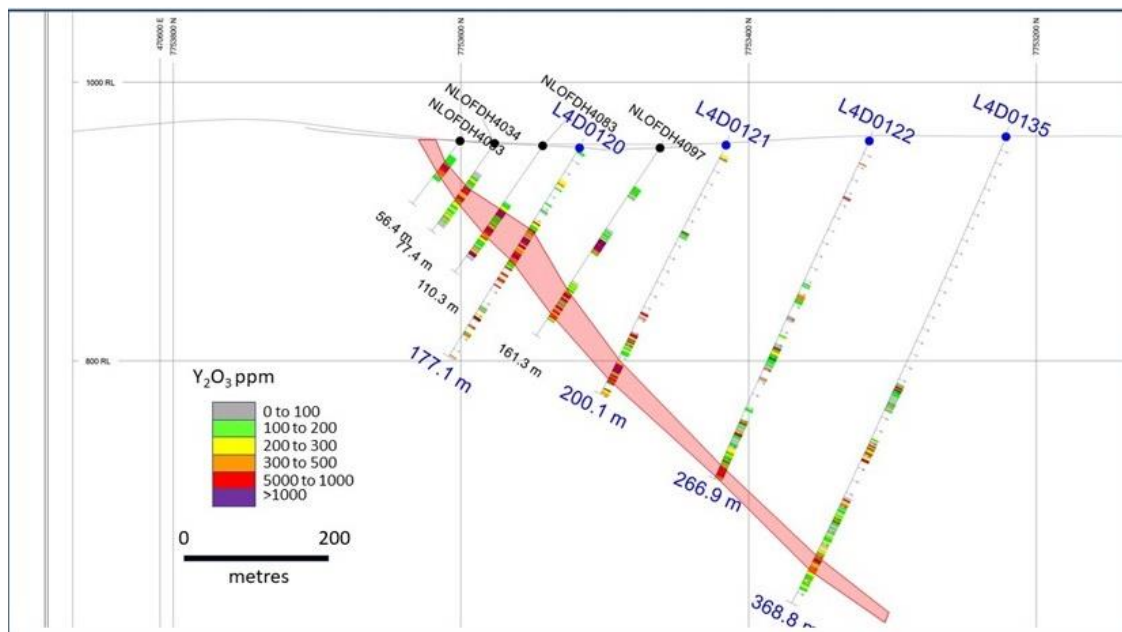
Geological, lithogeochemical and geophysical surveys delineated an ENE-trending, REE-bearing alteration zone in Area 4. This zone was subsequently delineated by the 2011 drilling campaign to a depth of approximately 200 m. The mineralization is associated with a zone of variably intense albitisation and carbonatization that is centered along a major sinistral fault system. The 2011-2012 drilling campaign achieved a nominal drill-hole spacing of 25 m for approximately 650 m along strike and demonstrated the down-dip continuity of the mineralised zone to vertical depths of more than 200 m. Four deep holes were drilled to test the continuity of the zone and it was found to be present at vertical depths of up to 300 m.

As with the 2011 drilling, the 2012 infill holes were drilled perpendicular to the strike direction and were angled between 55° and 75° to intersect the southerly dipping mineralization at approximately right angles in an attempt to obtain near true thickness intersections. Eight holes (NLOFDH4084, NLOFDH4084B, NLOFDH4085, NLOFDH4010, NLOFDH4011, NLOFDH4012, NLOFDH4013, and NLOFDH4014) were drilled down dip on the mineralization to recover sufficient drillhole core material for initial metallurgical testwork. The positions of the drillhole collars are illustrated in Figure 10-3. Figure 10-6 shows an example of a typical drill section through the mineral deposit. The results of these drilling campaigns resulted in the declaration of an initial mineral resource in Area 4 (Siegfried and Hall, 2012).

The 2020 drilling campaign was planned to extend the mineral resource both along strike to the west and to greater depths. The earlier 2012 drilling included four deep holes that indicated that the mineralization was continuous to a depth of up to 300 m vertically. However, these holes were not sufficiently closely spaced to be included in the previous mineral resource.

The present drilling has extended the strike length of the altered/mineralized zone to approximately 1.1 km and has intersected the mineralized zone in multiple drillholes below 300 m indicating that the mineralization is continuous to at least to this depth.

The orientation of the mineralized zone in 3D is well established by multiple drill intersections on close-spaced fences. Fence drilling indicated an orientation of between 070°E and 075°E and dips between 45°S and 60°S. This implies that the bulk of the drillholes intersected the targeted mineralization close to a 90° angle and that the difference between sample length and true thickness is therefore relatively minor.



Source: Swinden, 2021

Figure 10-6 Example of a Drilling Section through the Main Zone Mineralization at Area 4

Figure 10-6 illustrates that the Main Zone mineralization has variable grades of REE but consistently contains intervals with high Y concentrations (>0.1% Y).

Additional zones of REE mineralization with variable thickness occur up to 20 m to 40 m below the Main Zone and up to 25 m to 30 m above. These zones have potentially economic merit in an opencast mining scenario.

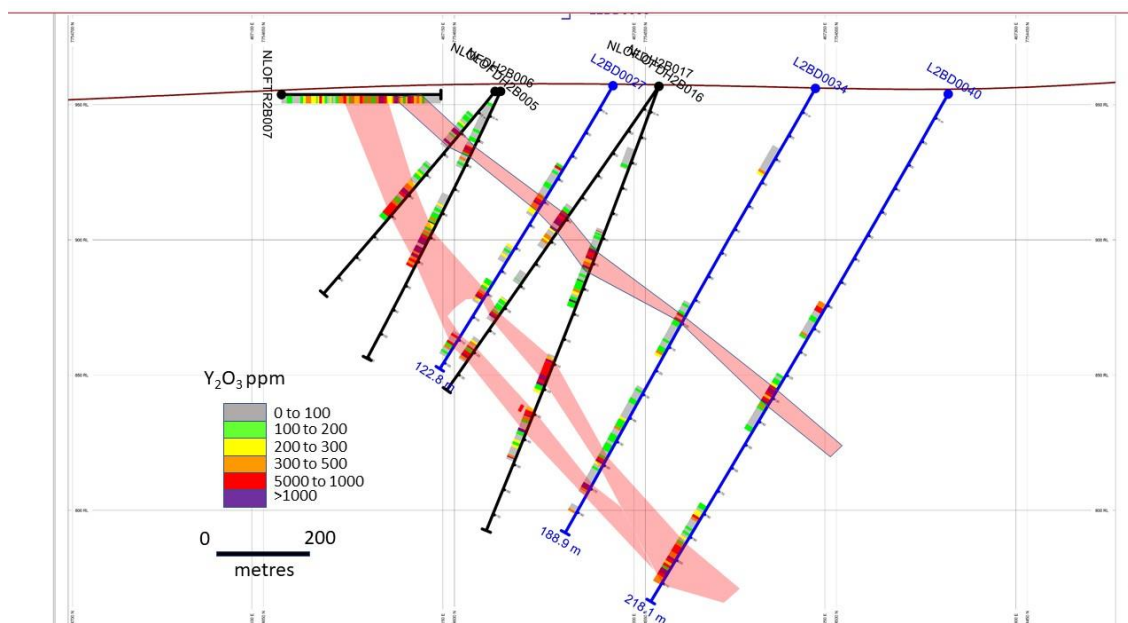
10.4.2 Area 2B

The results of the diamond drilling at Area 2B, coupled with the results of geological mapping and trenching, show that the Area 2B zone represents a portion of a rare earth mineralizing system that is structurally controlled and was formed by both hydrothermal and magmatic (intrusive) events. Significant concentrations of REE (>0.5% TREE+Y) occur in narrow (< 1 m), generally carbonatized, zones of veining, fracture fill and breccia fill related to late hydrothermal activity within broader (10 m to 30 m) zones of albititic and carbonate alteration that are characterized by anomalous concentrations of HREE and Y.

Within the broader Area 2B alteration zone, the mineralization is shown by the drilling to occur in multiple, sub-parallel zones that have been traced from surface to vertical depths of about 200 m (Figure 10-7). Multiple intersections in drilling sections demonstrate that the alteration zones generally strike at approximately 045° to 055° and dip between 45° and 60° to the southeast. All drillholes were oriented approximately perpendicular to the mineralized zone in both plan and section. Intersections are considered to approximately reflect true width.

A key result of the 2020 drilling was to demonstrate that the zone previously indicated by 2010 drilling could be followed along strike between the widely separated holes. The mineralisation is now well defined along a strike length of slightly more than 600 m and to depths of at least 150 m.

Most holes in the Area 2B zone have a large shear zone at or near the footwall of the alteration zone. Most of the alteration occurs in the structural hangingwall. The alteration zone is of variable intensity and is variably mineralized along the Area 2B zone. Although much of the alteration sequence in the Area 2B zone has been enriched in REE, mineralization with significant grades appears to be relatively late in the alteration sequence. The specific mineralized structures are considered to be late veinlets, fracture fill, and breccia fill, and appear to be related to late hydrothermal activity following the main episode of albitization, carbonatization and related hydrothermal alteration.



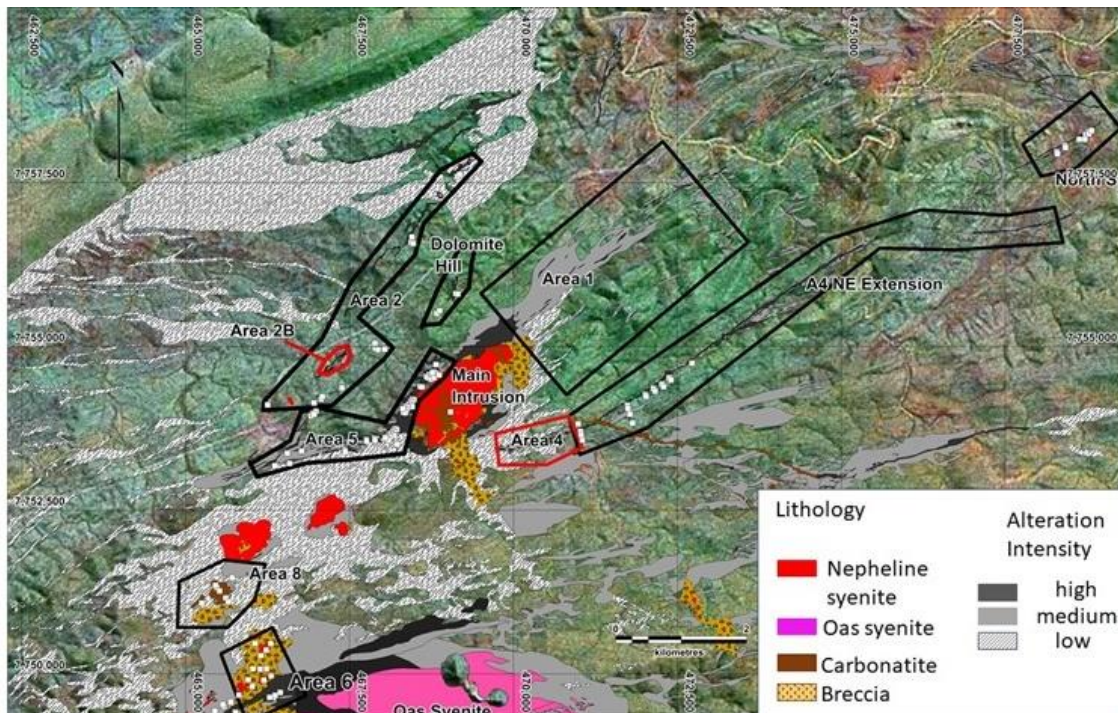
Note: Colour bands are ppm Y_2O_3 . Outline of mineralization and alteration are interpreted from assay data.
Source: Swinden, 2021

Figure 10-7 Example of Drilling Section through the Area 2B Mineralized Zone

10.5 Exploration Drilling Outside the Mineral Resource Areas

10.5.1 Location and Procedures

A total of 133 drillholes have been drilled on the Lofdal EPL that were not part of the mineral resource drilling in Areas 2B and 4. These holes were drilled between 2010 and 2020 on a variety of geological, lithogeochemical and radiometric targets. The location, orientation and length of these holes is given in Table 10-9 and collar locations are shown in Figure 10-8. All holes were drilled using HQ to start followed by NQ core. These drillholes were drilled at the same time as the resource drilling campaigns in 2010, 2011, 2012, 2014 and 2020, using contractors, equipment and Standard Operating Procedures identical to those described for the mineral resource drilling. The comprehensive QAQC program described for the mineral resource drilling was implemented for all exploration drillholes.



Note: Priority exploration areas are outlined by black lines. Resource areas (drillholes not shown) outlined in red lines.
Source: Background is Landsat Geocover Mosaic, 2000. Swinden, 2021

Figure 10-8 Location of Exploration (Non-Resource) Drillholes (White Squares) in EPL3400

Table 10-9 Location and Orientation Information for Exploration Drillholes on the Lofdal EPL. WGS84 UTM 33S

Hole ID	Area	Easting (m)	Northing (m)	Elevation (masl)	Azimuth (°)	Dip (°)	Depth (m)	End-Date
NLOFDH2021	Area 2	466248	7754110	950.371	330	-55	155.4	2012-11-29
NLOFDH2022	Area 2	468436	7756593	951.554	330	-55	62.3	2012-12-01
NLOFDH2023	Area 2	468451	7756670	955.749	300	-55	59.45	2012-11-30
NLOFDH2024	Area 2	468471	7756574	952.013	300	-55	83.3	2012-12-03
NLOFDH2025	Area 2	469141	7757594	933.55	320	-55	140.3	2012-12-06
NLOFDH2026	Area 2	469176	7757625	935.109	320	-55	101.3	2012-12-07
NLOFDH2A014	Area 2A	467890	7754970	947.524	10	-50	98.21	2010-10-22
NLOFDH2A015	Area 2A	467934	7754952	948.043	14	-50	80.3	2010-10-25
NLOFDH2A016	Area 2A	468033	7754959	947.326	8	-50	77.3	2010-10-27
NLOFDH2C017	Area 2C	467385	7754369	964.916	339	-50	88.23	2010-10-30
NLOFDH2C018	Area 2C	467067	7754192	986.665	337	-50	71.5	2010-10-28
NLOFDH2C019	Area 2C	467085	7754235	988.507	295	-50	170.4	2010-10-31
NLOFDH2C020	Area 2C	466959	7754140	976.524	336	-50	77.3	2010-11-03
NLOFDH4039	Area 4 NE Ext.	470984	7753817	951.566	345	-55	110.3	2011-09-08
NLOFDH4040	Area 4 NE Ext.	471736	7754103	959.319	345	-55	113.1	2011-09-09
NLOFDH4041	Area 4 NE Ext.	471023	7753704	952.819	345	-55	101.4	2011-09-10
NLOFDH4042	Area 4 NE Ext.	471775	7753965	965.661	345	-55	101.3	2011-09-11
NLOFDH4043	Area 4 NE Ext.	471050	7753574	950.024	345	-55	164.3	2011-09-12
NLOFDH4044	Area 4 NE Ext.	471818	7753832	975.461	345	-55	151.11	2011-09-13
NLOFDH4045	Area 4 NE Ext.	471028	7753654	952.216	345	-55	152.5	2011-09-14
NLOFDH4046	Area 4 NE Ext.	471746	7754055	963.953	345	-55	152.1	2011-09-15
NLOFDH4101	Area 4 NE Ext.	472008	7754265	952.729	330	-55	80.5	2012-09-07
NLOFDH4102	Area 4 NE Ext.	472133	7754360	963.786	330	-55	68.3	2012-09-12
NLOFDH4103	Area 4 NE Ext.	472263	7754426	973.797	330	-55	74.3	2012-09-11
NLOFDH4104	Area 4 NE Ext.	472376	7754525	977.995	330	-55	101.3	2012-09-10
NLOFDH4105	Area 4 NE Ext.	472399	7754487	974.818	330	-55	155.4	2012-09-14
NLOFDH4106	Area 4 NE Ext.	472023	7754237	952.787	330	-55	82.2	2012-09-15
NLOFDH4107	Area 4 NE Ext.	472158	7754317	956.442	330	-55	89.4	2012-09-18
NLOFDH4108	Area 4 NE Ext.	472289	7754386	965.731	330	-55	113.4	2012-09-19
NLOFDH4109	Area 4 NE Ext.	472724	7754747	976.635	330	-55	62.2	2012-09-21
NLOFDH5001	Area 5	468785	7754614	956.973	305	-55	80.05	2011-06-08
NLOFDH5002	Area 5	468783	7754586	957.114	305	-55	50.3	2011-06-09
NLOFDH5003	Area 5	468712	7754572	960.006	305	-55	68.2	2011-06-10

Hole ID	Area	Easting (m)	Northing (m)	Elevation (masl)	Azimuth (°)	Dip (°)	Depth (m)	End-Date
NLOFDH5004	Area 5	468673	7754540	962.062	305	-55	47.13	2011-06-11
NLOFDH5005	Area 5	468738	7754556	959.106	305	-55	101.2	2011-06-12
NLOFDH5006	Area 5	468576	7754390	967.638	305	-55	53.2	2011-06-13
NLOFDH5007	Area 5	468602	7754380	967.279	305	-55	84.2	2011-06-14
NLOFDH5008	Area 5	468460	7754197	967.535	305	-55	92.2	2011-06-15
NLOFDH5009	Area 5	468481	7754189	968.04	305	-55	137.2	2011-06-16
NLOFDH5010	Area 5	468430	7754162	970.155	305	-55	89.1	2011-06-17
NLOFDH5011	Area 5	468451	7754143	970.337	305	-55	150.2	2011-06-21
NLOFDH5012	Area 5	468400	7754118	972.429	305	-55	86.1	2011-06-22
NLOFDH5013	Area 5	468426	7754104	972.276	305	-55	116.3	2011-06-23
NLOFDH5014	Area 5	468195	7753748	983.625	305	-55	80.2	2011-06-24
NLOFDH5015	Area 5	468216	7753732	985.07	305	-55	104.1	2011-07-05
NLOFDH5016	Area 5	468165	7753705	983.968	305	-55	77.3	2011-07-06
NLOFDH5017	Area 5	468187	7753690	984.888	305	-55	110.3	2011-07-07
NLOFDH5018	Area 5	468128	7753670	981.441	305	-55	77.3	2011-07-08
NLOFDH5019	Area 5	468154	7753655	981.513	305	-55	89.3	2011-07-09
NLOFDH5020	Area 5	467879	7753594	983.669	345	-55	104.3	2011-07-10
NLOFDH5021	Area 5	467886	7753569	981.961	345	-55	128.3	2011-07-11
NLOFDH5022	Area 5	467754	7753578	983.777	345	-55	53.4	2011-07-14
NLOFDH5023	Area 5	467760	7753555	980.96	345	-55	77.4	2011-07-15
NLOFDH5024	Area 5	466911	7753940	962.626	305	-55	50.3	2011-07-16
NLOFDH5025	Area 5	466940	7753977	967.287	305	-55	62.4	2011-07-17
NLOFDH5026	Area 5	466987	7754009	973.681	305	-55	71.4	2011-07-18
NLOFDH5027	Area 5	466960	7753962	970.31	305	-55	86.2	2011-07-19
NLOFDH5028	Area 5	468694	7754524	961.798	305	-55	89.3	2011-07-20
NLOFDH5029	Area 5	468821	7754559	954.766	305	-55	92.3	2011-07-21
NLOFDH5030	Area 5	468820	7754593	955.399	305	-55	98.27	2011-07-22
NLOFDH5031	Area 5	468766	7754539	958.171	305	-55	137.4	2011-07-23
NLOFDH5032	Area 5	468739	7754500	959.657	305	-55	140.3	2011-07-27
NLOFDH5033	Area 5	468790	7754522	956.987	305	-55	140.3	2011-07-28
NLOFDH5034	Area 5	468866	7754565	951.044	305	-55	161.2	2011-07-29
NLOFDH5035	Area 5	468454	7754083	971.553	305	-55	191.3	2011-07-30
NLOFDH5036	Area 5	468361	7754088	973.498	305	-55	77.2	2011-07-31
NLOFDH5037	Area 5	468386	7754068	973.953	305	-55	122.4	2011-08-01
NLOFDH5038	Area 5	468335	7754042	974.685	292	-55	92.3	2011-08-02

Hole ID	Area	Easting (m)	Northing (m)	Elevation (masl)	Azimuth (°)	Dip (°)	Depth (m)	End-Date
NLOFDH5039	Area 5	468365	7754030	973.987	292	-55	122.3	2011-08-11
NLOFDH5040	Area 5	468321	7753996	974.764	292	-55	89.3	2011-08-12
NLOFDH5041	Area 5	468348	7753987	973.826	292	-55	131.5	2011-08-13
NLOFDH5042	Area 5	468180	7753638	980.529	305	-55	161.9	2011-08-14
NLOFDH5043	Area 5	468096	7753630	978.308	305	-55	80.5	2011-08-15
NLOFDH5044	Area 5	468120	7753614	977.561	305	-55	110.3	2011-08-16
NLOFDH5045	Area 5	468068	7753586	974.983	305	-55	71.2	2011-08-17
NLOFDH5046	Area 5	468093	7753575	975.476	305	-55	116.3	2011-08-20
NLOFDH5047	Area 5	468313	7754049	974.978	305	-55	80.4	2011-08-21
NLOFDH5048	Area 5	468763	7754600	958.037	305	-55	50.2	2011-08-22
NLOFDH5049	Area 5	468815	7754723	951.892	305	-55	53.2	2011-08-23
NLOFDH5050	Area 5	467426	7753416	980.268	345	-55	77.3	2011-08-31
NLOFDH5051	Area 5	467432	7753391	979.673	345	-55	107.4	2011-09-03
NLOFDH5052	Area 5	468043	7753607	975.263	305	-55	149.1	2011-09-16
NLOFDH5053	Area 5	468221	7753507	989.782	305	-55	110.0	2011-09-17
NLOFDH5054	Area 5	466360	7753181	963.971	330	-55	152.3	2012-11-12
NLOFDH5055	Area 5	466583	7753215	953.333	330	-55	98.15	2012-11-14
NLOFDH5056	Area 5	466546	7753371	951.566	345	-55	140.1	2012-11-28
NLOFDH5057	Area 5	466765	7753405	954.306	330	-55	215.4	2012-11-16
NLOFDH6001	Area 6	466303	7749643	1031.782	180	-55	122.3	2011-09-01
NLOFDH6002	Area 6	466337	7749643	1030.34	180	-55	122.4	2011-09-02
NLOFDH6003	Area 6	466364	7749678	1026.352	180	-55	122.3	2011-09-18
NLOFDH6004	Area 6	466279	7749644	1031.453	180	-55	122.3	2011-09-19
NLOFDH6005	Area 6	465759	7749415	967.483	330	-55	215.4	2012-09-24
NLOFDH6006	Area 6	465696	7749558	981.915	150	-55	182.4	2012-09-28
NLOFDH6007	Area 6	465688	7749575	982.414	150	-55	169.7	2012-10-04
NLOFDH6008	Area 6	465583	7749507	970.518	150	-55	145.3	2012-10-18
NLOFDH6009	Area 6	465567	7749535	976.727	150	-55	122.0	2012-10-20
NLOFDH6010	Area 6	466136	7750078	1026.085	150	-55	161.1	2012-10-25
NLOFDH6011	Area 6	466227	7750368	1053.167	150	-55	200.3	2013-06-04
NLOFDH6012	Area 6	466440	7749718	1021.493	150	-55	248.2	2013-05-04
NLOFDH6013	Area 6	466193	7749906	1035.132	150	-55	200.1	2013-05-08
NLOFDH6014	Area 6	466233	7750084	1028.034	150	-55	251.3	2013-05-12
NLOFDH6015	Area 6	466027	7750080	1010.091	150	-55	239.2	2013-05-29
NLOFDH6016	Area 6	466147	7750282	1044.728	330	-55	86.3	2013-05-21

Hole ID	Area	Easting (m)	Northing (m)	Elevation (masl)	Azimuth (°)	Dip (°)	Depth (m)	End-Date
NLOFDH6017	Area 6	466056	7750469	1050.965	150	-55	260.3	2013-05-25
NLOFDH6018	Area 6	466300	7750465	1034.338	150	-55	224.3	2013-06-08
NLOFDH6019	Area 6	465812	7749682	1019.716	150	-55	212.3	2013-06-14
NLOFDH6020	Area 6	465856	7749210	973.838	150	-55	167.0	2013-06-17
NLOFDH6021	Area 6	465445	7749436	979.531	150	-55	200.0	2013-06-20
NLOFDH6022	Area 6	466301	7749974	1026.312	150	-55	242.2	2013-07-06
NLOFDH6023B	Area 6	466015	7749814	1026.714	150	-55	221.0	2013-07-12
NLOFDH6024	Area 6	465954	7749914	1013.825	150	-55	257.4	2013-07-19
NLOFDH7001	Main Intrusion	469041	7754001	971.888	0	-90	239.6	2012-09-06
NLOFDH8001	Area 8	465610	7751163	998.082	315	-55	152.3	2011-07-12
NLOFDH8002	Area 8	465478	7751319	988.551	135	-55	152.3	2011-07-13
NLOFDH8003	Area 8	465708	7751470	981.675	0	-90	152.5	2011-07-24
NLOFDH8004	Area 8	465246	7751083	990.618	0	-55	80.2	2011-07-25
NLOFDH8005C	Area 8	465455	7751355	983.709	324	-55	182.1	2012-11-05
NLOFDH8006	Area 8	465516	7751270	995.812	324	-55	149.9	2012-11-07
NLOFDH8007	Area 8	465651	7751102	999.671	324	-55	152.0	2012-11-09
LDD0001	Dolomite Hill	468829	7755507	974.266	310	-60	65.6	2020-07-30
LDD0002	Dolomite Hill	468874	7755548	966.517	330	-80	53.9	2020-08-01
LDD0003	Dolomite Hill	469121	7755809	947.552	310	-55	104.4	2020-08-04
LDD0004	Dolomite Hill	469149	7755802	950.501	310	-55	152.9	2020-08-06
LND0001	North Splay	478726	7758238	943.392	330	-55	77.7	2020-06-26
LND0002	North Splay	478766	7758268	938.433	330	-55	68.7	2020-06-27
LND0003	North Splay	478777	7758247	939.553	330	-55	122.5	2020-07-08
LND0004	North Splay	478814	7758295	934.592	330	-55	83.7	2020-07-10
LND0005	North Splay	478715	7758163	937.839	330	-55	215.9	2020-07-14
LND0006	North Splay	478630	7758163	934.675	330	-55	116.8	2020-07-16
LND0007	North Splay	478409	7758063	932.468	330	-55	86.7	2020-07-18
LND0008	North Splay	478417	7758037	931.765	330	-55	143.8	2020-07-21
LND0009	North Splay	478285	7757971	938.997	330	-55	140.9	2020-07-23
LND0010	North Splay	478297	7757940	933.545	330	-55	218.9	2020-07-28

10.5.2 Exploration Drilling Results

None of the exploration drilling outside of Area 2B and Area 4 has identified mineralization of a size and grade that it could be included as part of the mineral resource and these exploration drilling results are not considered to be material to the mineral resource on the Lofdal EPL. Results are referenced to exploration

priority areas shown on Figure 10-8. Assays of typical significant mineralized intersections are given in Table 10-10.

10.5.2.1 Area 2

Exploration holes were drilled in Areas 2A and 2C as part of the initial drilling campaign that defined the alteration/mineralization zone in Area 2B. These holes targeted narrow exposed carbonatite dykes that yielded individual grab samples exhibiting high grades of HREE. The objective of this drilling was to test different styles and grades of dykes as a prelude to a more comprehensive drilling program in subsequent years. The drilling demonstrated that the narrow dykes exposed at surface in Area 2 did not widen with depth and anomalous mineralization was only present over widths of less than 2 m.

In 2012, six drillholes were drilled to test the northeastern extension of the Area 2B alteration system approximately 2 km to 3 km NE of Area 2B. These holes targeted the alteration/mineralization zone beneath where the best surface grab samples were taken from this part of the zone where albitic and carbonatitic alteration were well developed. Three of these holes intersected narrow widths (3 m to 4 m) of relatively low-grade mineralization, demonstrating that although the zone was carrying REE in this area, it is relatively low grade and discontinuous.

10.5.2.2 Area 4 NE Extension

The alteration zone that hosts the Area 4 mineral resource can be traced along strike in outcrop to the northeast for almost 9 km. Surface grab sampling outlined a number of areas with anomalous REE contents (described in Dodd et al., 2014). Seventeen drillholes were drilled in this zone in 2011 and 2012 to test the width and grade of the zone within approximately 2 km of the east end of the Area 4 mineral resource. Although the alteration was intersected in most holes, REE values were uniformly low grade, and anomalous over less than 4 m of core length.

10.5.2.3 Area 5

Area 5 is an extensive alteration zone with associated REE and thorite mineralization that outcrops along the northwest side of the Main Intrusion. Outcrops at its northeast end returned good REE values and the zone returned anomalous surface grab samples along more than 3.3 km of strike length. Targets were defined by surface outcrops with anomalous grab samples and by radiometric anomalies. 53 drillholes were drilled to test this zone. Mineralization was encountered in most drillholes, but it was found to be inconsistent in grade and width. The best intersections were between 10 m and 15 m wide (not true width; the orientation of the zone is not well defined by the drilling) with moderate grades of REE. However, significant intersections could not be connected between holes on the same section or along strike.

10.5.2.4 Area 6

Area 6 is an aerially extensive zone of fenetization associated with an intense radiometric anomaly in sedimentary rocks immediately north of the Oas Syenite. Surface outcrops are generally not well mineralized and where present, form highly radioactive, very narrow veins. 24 holes were drilled in this area in an attempt to define the nature and extent of any REE mineralization. The drilling found extensive zones of relatively low grade REE over a considerable vertical extent, but no areas were identified with significant grades that could be connected by several drillholes or that would be considered economically interesting. The alteration and mineralization do not seem to occupy consistent structures and there is no indication from the drilling as to whether the mineralized widths are true widths. The alteration contrasts in mineralogy and lithology with other mineralized alteration zones at Lofdal. The REE mineralization is also different, dominated by REE silicates (britholite, allanite), rather than phosphates, and typically associated with fluorite and locally molybdenite.

10.5.2.5 Area 8

Area 8 comprises the Emanyia Plug and a number of nearby carbonatite dykes. There was little indication of mineralization in this area from surface grab samples or radiometrics. Seven drillholes were completed to test the potential of this plug and only very low grade, sporadic mineralization was encountered.

10.5.2.6 Dolomite Hill

Dolomite Hill is a wide zone of alteration immediately north of the Main Intrusion that returned some relatively high grades from surface grab sampling. Four holes were drilled in 2020 to test whether the size and grade of this zone shows improvement with depth. The holes encountered only sporadic mineralization, with few consecutive samples exhibiting anomalous REE mineralisation.

10.5.2.7 North Splay

The North Splay is an outcropping albitite and carbonatite alteration system. It is the most northerly and the most distant mineralization from the Main Intrusion and Area 4. A number of anomalous grade grab surface samples resulted from sampling that occur over a strike length of approximately 1.4 km, as described by Dodd et al. (2014). Despite the fact that some alteration was encountered in the core, there were no REE values of potential economic interest in the core samples.

Table 10-10 Analyses of Typical Significant Altered/Mineralized Intersections in Exploration Drillholes

Hole ID	From m	To m	width m	La ppm	Ce ppm	Pr ppm	Nd ppm	Sm ppm	Eu ppm	Gd ppm	Tb ppm	Dy ppm	Ho ppm	Er ppm	Tm ppm	Yb ppm	Lu ppm	Y ppm	THREE+Y %	TREE+Y %
NLOFDH2A014	38	39	1	493	971	97	381	109	44	158	28	167	29	83	11	75	11	1045	0.16	0.36
NLOFDH2021	105	108	3	193	366	39	160	106	47	173	35	215	37	96	13	78	11	978	0.16	0.25
NLOFDH4104	35	38	4	73	137	15	63	66	29	121	21	111	18	41	5	28	4	437	0.08	0.11
NLOFDH5051	51	62	11	899	1543	163	582	166	55	190	27	135	23	62	9	57	9	616	0.20	0.54
NLOFDH5012	46	61	15	57	117	13	57	92	53	222	43	254	48	125	17	99	14	1258	0.27	0.31
NLOFDH6008	87	94	7	596	883	83	282	96	33	116	19	113	21	59	8	47	6	632	0.24	0.44
NLOFDH8005C	129	130	6	6400	10235	948	2776	263	61	154	18	94	17	43	6	31	4	565	0.26	2.32

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

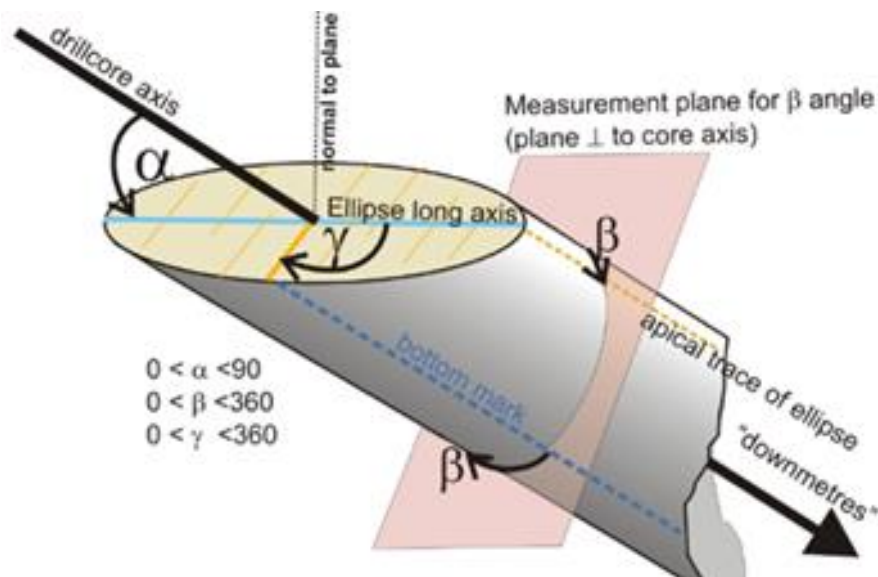
Sample preparation for the 2010 and 2011-2012 drilling campaigns and for lithogeochemical grab sampling has previously been described in detail by Swinden and Siegfried (2011) and Siegfried and Hall (2012). Each drilling campaign had its own set of Standard Operating Procedures. These were similar to those employed in the 2020 drilling and were implemented in accordance with the CIM Best Practice Exploration Guidelines (Refer to Table 10-2 and Table 10-5 for summaries of procedures during these drilling campaigns).

The standard operating procedures (SOPs) for geological and geotechnical logging, core splitting and sampling were compiled by Gecko and reviewed by MSA to ensure that the various activities were carried out in a consistent, transparent, auditable and appropriate manner in accordance with industry standards. The following descriptions refer to procedures followed during the 2020 drilling.

11.1 Drillhole Logging

Geological and geophysical logging (Gamma logging of all core; handheld PXRF for Y on core with anomalous radiometric readings) was carried out by Gecko geo-technicians and geologists and followed a comprehensive protocol.

Structural data, alpha and beta angles (Figure 11-1), were collected on the core to determine the spatial orientation of mineralising and barren vein systems. The alpha angle is the acute angle between the core axis and the long axis of the ellipse (0° - 90° ; Figure 11-1). The beta angle is the angle between the orientation line marking “Top of Hole” as reference line along the core and the ellipse apical trace measured in a clockwise sense (0° - 360° ; Figure 11-1). Alpha and beta angles data were collected using a goniometer in all mineralised zones and other zones as deemed necessary by the logging geologist.



Source: Holcombe, 2016

Figure 11-1 Illustration of the Alpha, Beta, and Gamma Angles in Core

The drillhole cores were logged in detail, recording lithology, alteration, intense near surface weathering / overburden and structure. The weathering at depth is visually indistinct but is evident from core recovery and geotechnical logging (RQD).

11.1.1 Sample Preparation

As mineralization is not always visually discernible, the core intervals to be sampled were determined at the discretion of the logging and sampling geologist using both logging information as well as gamma readings, measured with a RadEye PRD scintillometer, and XRF analyses, from an Olympus Delta 50 or Olympus Vanta handheld XRF analyser (PXRF). The RadEye was used to obtain indicative gamma readings along the entire length of core while in the box and the maximum and minimum values were recorded. Where the in-box gamma values were greater than 50 the core was removed from the box and the RadEye was used to determine gamma values again. PXRF readings for Y were taken along the entire core and the values noted. Both the RadEye gamma and PXRF Y values are indicative and are used solely for the purpose of identifying areas to be sampled for laboratory analysis.

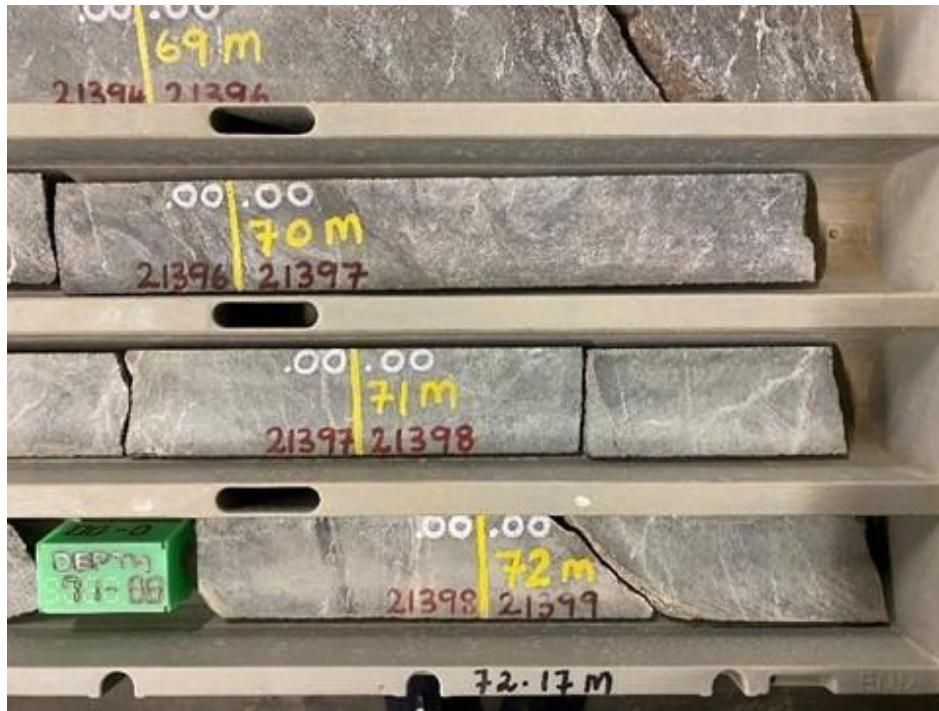
Sampling of the drillhole core was undertaken after metre marking, geological and geotechnical logging, and photographing of the core. All core cutting, sampling, bagging and dispatch procedures were undertaken at the Lofdal field camp. After this work was completed, the remaining core was transported to the warehouse in Khorixas for storage.

Mineralised intervals in the drillhole core were generally sampled at one metre intervals. In cases where lithological changes were observed within a one metre sampling interval, then each lithology was sampled separately, using a minimum 15 cm core length. Sampling was to at least 2 m above and below the zone identified as potentially mineralised. Narrower potentially mineralised zones away from the main zones of mineralisation were also sampled and, at the discretion of the sampling geologist, a single one metre sample was taken either side of the potentially mineralised zone.

11.1.2 Core marking and splitting

Prior to cutting the core, sampling intervals and unique sample numbers (sample ticket book number) were clearly marked above the core orientation line and below the core cutting line drawn on the drillhole core. In instances where the orientation of the core was unknown, then the cutting line and the orientation line were the same line. The start and end of each sample was marked with a yellow line around the core and a white dot on the core cutting line.

A designated geologist responsible for all core sampling carried out the core sampling. The colour convention of yellow (metre mark), white (sampling interval) and red (sample number) was used for all drillholes. (Figure 11-2).



Source: Ellmies, 2020

Figure 11-2 Examples of Drillhole Core Marking

The core was split in half using a commercial core cutter (Figure 11-3) with a 2.2 mm wide diamond core cutting blade. The split halves were returned to the labelled core boxes between the depth blocks and correctly orientated with the aid of the downhole arrows. The logging/sampling geologist checked the core boxes to ensure that all core and core markings were correct prior to removing the sample. The upper half of the core was used for analysis and the lower half of the core was retained in the core tray for future reference or additional testwork. Sample numbers were marked on each individual piece of core with a red waterproof marker and recorded in the customised sampling sheet, which was then captured in the project database.



Source: Ellmies, 2020

Figure 11-3 Core Cutting Device

11.1.3 Core sampling and sample dispatch

Each core sample was assigned a unique number on a wet strength, sequential sample number tag and the sample, generally representing one metre, was consistently taken from the same side of the core relative to the cutting line and placed in a thick plastic sample bag. Two sample number tags were placed in each sample bag, one inside the bag and the other clearly visible to the outside of the bag but located in the folded seal of the bag. Bags were securely sealed with staples and sequentially packed ready for dispatch. Drillhole number and sampling interval were recorded on the sampling book stub and entered into customized sampling sheets which were then digitally captured into the on-site computer. The sampling database was regularly transmitted and backed up at NMI's Windhoek office.

Gaps in the sample sequence were left for standards, blanks and duplicates during the sampling process. The standards and blanks were only packed and labelled with the assigned sample numbers after the core sampling process was completed, in order to minimise the possibility that sample numbers are inadvertently swapped between routine samples, standards or blanks.

The geologist responsible for sampling and dispatch verified the sample numbers and sequence before the samples were packaged in groups of ten into uniquely numbered heavy-duty bags, which were closed with cable ties. The bags were then re-checked against the final sample submission sheet and signed off by the geologist before being loaded and transported in a company vehicle to Actlabs Namibia (Pty) Ltd (Actlabs) in Windhoek.

The Gecko driver of the vehicle signed two copies of an acceptance/transportation sheet specifying the quantity of bags together with sample export documentation for the onward dispatch to Actlabs in Canada.

The samples were dispatched to Windhoek on a weekly or fortnightly basis and all sample transport documentation is filed at the NMI offices in Windhoek.

11.1.4 Density measurements

Rock density measurements using the Archimedes principle (weight in air versus weight in water) were taken for each core sample, after splitting and sampling. Each sample was between approximately 15 cm and 20 cm long. The density device comprises a 3 kg electronic scale, below which a water container was placed (Figure 11-4). A core sample holder attached to the balance was used to immerse core in water in the container. The method was as follows:

- the balance was reset to 0.00 g before each reading;
- a dry length of core was placed in the core holder and the mass of the core in air was recorded;
- the container was filled with water to submerge the sample and the mass of the core was determined in water.

The density (specific gravity or SG) was calculated using the formula:

$$SG = \frac{\text{Mass in Air}}{\text{Mass in Air} - \text{Mass in Water}} = \frac{W}{V}$$

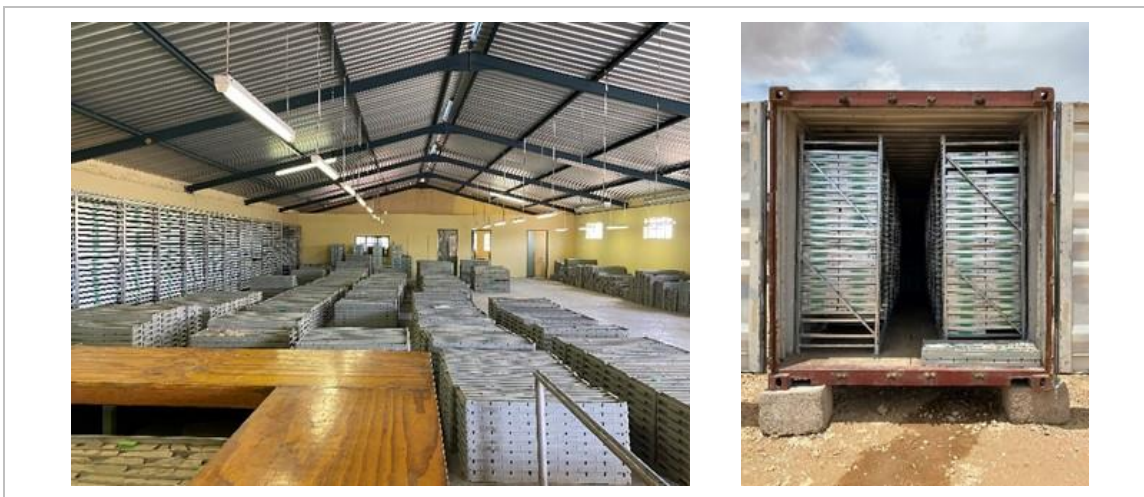


Source: Ellmies, 2020

Figure 11-4 Apparatus for Measuring Density (SG) by the Archimedes Principle

11.1.5 Core storage

The core trays with the unsampled intervals and remaining halves of the sampled intervals are permanently stored in a rented facility near Khorixas (Figure 11-5 and Figure 5-2). The fenced premises are locked, as is the warehouse and all storage containers. Only Gecko and NMI staff and NMI consultants have access to the building where the core is stored.



Source: Ellmies, 2020

Figure 11-5 Core Storage in the Khorixas Warehouse and in Containers in the Warehouse Yard

11.2 Sample Analyses

11.2.1 Sample preparation at the laboratory

At the Actlabs preparation facility in Windhoek, the samples were laid out and checked against the NMI sample list to verify that all samples are present and correctly numbered. An internal sample tracking sheet was prepared by Actlabs to track progress of the samples through the laboratory.

Using Actlabs' sample preparation package RX1, the samples were initially crushed in a jaw crusher to 90% passing two mm and then passed through a riffle splitter to obtain a 250 g split for pulverisation. The splits were pulverized with a swing mill in hardened steel bowls to 95% passing 105 µm. Samples were then homogenized in a stainless-steel riffle splitter and a 15 g sample and duplicate were drawn from the splitter for analysis. The splits were placed in Ziploc bags and prepared for shipping to Actlabs' analytical laboratory in Canada. The duplicate pulps were stored at the Actlabs facility in Windhoek.

11.2.2 Sample analyses at the laboratory

The pulp samples were couriered by air to the Actlabs analytical facility in Ancaster, Ontario, Canada, where they were analysed for major element oxides, rare earth elements and other trace elements.

Actlabs used their Code 8, REE Assay Package which involves a lithium metaborate fusion, multi acid digestion, and Inductively Coupled Plasma Analysis – Optical Emission Spectroscopy (ICP-OES) finish for

major element oxides; Sc, Be, V, Sr, Y, and Zr. An ICP-Mass Spectrometry (ICP-MS) finish was used for other trace elements including the REE. Nb₂O₅ and ZrO₂ were determined by sample fusion and standard X-ray fluorescence (XRF) method for samples with >0.3% P₂O₅.

Rare earth elements are among the most difficult elements to analyse to a high degree of analytical precision under a wide range of individual REE concentrations. The lithium metaborate fusion and ICP-MS finish is the current industry standard for high quality REE analyses.

Actlabs' quality system is accredited to international quality standards through the International Organisation for Standardisation / International Electrotechnical Commission (ISO/IEC) 17025, which includes ISO 9001 and ISO 9002 specifications, with CAN-P-1758 (Forensics), CAN-P-1579 (Mineral Analysis) and CAN-P-1585 (Environmental) for specific registered tests by the Standards Council of Canada (SCC).

11.3 Sample Security

All drillhole core handling, sampling and transportation were undertaken by Gecko staff. The individual procedures followed strict protocols outlined in a comprehensive SOP manual which was drafted by Gecko and reviewed by MSA. NMI's field camp has sample preparation facilities and is located in a relatively remote area to which only staff and contractors have access.

The core boxes were transported from the drilling rigs to the exploration camp on a daily basis by Gecko staff. To reduce movement of the core, the boxes were covered with foam sheets and ratchet-strapped to the loading bay of a utility vehicle for transport. Once the samples had been taken and prepared for dispatch, a Gecko staff member transported the samples in sealed bags to Actlabs' Windhoek preparation laboratory from where the material was couriered to Actlabs' Canadian facilities for sample analyses.

A "chain of custody" is maintained from the site to the laboratory via locked facilities and dispatch and receipt documentation. MSA considers that there was little or no opportunity for sample tampering by an outside agent due to the secure and auditable "chain of custody" implemented by Gecko and NMI personnel.

11.4 Quality Assurance and Quality Control

Appropriate quality assurance and quality control (QAQC) monitoring is a critical aspect of the sampling and assaying process in any exploration program. Monitoring the quality of laboratory analyses is fundamental in ensuring the highest degree of confidence in the analytical data and providing the necessary confidence to make informed decisions when interpreting all the available information. QA may be defined as information collected to demonstrate that the data used in the project are valid. QC comprises procedures designed to maintain a desired level of quality in the assay database. Effectively applied, QC leads to identification and correction of errors or changes in procedures that improve overall data quality. Appropriate documentation of QC measures and regular scrutiny of QC data are important as a safeguard for project data and form the basis for the quality assurance program implemented during exploration.

In order to ensure quality standards are met and maintained, planning and implementation of a range of external quality control measures is required. Such measures are essential for minimizing uncertainty and improving the integrity of the assay database and are aimed to provide:

- An integrity check on the reliability of the data;
- Quantification of accuracy and precision;
- Confidence in the sample and assay database;
- The necessary documentation to support database validation.

For all its drilling campaigns at Lofdal, NMI has adopted an industry standard QAQC program and inserted internal standards and certified reference material (CRM) and blanks each at a frequency of one in 20 (5%) into the batches prior to submission to Actlabs. These control samples were inserted as part of a continuous sample number sequence and the QAQC samples were not obviously different from routine samples after the pulverization process. Actlabs were requested in the sample submission sheet to split the pulp of predetermined samples (1 in 20) and insert the material in the empty and pre-numbered bags to create the required 5% duplicate samples. Actlabs in Canada was unaware which samples were QAQC samples and what their composition was. This allowed for monitoring of the sample preparation procedure as well as monitoring the accuracy and precision of analyses.

An additional 5% of the total samples were also submitted to a second laboratory in Canada for check analysis. Hence the overall number of control samples constituted 20% of all samples analysed which is in line with best practice procedures to ensure integrity of data and is independent from the internal QAQC methods applied by the laboratory itself.

Results of the 2010, 2011 and 2012 QAQC program were reported by Swinden and Seigfried (2011) and Seigfried and Hall (2012). The QAQC programs for these campaigns demonstrated that the analytical work was fully adequate and did not indicate any issues with the data quality.

11.4.1 Blanks, CRMs and Duplicates, 2020

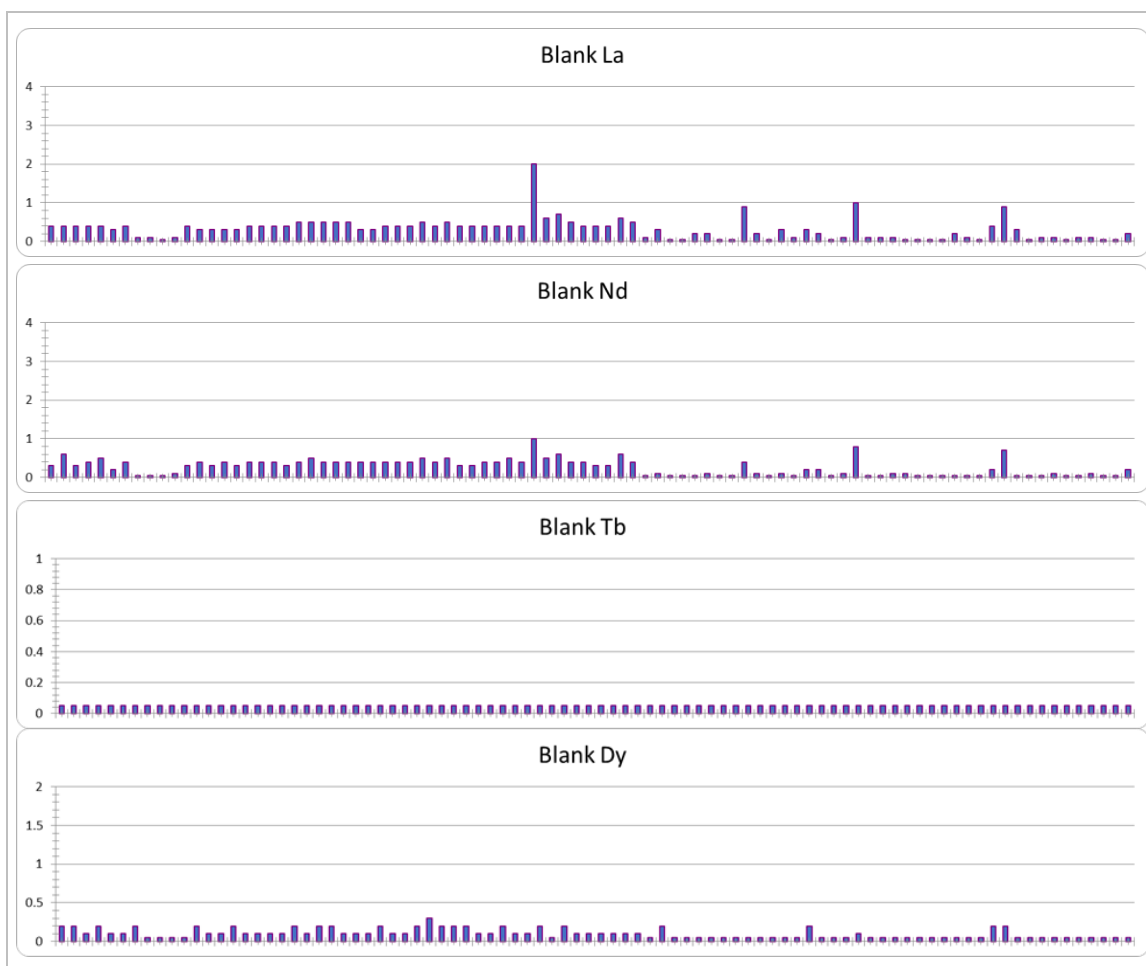
Data from duplicates, internal standards, CRM's and blanks were examined on a batch-by-batch basis to check for analytical data confidence. Data from duplicate, standard and blank analyses was examined numerically and graphically to determine the repeatability of the duplicate analyses, the precision of the standard analyses with respect to the accepted values, and the levels of REE present in the blanks.

11.4.1.1 Blank samples

Two different blank materials were used to evaluate sample preparation:

- Quartz pebble blank sourced from Ferreiras Garden Shop in Windhoek and pulverized by Actlabs. Used in drillholes L4D0115 through L4D00136 and in drillholes L2BD0027 through L2BD0041; and
- Dolomite sourced from Ferreiras Garden Shop in Windhoek (previously used and assayed by Namibia Rare Earth). Used in drillholes L4D0137 through L4D00170 and in drillholes L2BD0042 through L2BD0055.

The blank materials were supplied as coarse gravel and approximately 50 g was used. The blank samples were inserted with consecutive numbers within the core sample stream. The blank material experienced the same sample preparation and analytical stream as the routine field samples. Graphical representations of blank sample results for selected REE are shown in Figure 11-6 and Figure 11-7 and blank sample summaries for all drilling are given in Table 11-1. Repeated analyses show that these materials are acceptable for use as blanks for carbonatite hosted REE analyses with all REE present in concentrations near or below their detection limits. Background values are slightly higher in the quartz pebble blank and this is reflected in a number of slightly anomalous (i.e. between 1 and 3 ppm) Ce analyses in this material. However, this material is considered to have REE concentrations that are well within the range of values needed for a blank for QAQC purposes. There are no anomalies in the analyses that suggest anything more than normal difficulties of analysing these elements at very low concentrations. No further action was taken or required, and the results of the blank analyses are interpreted to indicate that there was no contamination or systematic analytical issues during the period of sample submission and analyses.



Note: vertical axis in ppm

Figure 11-6 Blank Analyses for Selected REE from Area 2B

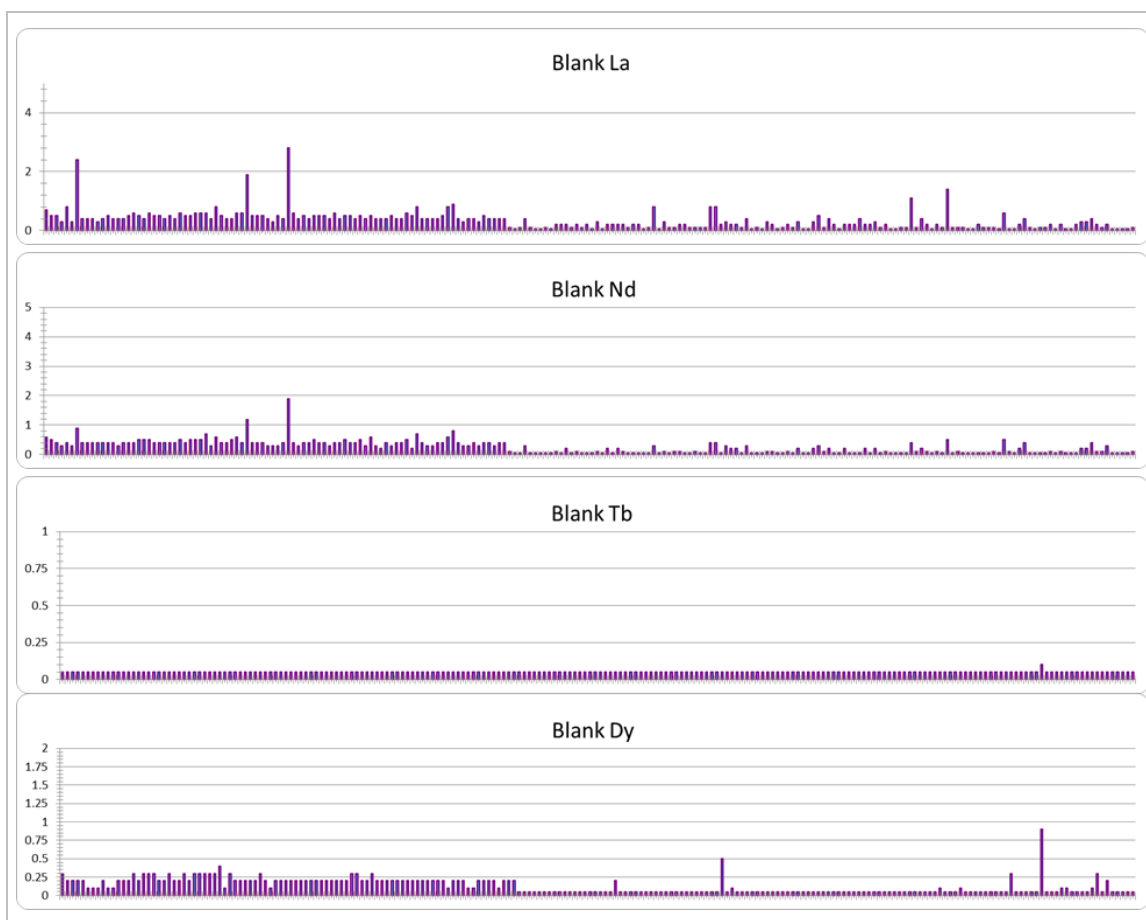


Figure 11-7 Blank Analyses for Selected REE from Area 4

Table 11-1 Number of Blank Failures (>10 times LDL*)

Number of Samples	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y
300	6	33	1	2	0	0	0	0	0	0	0	0	0	0	0

Note: *LDL=Lowest Detection Limit

11.4.1.2 Standards

During the 2020 drilling campaign, four different standards were used. Three NMI in-house standards with varying degrees of HREE enrichment, namely STD4, STD5 and STD6, were used. All in-house standards underwent 'round robin' analyses at four independent laboratories, but are not certified. Repeated analyses of STD4 and STD6 throughout the program demonstrated a generally consistent REE composition. The use of STD5, however, was discontinued after 20 holes because of reproducibility issues, particularly with

respect to Sm, Tb, Er and Tm. One commercial CRM, AMIS0185 was also used. The heavy rare earth elements Gd, Tb, Er, Yb and Lu in this CRM are neither certified nor provisional but only reported for informational purposes and were not used for the purposes of QAQC in this project. Representative QAQC plots for the 2020 drilling are shown in Figure 11-8 to Figure 11-14 and the QAQC data for all elements is summarized in Table 11-2 and Table 11-3.

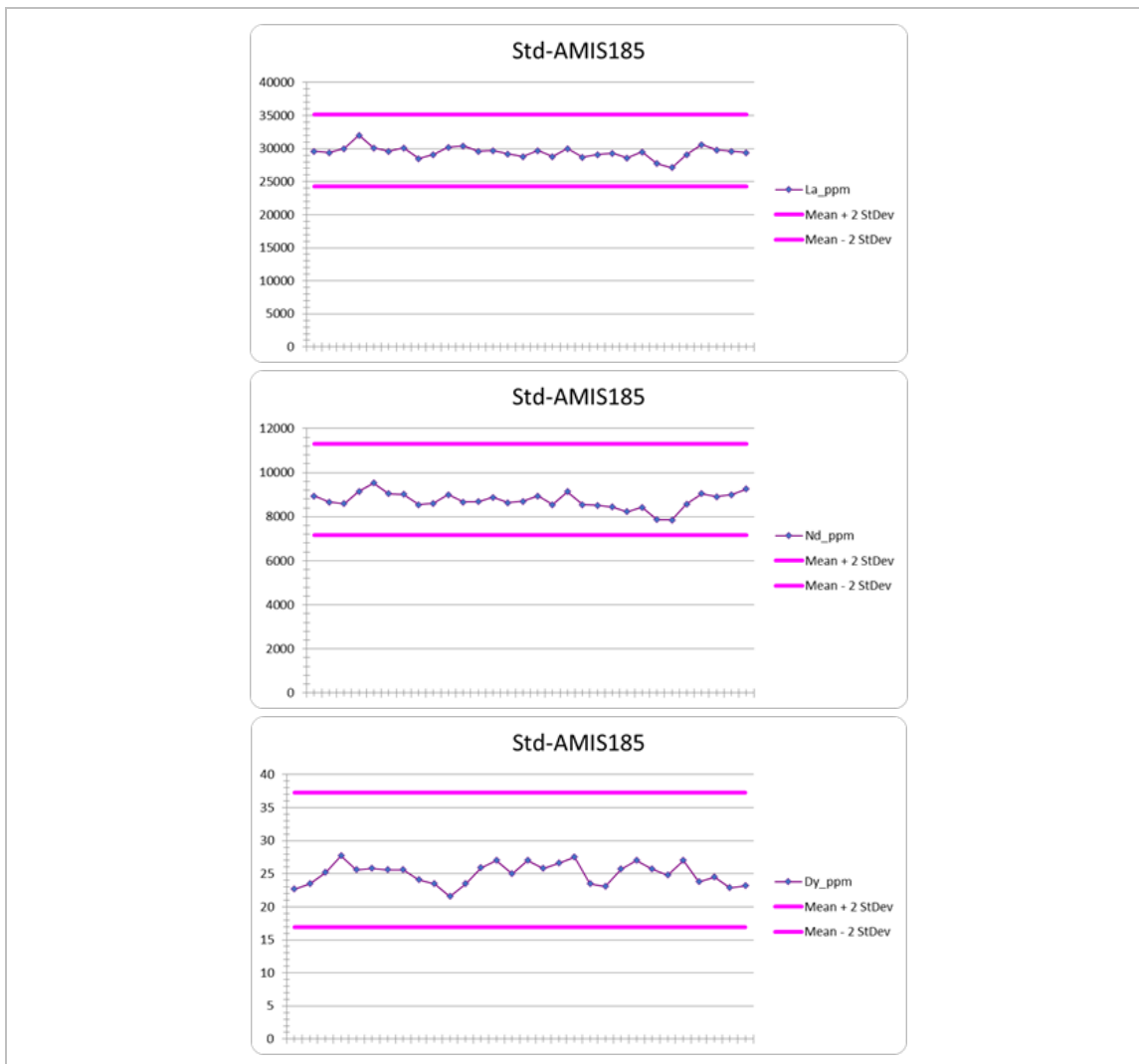


Figure 11-8 Analyses of the CRM AMIS0185 for Selected REE in Area 2B

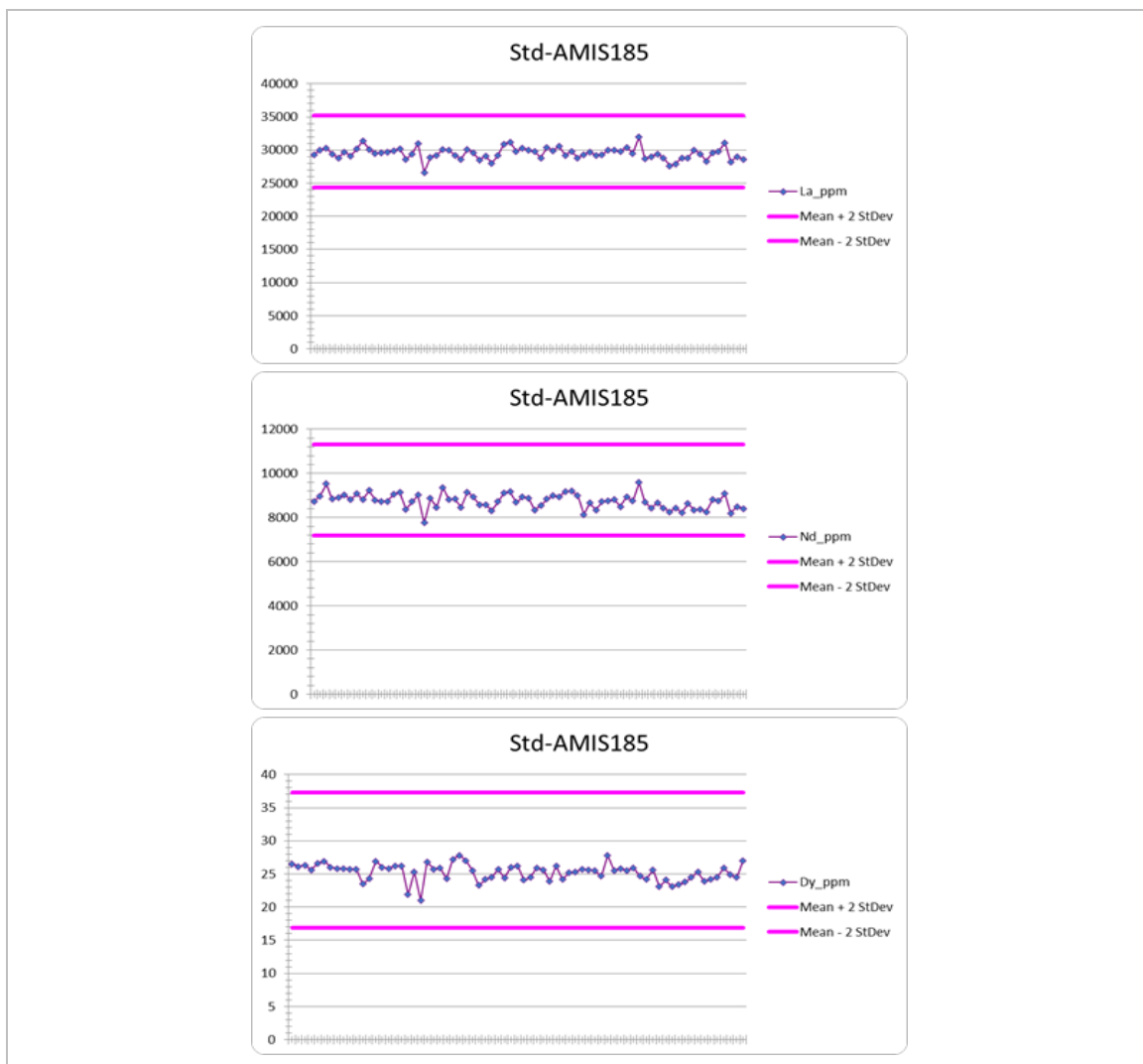


Figure 11-9 Analyses of the CRM AMIS0185 for Selected REE in Area 4

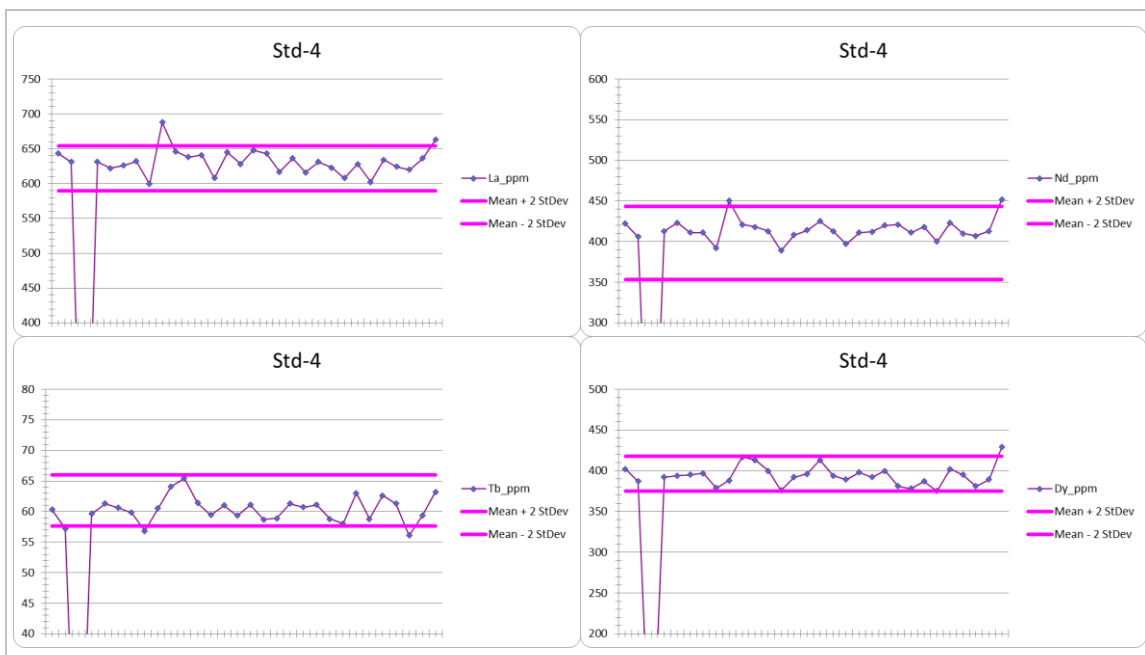


Figure 11-10 Analyses of the Standard STD4 for Selected REE in Area 2B

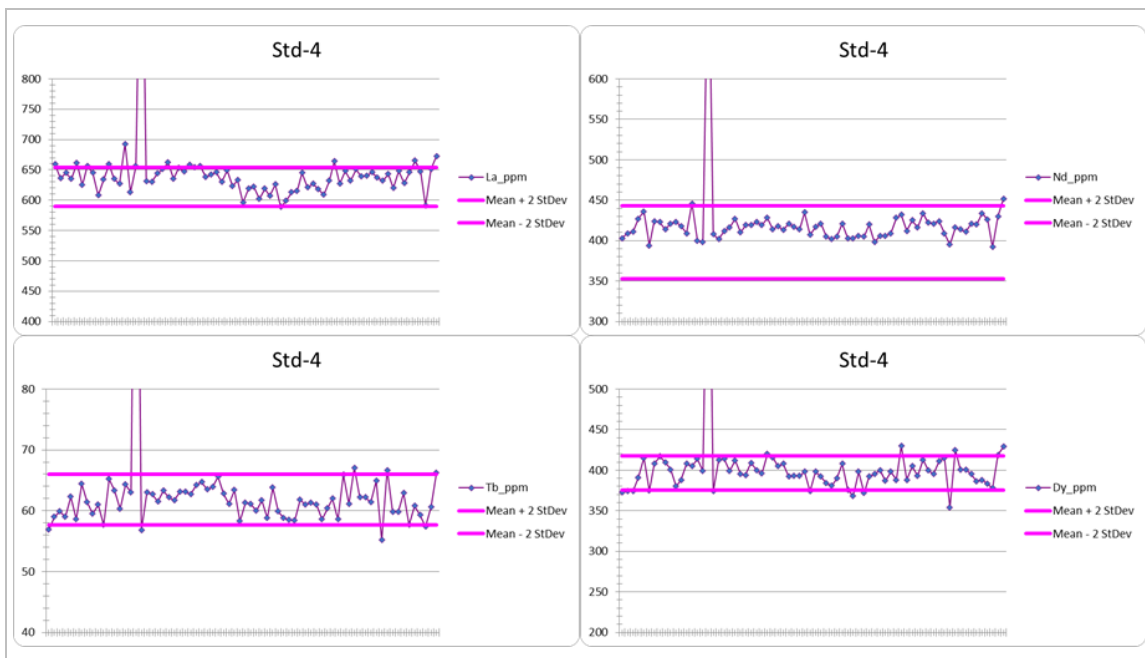


Figure 11-11 Analyses of the Standard STD4 for Selected REE in Area 4

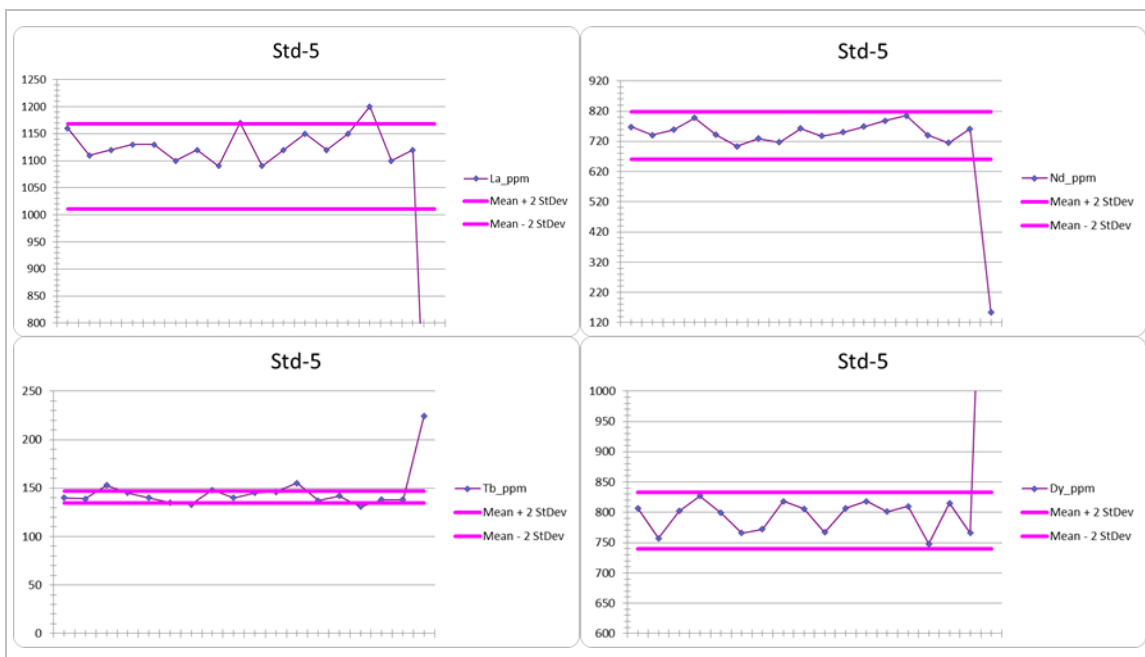


Figure 11-12 Analyses of the Standard STD5 for Selected REE in Area 4

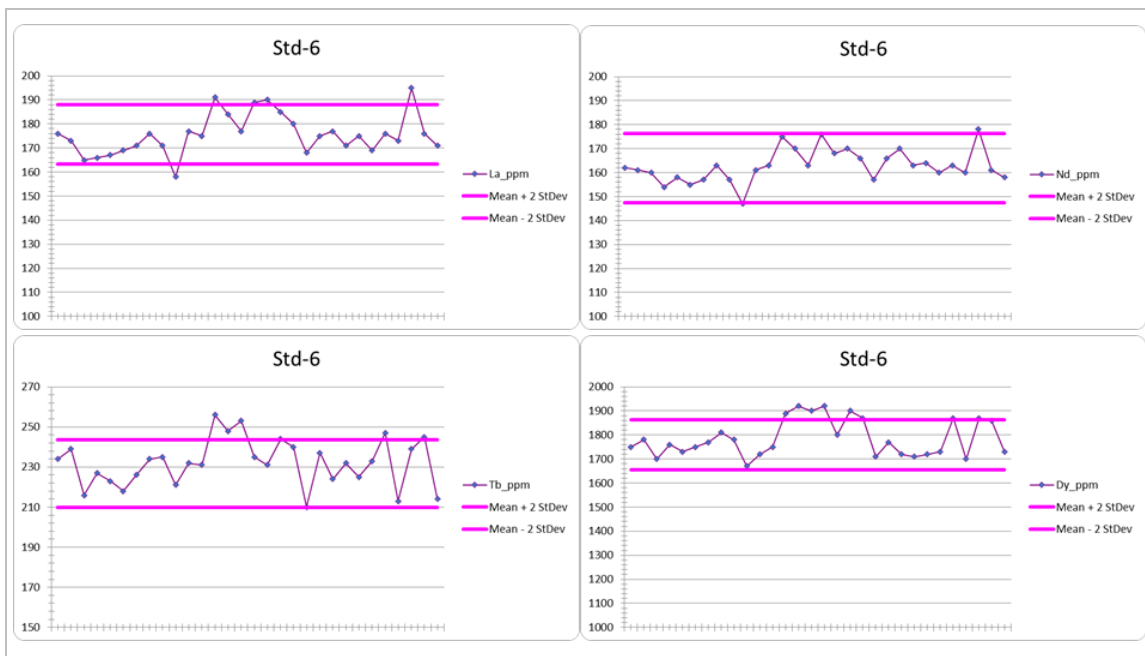


Figure 11-13 Analyses of the Standard STD6 for Selected REE in Area 2B

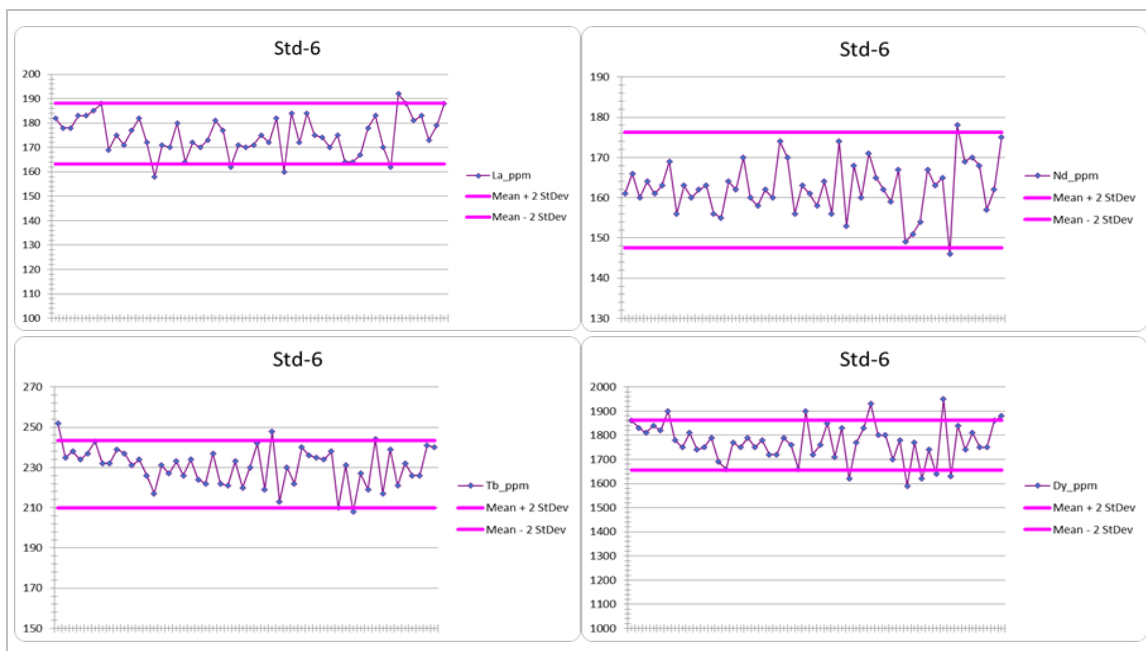


Figure 11-14 Analyses of the Standard STD6 for Selected REE in Area 4

Table 11-2 Statistics for the Reference Materials used in the 2020 Drilling Program

CRM Name	Number Used	Statistic	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y
STD 4	81	Accepted Mean	622.0	990.2	102.0	398.2	179.9	76.6	300.6	16.1	396.3	81.1	227.8	33.7	210.8	30.0	2439.7
		Standard Deviation	36.9	5.6	22.6	4.6	3.3	14.0	2.1	16.1	10.6	4.4	6.6	0.9	6.5	2.2	97.9
STD 5	18	Accepted Mean	1089.3	1666.2	173.9	739.2	465.1	201.4	787.6	39.4	785.9	142.8	370.1	49.7	276.0	36.8	4126.0
		Standard Deviation	65.3	7.9	39.3	8.2	9.7	25.5	3.0	39.4	23.3	6.9	5.5	0.6	9.2	2.4	162.6
STD 6	88	Accepted Mean	175.7	331.7	38.8	161.9	178.5	137.1	804.4	6.2	1758.9	399.2	1228.2	183.7	1106.7	146.4	12659.0
		Standard Deviation	11.9	1.2	7.2	7.6	5.5	38.1	8.4	6.2	51.9	12.6	63.1	4.9	34.8	10.9	468.0
AMIS0185*	106	Accepted Mean	29760.0	40750.0	3471.0	9238.0	556.0	94.2	29760.0	2720.0	27.1	3.2	-	0.4	-	-	62.0
		Standard Deviation	4610.0	343.00	1033.0	48.00	12.10	2720.0	-	-	5.10	0.50	-	0.08	-	-	7.70

Note: * Certified and provisional concentration only

Table 11-3 Failure Rate (Outside ± 3 SD) for CRMS Assayed by Actlabs during the 2020 Drilling Campaign

CRM Name	Failure Rate	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y
STD 4	Number of Samples	7	3	3	3	17	3	2	4	8	2	6	7	7	2	2
	Percentage	6.5 %	2.8 %	2.8 %	2.8 %	15.7 %	2.8 %	1.9 %	3.7 %	7.4 %	1.9 %	5.6 %	6.5 %	6.5 %	1.9 %	1.9 %
STD 5	Number of Samples	1	1	1	1	8	1	0	4	1	1	6	9	1	1	1
	Percentage	5.6 %	5.6 %	5.6 %	5.6 %	44.4 %	5.6 %	0.0 %	22.2 %	5.6 %	5.6 %	33.3 %	50.0 %	5.6 %	5.6 %	5.6 %
STD 6	Number of Samples	1	2	4	0	0	0	0	3	5	1	0	10	3	0	0
	Percentage	1.1 %	2.3 %	4.5 %	0.0 %	0.0 %	0.0 %	0.0 %	3.4 %	5.7 %	1.1 %	0.0 %	11.4 %	3.4 %	0.0 %	0.0 %
AMIS0185*	Number of Samples	0	0	0	0	0	0	-	-	0	0	-	-	-	-	1
	Percentage	0.0 %	0.0 %	0.0 %	0.0 %	0.0 %	0.0 %	-	-	0.0 %	0.0 %	-	-	-	-	0.9 %

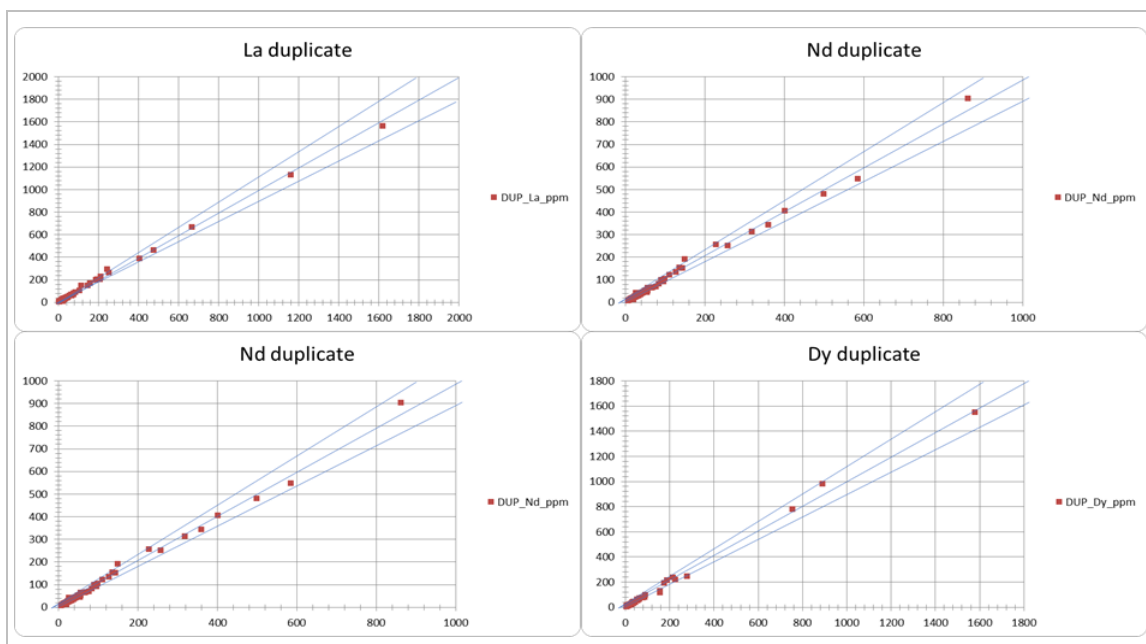
Four samples gave highly anomalous results and required further investigation. These samples are tabulated in Table 11-4. Three were found to result from inadvertent inclusion of the wrong material in the sample stream, probably taking place in the core yard, and the other was due to a missing Y assay. None indicate an issue with the sample preparation or analytical procedures.

Table 11-4 Resolution of Anomalous CRM Analyses

Area	Standard recorded as	Elements affected	Resolution
Area 4	STD 4	All REE display a positive spike	STD5 was mistakenly inserted
Area 4	STD5	Negative spike in LREE, positive spike in HREE	STD6 was mistakenly inserted
Area 4	AMIS 185	Zero value for one Y analysis	Cell was blank on the reporting spreadsheets
Area 2B	STD4	All REE display a negative spike	A blank was mistakenly inserted

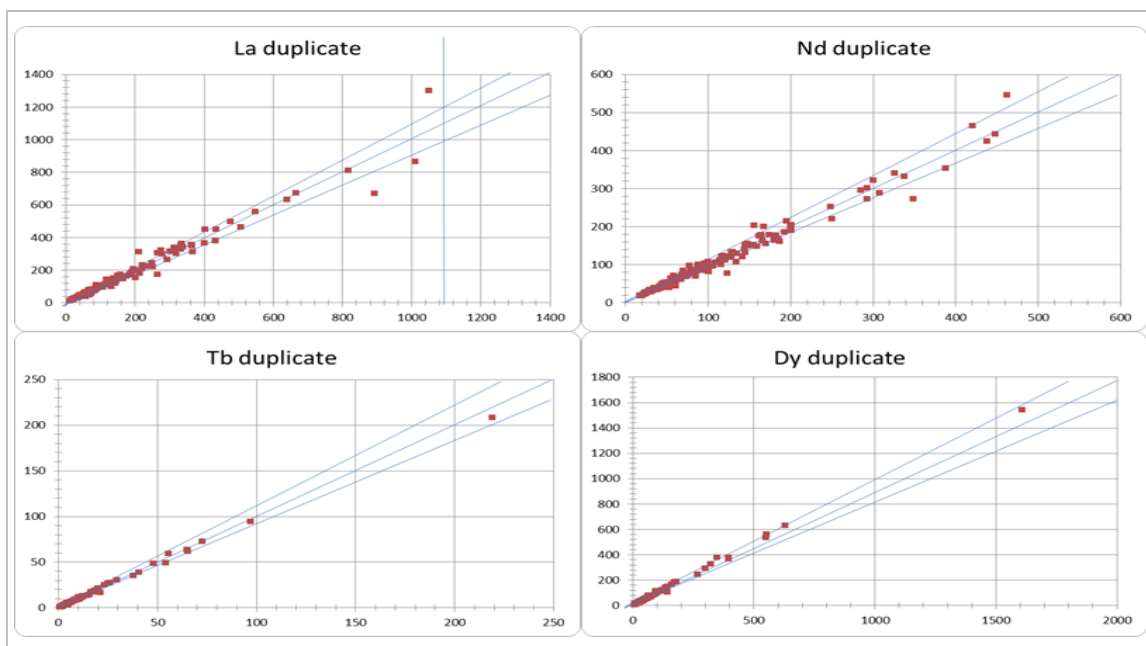
11.4.1.3 Pulp duplicates

Laboratory duplicates were prepared for 1 in 20 samples. The original and duplicate analyses were compared graphically to ensure repeatability. Where significant outliers from the expected values were observed in adjacent control samples, a subset of the batch was re-analysed. This was the case in one batch where several duplicates displayed an anomalous amount of divergence. Re-analyses produced acceptable duplicate analyses. No additional measures were taken or deemed necessary.



Note: Reference lines are: 1:1; +10% and -10%, values are ppm

Figure 11-15 Analyses of Laboratory Duplicates for Selected REE from Area 2B



Note: Reference lines are: 1:1; +10% and -10%, values are ppm

Figure 11-16 Analyses of Laboratory Duplicates for Selected REE from Area 4

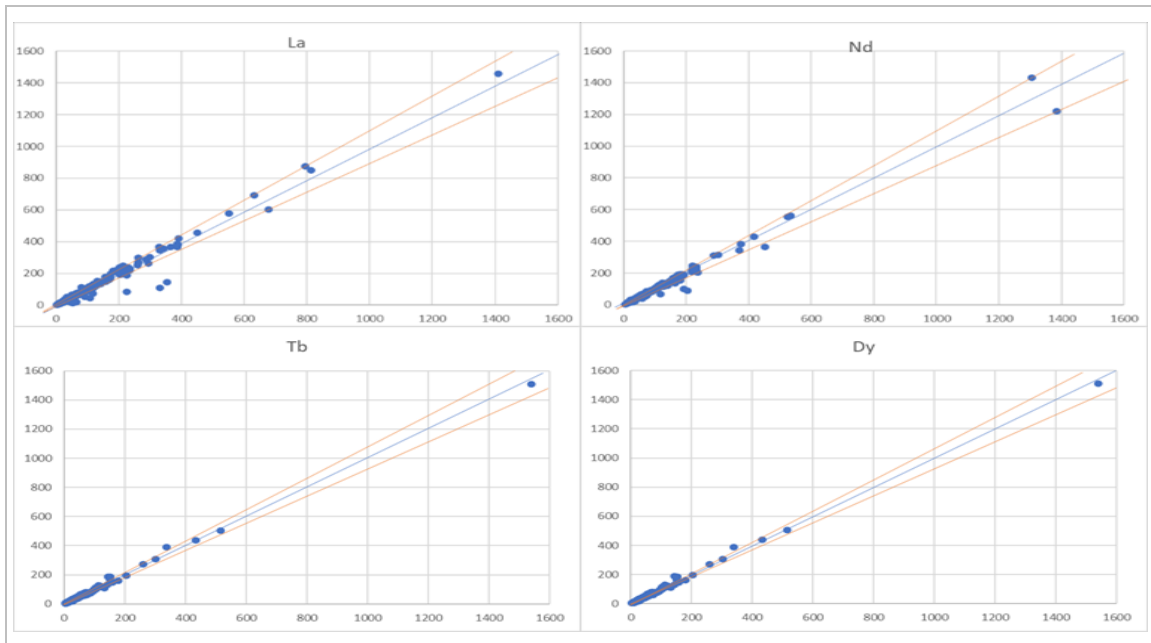
Table 11-5 Percentage of Assays within Mean Absolute Difference of 10% and 20% (Above 10x LDL) – Actlabs Duplicate versus Original

	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y
10 %	76	77	78	82	77	80	80	79	82	81	80	79	82	80	83
20 %	94	95	95	96	97	97	97	97	96	96	97	96	96	96	96

11.4.1.4 Second laboratory duplicate assays (“Umpire laboratory”)

Approximately 5% of samples were sent for check analyses at a second laboratory; ALS Minerals (ALS) in North Vancouver, Canada. Pulps were split from analytical samples at the Actlabs sample preparation facility in Windhoek and were shipped directly to ALS. The results of these analyses were plotted graphically against the original analysis. In the vast majority of samples, there was less than 10% discrepancy between the two laboratories. Representative plots of the umpire results for samples used in the MRE are presented in Figure 11-17 and Table 11-6.

ALS are registered to ISO 9001:2008 and have received ISO 17025 accreditation for laboratory procedures relevant for the purpose of the check assay exercise.



Note: ALS assays on horizontal axis. Actlabs on vertical axis, values in ppm
Blue reference lines is 1:1; red lines are +10% and -10%

Figure 11-17 Graphical Representation of Comparative Analyses by Actlabs and the Umpire Lab (ALS) for Selected REE

Table 11-6 Mean and Variance of Original and Duplicate Data – Actlabs versus ALS

Element	Actlabs		ALS	
	Mean	Variance	Mean	Variance
La	118.6	62340.2	117.0	67455.5
Ce	221.7	203810.4	216.7	204241.6
Pr	24.5	2609.9	24.1	2611.4
Nd	99.3	70015.8	96.1	63104.1
Sm	31.9	11530.6	30.8	10166.8
Eu	10.5	820.0	10.6	850.5
Gd	41.1	9203.8	38.6	7464.5
Tb	7.2	210.3	7.2	207.9
Dy	44.9	9701.8	45.4	9591.9
Ho	9.2	455.8	9.0	471.4
Er	27.5	5527.6	26.2	4656.1
Tm	3.9	108.0	3.8	113.1
Yb	25.1	5379.7	24.0	4765.2
Lu	3.3	80.8	3.5	106.2
Y	278.5	635284.9	267.6	509003.4

11.5 Adequacy of Sample Preparation, Security and Analytical Procedures

All aspects of core handling, marking, logging, cutting, bagging, labelling and sample submission to Actlabs' preparation facilities at Windhoek are covered by well-designed protocols to ensure that all routine activities are conducted with maximum consistency and followed industry standards.

Drillhole core handling and storage as well as core sampling and transport are conducted in a safe and secure manner. NMI followed an auditable chain of custody, which ensured high levels of security and integrity of the results.

The QP is of the opinion that the sampling and analytical procedures and the number of QAQC samples inserted into the sample stream are appropriate for the current level of the project, the type of the deposit and for the analytical techniques used. The CRMs and blanks show acceptable performance for the elements analysed over the period of the sampling campaign and precision was demonstrated to be acceptable.

The analytical results from the primary and the secondary laboratories correlate well and therefore the second laboratory assays confirm the element concentrations determined from the primary laboratory within acceptable analytical limits.

All assay data captured in the project MS Access database has passed internal QAQC criteria either on primary or secondary analyses. The quality control procedures have been effective in demonstrating the

quality of the Actlabs analytical results and any issues that were identified were quickly dealt with and resolved.

It is the QP's opinion that the sample assay results are acceptable for use in a Mineral Resource Estimate.

12 DATA VERIFICATION

The Lofdal site was visited by the QP for the Mineral Resource (Jeremy Witley) from the 28th to the 30th of October 2020. During the site visit, the following verification work was completed:

- The exploration processes were examined, and it was found that the work is being carried out according to the Lofdal procedures, which are appropriate for the purposes of evaluating the Mineral Resource.
- The logging, sampling and assay records were examined for a selection of drillholes from both previous and 2020 drilling for Area 4 (nine pre-2020 holes and eight recent holes with ICP assay results) and Area 2B (13 recent holes, three with ICP assay results) and were verified against observations made on the cores. The logging was found to be of good quality and the higher grade REE mineralisation was observed to be associated with the lithology, alteration and structures as described in sections 7.4.2, 7.4.3 and 7.4.4 of this report.
- Ad-hoc hand-held XRF readings were taken on several cores that confirmed the presence of elevated Y. Although the results of this exercise are not definitive, it served to verify the magnitude of the assayed Y grades.
- RadEye readings were taken on the core at the drilling rigs, at the field camp and core yard, as well as on outcrops of carbonatite dykes in the field. These readings confirmed the elevated gamma readings associated with the mineralised zones.
- Cores were observed being taken from the drillholes during the drilling process for drillhole L4D0148 and L4D0151. The cores observed being removed from the holes exhibited albitisation and iron alteration in the footwall to the mineralisation at Area 4.
- The drilling locations of completed drillholes in the 2020 and previous programs were observed in the field. Handheld GPS readings of the collar positions were taken for seven of the 2020 Area 4 drillholes and eight of the 2020 Area 4 drillholes. The handheld GPS coordinates were compared with the final DGPS surveys, and no material discrepancies (>5 m) were noted.
- The general site and the carbonatite dyke outcrops were examined. The outcrops observed are generally aligned with the mapping performed by NMI and its predecessors.

The Lofdal site was again visited by the QP for the Mineral Resource (Jeremy Witley) on the 10th of November 2022. Three of the drillholes completed since the previous site visit, that were included in the 2020 Mineral Resource, were inspected as well as the bulk sampling pit from which the metallurgical samples were extracted

Additional verification comprised:

- Spot checks of the database against the original borehole logs.
- Spots checks of database against original Assay Certificates.
- Examination of database used for mineral resource estimation for any errors.

In the opinion of the QP, the data verification processes demonstrate that the data collected are adequate for the purpose of Mineral Resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Historical Testwork Background

Testwork was done at both Mintek in Johannesburg, South Africa and Nagrom in Western Australia in 2014. The focus of the bulk of that testwork was to produce a 20% TREO concentrate by means of physical separation methods coupled to a HCL calcite leach step. Limited scoping type tests were completed on the subsequent cracking process for thorium removal.

The key reports cited by MDM, as part of previous PEA, were as follow:

1. Scoping metallurgical testwork on the Lofdal REE Deposit, Mintek, August 2013;
2. Phase 2 – Additional WHIMS testwork on the Lofdal High Grade REE Sample, Mintek, May 2013;
3. Phase 3 – Magnetic separation testwork on Lofdal High Grade and Low Grade ores, Mintek, January 2013;
4. Rare earth characterisation testwork, Nagrom, December 2013;
5. Test Report of sorting rare earth elements oxides, Tomra Sorting Solutions, April 2013.

On that basis, MDM aimed to firm up on the efficiencies of the various recovery steps, and to investigate various acid leach regimes to produce a high grade xenotime concentrate, which could be sent to a typical sulphuric acid bake cracking process to liberate locked thorium in the xenotime matrix. The cracking plant cost was provided to MDM by a third party on a confidential basis, and costed in terms of facilities required with associated capital and operating cost components.

Further testwork was to be required to provide the information to perform a detailed engineering study going forward for Lofdal, both in terms of the concentrate section as well as the cracking plant facilities earmarked to be located in the town of Walvis Bay, Namibia should this process option be selected.

13.2 Previous Testwork

The following is a brief discussion on the historical testwork, as well as flowsheet development over a period of time, prior to previous PEA execution by MDM. It must be noted that MDM was not involved during the preliminary testwork and thus provided a summary only of work completed as reflected in various reports mentioned above.

Initial scoping testwork was initiated at Mintek, Johannesburg on two HQ cores from the area of interest. The main focus being to determine what physical beneficiation techniques could be applied to the Lofdal material, and to evaluate the migration of thorium through the various steps, as well as to determine if thorium could successfully be rejected in the cracking step. The report that served as reference for the summary below is “Scoping Metallurgical testwork on the Lofdal REE Deposit, Report number 6570, 2 August 2013”.

Upon inspection of the drill cores it was evident that modern sorting technology would be able to serve as a primary coarse upgrading step and, as such, reduce tonnages that require milling and further physical upgrading.

The scoping work at Mintek was conducted on the drill core samples that were subjected to XRT sorting by Tomra. The cores were jaw crushed in Namibia and screened at 10 mm, before sorting was completed on the +10 mm fraction. The fines and sorted coarse fraction was then sent to Mintek.

The two drill cores represented a high-grade sample (Hole 4084) and a low-grade sample (Hole 4085).

The initial sorting tests were successful and Tomra believed that with further optimisation the sorters could yield a 50% mass reduction at a 90% recovery of TREO to the sorter product; the fines fraction would bypass the sorter.

Initial comminution tests indicated that the ore has a relatively low crushing work index as compared to the ball mill work index, and as such, with regard to power consumption, sorting would provide a valuable avenue to consider going forward.

Magnetic separation was then applied to the whole ore, sorter product and unsorted fines, milled to various product sizes. The xenotime mineralogy indicated that the mineral is fine and concentrated in the sub 38 µm size fraction. Initial whole ore, sorter product and fines WHIMS tests indicated that magnetic separation would provide acceptable upgrading of the xenotime mineral.

Initial flotation work was not successful and indicated poor selectivity between xenotime and calcite. Further work would be required and flotation was considered to still be an option for upgrading the product. The flotation work program was continued to attempt improving the flotation efficiency and selectivity.

The initial samples indicated good thorium rejection in the magnetic separation stage and also that the concentrate responds to a caustic “crack” followed by selective leaching of the thorium using HCL with subsequent precipitation.

The scoping level work highlighted several areas that would require further optimisation, but overall demonstrated that the material could be upgraded using physical processes and that thorium could be removed from the final product. This work consisted of a number of magnetic separation and flotation test permutations. As part of the previous PEA study, various other technologies were also investigated, including gravity recovery and gangue HCL leaching to upgrade the concentrate prior to the cracking stage and reject the carbonate minerals. Thorium rejection via direct acid leaching was also investigated with various precipitation s-curves produced in order to optimise the various precipitation stages post thorium removal. This work provided a solid base to scope the subsequent phases of the testwork campaign.

The phase 2 testwork report named “Phase 2 – Additional WHIMS Testwork on the Lofdal High Grade REE Sample, May 15, 2013” highlighted the below key points.

This report was, in essence, a continuation of the previous work and comprised of additional magnetic separation testwork in order to improve the overall TREO recovery to concentrate and TREO concentrate grade from 7.8% to 20%.

The testwork included higher intensity scavenger work as well as cleaner WHIMS work with the inclusion of regrind at various positions in the flowsheet. Regrind was investigated prior to scavenging as well as prior to cleaning.

This work highlighted that the highest overall TREO grade was achievable from the “as is” cleaning circuit at 17.4% TREO but at a low overall recovery. Regrind prior to cleaning did not provide further significant upgrading. It was also apparent that, in order to improve the recovery, the cleaner tails would need to be locked cycled in the WHIMS circuit, but further work would be required in this regard.

Regrinding on the rougher non-magnetics and processing it in a scavenger stage with higher intensities provided an additional TREO recovery of 17.8% with an overall recovery to concentrate (rougher and scavenger) of 62%. The work indicated that a 70-75% TREO recovery would be possible but would need confirmation during the phase 3 campaign should higher intensity regrind scavenger WHIMS to be used.

The Phase 3 testwork campaign at Mintek is covered in the report “Phase 3 – Magnetic Separation and Flotation Testwork on Lofdal High Grade and Low Grade Ores, Report no. 6936, 2 July 2014”.

Stage 1 and 2 campaigns showed that the fine grained xenotime could be upgraded with staged milling and WHIMS to a concentrate grade of 8-12% TREO at unit recoveries of 60-80%. The phase 3 work focused on treating the low grade sample through the process to demonstrate whether the size of the resource could be extended by reducing the cut-off grade. Phase 3 attempted to optimise the process conditions to produce a >20% TREO concentrate with an overall recovery in excess of 60%. During phase 3, attempts to improve the circuit also included flotation and gangue acid leaching into the overall flowsheet.

Both the high-grade and low-grade samples were amenable to upgrading with the WHIMS but the high-grade sample had a higher upgrade ratio than the low grade sample. This again highlighted the need to have upfront ore sorting in order to upgrade the low-grade sample prior to milling and magnetic separation. It was also noted that the low-grade sample contained significant amounts of hematite and ankerite that interfered with the low grade sample WHIMS treatment. It was confirmed that the low grade sample contained an anomalous amount of iron and as such is not representative of the ore body. This was to be further investigated during the phase 4 campaign on a larger sample.

During the phase 3 work, the focus shifted after initial WHIMS optimisation tests were completed, with the emphasis on producing an unoptimized bulk magnetic concentrate and then proceeded to optimise the flotation and gangue acid leaching steps. It became apparent that additional scavenger stages in the magnetic separation circuit would be required to improve on the overall TREO recovery. The bulk magnetic concentrate was then utilised in a flotation cleaning step using the Clariant SM15 reagent followed by HCL gangue acid leaching.

The flotation on the magnetic concentrate yielded upgrade ratios of 1.7-1.9 with recoveries in the range of 80-90% TREO. Performing a HCL leach on the magnetic and flotation concentrate at pH 1 successfully removed calcite with insignificant leaching of the other minerals. Recoveries of 98-99% were achieved with upgrade ratios in the order of 1.3-2.7. Flotation on the gangue acid leach residue also indicated potential to improve the overall TREO grade further.

Four circuits were evaluated where the primary concentration step was either flotation or WHIMS. The preferred option at that point in time, although not optimised, appeared to be magnetic separation followed by flotation, gangue acid leach with flotation on the gangue acid leach residue. The only concentrate available for this series of tests was the unoptimized LONGI magnetic concentrate and as such, this work would confirm the optimum flowsheet but would not serve as a definitive flow sheet test to produce >20% TREO concentrate.

That flowsheet provided the best TREO grade of 11.4% at an overall recovery of 60.4% with a head grade of 0.7% TREO. Optimisation testwork was planned for a phase 4 campaign. It was to be noted that during that phase of work, flotation and magnetic separation at various grind sizes were also extensively tested. Although flotation performed reasonably well at the coarser grind sizes, upgrade through the magnetic separation process step was compromised at coarser grind sizes. The optimum grind size selected at this stage was a P₈₀ of 38 µm.

The following key objectives from the phase 3 work was summarised as follows:

- The HCL leach at pH 1 successfully removed the calcite present in the concentrate with insignificant leaching of the other minerals. Thorium removal and dissolution of rare earth elements by HCL, nitric acid and acid boil was not successful. In the acid bake approach, both the thorium and yttrium dissolved with no selective impurity removal;
- Various dispersants were trialled in the magnetic separation process with Calgon proving to be the best, but it adversely affected the magnetic separation process and gave erratic results. Further work using dispersants was to be required;
- The effects of pulsation in the magnetic separation step were extensively tested and it seemed to improve the overall yttrium recovery and grade by reducing entrainment of albite and calcite.

Extensive mineralogy work was completed to improve the understanding of mineral migration through the various process steps in the testwork.

In parallel, WHIMS and flotation testwork had been initiated at Nagrom, using a split of the same sample that Mintek was working on.

Various characterisation tests were initially completed on the material, which ranged from dry magnetic separation tests and gravity separation tests. The gravity tests did not indicate gravity to be a viable option to pursue further.

WHIMS testwork followed evaluating magnetic recovery at a coarser grind as was employed at Mintek with a selected grind of $P_{80} = 106 \mu\text{m}$.

The circuit consisted of a primary rougher and a secondary scavenger in order to improve the overall recovery. Overall, an 87% TREO + yttrium recovery was realised into 37.8% of the mass. This work illustrated that the hematite and thorium upgraded into the magnetic fraction.

The concentrate from the magnetic separation work was then progressed to flotation work on the same grind size. Various test regimes were evaluated with the best test yielding 82% TREO + yttrium recoveries with 98% CaO recovery. The grade was diluted by the high calcite recovery to the concentrate and it was thought that the coarser grind did not provide sufficient liberation with the resulting flotation of composite particles.

Bulk testwork was then initiated based on a grind size of $P_{80} = 38\mu\text{m}$. The bulk testwork phase kicked off with evaluating the gravity response using a Falcon concentrator but with limited success.

The bulk testwork phase extensively tested wet high-intensity magnetic separation and flotation steps that aligned well with the Mintek test results. Nagrom also proceeded to test the various primary concentrates through the carbonate leach process utilising HCL. Similar results to those obtained at Mintek were reported with low losses of REEs to the leach solution and good upgrade ratios.

Flotation and magnetic separation of the gangue leach residue was also evaluated in order to clean the residue before the cracking step. Flotation proved to be the superior process for cleaning the acid leach residue.

Preliminary sorter testwork was completed at both Tomra as well as at IMS Engineering, evaluating various sorting technologies with the most promising being XRT followed by XRF sorting sensors. Although optimisation work on a larger bulk sample would be required as the project progresses, good results were obtained at Tomra using the XRT technology. It appeared that using the XRT sorter in the primary location with a XRF sorter on the waste stream as a secondary scavenger might prove to be the most viable option for primary waste rejection on the Lofdal material.

Further work was still in progress both at Mintek as well as Nagrom with the phase 4 testwork campaign focussed on further circuit optimisation and producing final product material.

13.3 2021 Ongoing Preliminary Testwork

During 2020, an 18-ton bulk trench sample was delivered to Light Deep Earth (LDE) (refer to Appendix D) Laboratory in Pretoria, South Africa. Several sized samples were prepared for bulk ore sorting testwork at Rados International using XRF sorter machines and at IMS Engineering for Steinert XRT sorter machines.

This ore sorting work showed potential for its inclusion in a flowsheet to upgrade ore prior to comminution and beneficiation. This work showed that at the feed grade of 0.3% TREO, a 90% TREO recovery could be anticipated at a 40% mass rejection resulting in advancing grade of ~0.5% TREO.

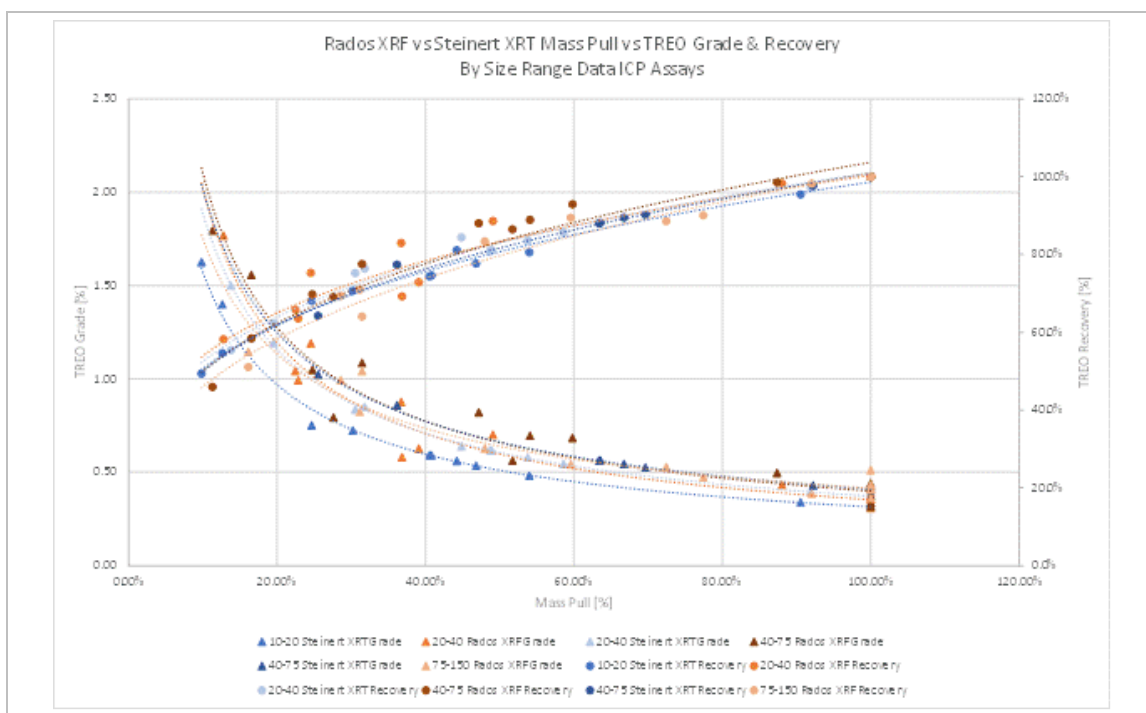


Figure 13-1 XRF and XRT Grade & Recovery by Mass Pull Test Data for Size Ranges (in mm) Tested

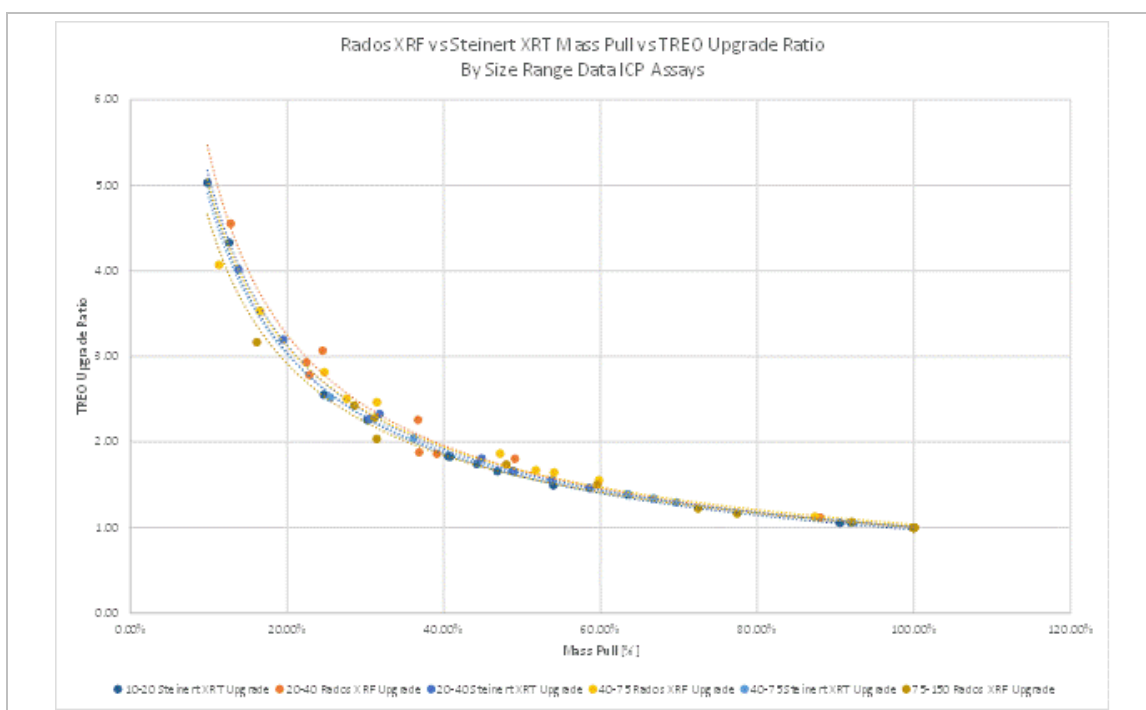


Figure 13-2 XRF and XRT Upgrade Ratios by Mass Pull Test Data for Size Ranges (in mm) Tested

Downstream gravity and magnetic beneficiation testwork followed on a sorted product sized into various size fractions at Light Deep Earth Laboratories (LDE). Mineralogy study was also carried out on the sorted product, where Qemscan evaluation indicated that the xenotime is reasonably (>60%) liberated below 45 µm.

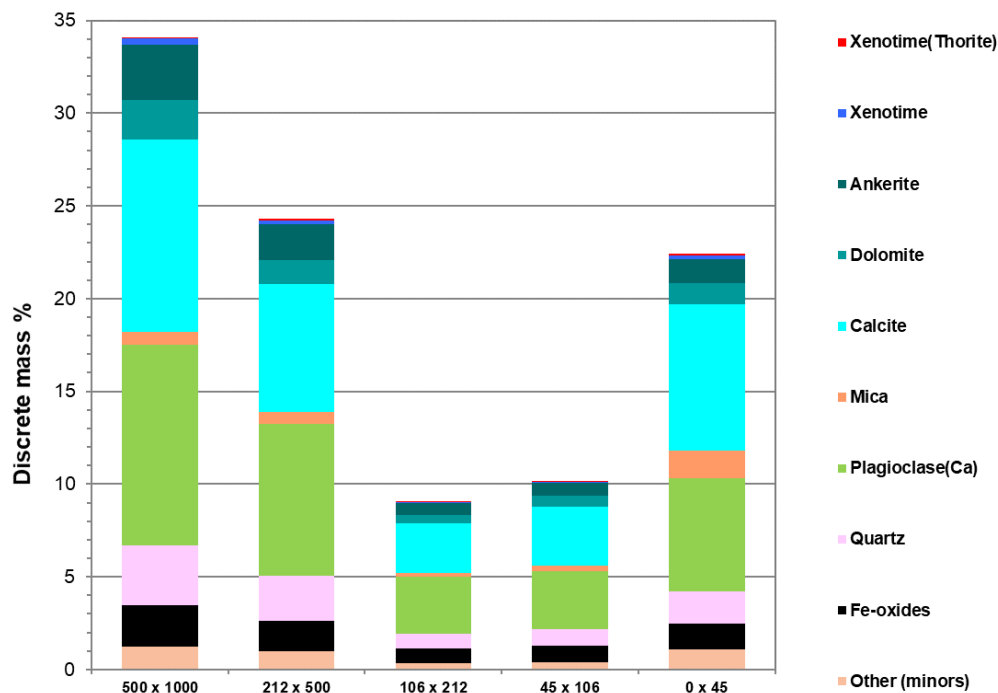


Figure 13-3 Mineral Phase Distribution in the 5 Prepared Feed Samples Investigated at LDE

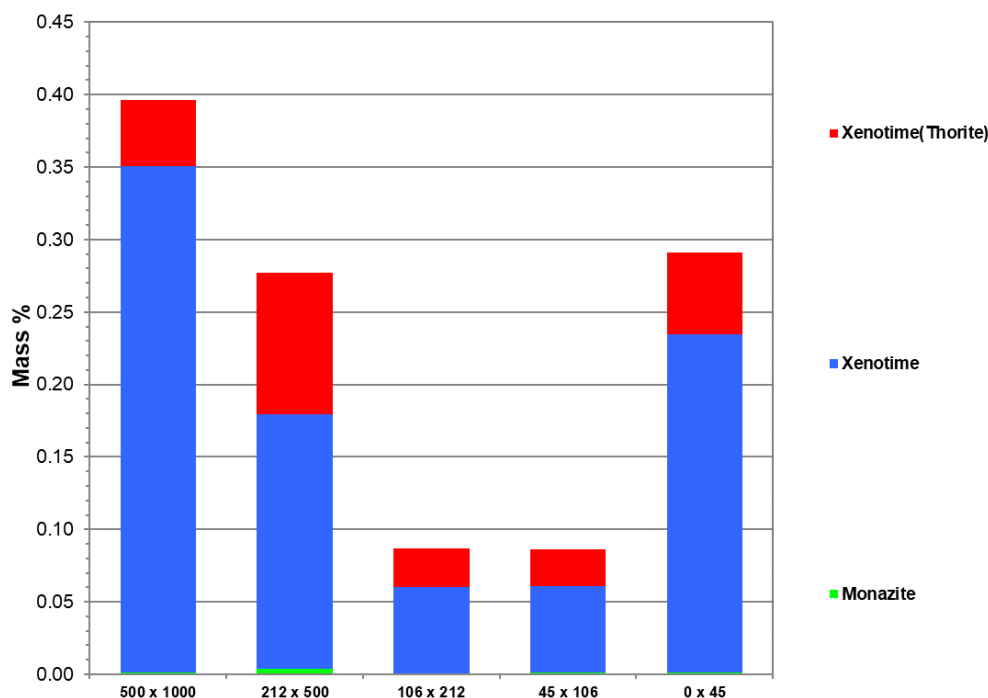


Figure 13-4 REE Mineral Phase Distribution in the 5 Prepared Feed Materials

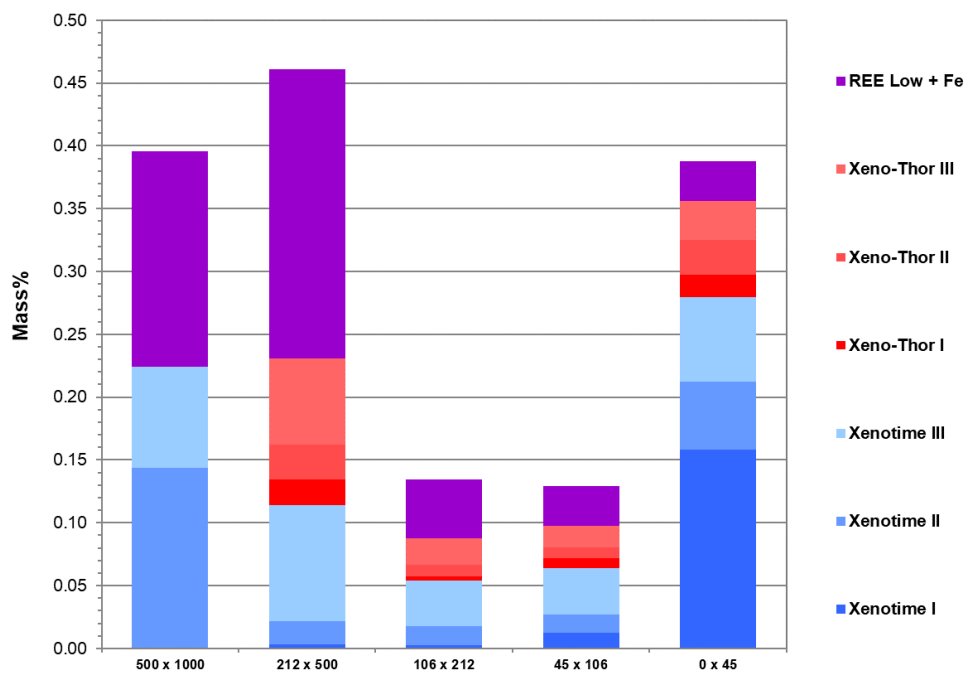


Figure 13-5 Mineral Particles (Degrees of Liberation) across the Prepared Feeds with I being >75% Liberated, II being 50-75% Liberated and III being 25-50% Liberated

The three coarser fractions (106 x 212 µm, 212 x 500 µm and 500 x 1000 µm) contained up to 6% ankerite, where the ankerite could not be isolated to a high-density fraction without the significant sacrifice of TREO recovery. Ankerite concentrated predominantly to the middling fractions, showing poor upgrade responses in these size fractions. There was however, a slightly improved upgrade response in 106 x 212 µm fraction as compared to coarser fractions. Magnetic separation (WHIMS) and gravity separation (shaking table) achieved between 30 to 40 % TREO recovery at a ~20% mass pull and 2 x upgrade ratio. DMS and coarse gravity separation was unable to isolate a high density fraction rich in ankerite with low levels of xenotime.

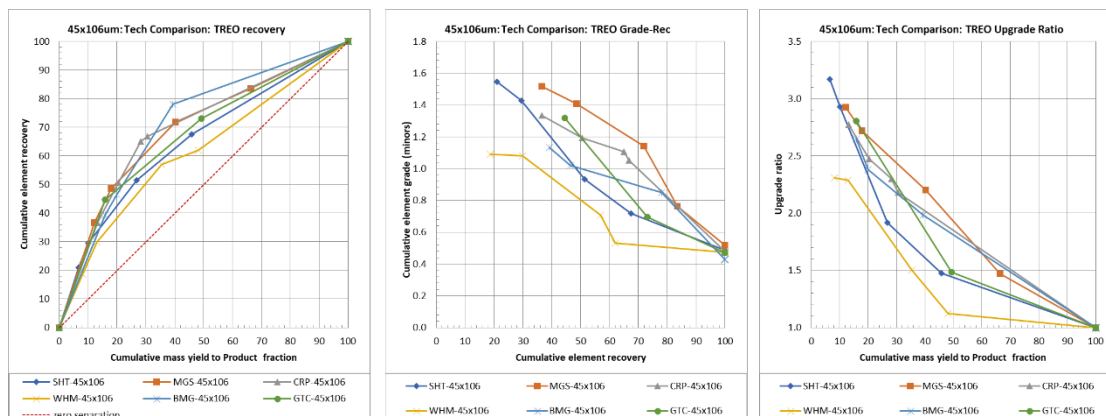


Figure 13-6 TREO Recovery-Yield, Grade-Recovery and Upgrade-Ratio-Yield Comparison per Technology on the 45x106µm Fraction

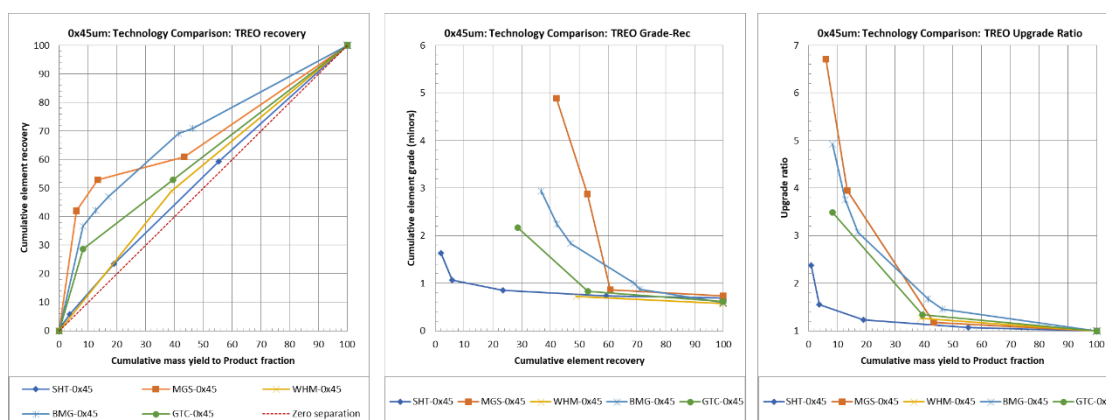


Figure 13-7 TREO Recovery-Yield, Grade-Recovery and Upgrade-Ratio-Yield Comparison per Technology on the 0x45µm Size Fraction

For the 45 x 106 µm fraction, gravity separation using multi-gravity separator (MGS) showed the most promise from a gravity separation point of view with 70% TREO recovery at 40% concentrate yield, whilst for magnetic separation, the Malvern Engineering Belt magnet (BMG) showed the most promise for TREO recovery (>75%) in 40% of the mass.

For the finer 0 x 45 µm fraction, gravity separation using multi-gravity separator (MGS) was able to produce high TREO grades (>5%) but recovery greater than 60% was challenging on the unit, whilst magnetic separation using the Malvern Engineering Belt magnet (BMG) showed the most promise for TREO recovery (>70%) in 45% of the mass.

Further gravity and magnetic separation testwork on a 0 x 106 µm fraction using MGS and BMG was conducted without significant change in beneficiation performance with recoveries greater than 60% TREO difficult to achieve.

Mineralogical studies showed that xenotime liberation only occurs below 45 µm and then only about 60% of the xenotime particles would be fully liberated. A large portion of the xenotime should be recoverable if high TREO grades were not pursued.

With 60% of xenotime liberated below 45µm, flotation performance testing required further investigation.

Then in early 2021, SGS Lakefield was mandated by NMI to conduct testwork, focussing on flotation, on XRF and XRT sorted products from the Rados and IMS testwork in South Africa. Two ~250 kg sorted samples were received by SGS Lakefield for mineral processing flowsheet development. The objectives were:

1. Simplify the flotation flowsheet developed in previous testwork, with similar or better flotation performance.
2. Evaluate the following two flowsheets a) direct flotation on the sorted product b) magnetic separation on the sorted product, followed by flotation on the magnetic separation concentrate.
3. Conduct hydrometallurgical amenability testing on concentrated product for caustic crack, acid bake and water leach, followed by impurity removal and rare earth precipitation.

Mineralogy studies were carried out as a starting point and the majority of the flowsheet development was focused on XRT sorted product. Rare earth minerals (REM) (collectively accounting for 1.15% and 1.13%) included mainly xenotime (0.84% and 0.80%), as well as trace amounts (<0.2%) of synchysite/parisite, monazite, bastnaesite, columbite/pyrochlore, and zircon (0.39% and 0.32%) (Figure 13-8).

The exposure of xenotime, calculated for the head samples, is presented in Figure 13-9. Exposure (>80%) was 27% and 29%, <80->20% was 43% and 48%, and <20% was 31% and 22%, in XRF SP and XRT SP samples, respectively.

Liberation for Thorite/Th-Y-Silicates in XRF SP and XRT SP samples was 22% and 38%, F-C-REE was 20% and 37%, ankerite was ~4%, Fe-oxides was 85% and 84%, respectively (Figure 13-10).

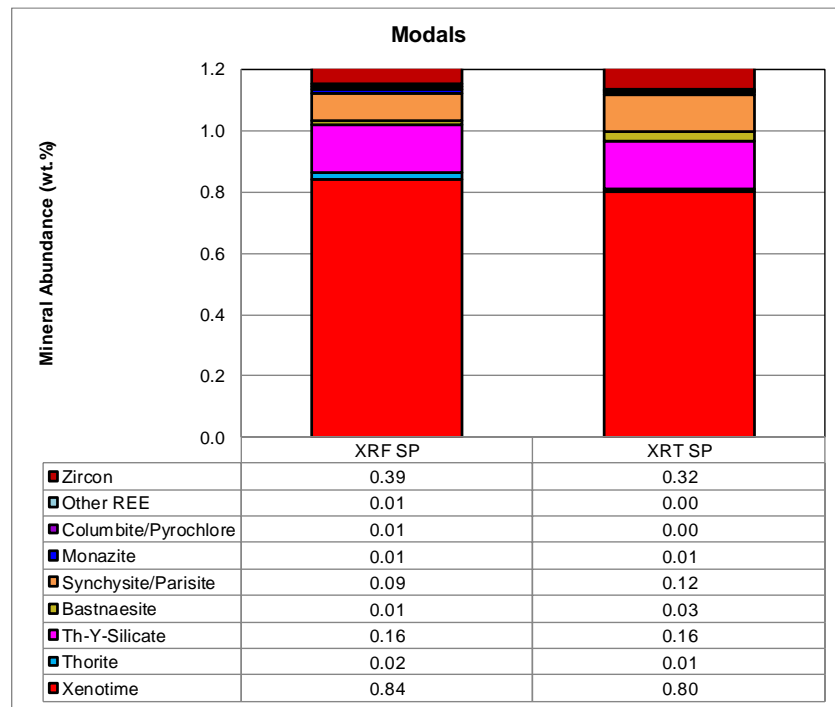


Figure 13-8 Summary of Mineral Distribution (Mass %) of REM, Zircon and Thorite for the XRF and XRT Head Samples

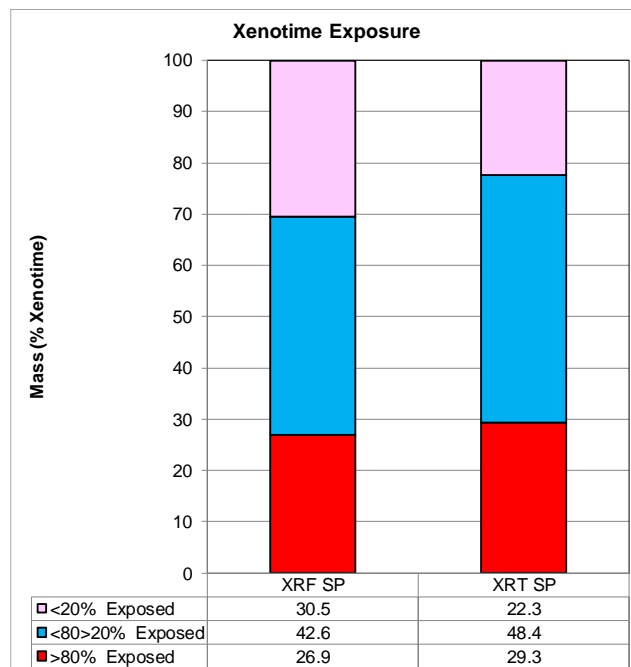


Figure 13-9 Summary of Exposure (Mass%) of Xenotime for the XRF SP and XRT SP Head Samples

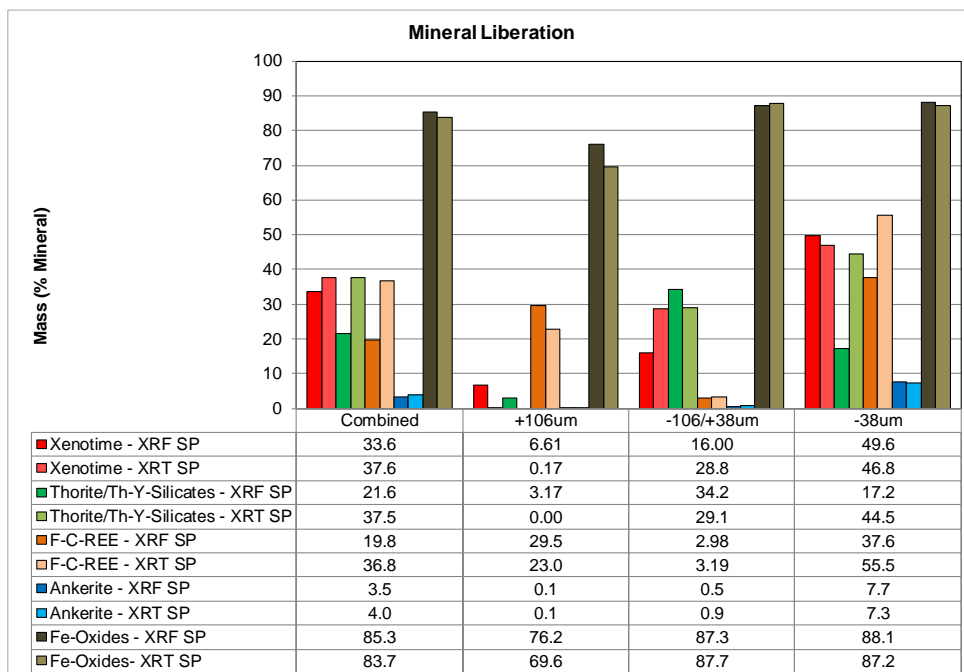


Figure 13-10 Summary of Liberation (Mass%) of Xenotime, Thorite/Th-Y-Silicates, F-C-REE, Ankerite and Fe-Oxides for the Head Samples

Flotation testwork was conducted on the XRF sorted product, blend of 75% XRT SP with 25% fines generated during sorting, and 100% fines. The flotation flowsheet was successfully simplified to REE flotation only from previously three flotation stages (sulphide flotation, carbonate flotation, and REE flotation). With testing on the selective collectors and effective depressants, in combination with an iron pre-removal, a preliminary flotation test produced a concentrate grading ~7.6% TREO (>13 times upgrade) achieving a ~70% recovery. Typical batch flotation test flowsheet shown in Figure 13-11.

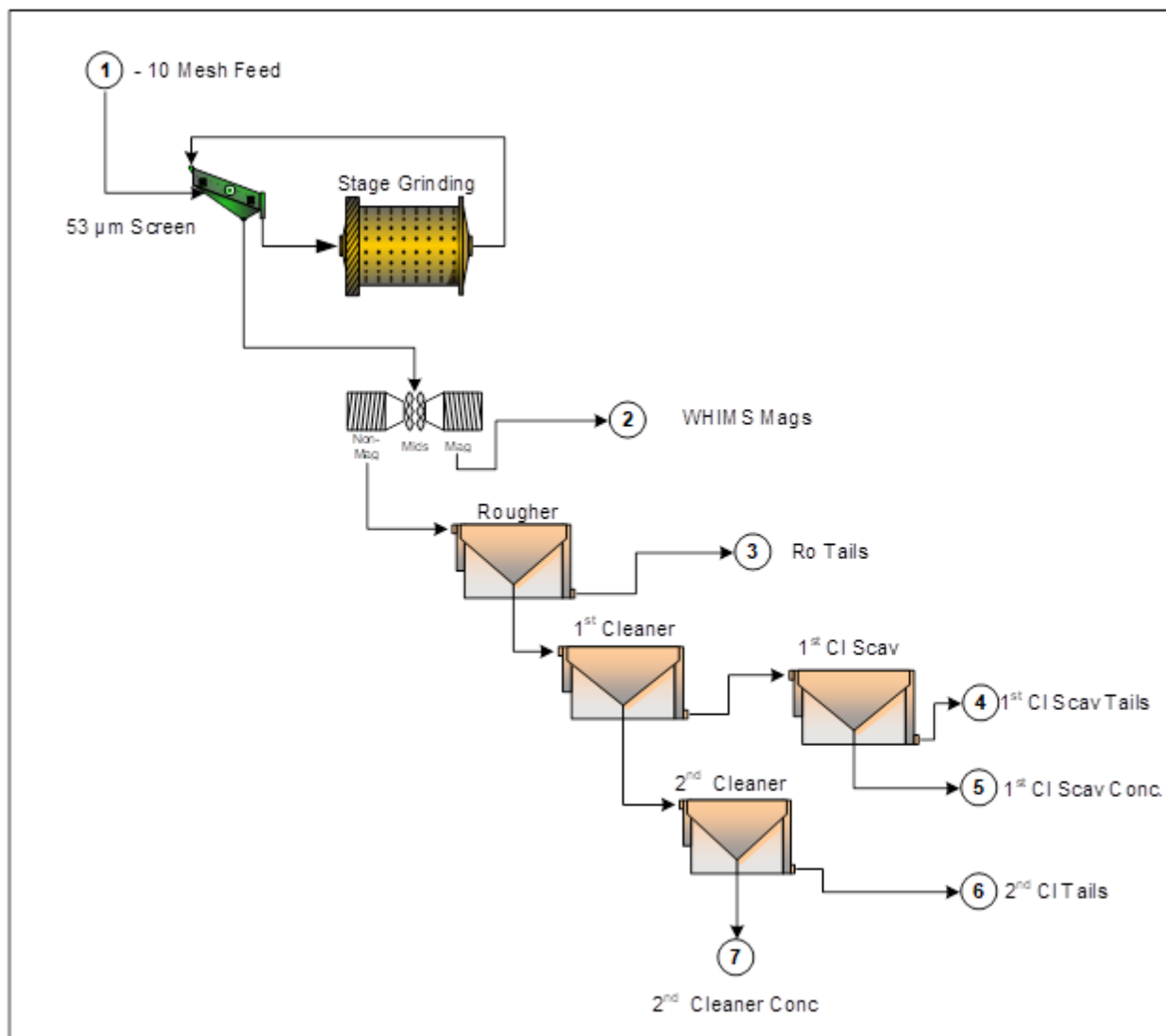


Figure 13-11 Typical Batch Flotation Flowsheet

Besides, the preliminary flotation test results with the direct flotation on the sorted product (flowsheet A) demonstrating higher REO recovery with similar REO upgrade than the flotation on the magnetic concentrate (flowsheet B). Other testing conditions, such as the grind size, pulp temperature, pulp density, were evaluated, to further reduce flowsheet CAPEX and OPEX costs. The flotation results of the collector screening and depressant screening tests are presented in Figure 13-12 and Figure 13-13, where:

- hydroxamate collector demonstrated better yttrium flotation performance and selectivity against calcium as compared to fatty acid or phosphoric acid ester collector
- calgon performed well in improving yttrium selectivity vs calcium

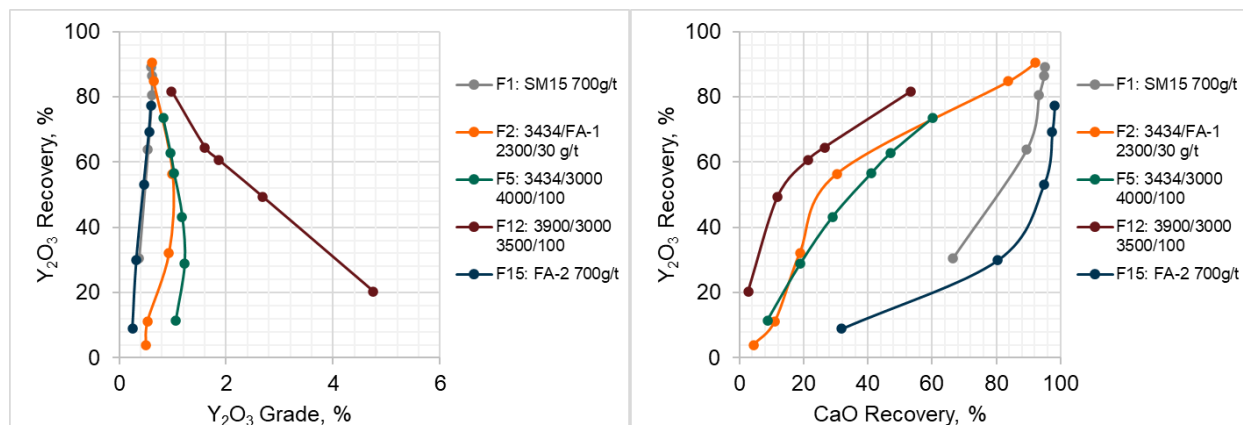


Figure 13-12 Preliminary Flotation Tests-- Collector Screening

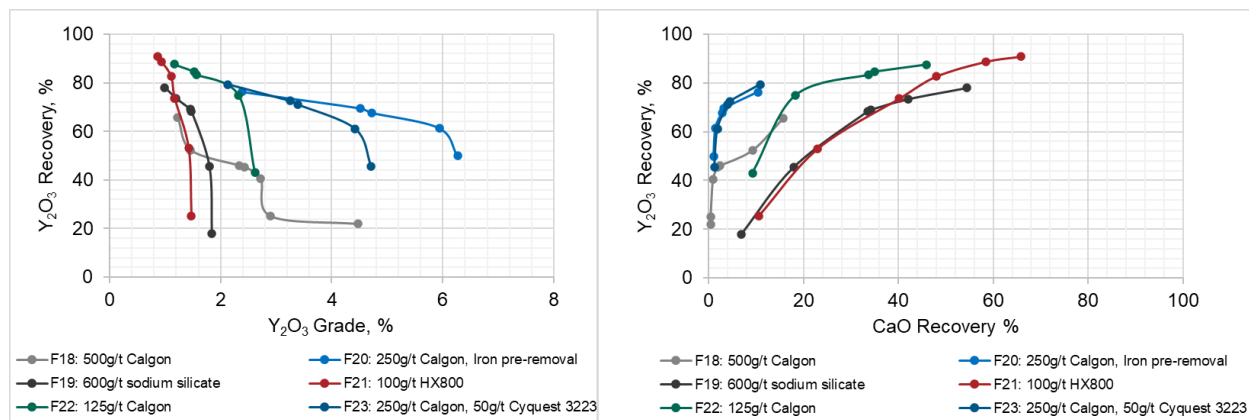


Figure 13-13 Preliminary Flotation Tests-- Depressant Screening

Bulk concentrates were produced for downstream hydrometallurgical testing, where concentrate at approximately 10% TREO grade and 61% recovery and 3.7% mass pull was targeted for the concentrate production.

Hydrometallurgical amenability testing on concentrates for the caustic crack process showed that this process route was not attractive as ~30% of the yttrium was in solution with 92% of the thorium and the balance of the yttrium after two cycles of caustic cracking and acid leaching remaining in the solids.

The acid bake amenability testing, however, demonstrated around 98% yttrium dissolution with water leach at 20% solids and 25°C with liquor composition of 14.2 g/L TREE (of which 8.2 g/L Y and 1170 mg/L Dy) and 37 g/L Fe in the solution. Half the concentrations at double the volumes were achieved in tests at 10% solids.

Preliminary impurity removal tests have demonstrated amenability for ~97-99% removal of thorium at pH 2.8-3.0 with less than 5% of the Nd and Y as shown in Figure 13-14.

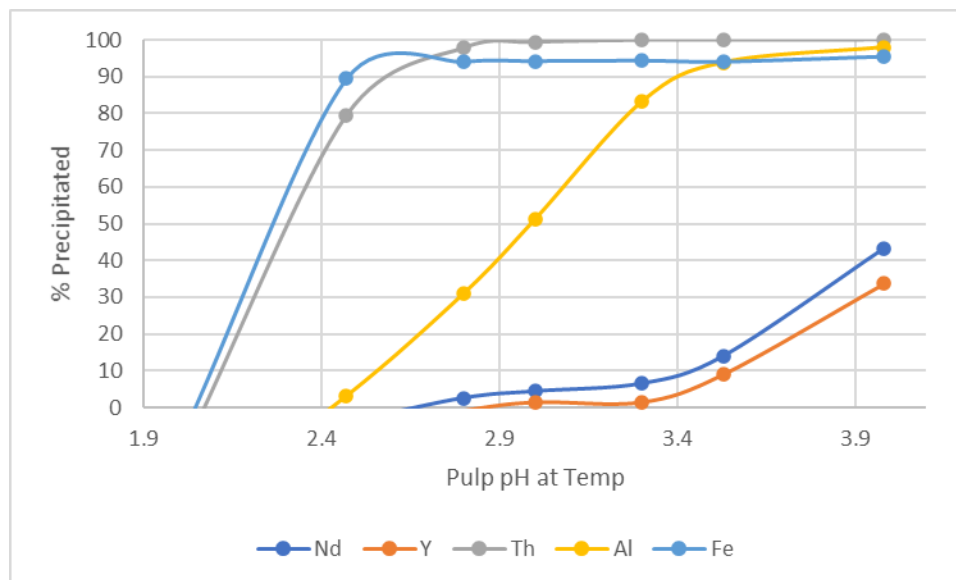


Figure 13-14 Extent of Precipitation of Select Elements in IR-4 with Magnesium Carbonate

A total of 12 acid bake and water leach tests were completed throughout the test program to investigate the dissolution of rare earth elements (REE) and the behaviour of gangue minerals through the addition of sulphuric acid at elevated temperatures (200-300°C). Optimum results were achieved with an acid bake process using 1250 kg/t H₂SO₄ at 300°C followed by a water leach with 20% solids by weight at 25°C. At this regime the tests showed very good REE recoveries with 97-98% for yttrium, 95% for dysprosium and 94-95% for terbium.

Impurity removal testwork resulted in the preference of using magnesium carbonate for a maximum precipitation of iron and thorium from the slurry while minimizing REE co-precipitation. The final impurity removal test in this program included a stoichiometric addition of hydrogen peroxide to oxidize iron in solution in order for it to precipitate.

Crude REE precipitation generated an intermediate product assaying at 43% total REE with 1.86% Al and less than 0.5% iron, thorium, and uranium when adjusting the liquor to pH 6.5. This mixed REE precipitate contained all of the yttrium and dysprosium along with 94.5% of the terbium.

REE precipitate re-leach consisted of a two-stage sulphuric acid process wherein solids were slurried in de-ionized water and heated to 50°C followed by addition of sulphuric acid to achieve pH 1.0. Following this, additional REE precipitate was added to the slurry to increase the pH to 3.5. This step resulted in a concentrated REE liquor representing 99% of the available REE and rejected 94% of the thorium, 85% of the aluminum, and 99% of the iron.

To remove residual uranium and thorium, the re-leach liquor was contacted sequentially with Purolite A660 for two contacts and then an organic mixture of 0.5% Primene JMT, 2.5% isodecanol in Aromatic 150ND. As a result, 99.9% of the uranium was removed in the first IX contact. 94% of the thorium was also collected in the ion exchange contacts while a further 75% was removed in a solvent extraction contact. The practically thorium-free raffinate from the solvent extraction step was advanced to final REE precipitation and calcination.

The final step of this test program was the precipitation of REE with minimal impurities, primarily Na, Mg, Si, and Ca. Oxalic acid precipitation and calcination produced a final solid containing 98.1% total rare earth oxides (TREO) representing 94% of the feed rare earth elements with the full suite of assays shown in Table 13-1.

Table 13-1 Oxalic Acid Precipitate Calcination (C-RP3) Assay Summary

Sample & Quant.	RP-3 Precip	C-RP3 Calcine
(mL or g)	22	10
La	11002	23800
Ce	19924	43100
Pr	2094	4530
Nd	7489	16200
Sm	3467	7500
Eu	2140	4630
Gd	13961	30200
Tb	3587	7760
Dy	28985	62700
Ho	6703	14500
Y	226056	489000
Er	21404	46300
Tm	2922	6320
Yb	17243	37300
Lu	2325	5030
Sc	<12	<25
Th	0.51	1.1
U	<0.2	<0.5
Si	878	1900
Al	<23	<50
Fe	32	70
Mg	1387	3000
Ca	740	1600
Na	1433	3100
K	<37	<80
Ti	<28	<60
P	18	40
Mn	<37	<80
Zn	20	44
S (%)		0.01
TREE (%)	36.9	79.9

Add'n Elements (g/t)	
Ag	<50
As	<200
Ba	15
Be	0.90
Bi	105
Cd	<3
Co	<200
Cr	100
Li	<30
Mo	381
Nb	
Ni	99
Pb	<200
Sb	35
Se	<50
Sn	40
Sr	16.7
Ta	
Tl	<50
V	<60
Zr	
LOI (%)	0.49

13.4 Bulk Fresh Sample Testwork

A further comprehensive metallurgical test program was carried at SGS out on fresh low grade feed ore sampled during the starter pit blasting program. The grade was lower as compared to the trench sample which had been used for the previous sorting and beneficiation testwork program - 0.19% TREO vs 0.36% TREO.

Sorting testwork both on XRF and XRT sorter techniques on the low grade fresh sample yielded Yttrium losses over the sorting step, which with the cumulative losses in the flotation stage demonstrated poor economic value. An intense machine learning program on the XRT sorting was done without adequate upside potential.

The purpose of the bulk sample metallurgical program was amended to include characterisation of the metallurgical behaviour of unsorted fresh low grade material to support early engineering design and economic evaluation of a low grade ROM process plant.

Comminution characterization testwork was carried out at Geolabs (refer to Appendix E) in South Africa on the fresh material samples (Low grade ROM plus sorted product with fines), which demonstrated lower ore hardnesses as compared to ore characterisation testwork having been completed historically at Mintek with BBWi of 19.2 kWh/t. The Bond Ball Milling indices on the fresh material was 13 to 18% lower than historic Mintek data.

	Sample ID	BBWi (kWh/t)	P ₁₀₀ (µm)	F ₈₀ (µm)	P ₈₀ (µm)	Gbp (g/rev)
1	ROM no sorting	16.7	53.0	2134	39	0.82
2		16.2	75.0	2134	60	1.05
3		15.7	106.0	2133	82	1.25
4	Sorted Product & Fines	15.7	53.0	2230	39	0.89
5		14.6	75.0	2230	55	1.12
6		14.2	106.0	2230	77	1.35

Figure 13-15 BBWi data on Fresh Material (Geolabs SA)

Flotation testwork was carried out at SGS Canada Inc. in Lakefield, Ontario. Two samples were delivered to SGS for flotation testing for purposes to confirm and validate flotation regime from the trench sample program as well as to test potential thrifting of flotation reagent recipes from the trench sample for improved project economics. The samples included a low grade ROM ore sample from the fresh starter pit as well as an upgraded sample post XRF ore sorting.

SGS conducted over 17 flotation tests starting with the selected regime from previous program on trench material as a baseline. Alternative collectors and thrifting flotation conditions were tested going forward to determine impact.

Mineralogical characterisation was done on the low grade feed sample using the TIMA (TESCAN Integrated Mineral Analyzer). TIMA is a fully automated, high throughput, analytical scanning electron microscope for mineralogical studies including mineral liberation analysis, measuring modal abundance, size-by-size liberation and mineral association.

Key results demonstrated median xenotime grain sizes of 26 µm and 10 µm for the two sized head fractions tested (+38 µm and -38 µm) respectively. Of the xenotime in the +38 µm fraction, 32% was pure to liberated xenotime with 76% pure to liberated xenotime in the -38 µm fraction (Table 13-2).

Table 13-2 Mineralogy Results Summary of Xenotime Liberation and Association

Mineral Name	LG XRF Comp	ROM Head Comb	ROM +38 um	ROM -38 um
Pure Xenotime	22.9	41.7	15.0	66.9
Free Xenotime	5.60	3.80	6.20	1.61
Lib Xenotime	9.90	9.40	11.1	7.78
Xnt: REM	0.00	0.10	0.03	0.23
Xnt: Zr Silicates	1.50	1.50	0.35	2.53
Xnt: Apatite	1.10	0.60	0.79	0.49
Xnt: Calcite/Dolomite	4.90	6.00	7.78	4.37
Xnt: Ankerite/Siderite	0.50	0.50	0.43	0.54
Xnt: Quartz/Feldspars	9.80	6.60	6.69	6.57
Xnt: Biotite/Chlorite/Muscovite	1.10	0.90	0.41	1.29
Xnt: Fe-Oxides	4.10	1.20	2.13	0.28
Xnt: Other	0.50	0.20	0.40	0.14
Complex	38.0	27.1	48.7	6.6
Total	100.0	100.0	100.0	100.0
Pure+Free + Liberated	38.4	54.9	32.3	76.3

The material performed well in flotation with upgrade ratios of between 20 and 27 times from the low grade feed material. By comparison, the sorted trench material demonstrated flotation upgrade ratios of around 10 to 13 times. The final flotation concentrate grades of slightly lower TREO grades (~6% TREO) to advance to hydrometallurgical were produced from the low feed grade as compared to the sorted trench material.

The selected Florrea reagent suite from the previous program still provided the most attractive flotation results on the low grade feed material (see Figure 13-16). Thrifting conditions were tested to test reduced depressants, reduced collector dosages (see Figure 13-17) and coarser grinds were also tested (see Figure 13-18). The impact of high intensity conditioning ahead of flotation was tested, which yielded improved flotation performance. Alternative collectors were tested. The lower calcites in the fresh feed material as compared to trench material resulted in lower depressant requirements in the flotation regime. A range of collector dosages were tested to determine the upper and lower envelopes for flotation performance.

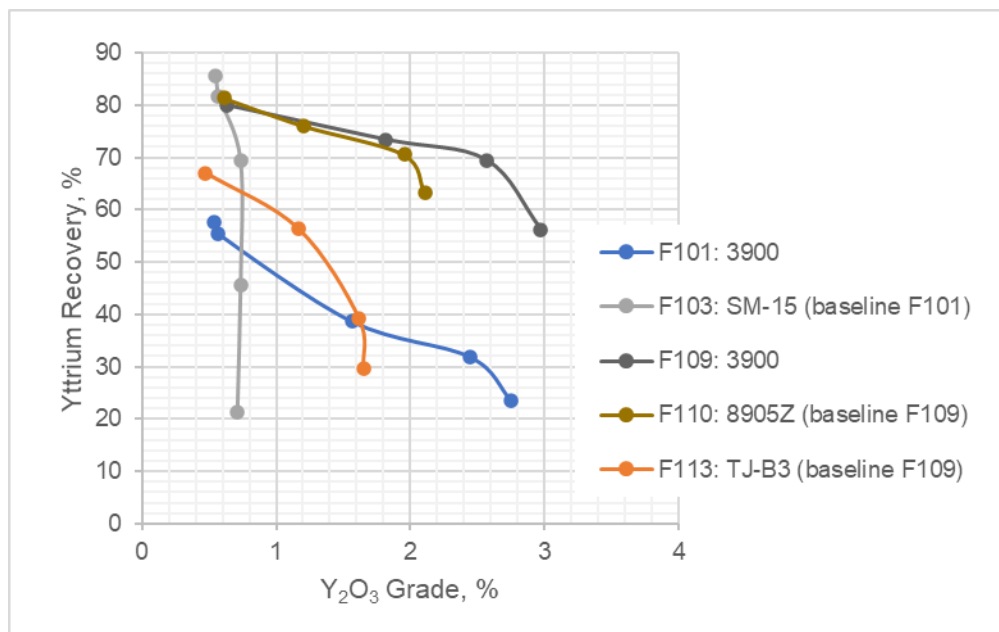


Figure 13-16 Effect of Collectors Testing

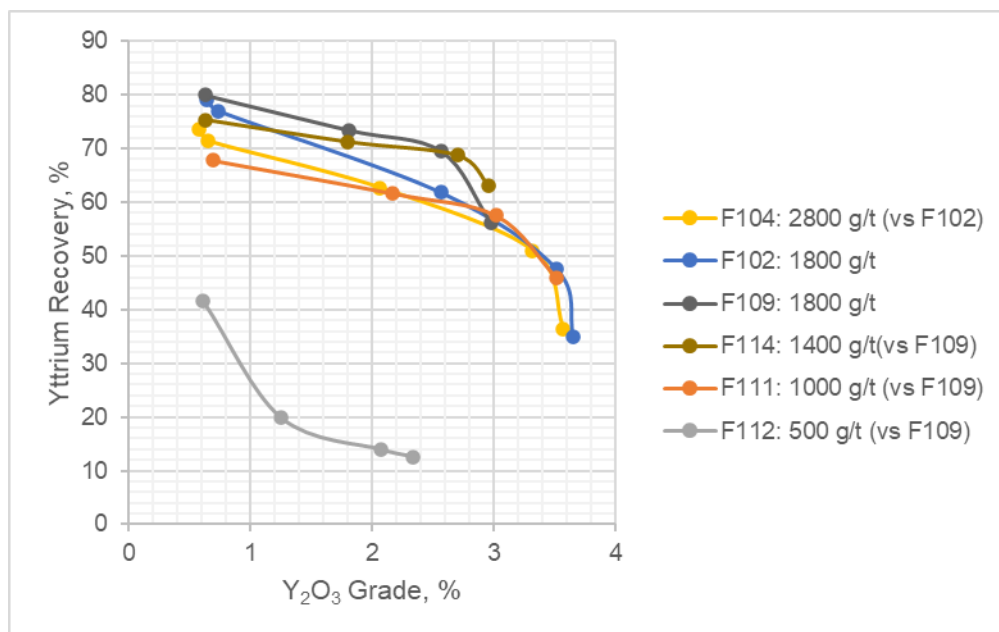


Figure 13-17 Effect of Collector Dosages

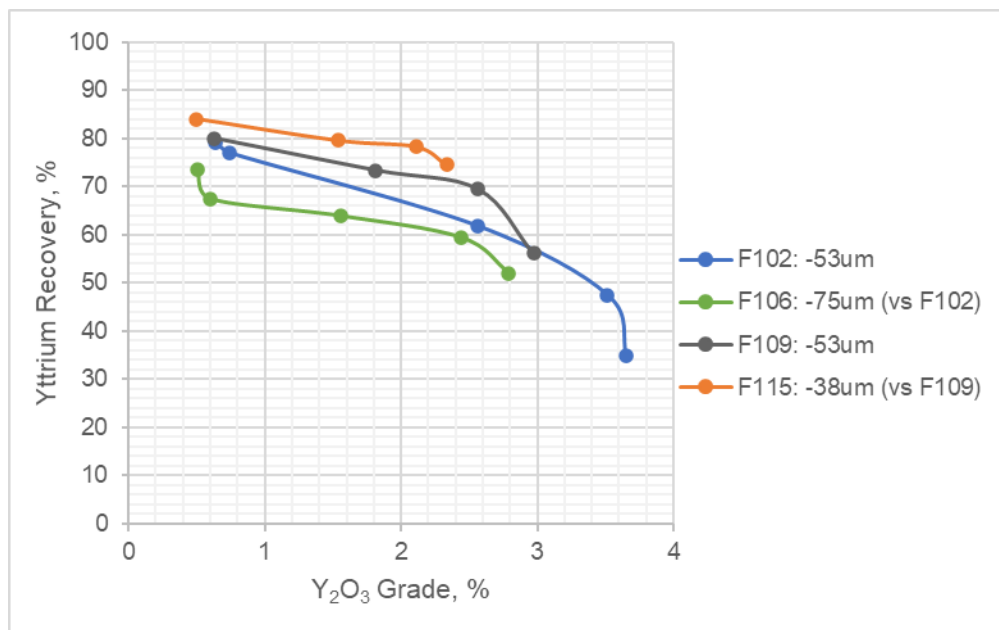


Figure 13-18 Effect of Grind

Additional thrifting testwork was done to determine flotation performance at several conditions: 1) reduced collector dosage between the selected point and the lower envelope (1000 g/t -1800 g/t); 2) effect of even finer grind to determine lower limit ($P_{100}=38\mu\text{m}$); 3) alternative collectors that may perform similar to Florrea with potential cost savings.

The recovery performance of the heavy rare earth oxides exceeded that of the light rare earth oxides (see Figure 13-19).

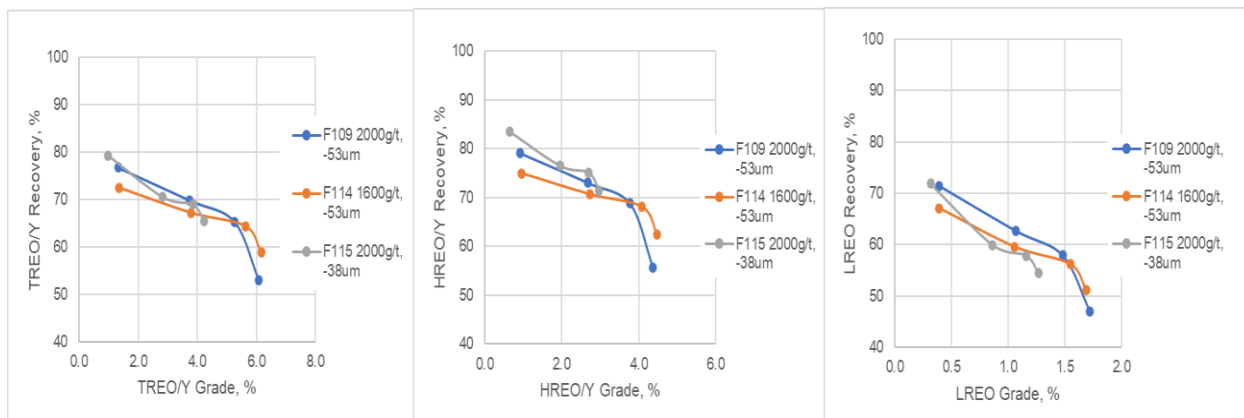


Figure 13-19 Flotation Results of the TREO, HREO and LREO

Bulk flotation tests were done in quadruplicate to produce a flotation concentrate sample for downstream amenability hydrometallurgical testing at close to optimum flotation conditions.

Iron removal steps in a WHIMS (wet high intensity magnetic separator) were done in both pre- and post flotation configurations with similar performances and low rare earth losses (~2%).

The produced flotation concentrates demonstrated repeatable flotation performances on the low grade feed material. See Figure 13-20. The average cleaner flotation concentrate from the bulk test runs (CP101 to CP104) was produced at a mass pull of 3% with a grade of ~5.6% TREO and recovery of 67%. Upgrade ratio of 25 times from the low head grade of 0.22% TREO. CaO and Fe₂O₃ grades of ~9% and 34% respectively in the concentrate.

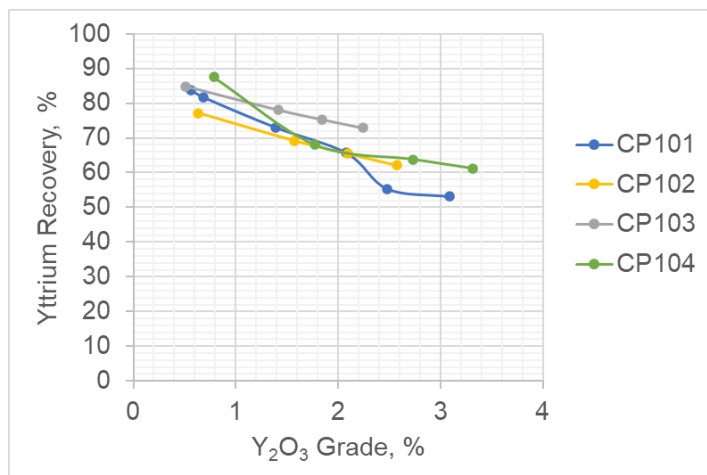


Figure 13-20 Flotation Results of Bulk Flotation Tests (CP101 to CP104)

Preferential upgrading of the HREO's was demonstrated as compared to the LREO's. The recovery of HREO (Eu to Lu) was in the 64 to 72% range. However, the recovery of the light rare earth elements (La to Sm) was lower in the 56 to 58% range.

Flotation concentrate was subjected to the downstream hydrometallurgical testing for validation of process route and efficiencies as per the previous acid bake test regime, where the previous hydrometallurgical testwork at SGS (see Section 13.3) had demonstrated the acid bake route is preferred due to lower reagent costs and higher recovery of the heavy rare earths compared to the caustic crack route.

A total of three acid bake and water leach tests were completed on the bulk flotation concentrate to investigate the dissolution of rare earth elements (REE) and the behaviour of gangue minerals through the addition of sulphuric acid at elevated temperatures (300°C) and at a range of acid dosages (1-1.5 t/t concentrate basis). Under previously determined optimum conditions (2021 test program at SGS Canada), these tests showed very good REE recoveries with 96% for yttrium, 95% for dysprosium and 94% for terbium. The water leach recovery of tracked elements for these three acid bake and water leach tests are shown in Figure 13-4 with a comparison to the 2021 bulk acid bake and water leach test results (WL-AB12).

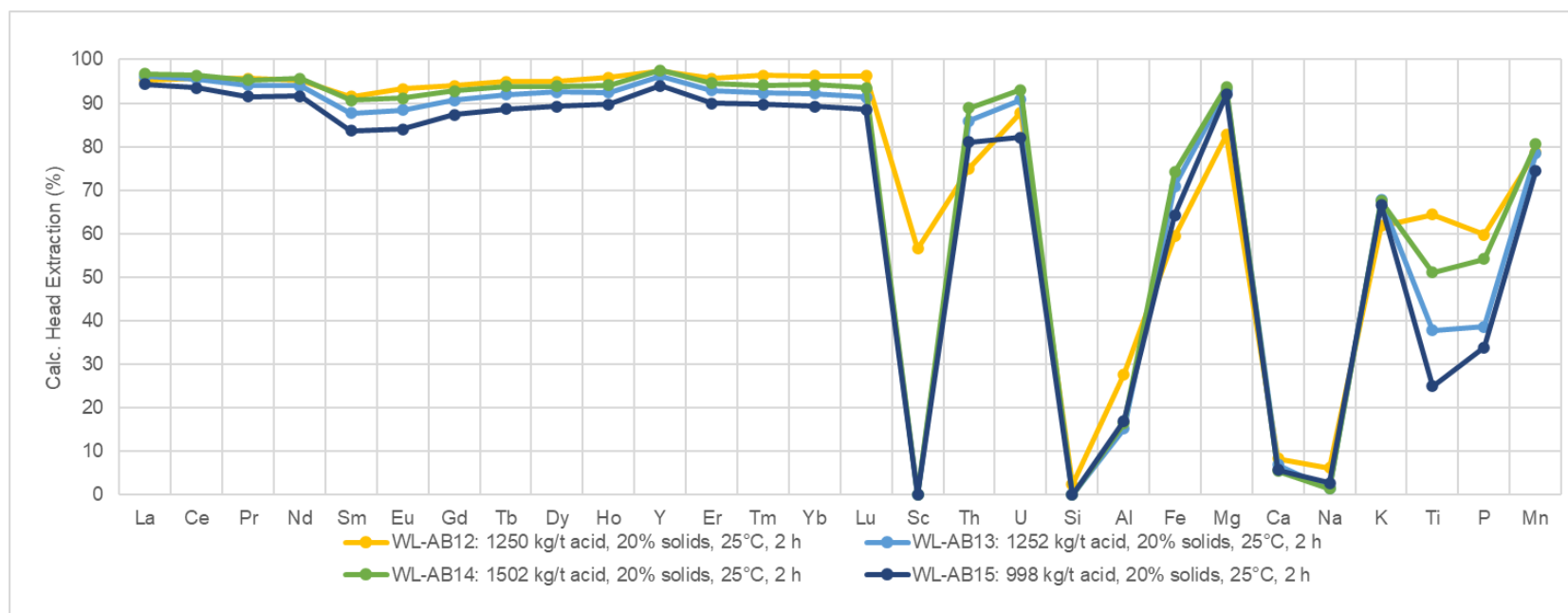


Figure 13-21 Acid Bake and Water Leach Extraction

Results of the impurity removal and crude REE precipitation tests on the leached solutions corroborated chemistries with the previous test programs on the two flotation concentrates with low co-precipitation of RE's in the impurity removal stage (between 1-9% as compared to between 1-15% in previous) with similar precipitation of impurities.

While the results are positive, there remains room to optimise these processes regarding OPEX and CAPEX as well as recoveries in continuous pilot plant testing during pre-feasibility study.

13.5 Basis of Design

The mineralogy and metallurgy of this orebody is complex. Multiple stages in the process route have a compounding effect on overall efficiency and recovery which, despite reducing advancing tonnage or removing deleterious elements advancing down the process route, recovery of valuable metals remains one of the biggest contributors to NPV, along with metal prices. Operating costs with reagents and power consumption being the largest portion, also contribute to project economics.

This therefore needed to inform the flowsheet selection for process engineering and design for PEA selection.

The conclusion from the flotation program on the low grade run of mine sample demonstrated that the flotation mechanism on the low grade ROM ore is promising, even with full tonnage reporting to the mill and flotation plant. That with the successful demonstration of the downstream hydrometallurgical testing on the concentrate produced from the low grade flotation formed the basis of design for the PEA flowsheet.

The repeatability of the results on the direct flotation with the thrifed SGS flotation regime confirm application for engineering design and costing at high upgrade ratios (~22 times) at acceptable recovery (between 64 to 67%) and grade (~5% TREO) at low mass pull (2.6 to 2.9%) for PEA evaluation. Figure 13-22 shows the selected data points from the flotation testwork based on detailed rare earth ICP analyses.

The flotation conditions include:

- Grind size P_{100} of 53 μ m.
- Temperature 50 degrees C.
- High intensity conditioning at 50% solids and 1800 rpm.
- Florrea 3900 collector at 1800g/t dosage
- Florrea 3000 co-collector at 90 g/t
- Calgon depressant at 100 g/t
- pH ~8.

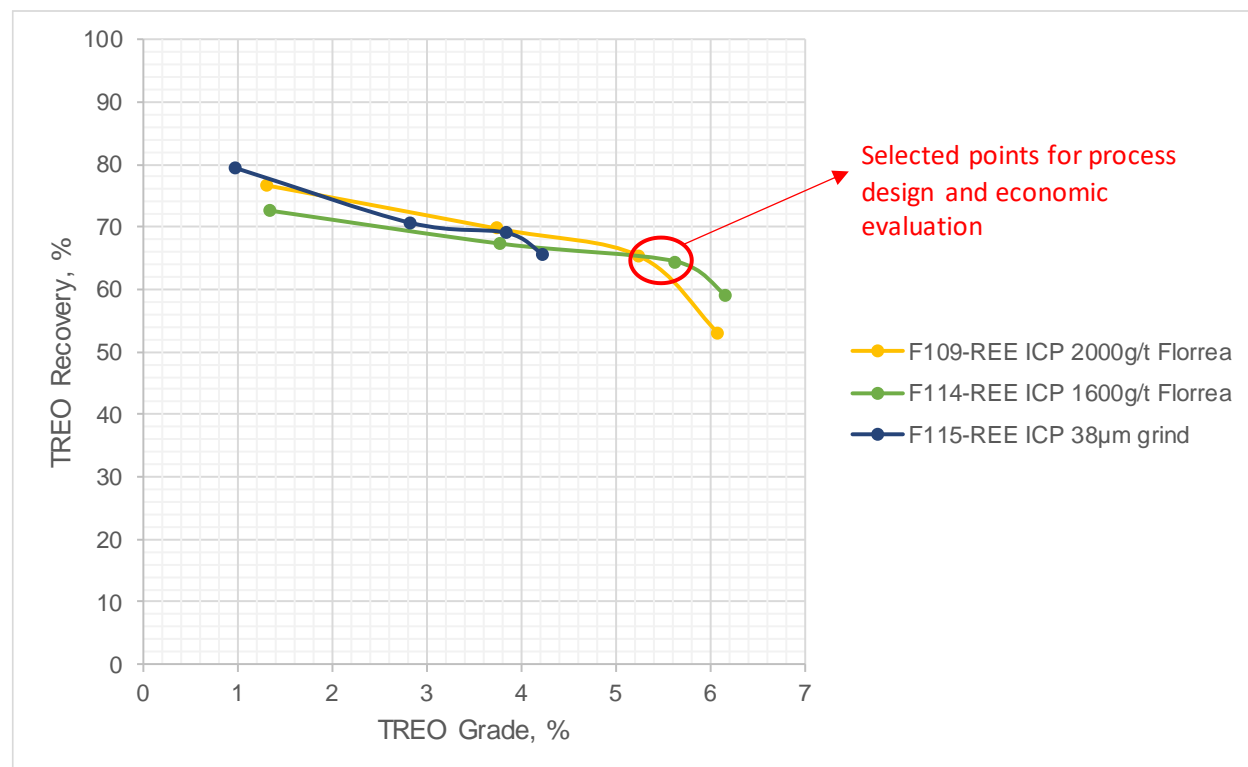


Figure 13-22 Flotation Test results for Process Design and Economic Evaluation

Flotation recovery for the economic evaluation is split between the light and the heavy rare earths according to the detailed REE ICP analysis from Test F114. See Table 13-3.

For flotation, the LREO recovery is 56.2%; HREO recovery is 68.1% with resultant TREO recovery at 64.4%.

Table 13-3 Flotation Test F114 Summary REE ICP Results

Products	Weight %	Assay, %									
		Y, %	La g/t	Ce g/t	Pr g/t	Nd g/t	Sm g/t	Eu g/t	Gd g/t	Tb g/t	Dy g/t
2nd CI Con 1	2.2	2.49	3540	6390	662	2400	750	345	1940	451	3570
2nd CI Con 1-2	2.6	2.27	3243	5862	608	2205	689	317	1780	413	3259
1st CI Con 1-3	4.1	1.51	2209	4003	416	1505	477	215	1206	278	2188
Ro Conc 1-4	12.4	0.53	817	1485	155	564	179	80	437	100	770
		Assay, % Cont									
		Ho g/t	Er g/t	Tm g/t	Yb g/t	Lu g/t	U g/t	Th g/t	LREO	HREO/Y	TREO
2nd CI Con 1	2.2	758	2340	309	1860	249	104	3430	1.7	4.5	6.2
2nd CI Con 1-2	2.6	693	2137	283	1706	228	99	3223	1.5	4.1	5.6
1st CI Con 1-3	4.1	465	1432	190	1147	154	73	2314	1.1	2.7	3.8
Ro Conc 1-4	12.4	164	502	67	403	55	28	931	0.4	1.0	1.3
Products	Weight %	Distribution, %									
		Y	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy
2nd CI Con 1	2.2	64.2	51.6	51.3	51.5	51.8	47.0	50.5	53.2	55.7	59.0
2nd CI Con 1-2	2.6	70.0	56.5	56.3	56.6	56.9	51.7	55.5	58.3	61.0	64.4
1st CI Con 1-3	4.1	72.5	59.7	59.7	60.1	60.3	55.5	58.5	61.4	63.8	67.1
Ro Conc 1-4	12.4	76.5	67.0	67.1	68.0	68.5	62.9	65.6	67.3	69.3	71.6
		Distribution, %									
		Ho	Er	Tm	Yb	Lu	U	Th	LREO	HREO/Y	TREO
2nd CI Con 1	2.2	63.5	59.6	61.0	61.0	58.4	57.0	38.1	51.2	62.4	58.9
2nd CI Con 1-2	2.6	69.5	65.1	66.8	66.9	63.9	64.9	42.8	56.2	68.1	64.4
1st CI Con 1-3	4.1	72.3	67.8	69.6	69.9	67.0	74.0	47.7	59.6	70.7	67.2
Ro Conc 1-4	12.4	77.3	71.9	73.9	74.4	72.0	86.0	58.2	67.1	74.9	72.5

The overall hydrometallurgical section recovery for economic evaluation is 93.5%.

The acid-bake water leach recovery is 96.2%. Future testwork for optimization of acid additions in this process stage is likely to improve recovery at least by 1.5 to 2%. Recovery over impurity removal stage is 97.7%; REE precipitation stage 99.5% with 100% recovery over the remaining hydrometallurgical stages.

Overall combined flotation and hydrometallurgical LREO recovery is 53%; HREO recovery is 64% with TREO recovery at 64.4%.

Sensitivities in the economic evaluation show the impact of recovery. Refer to Section 22.7.

Figure 13-23 shows the flowsheet schematic for the process flowsheet for the treatment of the low grade ROM ore into mixed rare earth oxide product.

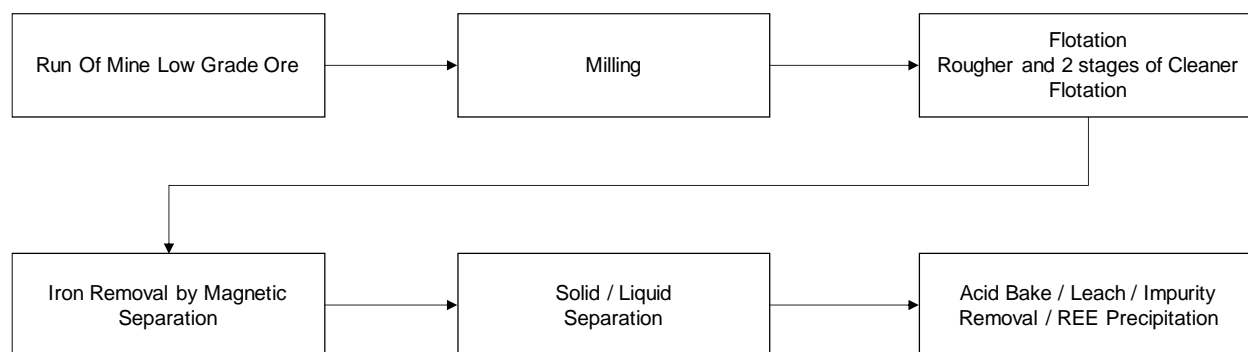


Figure 13-23 Basis of Design Flowsheet Schematic

An engineering sizing, capital and operating costing validation exercise was carried out by SGS Bateman for the revised direct low grade flotation flowsheet for the PEA economic evaluation on the 2021 mine plan. See Section 17.

14 MINERAL RESOURCE ESTIMATE

On behalf of NMI, MSA completed a Mineral Resource Estimate for the Area 4 and Area 2B deposits at the Lofdal Heavy Rare Earths project.

To the best of the QP's knowledge there are currently no title, legal, taxation, marketing, permitting, socio-economic or other relevant issues that may materially affect the Mineral Resource described in this Technical Report.

The Mineral Resources presented herein, with an effective date of May 12, 2021, represent an update to the previous Mineral Resource Estimate dated the July 31, 2012. The Mineral Resource estimate incorporates drilling data from holes completed by NMI from June 2010 until December 2020 inclusive, which in the QP's opinion were collected in accordance with The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Exploration Best Practices Guidelines", 2018.

The Mineral Resource was estimated using the 2019 CIM "Best Practice Guidelines for Estimation of Mineral Resources and Mineral Reserves" and classified in accordance with the "2014 CIM Definition Standards". It should be noted that Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

The Mineral Resource estimate was conducted using Datamine Studio RM software, together with Microsoft Excel, JMP and Snowden Supervisor for data analysis. The mineral resource estimation was carried out by Mr. Rui Goncalves under the supervision of Mr. Jeremy Witley (the Qualified Person).

14.1 Mineral Resource Estimation Database

The database provided by NMI to inform the Mineral Resource Estimate consists of:

- Information from diamond drillholes in the form of:
 - Collar surveys.
 - Downhole surveys.
 - Sampling and assay data.
 - Geology logs.
 - Specific gravity (SG) measurements.
 - Rock Quality Designation (RQD) measurements.
- Information from trench data.
- Topographic surveys were provided as contours in GIS shapefile format.

The drillhole and trench data were provided in Microsoft Excel files that were extracted from a Microsoft Access database managed by NMI. The principal sources of information used for the estimate are exploration drilling programs conducted by NMI from 2010 to 2012, in 2015 and in 2020. The trench data were used to guide the position of mineralised veins near surface but were not included in the grade estimation due to concerns on the representivity of the sampling.

A total of 173 drillholes were drilled and 28 trenches were dug at Area 4. One drillhole (NLOFDH4007) was excluded, due to the absence of downhole surveys. Thirteen drillholes located 800 m to the northeast of the main drilling area were too far from the main area to be considered. Additionally, seven holes drilled within the plane of the mineralisation at Area 4 were used in defining the wireframes for the mineralization but were excluded from the mineral resource estimate as these samples are not representative of the mineralised package.

The dataset for Area 2B consists of 46 drillholes and 25 trenches.

The cut-off date for inclusion of data into this estimate is March 13, 2021 at which time there was no outstanding information for Area 4 and Area 2B as the drilling was completed in 2020.

14.2 Exploratory Analysis of the Raw Data

The dataset examined consisted of sampling and logging data from diamond drillholes. The following attributes are of direct relevance to the estimate:

- REE oxide grades in ppm: Lanthanum (La_2O_3), Cerium (Ce_2O_3), Praseodymium (Pr_2O_3), Neodymium (Nd_2O_3), Samarium (Sm_2O_3), Europium (Eu_2O_3), Gadolinium (Gd_2O_3), Terbium (Tb_2O_3), Dysprosium (Dy_2O_3), Holmium (Ho_2O_3), Erbium (Er_2O_3), Thulium (Tm_2O_3), Ytterbium (Yb_2O_3) and Lutetium (Lu_2O_3), as well as Yttrium (Y_2O_3).
- Specific Gravity (SG) measurements.
- Rock Quality Designation (RQD) measurements.

14.2.1 Validation of the data

MSA undertook a high-level validation process which included the following checks:

- Examining the sample assay, collar survey, down-hole survey and geology data to ensure that the data were complete for all the drillholes,
- Examining the de-surveyed data in three dimensions to check for spatial errors,
- Examination of the assay and density data to ascertain whether they were within expected ranges,
- Checks for “FROM-TO” errors, to ensure that the sample data do not overlap one another or that there are no unexplained gaps in the sampling.

The data validation exercise revealed the following:

- There are no unresolved errors relating to missing intervals and any overlaps in the drillhole logging data. Absent assays correspond to intervals where no samples were taken.
- Examination of the drillhole data in three dimensions shows that the collars of the drillholes surveyed by DGPS plot in their expected positions relative to the topographic surface derived from the contour data.
- Extreme assays were checked, and no errors were found.
- Two methods were used to derive density measurements. Density measurements on the drillhole data pre-dating the 2020 campaign made use of downhole geophysical probe surveys, while the more recent campaign made use the Archimedes principle. A statistical comparison for Area 2B between these two methods indicates that the downhole densities reported higher average values and are of a statistically different population (Figure 14-1). These differences were less pronounced for Area 4 but resulted in the downhole geophysical probe densities being excluded from the estimation process.
- Several sampled intervals reported core recoveries above 100%. These were reported to NMI and rectified where possible. Abnormal core recoveries were set to 100% during the core recovery analysis.

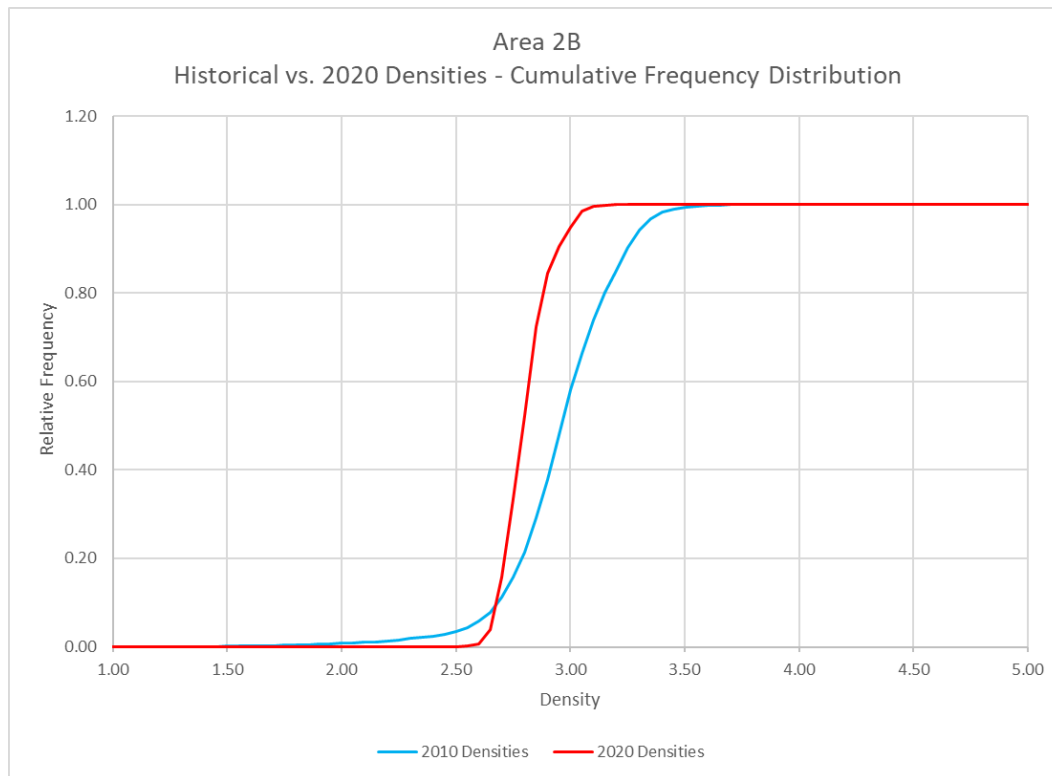


Figure 14-1 Cumulative Frequency Distribution Comparison Between Historical Downhole Probe and 2020 Archimedes Densities

14.2.2 Statistics of the Raw Sample Data

14.2.2.1 Sample lengths

A total of 10,082.09 metres of drillhole samples were assayed for Area 4 and 2,202.52 metres were assayed for Area 2B. Sample lengths vary from 0.09 m to 6.00 m in Area 4 and 0.19 m to 2.23 m in Area 2B with the dominant sample length being 1 metre for both areas as illustrated in the histograms in Figure 14-2.

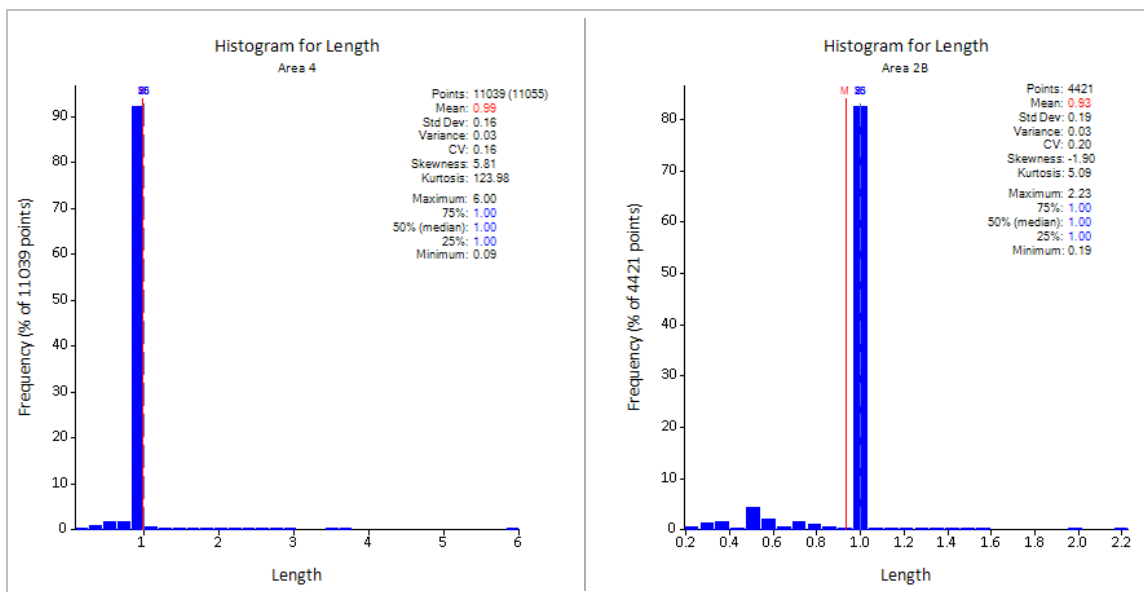


Figure 14-2 Histogram of Sample Lengths for Area 4 and Area 2B

14.3 Bivariate Analysis

The relationships between individual rare earth oxides were studied using scatterplots to understand the existence of any correlation between variables which should be preserved in the mineral resource estimate. A strong linear relationship between the grades of certain REE exist, with some elements displaying this relationship with multiple elements. As an example, Figure 14-3 shows the relationships of Tb_2O_3 with Dy_2O_3 and Ho_2O_3 for Area 4.

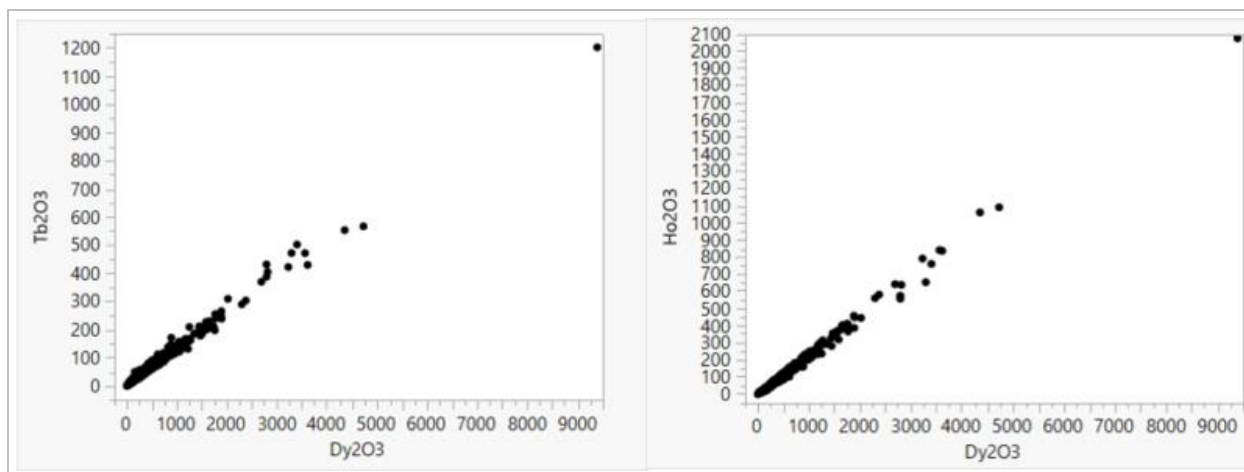


Figure 14-3 Scatter Plot of Sample Tb_2O_3 , Dy_2O_3 and Ho_2O_3 for Area 4

14.3.1 Core Recovery

The average core recovery is 93% for Area 4 and 94.64% for Area 2B. A broad depth-core recovery relationship exists showing increasing core recovery with increasing depth (Table 14-1). There is no discernible relationship between grade and core recovery.

Table 14-1 Core Recovery in Percent per Depth Interval below Surface

Area	Recovery in Percent per Depth Interval (m)						
	0 – 5	5 – 10	10 – 20	20 – 30	30 – 40	40 – 50	Overall
Area 4	64.8	82.9	87.0	91.1	93.9	92.6	93.36%
Area 2B	70.5	70.9	92.9	96.0	94.9	95.7	94.64%

14.4 Geological Modelling

Leapfrog Geo was used to generate three-dimensional volumes and surfaces representing the mineralised zones and weathering surfaces.

14.4.1 Topography

A topographic survey was provided by NMI which was conducted by UAS Flightec Solutions (Pty) Ltd. This survey consists of topographic contours which were used to generate a three-dimensional surface in Leapfrog Geo.

The surveyed drillhole collars correspond well with the resultant topographic surface. The trench data was draped onto the topographic surface, which was used to guide the modelling of the mineralised wireframes near surface.

14.4.2 Mineralised Zones

The modelling procedure examined the continuity of dysprosium oxide (Dy_2O_3) grades along strike and down-dip to generate mineralised wireframes using a statistical threshold of 10 ppm Dy_2O_3 for Area 4 and 12 ppm Dy_2O_3 for Area 2B. The use of these thresholds resulted in generally continuous zones that form a suitable framework for block model grade estimation. The modelled zones (or domains) were individually coded into the drillhole data and volumes were generated using Leapfrog Geo. Where necessary, manual edits were incorporated to provide for geologically realistic shapes.

The modelling resulted in fourteen individual mineralised zones for Area 4 (Figure 14-4) and seven for Area 2B (Figure 14-5).

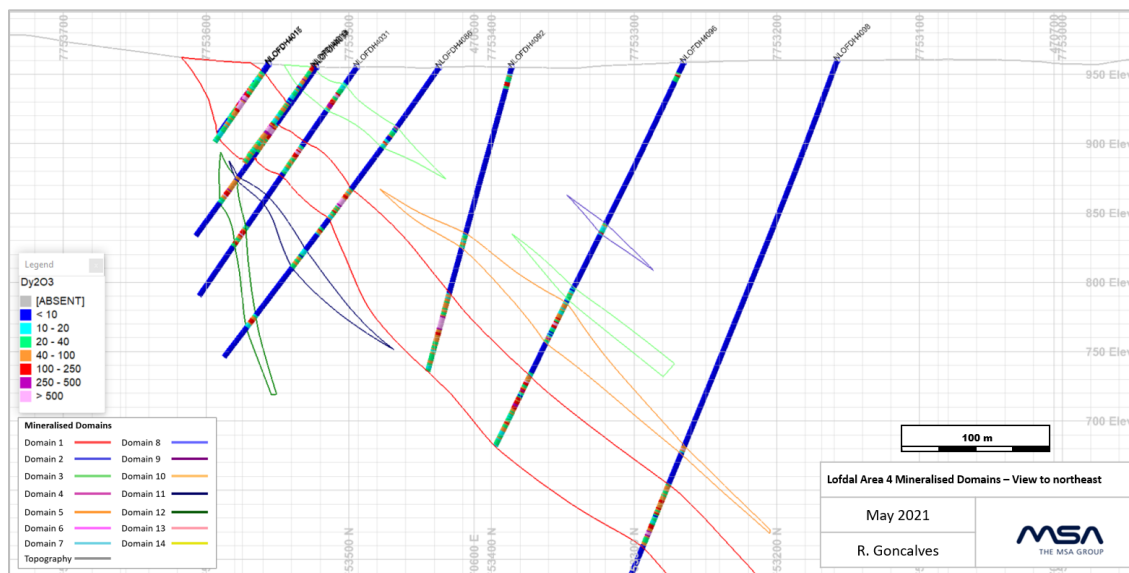


Figure 14-4 Cross-Section Illustrating Modelled Mineralised Zones for Area 4

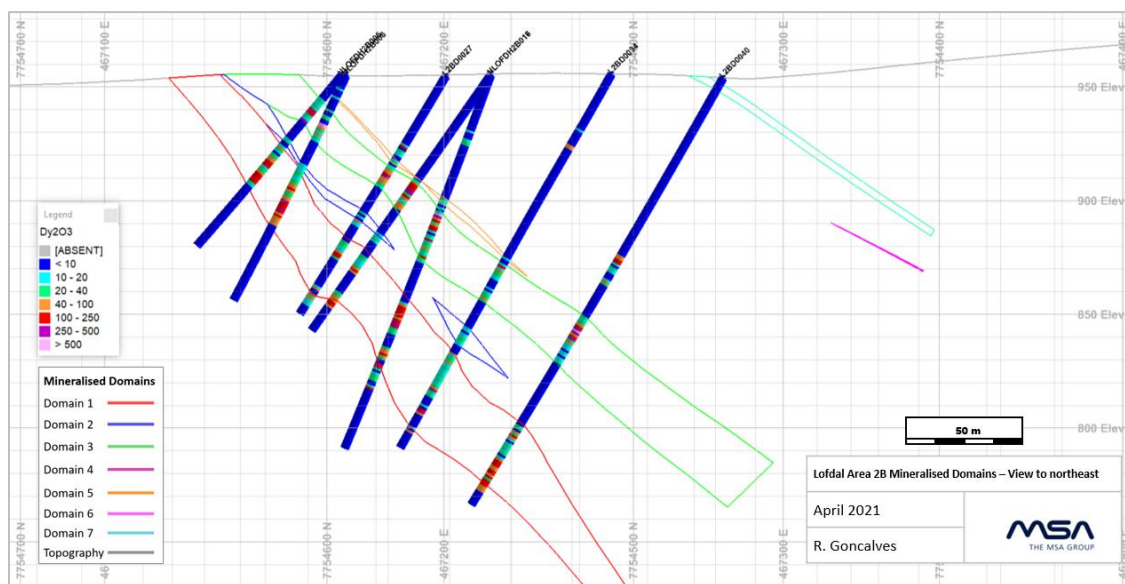


Figure 14-5 Cross-Section Illustrating Modelled Mineralised Zones for Area 2B

14.4.3 Oxidation/Weathering Surface

Due to the lack of detailed visual weathering logging, the rock quality designation (RQD) values were used as a proxy for weathering. The assumption being that lower RQD values will be associated with alteration due to weathering.

Log-probability plots were used to identify a RQD threshold value of between approximately 40 % to 45 % to represent the threshold on which to base a partially weathered surface. This threshold correlates well with areas near surface and highlighted zones of deeper weathering associated with structural features (Figure 14-6).

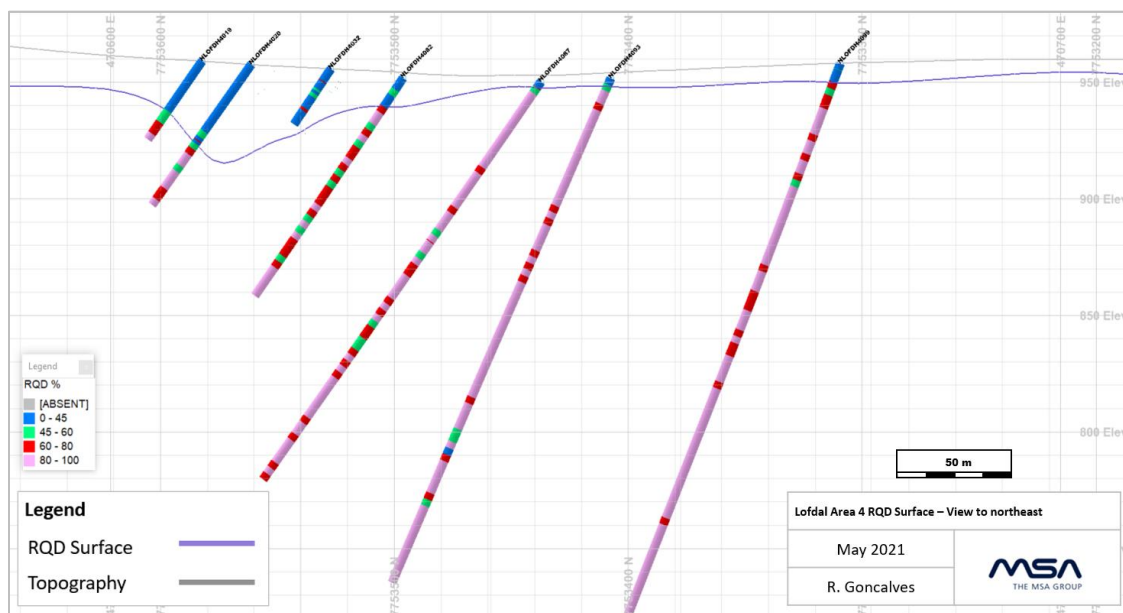


Figure 14-6 Cross-Section Illustrating Modelled RQD Weathering Surface for Area 4

14.5 Statistical Analysis of the Composite Data

Samples were composited to one metre lengths based on the dominant sample interval. Compositing was carried out inside the mineralised domain and statistics were analysed for the fifteen rare earth oxides. Log histograms of the composites for total rare earth oxides (TREO %), heavy rare earth oxides (HREO %), light rare earth oxides (LREO %) and dysprosium oxide (Dy_2O_3 ppm) are shown in Figure 14-7 for Area 4 and Figure 14-8 for Area 2B.

The following observations were made:

- The distributions for the individual REO grades are positively skewed.
- Area 4 grades are less variable than Area 2B as exhibited by the coefficient of variation for TREO grade.
- For Area 4, the CV of HREO grade is higher than for LREO, for Area 2B the converse is true, with the heavy oxides being less variable than the light oxides.
- The average grade of Dy_2O_3 ppm in the mineralised domain is 84 ppm for Area 4 and 63 ppm for Area 2B.
- Both deposits show similar proportions of HREO and LREO in TREO (Table 14-2).

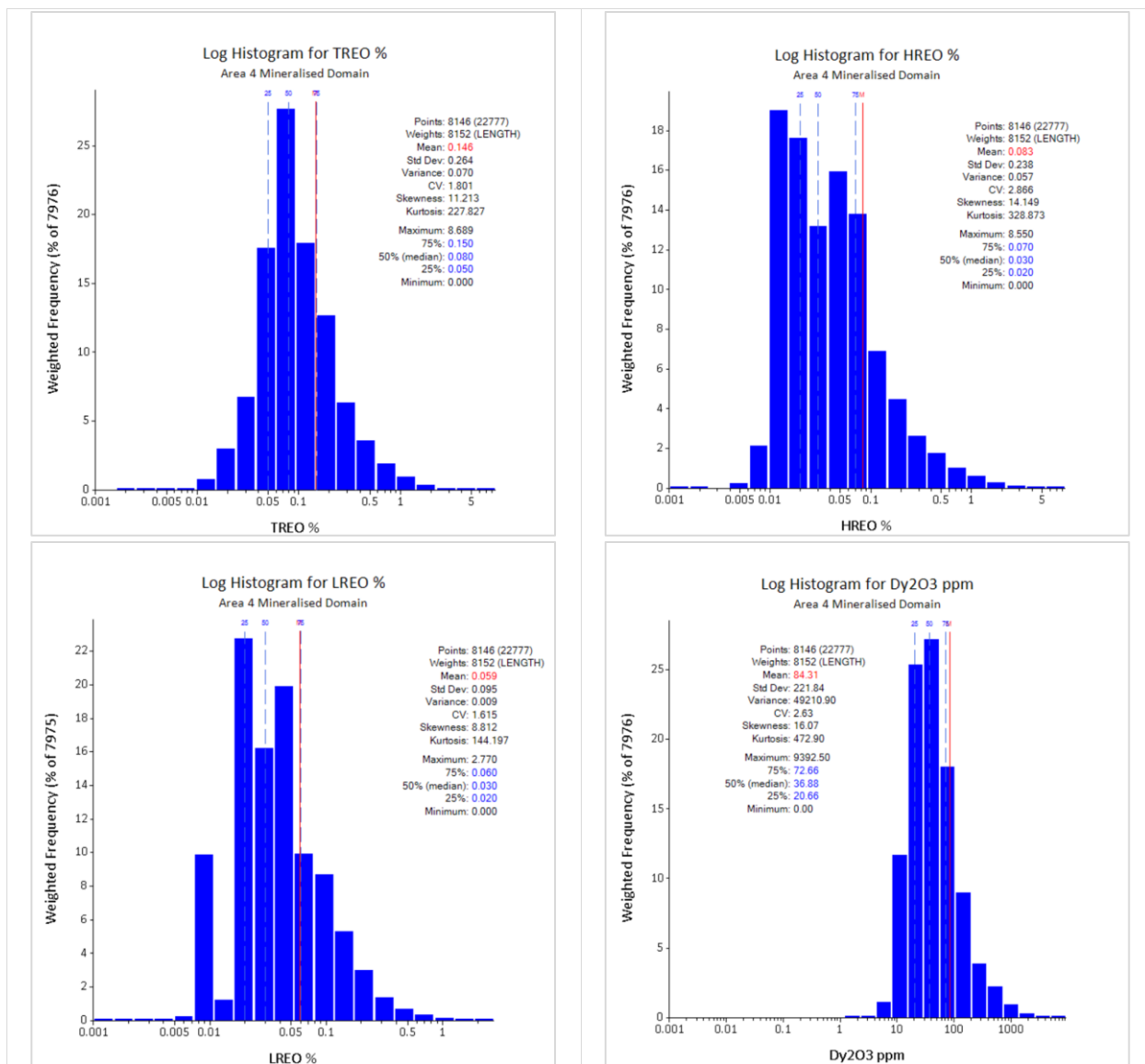


Figure 14-7 Histograms of TREO, LREO, HREO and Dy₂O₃ ppm for Area 4

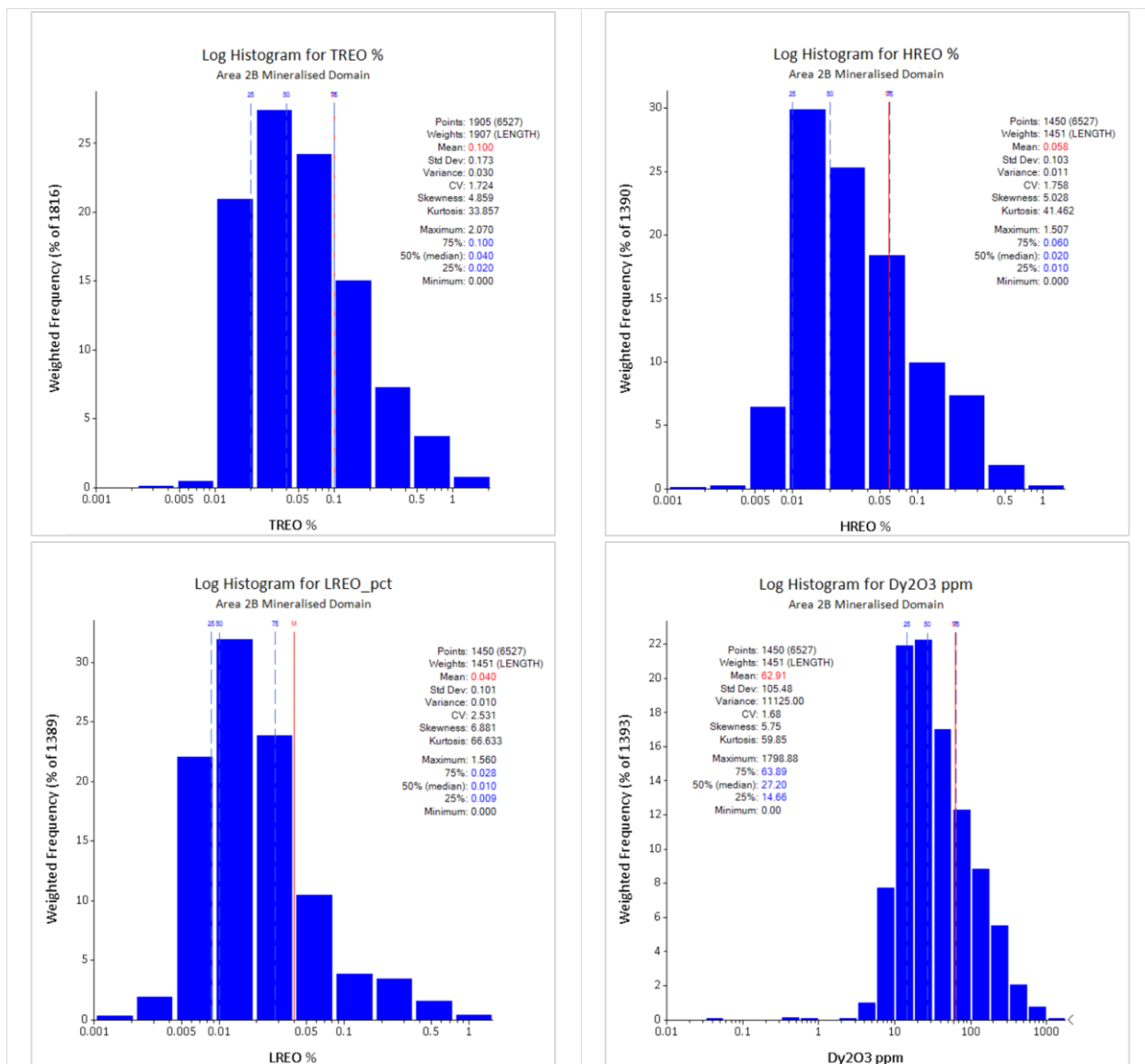


Figure 14-8 Histograms of TREO, LREO, HREO and Dy₂O₃ ppm for Area 2B

Table 14-2 Individual REO Proportions for Area 4 and Area 2B

REO	Percentage of REO in TREO	
	Area 4	Area 2B
La ₂ O ₃	10.93%	10.88%
Ce ₂ O ₃	19.70%	18.43%
Pr ₂ O ₃	2.09%	2.03%
Nd ₂ O ₃	7.67%	8.59%
Sm ₂ O ₃	2.34%	3.68%
Total LREO	42.73%	43.61%
Eu ₂ O ₃	0.91%	1.36%
Gd ₂ O ₃	3.78%	4.76%
Tb ₂ O ₃	0.82%	0.91%
Dy ₂ O ₃	5.52%	5.66%
Ho ₂ O ₃	1.17%	1.11%
Er ₂ O ₃	3.40%	3.19%
Tm ₂ O ₃	0.51%	0.47%
Yb ₂ O ₃	3.09%	2.98%
Lu ₂ O ₃	0.45%	0.43%
Y ₂ O ₃	37.62%	35.52%
Total HREO	57.27%	56.39%

14.5.1 Cutting and Capping

An outlier analysis was completed on the composite data for the individual mineralised domains and capping was applied where applicable. The capping exercise was cognisant of the bivariate relationship between rare earth oxides and impacted only between one and ten samples per domain.

14.6 Geostatistical Analysis

14.6.1 Semivariograms

Experimental semivariograms were calculated on the normal scores transformed composite data for total HREO and total LREO grades using Datamine Supervisor (previously Snowden Supervisor) software. Normalised semivariograms were calculated so that the sum of the variance is equal to one.

Variogram maps for Area 4 indicated the presence of anisotropy with the longest direction along strike (240°). The across strike continuity was the shortest as expected in tabular vein style deposit. Double structured, spherical semivariogram models were modelled for both HREO and LREO grade. Table 14-3 summarises the semivariogram parameters for Area 4.

Table 14-3 Semivariogram Parameters for Area 4

Attribute	Rotation Angle			Rotation Axis			Nugget Effect (C0)	Sill 1 (C1)	Range of First Structure (m)			Sill 2 (C2)	Range of Second Structure (m)		
	1	2	3	1	2	3			1	2	3		1	2	3
HREO	150	45	10	Z	X	Z	0.19	0.70	60	40	7	0.11	200	120	15
LREO	150	45	10	Z	X	Z	0.30	0.47	50	40	4	0.23	200	120	25

Data for Area 2B did not suggest the presence of strong anisotropy as observed for Area 4, with the LREO grade variogram fans indicating isotropic semivariogram models in directions 1 and 2 (within the plane of mineralisation). Single structure, spherical models were fitted to the experimental points for both HREO and LREO. Table 14-4 summarises the semivariogram parameters for Area 2B.

Table 14-4 Semivariogram Parameters for Area 2B

Attribute	Rotation Angle			Rotation Axis			Nugget Effect (C0)	Sill 1 (C1)	Range of First Structure (m)			Sill 2 (C2)	Range of Second Structure (m)		
	1	2	3	1	2	3			1	2	3		1	2	3
HREO	140	50	0	Z	X	Z	0.15	0.85	60	55	5	-	-	-	-
LREO	140	50	0	Z	X	Z	0.22	0.78	60	60	6	-	-	-	-

14.7 Block Modelling

Block models were generated for each project using 10 m by 10 m blocks in the X (easting) and Y (northing) direction and 5 m blocks in the Z (elevation) direction. The block model was not rotated.

Sub-celling was applied to optimally fill the modelled wireframes, resulting in minimum sub-cell of 2 m x 2 m x 1 m in X, Y and Z, respectively.

The common origins for the block models for Area 4 and Area 2B are shown Table 14-5.

Table 14-5 Block Model Origins Area 4 and Area 2B

Area	Easting (m)	Northing (m)	Elevation (m)
Area 4	469,500	7,752,800	500
Area 2B	466,900	7,754,300	600

14.7.1 Estimation Parameters

The search distance and rotation angles were based on the semivariogram. Kriging Neighbourhood

Analysis (KNA) was used to determine the minimum and maximum number of samples to be included in the search neighbourhood and the appropriate number of discretisation points to be used in a parent block. The KNA exercise looked at Kriging Efficiency as a metric of estimation quality and slope of regression was used to quantify the level of conditional bias when selecting the optimal parameters.

The search parameters are shown in Table 14-6 for Area 4 and Table 14-7 for Area 2B.

Table 14-6 Search Parameters for Area 4

Attribute	Rotation Angles			Rotation Axis			Search Distance (m)			Number of Composites	
	1	2	3	1	2	3	1	2	3	Min	Max
HREO	150	45	10	Z	X	Z	200	120	15	6	12
LREO	150	45	10	Z	X	Z	200	120	25	6	12

Table 14-7 Search Parameters for Area 2B

Attribute	Rotation Angles			Rotation Axis			Search Distance (m)			Number of Composites	
	1	2	3	1	2	3	1	2	3	Min	Max
HREO	140	50	0	Z	X	Y	60	55	6	5	10
LREO	140	50	40	Z	X	Y	60	60	6	5	10

Block grades were estimated in three passes, with the first pass using the search parameters shown in Table 14-6 and Table 14-7. The second search was expanded by a factor of 1.5 with a minimum of 5 and maximum of 12 samples included for Area 4 and a minimum of 4 and maximum of 10 for Area 2B. The third search made use of an expansion factor of 10, with a minimum of 4 and maximum of 16 samples included for Area 4 while Area 2B included a minimum of 3 and maximum of 20 samples in the search neighbourhood. Estimates using the third search parameter are of relatively low confidence with the parameters designed to estimate local average values.

Ordinary Kriging (OK) was used for the estimation of the rare earth oxides. The modelled semivariogram and search parameters were applied to the individual rare earth oxides. Estimates were completed for each individual mineralised zone comprising of fourteen zones for Area 4 and seven for Area 2B.

Density was estimated independently for each zone using inverse distance weighting and applying the same search parameters as the HREO attribute. Where blocks were not interpolated with a density estimate, the average value of the zone was assigned.

Dynamic anisotropy was used to align the search ellipsoids to account for local changes in the orientation of the mineralised zones along strike and dip. The dynamic search for each zone was orientated using trend surfaces created in Leapfrog Geo.

14.8 Validation of Estimates

The models were validated by:

- Comparison of the global estimates against the average composite sample grades.
- Swath plot validation.
- Visual examination of the input data against the block model estimates.

The average grade of the block model for each individual zone were validated against the declustered composite grades (declustered to 100 mX by 100 mY by 50 mZ). Globally, the estimated block grades compare favourably to the input data, with relative differences less than ten percent for the main mineralised zones. Larger percentage differences are noted for the smaller zones, which can be attributed to factors such the spatial arrangement and paucity of the data.

Swath plot validations in the X, Y and Z directions were used to locally validate the block estimates against the declustered sample composites. No material biases in the estimates of the individual elements were identified. Examples of a swath plot validation are shown for Dy₂O₃ in Figure 14-9.

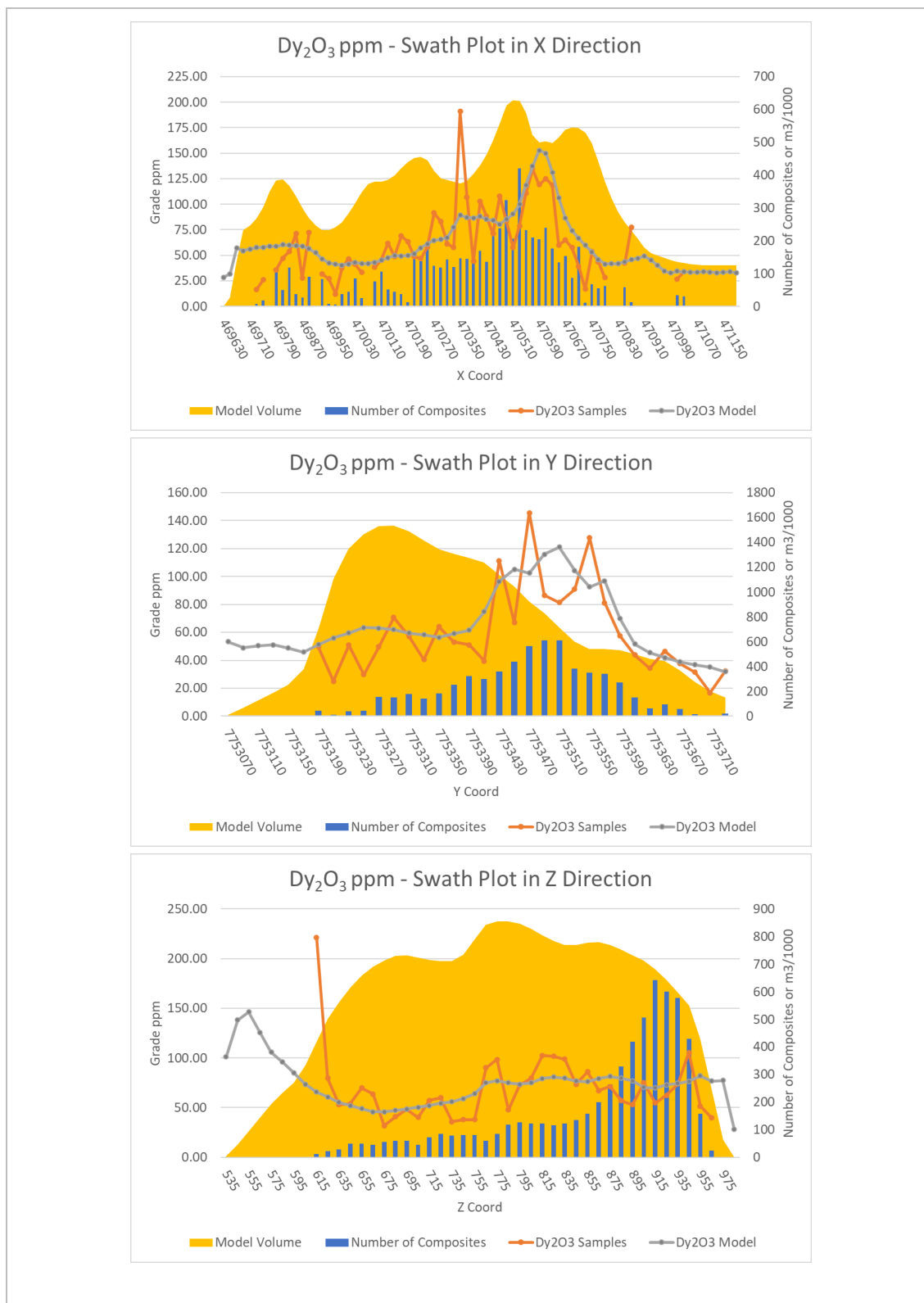


Figure 14-9 Swath Plot Validation for Dy_2O_3

The block model was examined visually to ensure that the drillhole grades were locally well represented by the model and it was found that the model validated reasonably well against the data. The model is less well locally representative of the data when extrapolating down dip, which was considered in the classification. Examples of this validation for Dy_2O_3 ppm are illustrated for Area 4 (Figure 14-10) and Area 2 (Figure 14-11).

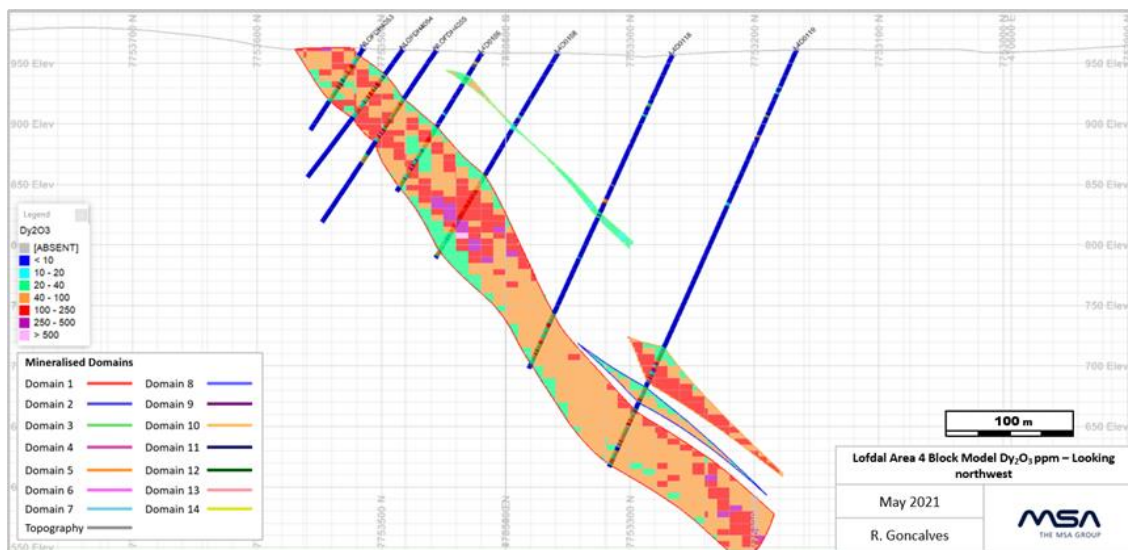


Figure 14-10 Area 4 Block Model Cross-Section view to Northeast: All Domains Dy_2O_3 ppm

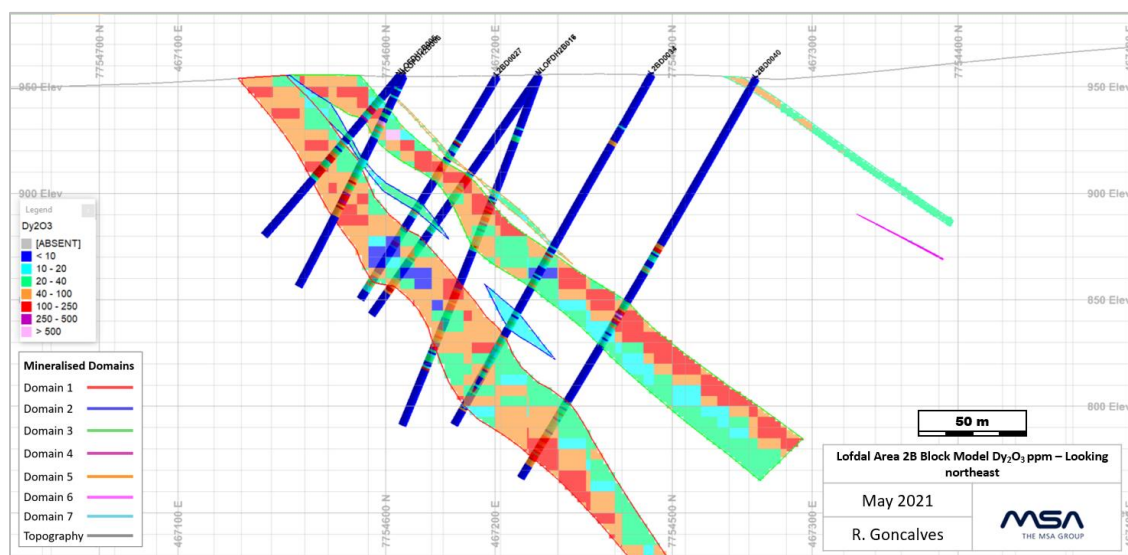


Figure 14-11 Area 2B Block Model Cross-Section View to Northeast: All Domains Dy_2O_3 ppm

14.9 Mineral Resource Classification

Classification of the Area 4 and Area 2B Mineral Resources was based on the degree of geological uncertainty, grade continuity and variability, frequency of the drilling data and the confidence parameter outputs from the kriging estimates. The main considerations in the classification are as follows:

- All the data that inform the Mineral Resource have been collected by NMI, using acceptable principles and the assays passed the relevant QAQC tests.
- The geological model is robust and the grade shells exhibit good continuity with low variability within and between drilling sections.
- Semivariogram ranges for the attributes are more than the general drillhole spacing in most areas.

Given the aforementioned factors, the Mineral Resources have been classified using the following criteria:

- The Mineral Resource was classified as Measured where the level of confidence in the estimates is high. This is underpinned by data on a drilling grid of 30 m spacing or less. The kriging efficiency is between 50% and 80% and the slope of regression is higher than 0.8 for the majority of the blocks in the model.
- the Indicated Mineral Resource is underpinned by data on a drilling grid of approximately 50 m spacing. The kriging efficiency and the slope of regression are lower than for Measured and the drillhole spacing is too wide to interpolate grades to a high level of accuracy, despite drillhole spacing being within the modelled variogram range. The Indicated areas are directly adjacent to the Measured areas.
- the Inferred Mineral Resource was classified where the confidence for the estimates is low. In these areas the drillholes are sparse and local estimates cannot be reliably made. The Inferred area is directly adjacent to the Indicated areas and occur in the deeper portions and periphery of the Mineral Resource.

The classified block model for Area 4 is shown in Figure 14-12.

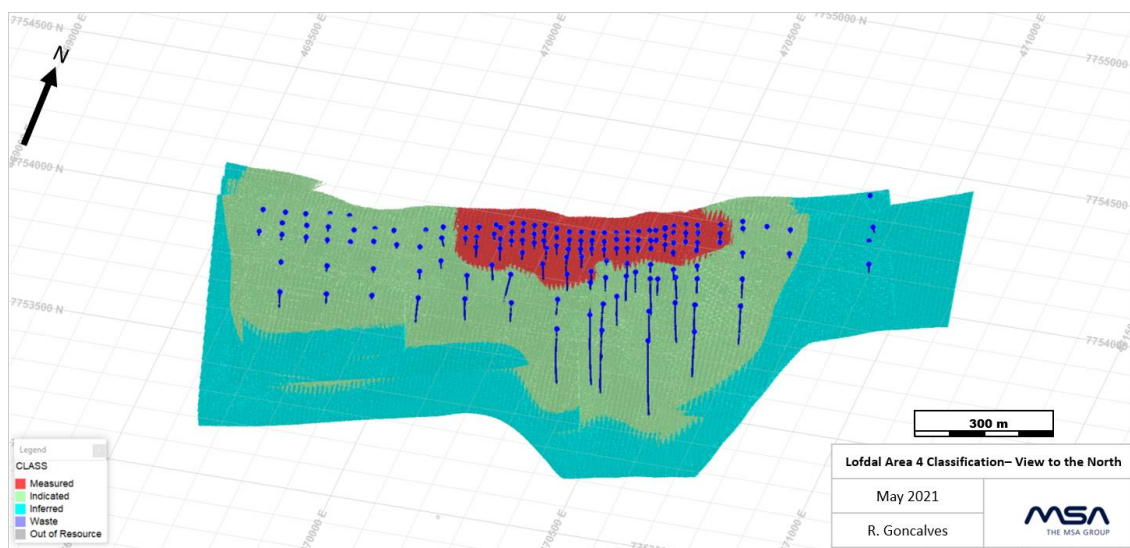


Figure 14-12 Mineral Resource Classification for Area 4

Mineral Resources for Area 2B were classified as Indicated and Inferred in the same way as for Area 4. Areas that fall outside of this classification, where significant extrapolation of grades occurs beyond the data coverage, were not included in the Mineral Resource and were assigned a code of OOR (out of resource) in the model (Figure 14-13).

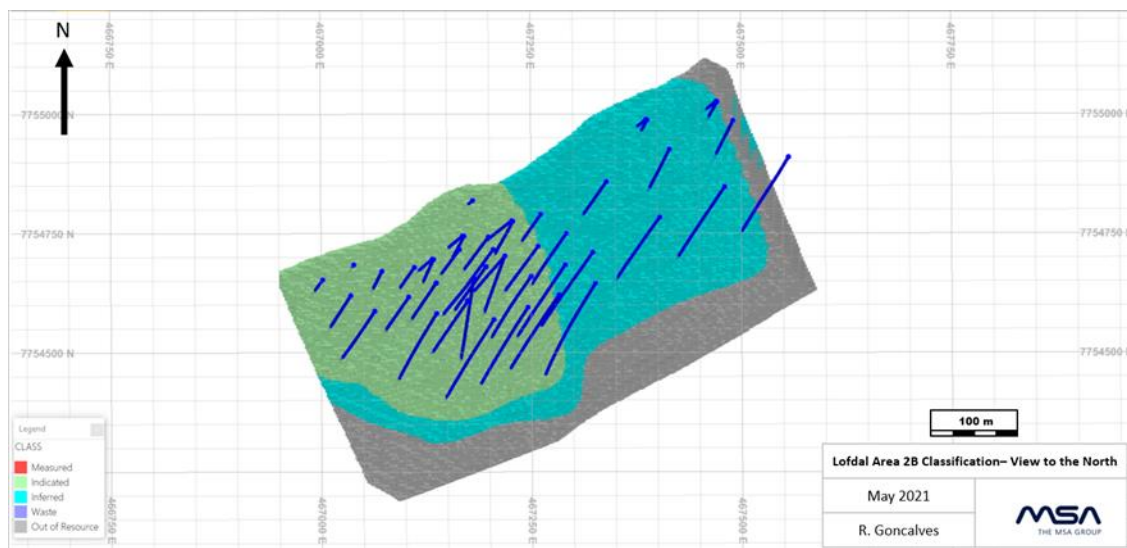


Figure 14-13 Mineral Resource Classification for Area 2B

The Mineral Resources could be affected by further infill drilling, which may result in increases or decreases in subsequent Mineral Resource estimates. Inferred Mineral Resources are high-risk estimates that may change significantly with additional data. It cannot be assumed that all or part of an Inferred Mineral Resource will necessarily be upgraded to an Indicated Mineral Resource due to continued exploration. The Mineral Resources may also be affected by subsequent assessments of mining, environmental, processing, permitting, taxation, socio-economic and other factors.

14.10 Mineral Resource Statement

The Mineral Resource estimate as of May 12, 2021 is presented in Table 14-8 for Area 4 and Table 14-9 for Area 2B. The Mineral Resource is stated at a cut-off of 0.10% total rare earth oxides (TREO).

In the QP's opinion, the Mineral Resources reported herein at the selected cut-off grade have "reasonable prospects for eventual economic extraction", taking into consideration mining and processing assumptions (refer to 14.11).

Table 14-8 Area 4, Measured, Indicated and Inferred Mineral Resource Estimates above 0.1% TREO cut-off grade – May 12, 2021

Category	Tonnes (Mt)	TREO* %	HREO** %	LREO*** %	Dy ₂ O ₃ ppm	TREO (Kt)
Measured	5.93	0.21	0.14	0.07	138	12.71
Indicated	36.63	0.16	0.08	0.08	82	59.97
Measured & Indicated	42.57	0.17	0.09	0.08	90	72.68
Inferred	6.09	0.17	0.07	0.09	72	10.12

Notes:

- (1) All tabulated data have been rounded and as a result minor computational errors may occur.
- (2) Mineral Resources, which are not Mineral Reserves, have no demonstrated economic viability.
- (3) Quantities reported are the total quantities for the project regardless of ownership.
- (4) *TREO = Total Rare Earth Oxides and includes Y₂O₃
- (5) **HREO = Heavy Rare Earth Oxides and includes Y₂O₃
- (6) ***LREO = Light Rare Earth Oxides
- (7) Mt = Million tonnes, kt = Thousand tonnes.

Table 14-9 Area 2B, Indicated and Inferred Mineral Resource Estimates above 0.1% TREO Cut-Off Grade – May 12, 2021

Category	Tonnes (Mt)	TREO* %	HREO** %	LREO*** %	Dy ₂ O ₃ ppm	TREO (kt)
Indicated	2.20	0.19	0.10	0.09	104	4.27
Inferred	2.58	0.19	0.09	0.09	92	4.80

Notes:

- (1) All tabulated data have been rounded and as a result minor computational errors may occur.
- (2) Mineral Resources, which are not Mineral Reserves, have no demonstrated economic viability.
- (3) Quantities reported are the total quantities for the project regardless of ownership.
- (4) *TREO = Total Rare Earth Oxides and includes Y₂O₃
- (5) **HREO = Heavy Rare Earth Oxides and includes Y₂O₃
- (6) ***LREO = Light Rare Earth Oxides
- (7) Mt = Million tonnes, kt = Thousand tonnes.

The Mineral Resource for Area 4 is presented at a variety of cut-off grades as shown in Table 14-10 for the combined Measured and Indicated Resources and Table 14-11 for the Inferred Mineral Resource.

Table 14-10 Area 4, Measured and Indicated Mineral Resource Grade-Tonnage – May 12, 2021

Cut-off TREO%	Tonnes (Mt)	TREO* %	HREO** %	LREO*** %	Dy₂O₃ ppm	TREO* (kt)
0.10	42.57	0.17	0.09	0.08	90	72.68
0.15	17.50	0.24	0.15	0.09	141	42.09
0.20	7.63	0.33	0.23	0.10	216	25.21
0.25	4.05	0.43	0.33	0.10	305	17.29
0.30	2.67	0.51	0.40	0.10	371	13.52

Notes:

- (1) All tabulated data have been rounded and as a result minor computational errors may occur.
- (2) Mineral Resources, which are not Mineral Reserves, have no demonstrated economic viability.
- (3) Quantities reported are the total quantities for the project regardless of ownership.
- (4) *TREO = Total Rare Earth Oxides and includes Y₂O₃
- (5) **HREO = Heavy Rare Earth Oxides and includes Y₂O₃
- (6) ***LREO = Light Rare Earth Oxides
- (7) Mt = Million tonnes, kt = Thousand tonnes.

Table 14-11 Area 4, Inferred Mineral Resources Grade-Tonnage – May 12, 2021

Cut-off TREO%	Tonnes (Mt)	TREO* %	HREO** %	LREO*** %	Dy₂O₃ ppm	TREO* (kt)
0.10	6.1	0.17	0.07	0.09	72	10.12
0.15	3.0	0.21	0.10	0.12	94	6.31
0.20	1.4	0.26	0.13	0.13	123	3.54
0.25	0.6	0.31	0.21	0.10	198	1.85
0.30	0.2	0.36	0.27	0.09	252	0.80

Notes:

- (1) All tabulated data have been rounded and as a result minor computational errors may occur.
- (2) Mineral Resources, which are not Mineral Reserves, have no demonstrated economic viability.
- (3) *TREO = Total Rare Earth Oxides and includes Y₂O₃
- (4) **HREO = Heavy Rare Earth Oxides and includes Y₂O₃
- (5) ***LREO = Light Rare Earth Oxides
- (6) Mt = Million tonnes, kt = Thousand tonnes.

The Mineral Resource for Area 2B is presented at a variety of cut-off grades in Table 14-12 for the Inferred Mineral Resource and Table 14-13 for the Indicated Mineral Resource.

Table 14-12 Area 2B, Indicated Resources Grade-Tonnage Table – May 12, 2021

Cut-off TREO%	Tonnes (Mt)	TREO* %	HREO** %	LREO*** %	Dy₂O₃ ppm	TREO* (kt)
0.10	2.20	0.19	0.10	0.09	104	4.27
0.15	1.24	0.25	0.12	0.13	126	3.10
0.20	0.76	0.30	0.14	0.16	143	2.27
0.25	0.47	0.35	0.15	0.20	158	1.61
0.30	0.29	0.39	0.16	0.23	171	1.13

Notes:

- (1) All tabulated data have been rounded and as a result minor computational errors may occur.
- (2) Mineral Resources, which are not Mineral Reserves, have no demonstrated economic viability.
- (3) Quantities reported are the total quantities for the project regardless of ownership.
- (4) *TREO = Total Rare Earth Oxides and includes Y₂O₃
- (5) **HREO = Heavy Rare Earth Oxides and includes Y₂O₃
- (6) ***LREO = Light Rare Earth Oxides
- (7) Mt = Million tonnes, kt = Thousand tonnes.

Table 14-13 Area 2B, Inferred Resources Grade-Tonnage Table – May 12, 2021

Cut-off TREO%	Tonnes (Mt)	TREO* %	HREO** %	LREO*** %	Dy₂O₃ ppm	TREO* (kt)
0.10	2.6	0.19	0.09	0.09	92	4.80
0.15	1.4	0.24	0.13	0.11	123	3.29
0.20	0.7	0.31	0.17	0.14	168	2.10
0.25	0.5	0.36	0.20	0.16	196	1.60
0.30	0.3	0.39	0.23	0.16	226	1.23

Notes:

- (1) All tabulated data have been rounded and as a result minor computational errors may occur.
- (2) Mineral Resources, which are not Mineral Reserves, have no demonstrated economic viability.
- (3) Quantities reported are the total quantities for the project regardless of ownership.
- (4) *TREO = Total Rare Earth Oxides and includes Y₂O₃
- (5) **HREO = Heavy Rare Earth Oxides and includes Y₂O₃
- (6) ***LREO = Light Rare Earth Oxides
- (7) Mt = Million tonnes, kt = Thousand tonnes.

The grades for the individual REE for each class are shown for Area 4 in Table 14-14 and in Table 14-15 for Area 2B.

Table 14-14 Area 4, Individual REO Measured, Indicated and Inferred Mineral Resources above 0.1% TREO Cut-Off Grade – May 12, 2021

Class	Tonnes Mt	TREO* %	La ₂ O ₃ ppm	CeO ₃ ppm	Pr ₂ O ₃ ppm	Nd ₂ O ₃ ppm	Sm ₂ O ₃ ppm	Eu ₂ O ₃ ppm	Gd ₂ O ₃ ppm	Tb ₂ O ₃ ppm	Dy ₂ O ₃ ppm	Ho ₂ O ₃ ppm	Er ₂ O ₃ ppm	Tm ₂ O ₃ ppm	Yb ₂ O ₃ ppm	Lu ₂ O ₃ ppm	Y ₂ O ₃ ppm
Measured	5.93	0.21	177	320	34	127	44	19	85	20	138	30	86	13	78	11	960
Indicated	36.63	0.16	208	371	39	139	40	15	62	13	82	17	49	7	44	6	546
M&I	42.57	0.17	204	364	38	137	41	16	65	14	90	19	54	8	48	7	603
Inferred	6.09	0.17	247	436	45	158	41	14	54	11	72	15	45	7	41	6	470

Notes:

- (1) All tabulated data have been rounded and as a result minor computational errors may occur.
- (2) Mineral Resources, which are not Mineral Reserves, have no demonstrated economic viability.
- (3) Quantities reported are the total quantities for the project regardless of ownership.
- (4) *TREO = Total Rare Earth Oxides and includes Y₂O₃
- (5) Mt = Million tonnes, kt = Thousand tonnes.

Table 14-15 Area 2B, Individual REO Measured, Indicated and Inferred Mineral Resources above 0.1% TREO grade – May 12, 2021

Class	Tonnes Mt	TREO* %	La ₂ O ₃ ppm	CeO ₃ ppm	Pr ₂ O ₃ ppm	Nd ₂ O ₃ ppm	Sm ₂ O ₃ ppm	Eu ₂ O ₃ ppm	Gd ₂ O ₃ ppm	Tb ₂ O ₃ ppm	Dy ₂ O ₃ ppm	Ho ₂ O ₃ ppm	Er ₂ O ₃ ppm	Tm ₂ O ₃ ppm	Yb ₂ O ₃ ppm	Lu ₂ O ₃ ppm	Y ₂ O ₃ ppm
Indicated	2.20	0.19	255	398	41	161	64	25	87	17	104	20	58	8	54	8	638
Inferred	2.58	0.19	243	385	41	178	85	29	94	16	92	17	49	7	45	7	575

Notes:

- (1) All tabulated data have been rounded and as a result minor computational errors may occur.
- (2) Mineral Resources, which are not Mineral Reserves, have no demonstrated economic viability.
- (3) Quantities reported are the total quantities for the project regardless of ownership.
- (4) *TREO = Total Rare Earth Oxides and includes Y₂O₃
- (5) Mt = Million tonnes, kt = Thousand tonnes.

14.11 Assessment of Reasonable Prospects for Eventual Economic Extraction (RPEEE)

In assessing “reasonable prospects for eventual economic extraction” (RPEEE) the Mineral Resource was reported from within a Whittle optimised pit shell using the following assumed parameters and a cut-off grade of 0.1% TREO.

Mining will be by open-pit methods:

- 45° slope angle in the partially weathered rock and 55° slope angle in the fresh rock
- 5% mining dilution
- 5% mining loss
- 10 m bench height
- Ore production rate of 1.68 million tonnes per annum.
- 75% final metallurgical recovery of TREO

Costs were assumed as follows:

- Mining cost for drill and blast: USD 2.75 / tonne mined.
- Processing costs: USD 30.6 / tonne milled
- G&A cost: USD 7.36 / tonne milled
- NMI price USD 66.20 per Kg TREO+Y₂O₃ (based on a discounted price deck from ARGUS Rare Earths Analytics and the estimated TREO proportion in concentrate (Table 14-16)).
- Offshore treatment cost and shipment priced in discounted basket price.

Table 14-16 Distribution of TREO in Concentrate

REO	Distribution of Individual REO in TREO % (concentrate values)
La ₂ O ₃	0.47
CeO ₂	0.78
Pr ₆ O ₁₁	0.08
Nd ₂ O ₃	0.32
Sm ₂ O ₃	0.51
Eu ₂ O ₃	0.53
Gd ₂ O ₃	3.52
Tb ₄ O ₇	1.14
Dy ₂ O ₃	9.09
Ho ₂ O ₃	2.04
Er ₂ O ₃	6.3
Tm ₂ O ₃	0.94
Yb ₂ O ₃	5.56
Lu ₂ O ₃	0.78
Y ₂ O ₃	67.94

A plan showing the extents of the block model and surveyed topography in relation to the conceptual pit shell boundaries is shown in Figure 14-14 for Area 4 and Figure 14-16 for Area 2B and a section through the deepest part of the modelled pit shell is shown in Figure 14-15 for Area 4 and Figure 14-17 for Area 2B. The pit shell covers the majority of the Area 4 grade block model both aerially and at depth, however the narrower mineralisation at Area 2B resulted in the mineral resource being constrained at depth by the limits of the pit shell. The modelled pit shell areas lie entirely within EPL 3400 and the nearest boundary of the licence is approximately 6 km to the north and 9 km to the east. The two pits are far enough away from each other to be operated as separate pits, although close enough so that ore will be transported to a central facility for processing. There is no infrastructure, such as major roads, power lines, water courses or settlements, within or within the immediate vicinity of the pit shell outline.

The reader is advised that the assessment of economic potential that is incorporated in the Mineral Resource is solely for the purpose of reporting Mineral Resources and does not represent an attempt to estimate Mineral Reserves.

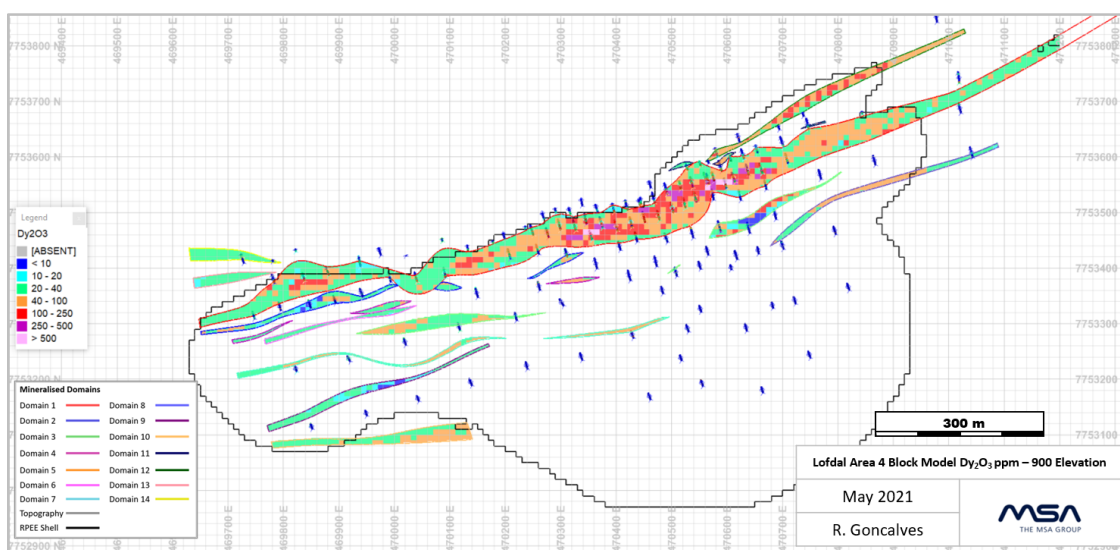


Figure 14-14 Area 4 – Plan showing Block Model Relative to Pit Shell Extents

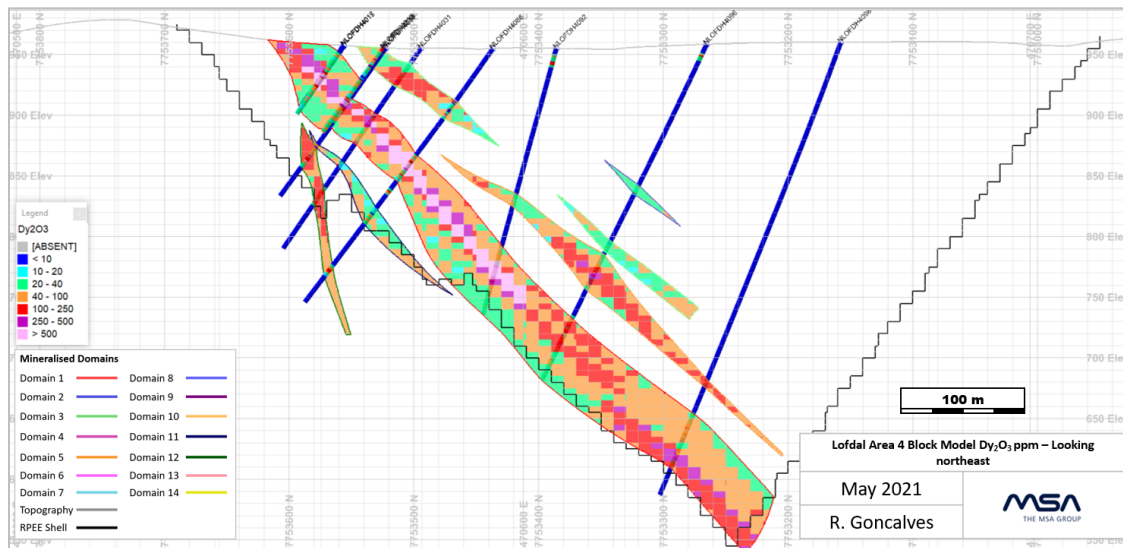


Figure 14-15 Area 4 Section looking Northeast showing Block Model Relative to Pit Shell Extents and Topography

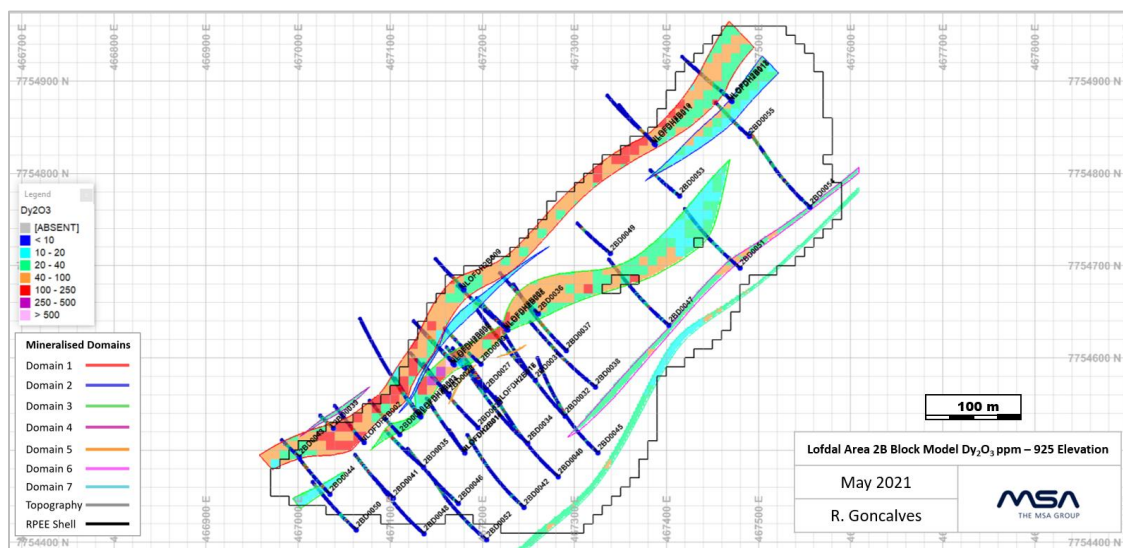


Figure 14-16 Area 2 – Plan showing Block Model Relative to Pit Shell Extents

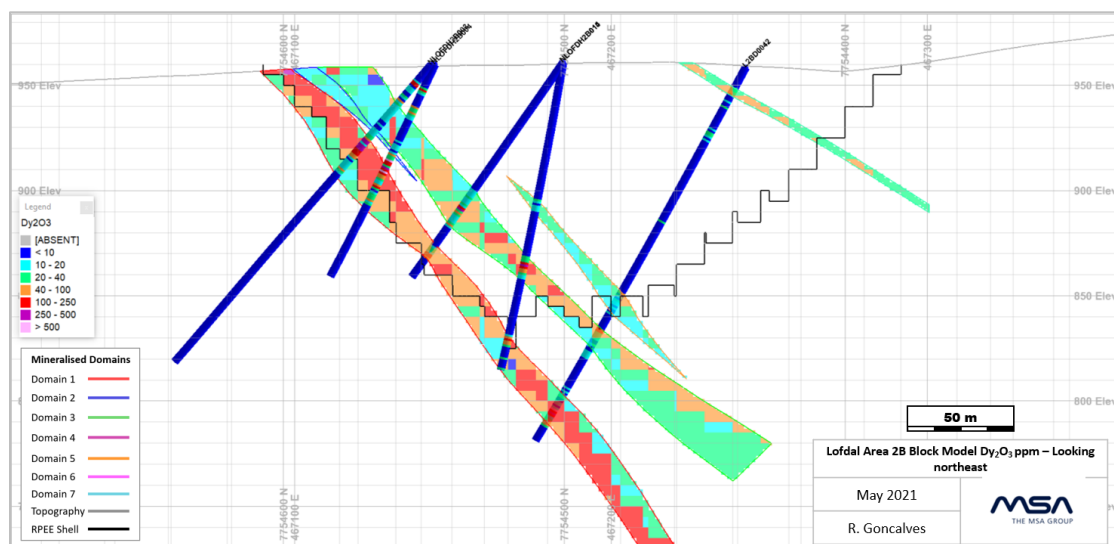


Figure 14-17 Area 2 Section looking Northeast showing Block Model Relative to Pit Shell Extents and Topography

14.12 Comparison with Previous Estimate

The Mineral Resource estimate detailed in this report represents the second Mineral Resource Estimate reported for Area 4 and a Maiden Mineral Resource for Area 2B. A comparison between the previous estimate for Area 4, with an effective date July 31, 2012, and the current estimate is shown in Table 14-17 at a 0.10% TREO cut-off.

Table 14-17 Area 4 – May 12, 2021 Mineral Resource Estimate compared with July 31, 2012 Mineral Resource Estimate

Classification	July 31, 2012			May 12, 2021		
	Tonnes (Mt)	TREO* %	TREO (kt)	Tonnes (Mt)	TREO* %	TREO (kt)
Measured	-	-	-	5.93	0.21	12.71
Indicated	2.88	0.32	9.23	36.63	0.16	59.97
M&I	2.88	0.32	9.23	42.57	0.17	72.68
Inferred	3.28	0.27	8.97	6.09	0.17	10.12

Notes:

- (1) All tabulated data have been rounded and as a result minor computational errors may occur.
- (2) Mineral Resources, which are not Mineral Reserves, have no demonstrated economic viability.
- (3) *TREO = Total Rare Earth Oxides and includes Y_2O_3
- (4) Mt = Million tonnes, kt = Thousand tonnes.

The total Mineral Resource increased from July 2012 to May 2021 as a result of the additional drilling which expanded the extent of the Mineral Resource both along strike and down-dip from the previously defined area. Given the high value and irregular nature of the high grade contacts a lower grade threshold was used to define the mineralisation. Together with extending the mineral resource drilling into lower grade areas, the overall grade decreased and the tonnage increased. The reader is advised that the July 31, 2012 Mineral Resource Estimate for Area 4 has been superseded by that of May 12, 2021 and is presented purely for comparative purposes.

15 MINERAL RESERVE ESTIMATES

There are no current Mineral Reserve estimates stated on this Property. This section does not apply to the Technical Report.

16 MINING METHODS

16.1 Caution to the Reader

The reader is cautioned that this PEA uses Inferred Mineral Resources. NI 43-101 Part 2, Section 2.3(1)(b) and Companion Policy 43-101 CP, Part 2, Section 2.3(1) Restricted Disclosure, prohibits the disclosure of the results of an economic analysis that includes or is based on Inferred Mineral Resources, an historical estimate, or an exploration target.

However, under NI 43-101, Part 2, Section 2.3(3) and Companion Policy 43-101CP, Part 2 Section 2.3(3), a PEA is allowed to use inferred mineral resources and to carry out an economic assessment in order to inform investors of the potential of the property. Investors must be informed that the preliminary economic assessment is preliminary in nature, that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised. Mineral resources that are not mineral reserves do not have demonstrated economic viability. To emphasise, no consideration was given to the resource classification in the pit optimisation and mine design.

16.2 Overview

The proposed mining method is conventional open pit mining. Mineralised rock and waste would be drilled, blasted, loaded by hydraulic shovels and hydraulic excavators into off-highway dump trucks, and hauled to the processing plant.

The basis for the pit design work was the mineral resource block model that was developed by MSA as part of a NI 43-101-compliant mineral resource estimate (refer to Section 14).

There are two primary Lofdal deposits currently under consideration. These are to be mined as open pits, with the normal sequence of drilling, blasting and hauling. Due to the nature of the deposit, the resultant pits are narrow and deep. Currently no backfilling is contemplated.

The proposed mining method is the development of a slot in front of the mineralised zone at each level. The centralised slot will enable waste mining on one side while mining mineralised material on the other side. This methodology will also facilitate separating mineralised material from waste material.

The target ROM feed to the processing plant is 2 000 000 tonnes/annum. The plant feed is mineralised material with a TREO \geq 0.1%. In the current schedule the production ramps up to this over 4 years.

Mineralised material with TREO \geq 0.05% and $<$ 0.1% is sent to a stockpile so it may be possible to process at a later stage if economics allow.

The combined Life of Mine of the two pits is 16 years and note that all prices and costs are in \$US.

16.3 Geotechnical Evaluation

No geotechnical or hydrological studies have yet been undertaken. An assumed maximum pit slope angle of 63° is applied to all slopes other than the low wall side, which follows the orebody and is at approximately 50°.

With the inclusion of the ramp system the overall highwall slope in Pit 4 is approximately 55°.

The pit design parameters are shown in Table 16-1.

Table 16-1 Pit Design Parameters

Lofdal Pit Design Parameters	
Bench Height	10 m
Face Batter Angle	85°
Berm Width	4.2 m
Ramp Width	15 m
Ramp Gradient	1:10

16.4 Hydrogeological Evaluation

No hydrogeological inputs are available at this time.

16.5 Open Pit Optimisation

16.5.1 Optimisation Parameters

The optimisation parameters are summarised in Table 16-2.

Table 16-2 Whittle Optimisation Parameters

Mining Areas A2 and 4		
Geological loss	5.0%	
Mining loss	2.5%	
Dilution	2.0%	
Pit slope	63°	
Mining cost (Waste)	\$2.76	At reference elevation
Mining cost (Mineralised)	\$2.76	At reference elevation
Reference elevation Area 4	880	m at 1/3 PEA design pit depth
Reference elevation Area 2B	910	m at 1/3 PEA design pit depth
Mining cost change with depth	\$0.02	Change per 10m
Mining cost change with depth	0.72%	Change per 10m
Annual InSitu target	2 160 000	tonnes

Annual ROM target	2 000 000	tonnes
Processing		
Processing cost	\$28.00	Cost/ROM tonne
Process recovery	58.00%	
Overheads (Annual)	\$12 364 800	Calculated from MSA Inputs (\$7.36 x 1 680 000)
Overheads	\$6.18	
Selling Costs		
Namibian Royalties	3.0%	Factor applied to revenue
Landowner Royalty	1.0%	Percentage of Revenue less operating costs. Ignored, changes with pit size.
Concentrate transport costs	\$36.31	/tonne. Prices FOB Walvis Bay 351km @ R1.50/tonne km
Rand \$US Exchange rate	14.5	

16.5.2 Geological Block Model Input to Whittle and Validation

Variably sub-blocked Datamine geological models were provided by the client as inputs. These are:

- A4MODFIN_2021.dm
- A2BMODFIN_2021.dm

The original geological models were left unchanged, and copies were made prior to adding attributes to facilitate the transfer of block values. The copies are:

- a4modfin_dep_10-5.dm
- a2bmodfin_dep_10-5.dm

The block model values in the Datamine models were transferred to Surpac models. Additional attributes were incorporated into the Datamine model to facilitate column, row and level numbering of mining blocks. The framework for the transfer is 10m x 10m x 5m which is the parent block size of the Datamine models and correspond with the dimensions of the Surpac models which were not sub-blocked.

The Datamine models contain multiple mineralised domains. Waste is domain 0, Area 4 has domains 1 to 14, and Area 2B domains 1 to 7. The volumes, tonnages and quality parameters were exported separately for each domain in the Datamine models and these were recalculated into a single domain per block in the Surpac models.

After transferring the model values reports were generated from both the Datamine and Surpac models to validate the transfer. The results are shown in Table 16-3 for Area 4 and Table 16-4 for Area 2B. It is noted that the difference in waste tonnes for Area 2B is -3.73%. This is considered acceptable for this level of study.

The Datamine model for Area 4 does not occupy the entire model framework. Whittle will consider this unoccupied volume as air which could influence the optimisation result. In the Surpac model this volume was designated as waste, using a default RD of 2.764.

The extended model for Area 4 was saved as a4_dep_10-5_whittle_vf_crd.mdl, and that for Area 2B as a2b_dep_10-5_whittle_vf_crd.mdl to maintain the same naming convention.

Merging the Datamine domain values into the Surpac models results in blocks containing a combination of mineralised material and waste, and for surface blocks air is included. To accommodate this a volume factor, “vf114”, for the mineralised component was added to the Surpac model. However, Whittle can only accept a single RD value so a combined RD was calculated for the blocks in the attribute “crdb”.

Attributes were added for mining and processing cost adjustment factors used by Whittle, and for REO prices and the calculated revenue for HREO and LREO.

The processing cost adjustment factor (PCAF) was set at 1, that is all mineralised blocks incur the same processing cost.

The models were then exported to Whittle using the revenue values vhre, vlre as grade values. The total revenue vtot, and grade values for hreo, lreo and treo were also exported for reporting purposes, although these do not play any part in the optimisation.

Tonnages for mineralised material and waste in Whittle were compared with the Surpac values to validate the export. The results are shown in Table 16-5 and Table 16-6.

It is noted that there is again a difference in the waste tonnes for Area 2B, this time of 3.71%, effectively cancelling out the difference that occurred in the transfer from Datamine to Surpac.

The difference in waste tonnes for Area 4 is 2.02%, which is acceptable. Mineralised material differences are well below 0.5%.

Table 16-3 Area 4 Datamine-Surpac Validation

Datamine Domain Values																					
Domain	Volume	Tonnes	Density	Ce2o3	Dy2o3	Er2o3	Eu2o3	Gd2o3	Ho2o3	Hreo	La2o3	Lreo	Lu2o3	Nd2o3	Pr2o3	Sm2o3	Tb2o3	Tm2o3	Y2o3	Yb2o3	Treo
0	895 848 343	2 476 293 366	2.764	0.714	0.454	0.379	0.169	0.451	0.162	0	0.604	0	0.068	0.553	1.028	0.451	0.113	0.067	0.596	0.382	0.001
1	26 422 092	72 285 732	2.736	293.979	68.955	41.483	13.035	52.216	14.275	0.071	161.684	0.064	5.473	114.35	31.43	35.295	10.581	6.142	458.795	37.687	0.135
2	2 310 008	6 386 501	2.765	274.233	35.734	19.977	8.926	32.248	7.006	0.035	153.447	0.059	2.534	103.04	28.67	26.324	5.815	2.805	217.372	17.415	0.094
3	2 027 120	5 568 309	2.747	291.592	40.53	24.602	8.494	32.483	8.297	0.041	163.654	0.062	3.499	111.626	30.996	26.28	6.324	3.681	260.641	23.5	0.104
4	1 093 604	3 085 641	2.822	217.775	33.148	19.114	7.749	28.178	6.683	0.033	118.905	0.047	2.309	89.059	23.925	22.408	5.318	2.676	209.578	16.43	0.08
5	580 904	1 590 467	2.738	222.622	110.204	65.284	13.599	62.052	25.126	0.107	118.191	0.049	8.719	93.487	24.743	28.165	14.843	9.806	704.25	58.285	0.156
6	243 300	688 044	2.828	306.632	23.666	12.635	6.968	22.666	4.557	0.023	141.345	0.062	1.696	119.336	29.735	22.827	3.966	1.819	141.808	11.535	0.085
7	951 164	2 662 640	2.799	694.12	39.799	20.841	12.904	39.499	7.521	0.038	430.872	0.146	2.471	233.146	66.305	39.748	6.859	2.882	230.287	17.383	0.184
8	1 066 440	2 954 327	2.77	237.365	37.702	20.897	8.505	32.436	7.395	0.037	125.862	0.05	2.725	93.29	25.009	22.863	6.089	2.97	228.612	18.819	0.087
9	887 952	2 501 085	2.817	348.969	21.128	11.143	7.881	23.03	4.04	0.021	191.55	0.074	1.427	132.331	36.273	27.789	3.698	1.565	129.246	9.76	0.095
10	737 272	2 051 184	2.782	330.581	40.536	22.068	11.171	38.484	7.987	0.042	171.299	0.07	2.87	127.873	35.576	31.229	6.786	3.117	269.219	19.411	0.112
11	870 864	2 349 244	2.698	315.36	46.997	24.388	12.101	45.139	9.165	0.046	172.957	0.068	3.071	125.11	33.76	36.778	8.144	3.523	283.715	21.001	0.114
12	839 264	2 312 172	2.755	240.171	74.977	39.256	15.792	65.094	14.892	0.069	126.25	0.055	5.119	108.626	26.708	45.662	12.745	5.853	423.222	34.633	0.124
13	293 212	792 208	2.702	174.153	20.473	11.425	6.293	21.029	4.012	0.019	94.761	0.037	1.77	68.646	18.856	17.573	3.494	1.672	112.25	11.459	0.057
14	479 736	1 295 315	2.7	156.255	22.573	12.791	7.022	22.726	4.577	0.024	77.53	0.034	2.432	70.653	18.098	19.667	4.019	2.003	147.664	14.168	0.058
Grand Total	934 651 275	2 582 816 235	2.76	12.919	2.919	1.835	0.657	2.37	0.667	0.003	7.337	0.003	0.26	5.266	2.284	1.801	0.494	0.281	16.817	1.698	0.006

Datamine	Volume	Tonnes	Density	Ce2o3	Dy2o3	Er2o3	Eu2o3	Gd2o3	Ho2o3	Hreo	La2o3	Lreo	Lu2o3	Nd2o3	Pr2o3	Sm2o3	Tb2o3	Tm2o3	Y2o3	Yb2o3	Treo
Waste	895 848 343	2 476 293 366	2.76	0.71	0.45	0.38	0.17	0.45	0.16	-	0.60	-	0.07	0.55	1.03	0.45	0.11	0.07	0.60	0.38	0.00
Total mineralised	38 802 932	106 522 869	2.746	296.645	60.220	35.661	11.998	46.964	12.391	0.061	163.853	0.064	4.704	114.827	31.494	33.174	9.350	5.257	393.883	32.299	0.126

Surpac	Volume	Tonnes	Density	Ce2o3	Dy2o3	Er2o3	Eu2o3	Gd2o3	Ho2o3	Hreo	La2o3	Lreo	Lu2o3	Nd2o3	Pr2o3	Sm2o3	Tb2o3	Tm2o3	Y2o3	Yb2o3	Treo
Waste	895 718 396	2 475 938 692	2.764	0.714	0.454	0.379	0.169	0.451	0.162	0	0.604	0	0.068	0.553	1.028	0.451	0.113	0.067	0.596	0.382	0.001
Total mineralised	38 802 932	106 522 984	2.745	296.645	60.219	35.661	11.997	46.964	12.391	0.061	163.854	0.064	4.705	114.828	31.494	33.174	9.301	5.257	393.884	32.299	0.125

Difference Waste	-0.01%	-0.01%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Difference mineralised	0.00%	0.00%	-0.02%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	-0.57%	0.00%	-0.32%	0.01%	0.00%	0.00%	0.00%	-0.52%	0.00%	0.00%	0.00%	-0.49%

Table 16-4 Area 2B Datamine-Surpac Validation

Datamine Domain Values																					
Domain	Volume	Tonnes	Density	Ce2o3	Dy2o3	Er2o3	Eu2o3	Gd2o3	Ho2o3	Hreo	La2o3	Lreo	Lu2o3	Nd2o3	Pr2o3	Sm2o3	Tb2o3	Tm2o3	Y2o3	Yb2o3	Treo
0	202 833 980	580 265 128	2.86	0.53	0.332	0.115	0.059	0.192	0.055	0	0.43	0	0.019	0.482	0.119	0.163	0.038	0.019	1.175	0.113	0
1	3 809 432	10 555 710	2.77	167.625	67.577	39.025	16.321	56.408	13.431	0.069	98.248	0.041	5.325	83.404	18.791	42.582	10.772	5.718	436.003	36.79	0.11
2	1 528 212	4 279 365	2.80	84.818	30.862	14.921	10.168	30.165	5.462	0.029	41.41	0.023	1.82	67.096	12.58	28.674	5.348	2.047	174.472	12.697	0.052
3	3 442 872	9 571 316	2.78	281.612	42.424	20.261	14.548	46.534	7.575	0.039	177.479	0.065	2.536	116.501	29.286	42.48	7.719	2.778	231.503	17.472	0.104
4	25 312	71 914	2.84	81.934	23.435	12.85	5.401	18.398	4.607	0.023	44.632	0.019	1.721	36.493	8.987	14.09	3.592	1.827	144.133	11.691	0.041
5	42 008	117 999	2.81	112.164	29.787	15.419	6.865	25.047	5.622	0.028	61.903	0.025	1.966	49.337	12.361	17.6	4.869	2.173	177.327	13.627	0.054
6	262 876	733 959	2.79	357.497	33.57	17.344	12.811	39.442	6.271	0.033	221.93	0.081	2.125	146.294	37.133	46.443	6.012	2.368	199.947	14.545	0.114
7	234 484	645 076	2.75	197.466	34.258	16.734	13.284	40.148	6.219	0.034	102.841	0.049	1.953	121.384	25.477	47.024	6.213	2.233	203.355	13.854	0.083
Grand Total	212 179 176	606 240 468	2.86	9.145	2.468	1.258	0.673	2.212	0.46	0.002	5.612	0.002	0.169	4.547	1.067	1.881	0.399	0.182	14.113	1.151	0.005

Datamine	Volume	Tonnes	Density	Ce2o3	Dy2o3	Er2o3	Eu2o3	Gd2o3	Ho2o3	Hreo	La2o3	Lreo	Lu2o3	Nd2o3	Pr2o3	Sm2o3	Tb2o3	Tm2o3	Y2o3	Yb2o3	Treo
Waste	202 833 980	580 265 128	2.86	0.53	0.33	0.12	0.06	0.19	0.06	-	0.43	-	0.02	0.48	0.12	0.16	0.04	0.02	1.18	0.11	-
Total mineralised	9 345 196	25 975 339	2.780	201.601	50.178	26.794	14.406	47.315	9.519	0.049	121.374	0.048	3.520	95.348	22.263	40.280	8.459	3.822	303.132	24.330	0.097

Surpac	Volume	Tonnes	Density	Ce2o3	Dy2o3	Er2o3	Eu2o3	Gd2o3	Ho2o3	Hreo	La2o3	Lreo	Lu2o3	Nd2o3	Pr2o3	Sm2o3	Tb2o3	Tm2o3	Y2o3	Yb2o3	Treo
Waste	195 186 576	558 598 841	2.862	0.469	0.306	0.104	0.054	0.175	0.05	0	0.391	0	0.017	0.438	0.108	0.149	0.035	0.017	1.063	0.102	0
Total mineralised	9 321 056	25 908 813	2.780	201.577	50.187	26.795	14.408	47.321	9.520	0.049	121.365	0.048	3.520	95.338	22.260	40.278	8.460	3.822	303.166	24.329	0.097

Difference Waste	-3.77%	-3.73%	0.03%	-11.51%	-7.83%	-9.57%	-8.47%	-8.85%	-9.09%	0.00%	-9.07%	0.00%	-10.53%	-9.13%	-9.24%	-8.59%	-7.89%	-10.53%	-9.53%	-9.73%	0.00%
Difference mineralised	-0.26%	-0.26%	0.02%	-0.01%	0.02%	0.00%	0.01%	0.01%	0.01%	-0.32%	-0.01%	-0.15%	-0.01%	-0.01%	-0.01%	-0.01%	0.01%	0.00%	0.01%	0.00%	-0.24%

Table 16-5 Area 4 Surpac Whittle Validation

Surpac	Volume	Tonnes
Waste	1 133 421 596	3 132 941 228
Total Mineralised	38 802 932	106 522 984
Whittle Model	Volume	Tonnes
Waste	N/A	3 196 074 037
Mineralised	N/A	106 563 314
Difference Waste	N/A	2.02%
Difference Mineralised	N/A	0.04%

Table 16-6 Area 2B Surpac Whittle Validation

Surpac	Volume	Tonnes
Waste	195 186 576	558 598 841
Total Mineralised	9 321 056	25 908 813
Whittle Model	Volume	Tonnes
Waste	N/A	579 316 316
Mineralised	N/A	25 994 142
Difference Waste	N/A	3.71%
Difference Mineralised	N/A	0.33%

16.5.3 Time Costs

The overheads cost was derived from the value of \$12 364 800 per/annum used in a previous optimisation. This was divided by the planned production rate of 2 000 000 ROM tonnes/annum to give the applied value of \$6.18/ROM tonne. This value is added to the processing cost.

16.5.4 Processing Plant Capacity

The planned processing plant capacity for scheduling purposes, as supplied by the client is 2 000 000 ROM tonnes per annum.

16.5.5 Processing Recovery

The applied processing recovery that was used for Whittle optimisation is 58% for all of the rare earth oxides. This value was supplied by the client based on testwork.

(Note – this value is 57% in the plant processing section)

16.5.6 Mining and Transportation Costs

16.5.6.1 Mining Costs

The average mining cost of \$2.76/tonne, for both mineralised material and waste, was derived from work done in a previous phase. This cost was calculated using values for vertical and horizontal hauling distances within the pit designs used at that time. This was assumed to be the cost applicable at 1/3 of the pit depth, with the cost varying with depth each side of a reference elevation by an assumed value of \$0.02, or 0.72%, per 10m. This cost adjustment was included in the mining cost adjustment factor (MCAF) used by Whittle.

The reference elevation for Area 4 is 880m, and for Area 2B 910m, determined from the previous pit designs.

16.5.6.2 Selling Costs

The cost of transporting the concentrate to Walvis Bay is R1.50/tonnes km for a distance of 351 km. This equates to a total cost of \$36.31/tonne.

This selling cost per ROM tonne is calculated in Whittle as:

- Grade x mining recovery x process recovery x Transport cost/revenue value

This is calculated separately for LREO and HREO.

16.5.6.3 Royalties

The applicable royalties are a Namibian royalty of 3.0% on revenue and 1.0% Landowner royalty payable on revenue less operating costs.

The Landowner royalty was ignored in the Whittle exercise as it varies with the mined tonnage, and so cannot be calculated within Whittle.

As the revenue is calculated within the block model the price less royalty in Whittle is applied as a factor of 0.97.

16.5.7 Processing Costs

The applied processing cost of \$28.00/ ROM tonne was supplied by the client.

16.5.8 Open Pit Constraints and Mining Limits

There are no pit constraints applicable for either of the two pit areas. However, the model frameworks were extended in Whittle to ensure that the pit shells did not hit the edge of the model. The blocks in the extended areas are populated as waste.

16.5.9 Mining Recovery and Dilution

The assumed values applied for geological loss and mining loss are 5% and 2.5% respectively.

A dilution value of 2% was also applied in Whittle.

16.5.10 Product Prices

The REO prices as shown in Table 16-7 were applied to the relevant grade values within the mining block model. The revenue values were totalled separately for the light and heavy rare earth oxides as vlre and vhre. The combined value vtot was also calculated but was not utilised in the Whittle model except for reporting purposes.

Table 16-7 Applied REO Prices

Revenue		
<i>Calculated from insitu grade per mineral</i>		
Pricing Used For PEA	Pricing per kg	Classification
La2O3	\$6.48	LREO
Ce2O3	\$8.47	LREO
Pr2O3	\$100.00	LREO
Nd2O3	\$110.00	LREO
Sm2O3	\$2.32	LREO
Eu2O3	\$44.09	HREO
Gd2O3	\$43.96	HREO
Tb2O3	\$1 500.00	HREO
Dy2O3	\$600.00	HREO
Ho2O3	\$118.80	HREO
Er2O3	\$34.78	HREO
Yb2O3	\$17.74	HREO
Lu2O3	\$800.00	HREO
Y2O3	\$7.42	HREO
Tm2O3	\$500.00	HREO

16.5.11 Applied Revenue

The revenue for IREO and HREO were calculated in the mining model and exported to Whittle as grade values. The price in Whittle was therefore set at a value of 1, less the 3% royalty, therefore \$0.97.

16.6 Optimisation Results

The Whittle pit shells selected as the templates for the pit designs for Area 4 and Area 2B are those with a revenue factor of 1, corresponding to shell 36 in each case, with the tonnages shown in Table 16-8 and Table 16-9.

Table 16-8 Area 4 Whittle Results

Area 4 Whittle shell 36 (R58-S63)	
Ore	23 339 122
Waste	103 608 825
Total tonnes	126 947 947
SR	4.44
TREO	0.186

Table 16-9 Area 2B Whittle Results

Area 2B Whittle shell 36 (R58-S63)	
Ore	5 131 897
Waste	25 649 305
Total tonnes	30 781 202
SR	5.00
TREO	0.176

16.7 Open Pit Design

As standard practice in mine design, the Whittle pit shells are used as templates to guide the pit design process.

The initial step was to design a pit shell without ramps to determine how closely the design could be matched to the Whittle shell while applying batter angles and berm widths.

Whittle adds blocks to the pit until the maximum value is reached without consideration for the practicality of mining the resultant pit. This results in drop-cuts of single blocks or small groups of blocks into the pit floor. Consequently, in a narrow deposit such as Lofdal it is not practical to design a pit as deep as the Whittle shell as the pit bottom becomes too small to deploy equipment. Removing these drop-cuts results in a more practical layout.

These initial designs were then reviewed to determine the number and location of ramps to ensure access for all operating benches.

Final designs were then developed incorporating the ramp systems as shown in Figure 16-1 and Figure 16-2.

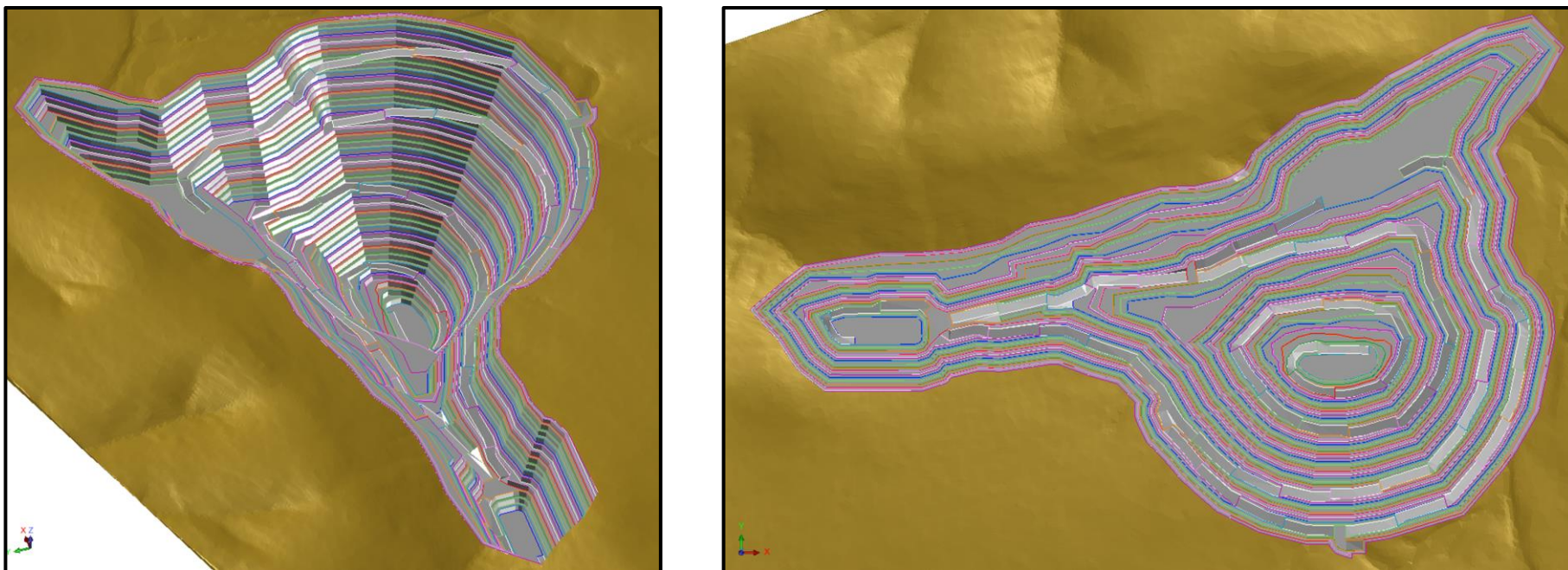


Figure 16-1 Area 4 Pit Design

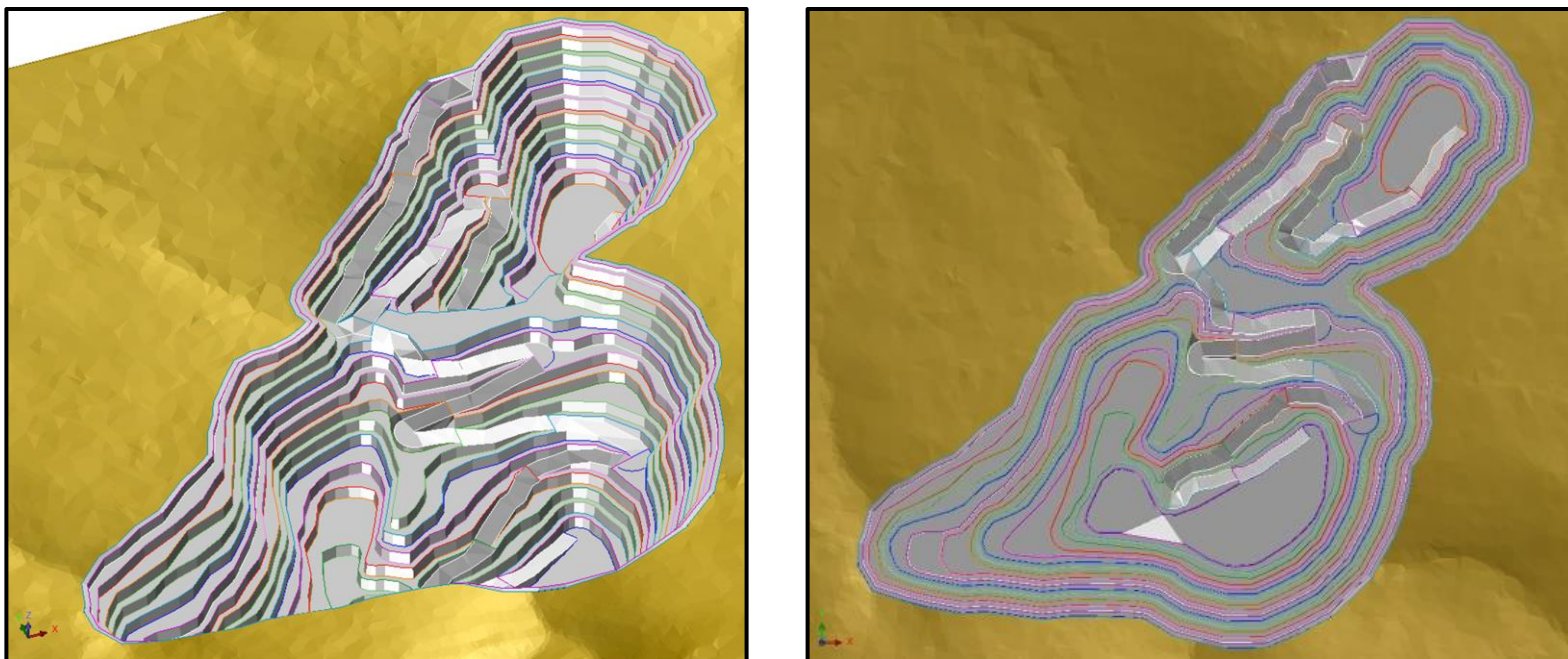


Figure 16-2 Area 2B Pit Design

16.8 Dump Design

At this stage of the mine design (PEA) no optimisation of the relevant waste dumps or topsoil stockpile dumps has been attempted. The only criteria applied was that waste could not be placed between the pits for environmental reasons and should not be too close to the pit area where it may restrict further pit expansion. The mine layout is shown in Figure 16-3.

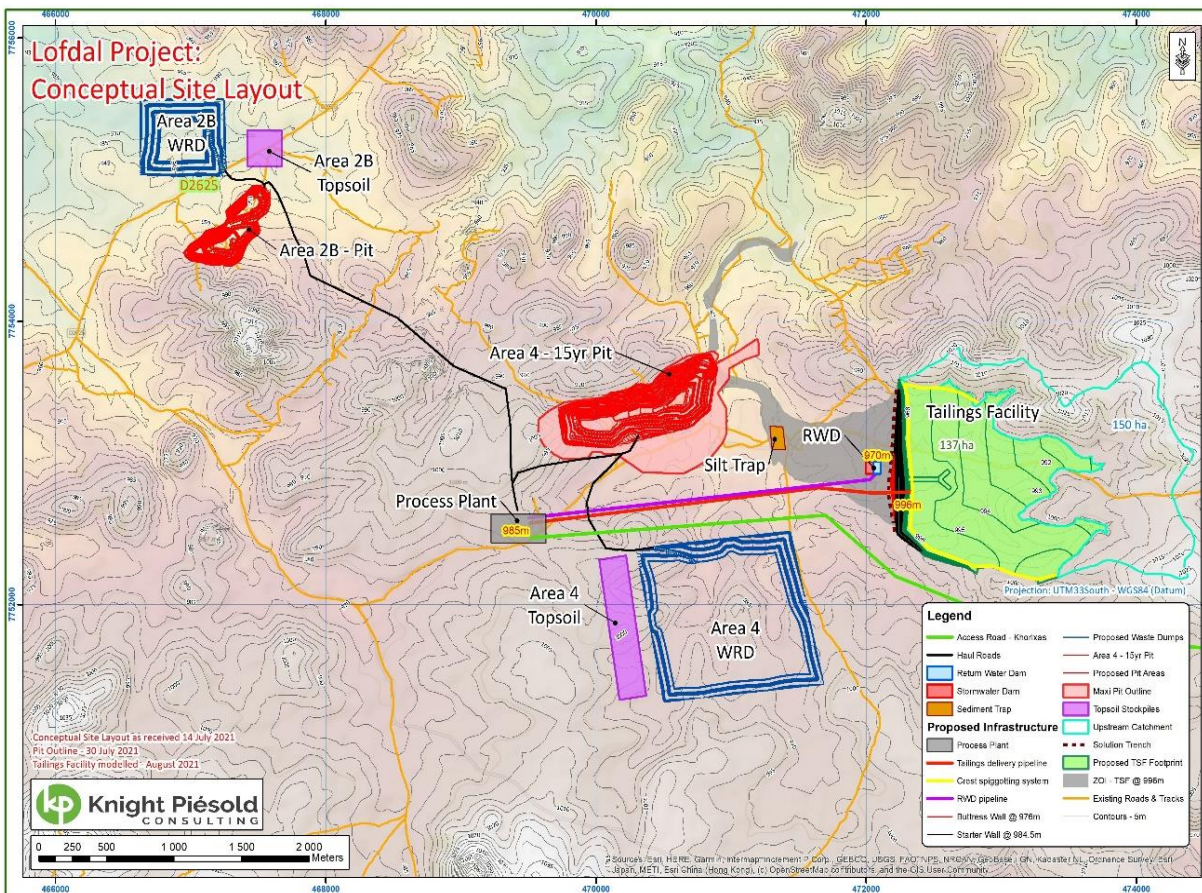


Figure 16-3 Mine Site Layout

Preliminary dump designs were produced for both Area 4 and Area 2B at the positions shown using the parameters in Figure 16-4.

The nominal lift height is 15m. For both waste dumps the first lift fills low areas of the topography. The waste dump volumes are shown in Table 16-10 and Table 16-11.

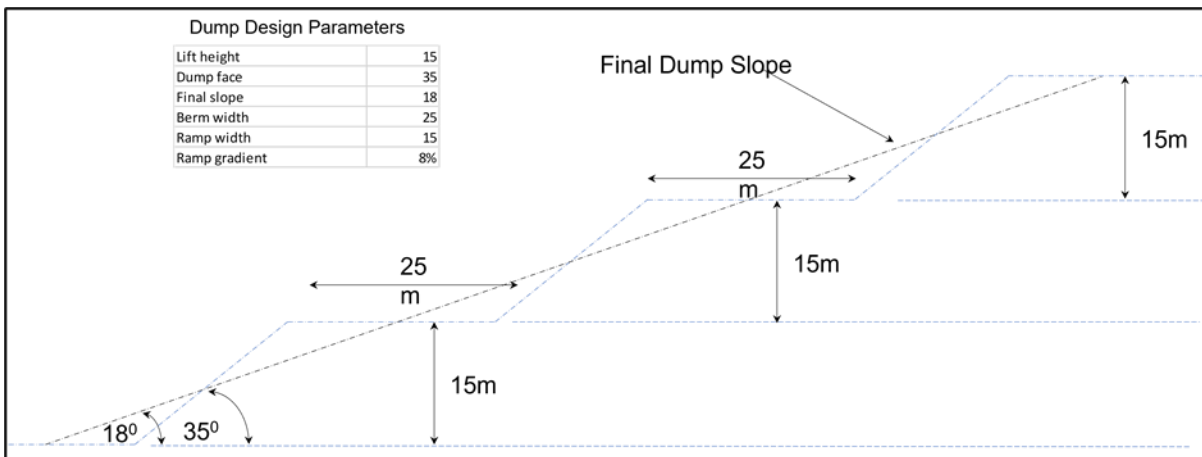


Figure 16-4 Waste Dump Design Parameters

Table 16-10 Area 4 Waste Dump Volume

Toe Elevation	Crest Elevation	Volume
965	980	2 100 352
980	995	13 307 908
995	1010	15 591 009
1010	1025	12 798 231
Total		43 797 500

Table 16-11 Area 2B Waste Dump Volume

Toe Elevation	Crest Elevation	Volume
925	940	1 224 741
940	955	4 673 239
955	970	3 572 135
970	985	2 248 818
Total		11 718 933

Provision has been made to stockpile topsoil stripped from the pit and waste dump areas. However, topsoil is understood to be minimal and an assumed thickness of 0.3m has been used to estimate the stockpile area. A swell of 15% and a maximum height of 3m have been assumed, to give the stockpile requirements shown in Table 16-12 and Table 16-13.

Table 16-12 Area 4 Topsoil Stockpile

Area 4	Area	Volume (Incl swell)
Pit	488 717	168 607
Dump	1 287 528	444 197
Total	1 776 245	612 805
Dump Area	204 268	
Dump side	452	

Table 16-13 Area 2B Topsoil Stockpile

Area 2B	Area	Volume (Incl swell)
Pit	195 148	67 326
Dump	344 045	118 696
Total	539 193	186 022
Dump Area	62 007	
Dump side	249	

16.9 Dewatering

Due to the arid nature of the region, no serious dewatering issues are expected. However, minimal pumping capacity has been allowed for. Water ingress due to rainfall will be managed with berms and cut-off drains.

16.10 Operating Hours

Total yearly hours used was 8760, with 730 hrs / month.

Weather consideration was calculated as follows:

Table 16-14 Weather Consideration

Weather		
Rain factor	0.50	hrs/1mm of rain
Average rainfall/year	150	mm/year
Hours lost/year	75	hrs/year
Hrs/day	24	hrs
Days lost/year	3.13	days/ year

To the above 3 public holidays were added.

Other factors that were included were mobilise and start-up checks, lunch and other breaks, end of shift, blasting and demobilising time. Adding Utilisation and Availability together with overall job efficiency results in 476 actual working hours per month. This was then used in the equipment calculations.

16.11 Mining Equipment

Separate sets of mining equipment are envisaged for waste mining and ROM production.

- Waste mining: Komatsu PC1250 Hex with 50t Komatsu HD465RDT trucks.
- ROM mining; Komatsu PC500 Hex with 30t Komatsu HM300ADT trucks.
- Drilling Epiroc D65

The smaller mining equipment is envisaged to facilitate cleaner mining of the ROM.

Pre-split drilling has been allowed to facilitate steeper pit slopes. Note that geotechnical work was not done and therefore the slope parameters are based on what is termed a “reasonable” assumption. This will be verified during following studies.

Ramp width was set at 15m which is in line with the physical size of the selected fleet.

16.12 Personnel

Personnel was calculated using a 4 shift FULCO system. There are 4 operators allowed for each piece of production equipment excluding annual and sick leave.

16.13 Production Schedule

16.13.1 Imported Values

The production schedule was developed using Xpac scheduling software.

A 20m x 20m grid was created for each area such that the block orientation is approximately aligned with the general strike of the deposit. This grid was then used to reserve the mining model within the design pit to create input files for Xpac.

The mineralised material has been previously classified by TREO content into three categories, “Low grade” $TREO < 0.05\%$, “Stockpile” $TREO \geq 0.05\%$ and $< 0.1\%$, and “ROM” $TREO \geq 0.1\%$. The values for these classifications, together with “Waste” were imported into separate Xpac database fields.

The imported values were compared with reported values from the Surpac model within the design pits. The comparisons are shown in Table 16-15 and Table 16-16. It is noted that there is a discrepancy in the waste tonnage for A2B of -3.49%. However, this is not significant for this level of study.

Table 16-15 Area 4 Surpac Validation

	Waste		Lowgrade		Stockpile		ROM	
Area 4 Pit Rev4	Xpac	Surpac	Xpac	Surpac	Xpac	Surpac	Xpac	Surpac
Tonnes	104 098 217	104 094 702	1 317 589	1 317 721	10 520 335	10 519 645	24 614 992	24 613 987
RD	2.78	2.78	2.77	2.77	2.75	2.75	2.740	2.74
Ce2O3	3.158	3.16	74.414	73.869	189.17	189.171	333.857	333.513
Dy2O3	0.538	0.54	14.857	14.576	36.17	36.446	112.780	113.952
Er2O3	0.400	0.401	8.904	8.713	21.59	21.76	69.575	70.302
Eu2O3	0.166	0.166	3.502	3.464	7.85	7.896	17.282	17.419
Gd2O3	0.504	0.506	13.073	12.916	29.56	29.757	74.232	74.888
Ho2O3	0.161	0.161	3.041	2.979	7.44	7.502	24.040	24.302
La2O3	1.944	1.946	39.018	38.733	101.24	101.226	187.598	187.348
Lu2O3	0.064	0.064	1.264	1.242	2.94	2.967	9.220	9.32
Nd2O3	1.362	1.365	33.343	33.084	78.83	78.875	127.612	127.577
Pr2O3	0.917	0.921	8.537	8.471	20.86	20.865	35.156	35.132
Sm2O3	0.503	0.505	9.929	9.883	23.00	23.106	41.648	41.849
Tb2O3	0.112	0.113	2.380	2.342	5.69	5.731	16.623	16.787
Tm2O3	0.066	0.067	1.332	1.304	3.22	3.246	10.470	10.583
Y2O3	1.851	1.848	96.677	94.397	237.15	239.058	775.812	783.722
Yb2O3	0.382	0.384	8.503	8.343	20.11	20.271	63.407	64.083
LREO	0.001	0.001	0.017	0.016	0.04	0.041	0.073	0.073
HREO	0.000	0.000	0.015	0.015	0.04	0.037	0.117	0.119
TREO	0.001	0.001	0.032	0.031	0.08	0.079	0.190	0.191

Table 16-16 Area 2B Surpac Validation

	Waste		Lowgrade		Stockpile		ROM	
Area 2B Pit Rev5	Xpac	Surpac	Xpac	Surpac	Xpac	Surpac	Xpac	Surpac
Tonnes	25 833 688	26 767 848	2 231 644	2 237 112	3 373 387	3 372 320	4 357 487	4 356 437
RD	2.34	2.83	2.78	2.77	2.77	2.76	2.767	2.76
Ce2O3	2.510	2.545	72.677	72.087	132.53	132.991	396.351	400.547
Dy2O3	0.873	0.884	20.421	20.388	41.18	41.422	98.775	99.073
Er2O3	0.371	0.378	10.977	10.967	22.51	22.681	54.272	54.346
Eu2O3	0.197	0.200	5.624	5.588	10.87	10.916	27.045	27.505
Gd2O3	0.614	0.625	19.182	19.099	36.16	36.279	91.134	92.277
Ho2O3	0.155	0.157	3.920	3.916	7.94	7.995	19.047	19.078
La2O3	1.624	1.643	37.056	36.759	71.91	72.256	252.580	255.286
Lu2O3	0.054	0.055	1.456	1.455	3.01	3.038	7.221	7.259
Nd2O3	1.661	1.689	40.375	39.957	72.60	72.736	171.641	174.158
Pr2O3	0.399	0.404	8.933	8.85	15.85	15.882	41.274	41.77
Sm2O3	0.588	0.600	16.454	16.303	30.42	30.515	75.022	76.507
Tb2O3	0.123	0.125	3.415	3.405	6.67	6.7	16.452	16.556
Tm2O3	0.056	0.057	1.555	1.553	3.24	3.265	7.826	7.845
Y2O3	4.081	4.161	120.249	120.079	250.22	251.948	613.109	612.234
Yb2O3	0.344	0.351	9.886	9.876	20.62	20.834	49.960	50.135
LREO	0.001	0.001	0.018	0.017	0.03	0.032	0.094	0.095
HREO	0.001	0.001	0.020	0.020	0.04	0.040	0.098	0.099
TREO	0.001	0.001	0.037	0.037	0.07	0.073	0.192	0.193

16.13.2 Scheduling Values

The imported values were processed by a set of XCM's (Scripts which calculate values in the Xpac database).

These applied 5% geological and 2.5% mining loss to calculate a ROM value. Dilution having been included in the re-blocking and reserving process.

The waste, low grade and stockpile tonnes were combined to a single value for scheduling purposes, and the mining loss added to this. However, provision was made to report the stockpile values separately. The schedule input values are shown in Table 16-17, Table 16-18 and Table 16-19.

As noted in Section 16.8, topsoil is minimal with an assumed thickness of 0.3m. Consequently, topsoil is included in the waste tonnage as it represents less than 0.5% of the waste moved, and the haul distance for both is very similar.

Table 16-17 Scheduling Values (Waste)

Pit Area	A4	A2B
Scheduling Tonnes		
Waste + LG + Loss		
Tonnes	106 031 181	28 174 269
RD	2.78	2.75
Ce ₂ O ₃	5.977	9.650
Dy ₂ O ₃	1.370	2.821
Er ₂ O ₃	0.908	1.430
Eu ₂ O ₃	0.307	0.736
Gd ₂ O ₃	1.091	2.453
Ho ₂ O ₃	0.336	0.530
La ₂ O ₃	3.490	5.433
Lu ₂ O ₃	0.132	0.194
Nd ₂ O ₃	2.499	5.419
Pr ₂ O ₃	1.212	1.241
Sm ₂ O ₃	0.861	2.147
Tb ₂ O ₃	0.237	0.450
Tm ₂ O ₃	0.143	0.206
Y ₂ O ₃	7.536	15.753
Yb ₂ O ₃	0.851	1.302
LREO	0.001	0.002
HREO	0.001	0.003
TREO	0.003	0.005

Table 16-18 Scheduling Values (Stockpile + Total Waste)

Pit Area	A4	A2B
Scheduling Tonnes		
Stockpile		
Tonnes	10 520 335	3 373 387
RD	2.75	2.76
Ce ₂ O ₃	189.166	132.527
Dy ₂ O ₃	36.167	41.183
Er ₂ O ₃	21.591	22.506
Eu ₂ O ₃	7.846	10.867
Gd ₂ O ₃	29.563	36.156
Ho ₂ O ₃	7.443	7.943
La ₂ O ₃	101.237	71.909
Lu ₂ O ₃	2.944	3.005
Nd ₂ O ₃	78.830	72.604
Pr ₂ O ₃	20.860	15.848
Sm ₂ O ₃	22.999	30.419
Tb ₂ O ₃	5.689	6.667
Tm ₂ O ₃	3.220	3.236
Y ₂ O ₃	237.154	250.220
Yb ₂ O ₃	20.109	20.622
LREO	0.041	0.032
HREO	0.037	0.040
TREO	0.078	0.073
Waste + LG + Loss + Stockpile		
Volume	41 918 373	11 440 994
Tonnes	116 551 516	31 547 656
RD	2.78	2.76

Table 16-19 Scheduling Values (ROM)

Pit Area	A4	A2B
Scheduling Tonnes		
ROM		
Tonnes	22 799 636	4 036 123
RD	2.74	2.76
Ce2O3	333.857	396.351
Dy2O3	112.780	98.775
Er2O3	69.575	54.272
Eu2O3	17.282	27.045
Gd2O3	74.232	91.134
Ho2O3	24.040	19.047
La2O3	187.598	252.580
Lu2O3	9.220	7.221
Nd2O3	127.612	171.641
Pr2O3	35.156	41.274
Sm2O3	41.648	75.022
Tb2O3	16.623	16.452
Tm2O3	10.470	7.826
Y2O3	775.812	613.109
Yb2O3	63.407	49.960
LREO	0.073	0.094
HREO	0.117	0.098
TREO	0.190	0.192

16.13.3 Schedule Control

The mining sequence of scheduling blocks is controlled by sets of dependency rules that guide the development of the pit.

The proposed mining methodology is to develop a slot on each level in front of the mineralised zone, and from this mine predominantly ROM on one side and predominantly waste on the other. This slot location was determined manually by identifying the row of blocks on each level that best fits the mineralised contact and loading this row value into an Xpac parameter database. An XCM then reads these values to create dependency rules that reference the slot as the starting point for each bench.

Further rules prevent waste benches mining too far ahead, while maintaining a lead ahead of lower benches.

16.13.4 Schedule Targets

The mining fleet size was not considered in the scheduling process. Instead targets for ROM and waste are specified for monthly increments in a calendar database, and the Xpac auto-scheduler then attempts to meet these targets while following the dependency rules. Note that the scheduled waste value is the total of waste, lowgrade, stockpile and losses. The stockpile material is reported separately. It is assumed that this will be stockpiled by TREO range as a potential future plant feed.

As several scheduling resources can be operating simultaneously, on each side of the bench slots, and in different pits, the total ROM tonnage is capped at 2 000 000 tonnes/annum.

16.13.5 Schedule Targets

The schedule tonnage output by pit and material type is shown graphically in Figure 16-5.

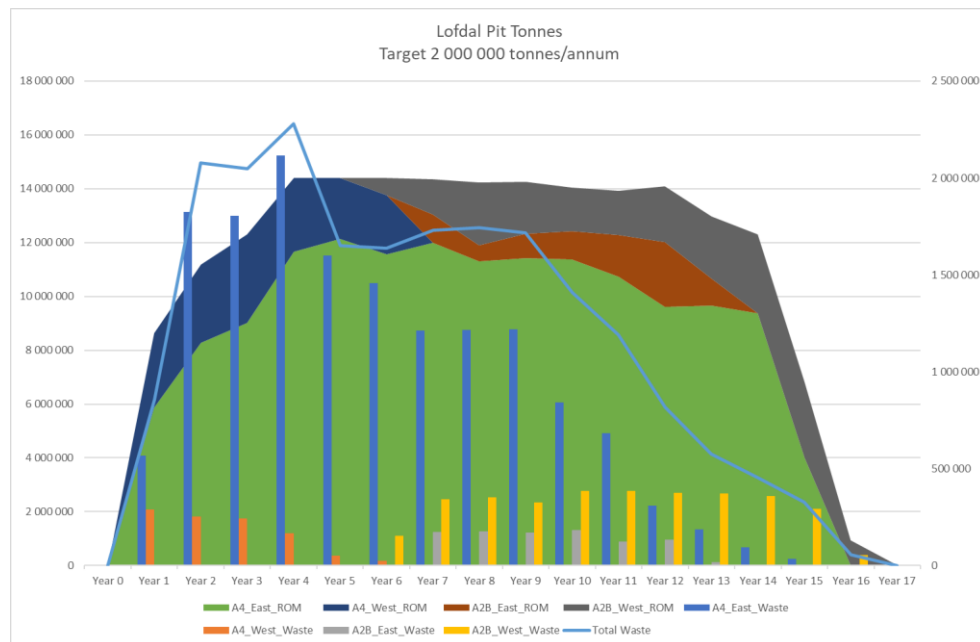


Figure 16-5 Mining Schedule

The scheduled plant feed tonnages and quality values are tabulated in Table 16-20, Table 16-21 and Table 16-22. The life of mine (LOM) is 16 years at an overall average tonne to tonne stripping ratio of 5.52, with Area 4 being 5.11 and Area 2B 7.82.

Table 16-20 Area 4 Plant Feed

A4 Plant Input	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
Tonnes	1 199 876	1 553 560	1 708 732	2 000 000	2 000 000	1 911 999	1 666 161	1 569 994	1 587 873	1 578 716	1 491 520	1 333 248	1 340 584	1 300 000	557 372		
Ce2O3	359.701	328.372	329.184	323.748	321.585	297.229	300.455	299.607	367.406	421.559	309.568	312.658	383.688	363.842	292.051	0.000	0.000
La2O3	215.253	192.139	190.919	183.716	178.779	166.257	164.376	164.207	202.222	234.847	165.841	169.212	216.568	208.429	167.763	0.000	0.000
Nd2O3	132.115	122.973	124.239	123.456	124.610	116.035	119.503	118.397	138.025	153.819	124.876	123.576	142.103	137.029	113.051	0.000	0.000
Pr2O3	37.323	34.163	34.255	33.845	33.892	31.572	32.135	32.026	38.434	44.008	33.776	33.701	39.943	37.817	30.436	0.000	0.000
Sm2O3	39.302	38.971	40.045	40.247	43.391	41.875	41.166	42.081	43.787	50.681	40.944	37.446	40.348	41.608	40.784	0.000	0.000
Dy2O3	109.918	111.148	100.171	107.199	119.834	123.785	116.857	126.947	132.269	155.433	110.198	83.233	76.515	80.621	130.516	0.000	0.000
Er2O3	68.411	69.317	61.168	66.429	72.755	75.353	71.092	76.232	82.033	97.666	67.911	51.170	46.020	50.462	89.985	0.000	0.000
Eu2O3	16.570	16.500	16.273	16.389	18.367	18.523	17.702	18.987	19.110	21.603	16.773	14.183	14.271	14.816	17.422	0.000	0.000
Gd2O3	70.535	71.020	68.629	70.534	80.042	81.399	77.411	83.370	84.053	96.677	71.602	57.554	56.755	58.170	75.810	0.000	0.000
Ho2O3	23.539	23.822	21.266	22.918	25.253	26.310	24.847	26.512	28.174	33.784	23.764	17.886	16.119	16.917	28.484	0.000	0.000
Lu2O3	9.105	9.328	8.026	8.889	9.461	9.864	9.411	9.695	10.803	12.914	9.079	6.990	6.181	6.868	12.576	0.000	0.000
Tb2O3	16.115	16.216	15.157	15.923	17.974	18.327	17.392	18.744	19.066	22.416	16.058	12.313	11.635	12.050	18.003	0.000	0.000
Tm2O3	10.250	10.432	9.135	9.983	10.794	11.319	10.790	11.341	12.417	14.957	10.298	7.673	6.815	7.545	13.902	0.000	0.000
Y2O3	747.343	754.700	678.688	736.014	823.857	861.102	813.226	872.269	920.673	1125.040	760.944	555.502	504.199	526.616	887.421	0.000	0.000
Yb2O3	62.438	63.559	55.012	60.871	65.437	67.987	64.636	67.996	75.354	88.977	61.993	47.636	41.965	46.582	86.174	0.000	0.000
HREO	0.113	0.115	0.103	0.112	0.124	0.129	0.122	0.131	0.138	0.167	0.115	0.085	0.078	0.082	0.136	0.000	0.000
LREO	0.078	0.072	0.072	0.071	0.070	0.065	0.066	0.066	0.079	0.091	0.068	0.068	0.082	0.079	0.064	0.000	0.000
TREO	0.192	0.186	0.175	0.182	0.195	0.195	0.188	0.197	0.217	0.257	0.182	0.153	0.160	0.161	0.200	0.000	0.000

Table 16-21 Area 2B Plant Feed

A2B Plant Input	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
Tonnes						88 001	326 351	406 527	390 610	373 120	441 590	622 474	461 261	407 114	389 256	129 820	
Ce2O3	0.000	0.000	0.000	0.000	0.000	437.802	345.131	415.299	372.609	419.272	322.389	263.315	313.719	415.534	672.371	738.515	0.000
La2O3	0.000	0.000	0.000	0.000	0.000	288.144	214.303	266.535	235.596	270.053	204.162	163.434	197.293	264.370	437.888	477.861	0.000
Nd2O3	0.000	0.000	0.000	0.000	0.000	151.058	161.138	164.558	167.136	175.998	141.199	123.971	131.788	168.941	293.033	353.412	0.000
Pr2O3	0.000	0.000	0.000	0.000	0.000	42.255	37.026	41.759	39.056	43.078	33.622	28.318	32.372	42.822	70.100	79.760	0.000
Sm2O3	0.000	0.000	0.000	0.000	0.000	55.491	77.174	68.990	80.943	77.411	68.122	64.109	60.249	61.219	111.782	138.411	0.000
Dy2O3	0.000	0.000	0.000	0.000	0.000	92.064	80.721	84.899	92.516	94.253	109.254	132.345	100.779	83.124	90.065	95.454	0.000
Er2O3	0.000	0.000	0.000	0.000	0.000	52.351	44.489	46.858	50.585	50.893	62.878	78.618	57.699	45.353	39.534	38.174	0.000
Eu2O3	0.000	0.000	0.000	0.000	0.000	21.898	25.984	24.342	28.439	27.349	25.602	25.900	22.805	22.092	37.746	45.510	0.000
Gd2O3	0.000	0.000	0.000	0.000	0.000	79.464	84.140	81.863	92.966	90.353	89.434	96.175	81.042	73.675	116.925	137.288	0.000
Ho2O3	0.000	0.000	0.000	0.000	0.000	18.269	15.590	16.441	17.779	17.990	21.653	26.713	20.036	16.015	15.170	15.285	0.000
Lu2O3	0.000	0.000	0.000	0.000	0.000	6.360	6.263	6.317	7.182	6.885	8.432	10.390	7.556	6.059	4.844	4.399	0.000
Tb2O3	0.000	0.000	0.000	0.000	0.000	15.234	14.042	14.485	15.966	16.019	17.332	19.907	15.712	13.695	17.773	19.965	0.000
Tm2O3	0.000	0.000	0.000	0.000	0.000	7.436	6.501	6.800	7.437	7.355	9.124	11.429	8.318	6.562	5.378	5.021	0.000
Y2O3	0.000	0.000	0.000	0.000	0.000	587.707	492.455	524.024	556.559	572.479	720.713	902.611	647.055	499.215	456.015	452.964	0.000
Yb2O3	0.000	0.000	0.000	0.000	0.000	45.522	41.924	43.487	48.481	47.182	58.465	72.848	52.879	41.910	33.649	30.980	0.000
HREO	0.000	0.000	0.000	0.000	0.000	0.093	0.081	0.085	0.092	0.093	0.112	0.138	0.101	0.081	0.082	0.085	0.000
LREO	0.000	0.000	0.000	0.000	0.000	0.098	0.083	0.096	0.090	0.099	0.077	0.064	0.074	0.095	0.158	0.179	0.000
TREO	0.000	0.000	0.000	0.000	0.000	0.190	0.165	0.181	0.181	0.192	0.189	0.202	0.175	0.176	0.240	0.263	0.000

Table 16-22 Total Plant Feed

Total Plant Input	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
Tonnes	1 199 876	1 553 560	1 708 732	2 000 000	2 000 000	2 000 000	1 992 512	1 976 521	1 978 483	1 951 836	1 933 110	1 955 722	1 801 845	1 707 114	946 628	129 820	
Ce2O3	359.701	328.372	329.184	323.748	321.585	303.414	307.772	323.402	368.434	421.122	312.497	296.953	365.776	376.170	448.440	738.515	0.000
La2O3	215.253	192.139	190.919	183.716	178.779	171.620	172.553	185.254	208.811	241.577	174.595	167.373	211.634	221.770	278.839	477.861	0.000
Nd2O3	132.115	122.973	124.239	123.456	124.610	117.576	126.322	127.892	143.772	158.059	128.605	123.702	139.462	144.640	187.060	353.412	0.000
Pr2O3	37.323	34.163	34.255	33.845	33.892	32.042	32.936	34.028	38.557	43.830	33.741	31.988	38.005	39.010	46.746	79.760	0.000
Sm2O3	39.302	38.971	40.045	40.247	43.391	42.474	47.064	47.616	51.123	55.791	47.153	45.932	45.442	46.285	69.978	138.411	0.000
Dy2O3	109.918	111.148	100.171	107.199	119.834	122.389	110.938	118.298	124.420	143.738	109.983	98.865	82.726	81.218	113.882	95.454	0.000
Er2O3	68.411	69.317	61.168	66.429	72.755	74.340	66.735	70.190	75.824	88.725	66.761	59.906	49.010	49.244	69.239	38.174	0.000
Eu2O3	16.570	16.500	16.273	16.389	18.367	18.672	19.058	20.088	20.952	22.701	18.790	17.912	16.456	16.551	25.780	45.510	0.000
Gd2O3	70.535	71.020	68.629	70.534	80.042	81.313	78.513	83.060	85.813	95.468	75.676	69.846	62.972	61.867	92.717	137.288	0.000
Ho2O3	23.539	23.822	21.266	22.918	25.253	25.956	23.331	24.440	26.122	30.764	23.282	20.696	17.122	16.702	23.009	15.285	0.000
Lu2O3	9.105	9.328	8.026	8.889	9.461	9.709	8.895	9.000	10.088	11.762	8.931	8.072	6.533	6.675	9.397	4.399	0.000
Tb2O3	16.115	16.216	15.157	15.923	17.974	18.191	16.843	17.868	18.454	21.193	16.349	14.730	12.679	12.442	17.908	19.965	0.000
Tm2O3	10.250	10.432	9.135	9.983	10.794	11.149	10.088	10.407	11.434	13.504	10.030	8.868	7.200	7.311	10.397	5.021	0.000
Y2O3	747.343	754.700	678.688	736.014	823.857	849.072	760.687	800.643	848.786	1019.410	751.754	665.981	540.769	520.081	710.026	452.964	0.000
Yb2O3	62.438	63.559	55.012	60.871	65.437	66.999	60.916	62.956	70.049	80.987	61.187	55.660	44.758	45.468	64.575	30.980	0.000
HREO	0.113	0.115	0.103	0.112	0.124	0.128	0.116	0.122	0.129	0.153	0.114	0.102	0.084	0.082	0.114	0.085	0.000
LREO	0.078	0.072	0.072	0.071	0.070	0.067	0.069	0.072	0.081	0.092	0.070	0.067	0.080	0.083	0.103	0.179	0.000
TREO	0.192	0.186	0.175	0.182	0.195	0.194	0.184	0.193	0.210	0.245	0.184	0.169	0.164	0.165	0.217	0.263	0.000

16.13.6 Period Progress Plots

The pit development for Area 4 is shown in Figure 16-6 to Figure 16-20, and for Area 2B in Figure 16-21 to Figure 16-31.

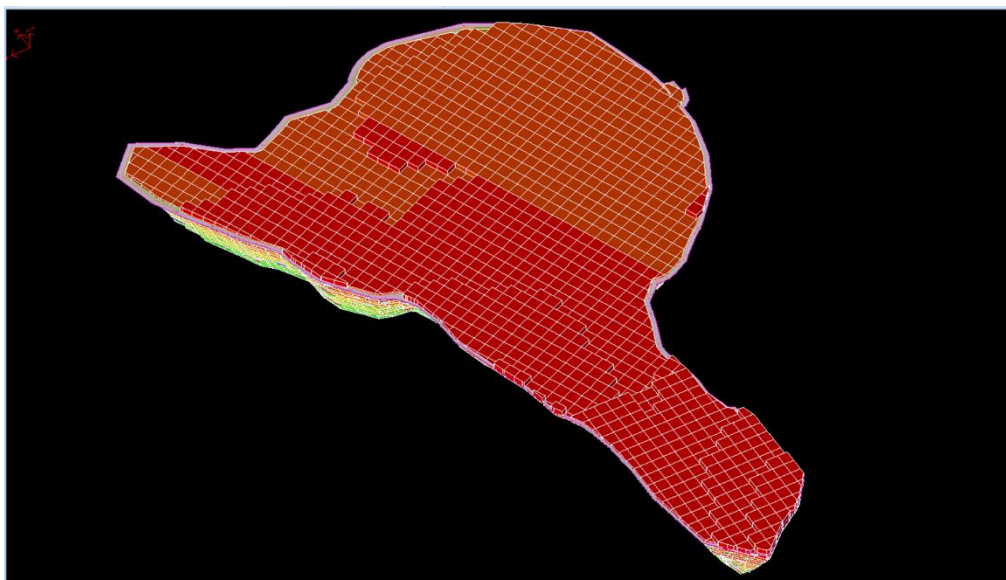


Figure 16-6 Area 4 Year 1

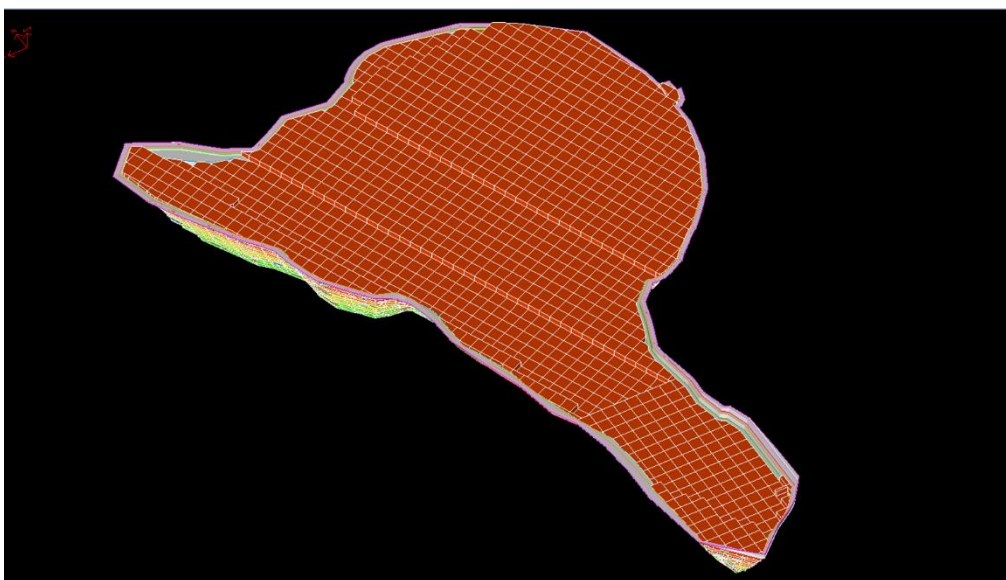


Figure 16-7 Area 4 Year 2

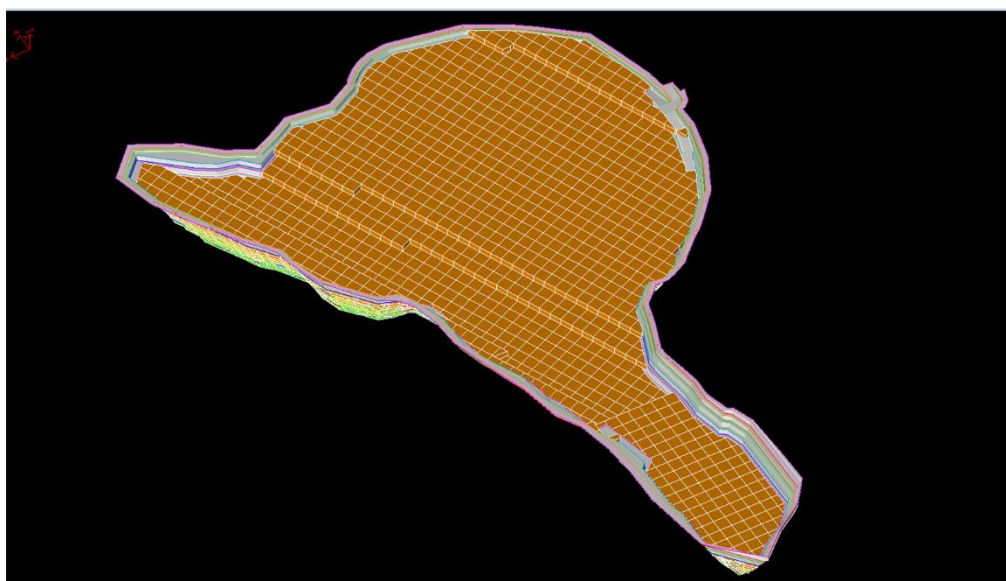


Figure 16-8 Area 4 Year 3

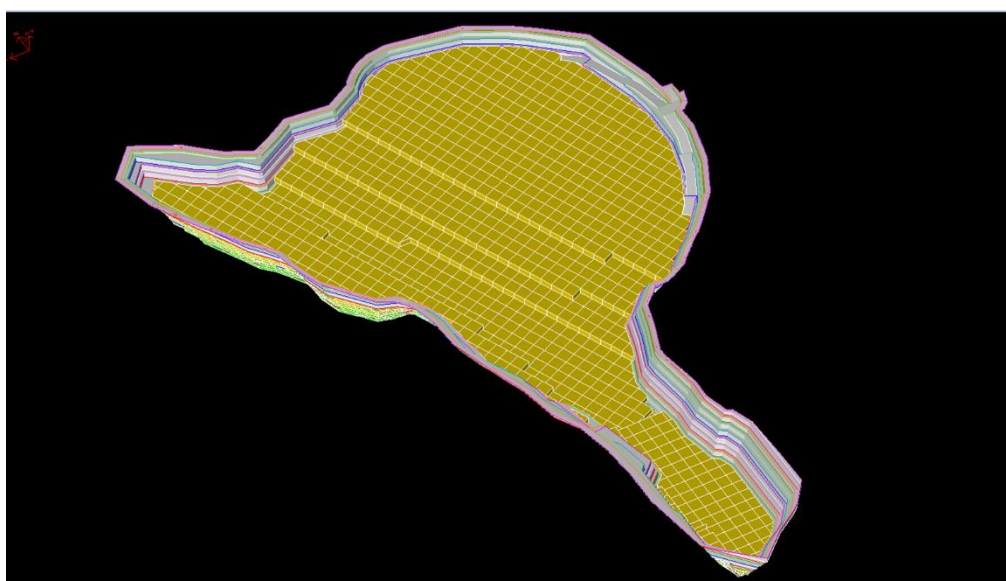


Figure 16-9 Area 4 Year 4

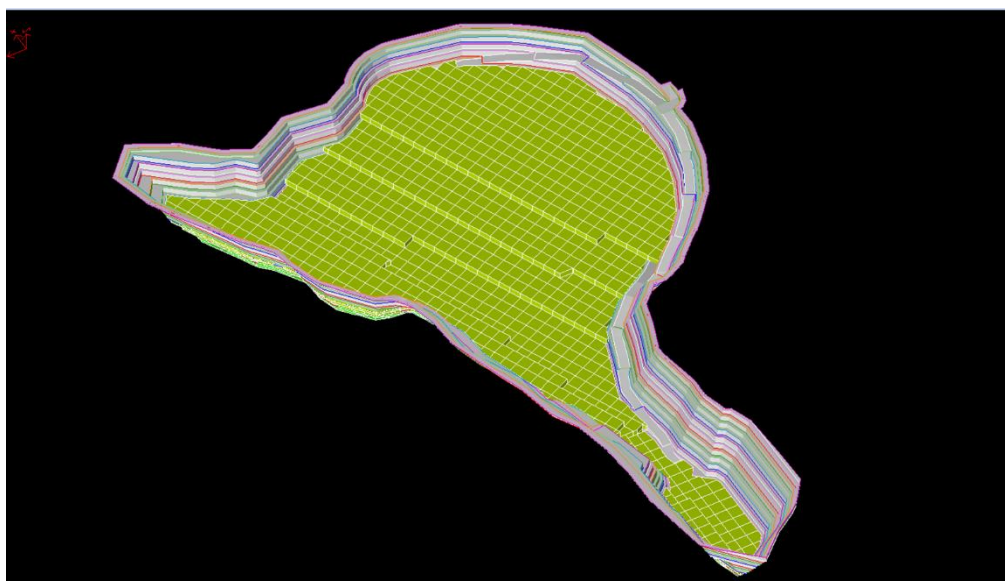


Figure 16-10 Area 4 Year 5

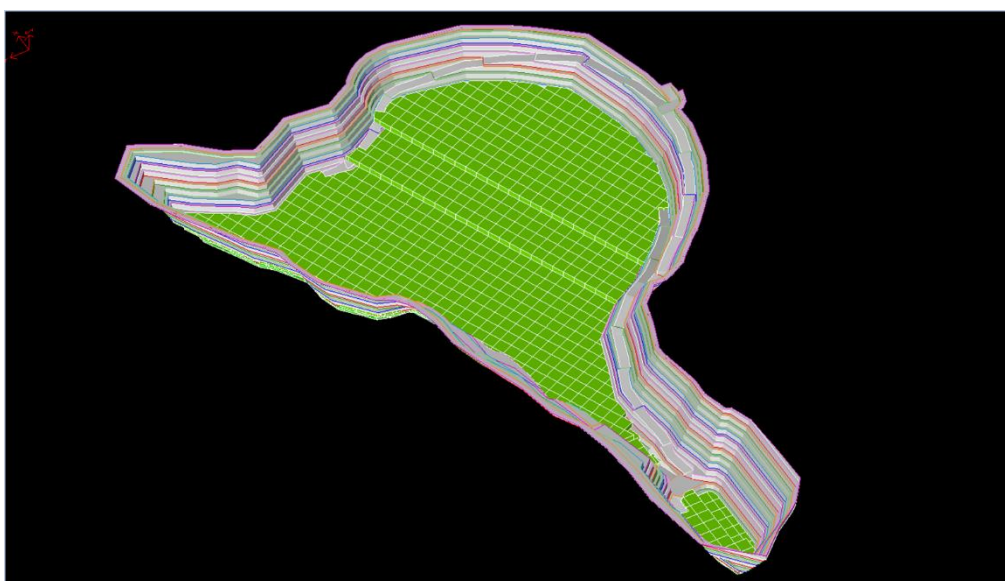


Figure 16-11 Area 4 Year 6

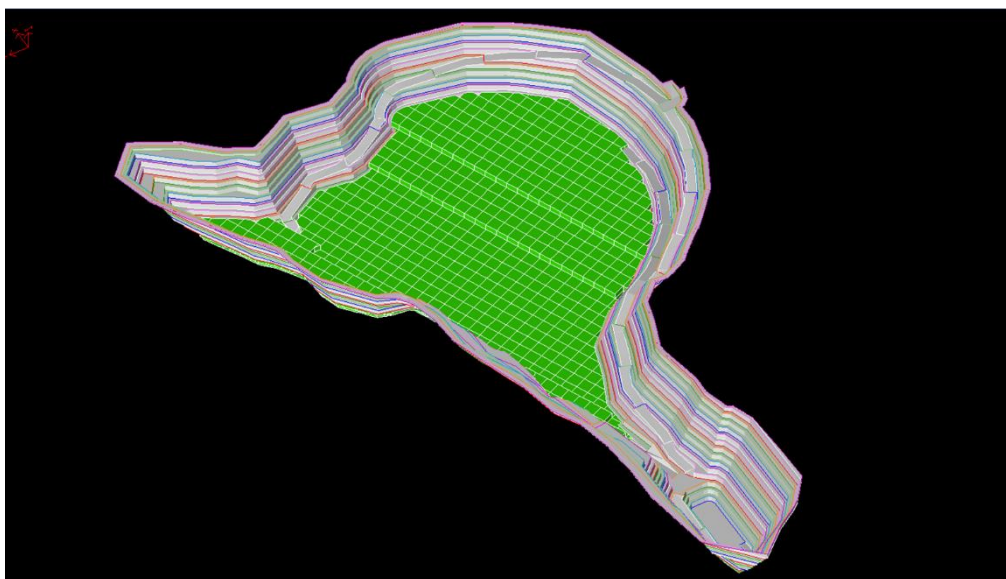


Figure 16-12 Area 4 Year 7

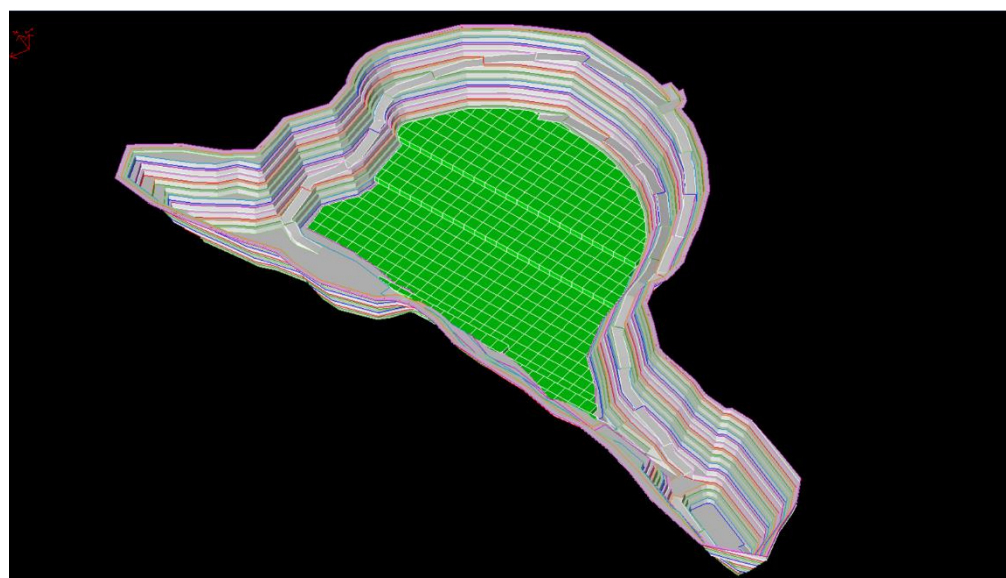


Figure 16-13 Area 4 Year 8

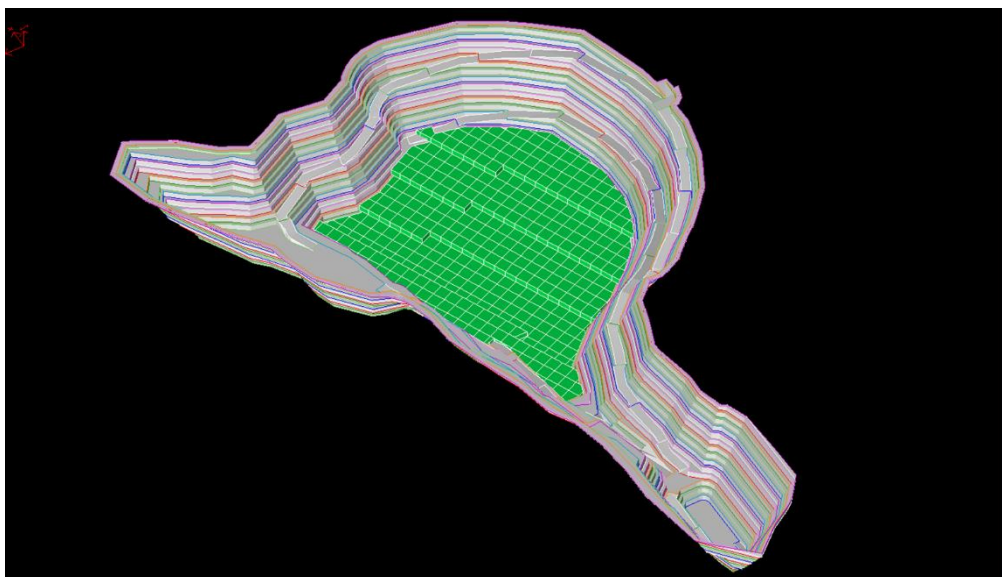


Figure 16-14 Area 4 Year 9

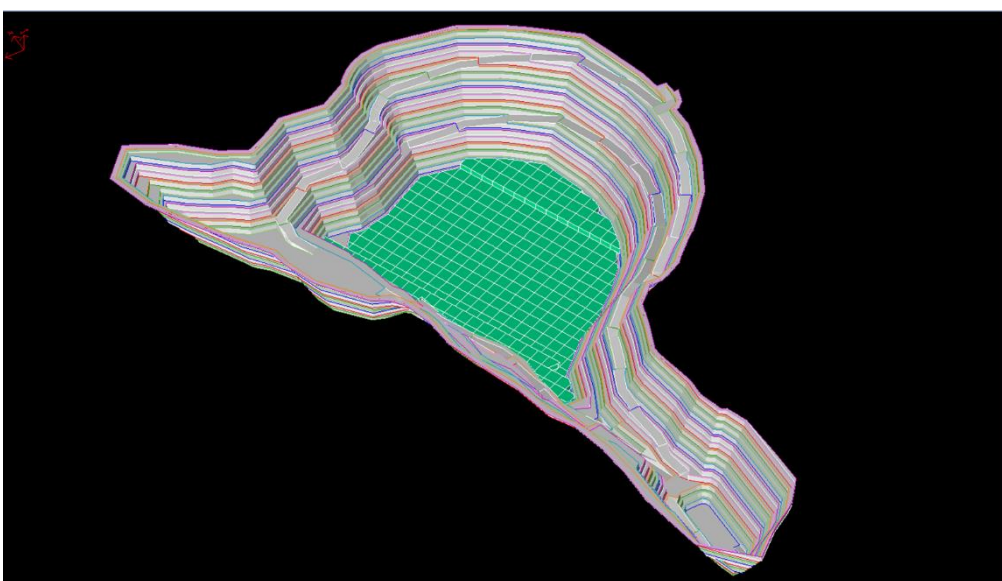


Figure 16-15 Area 4 Year 10

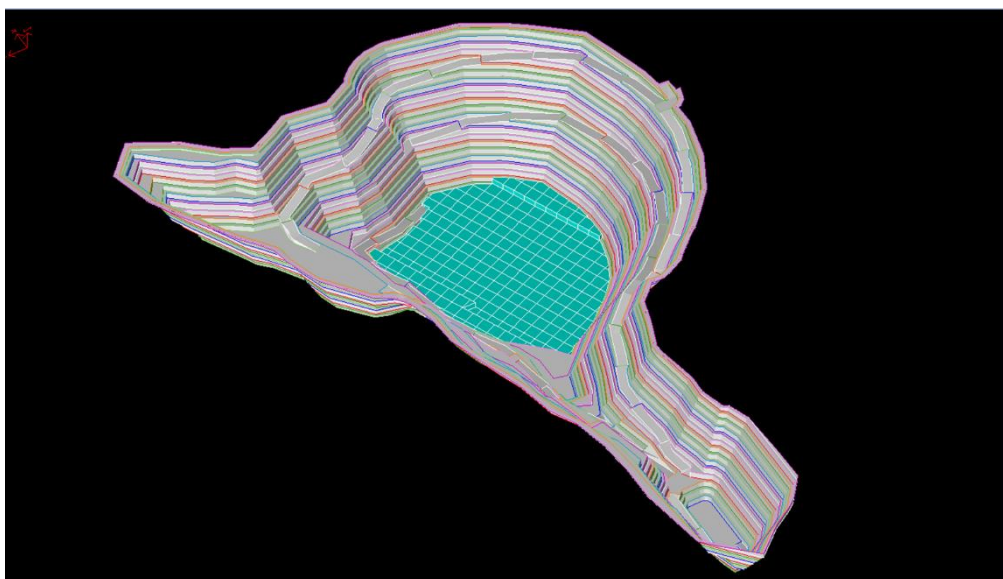


Figure 16-16 Area 4 Year 11

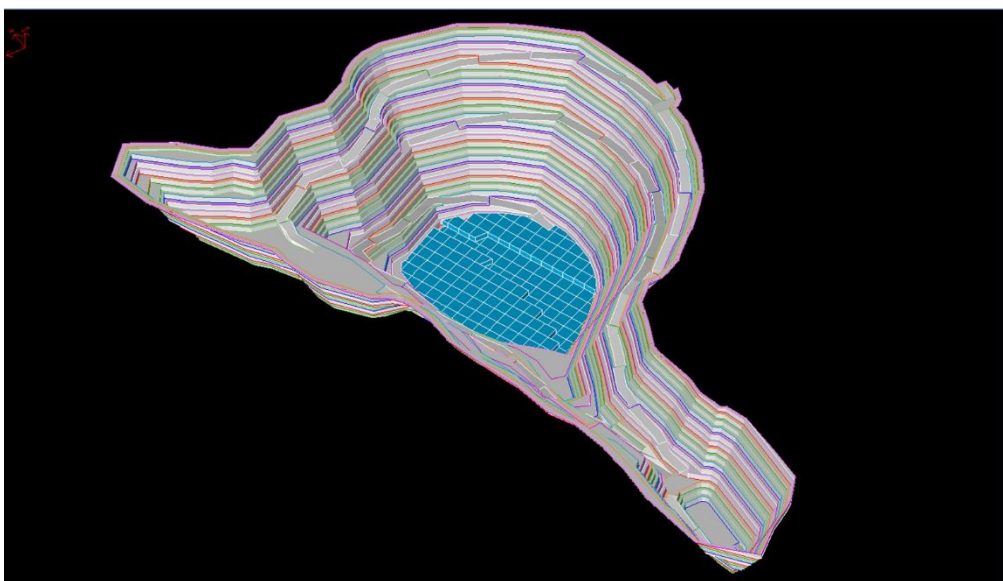


Figure 16-17 Area 4 Year 12

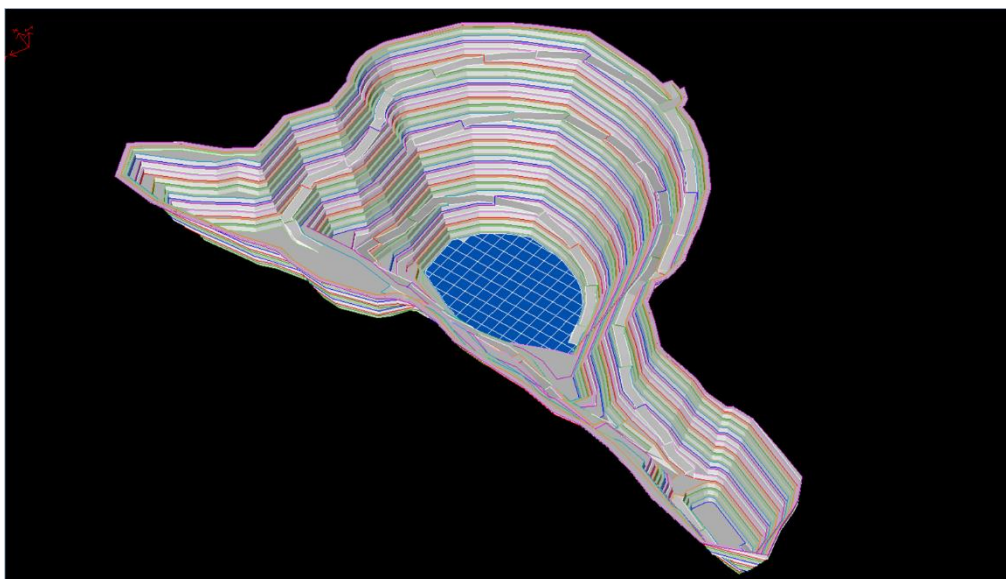


Figure 16-18 Area 4 Year 13

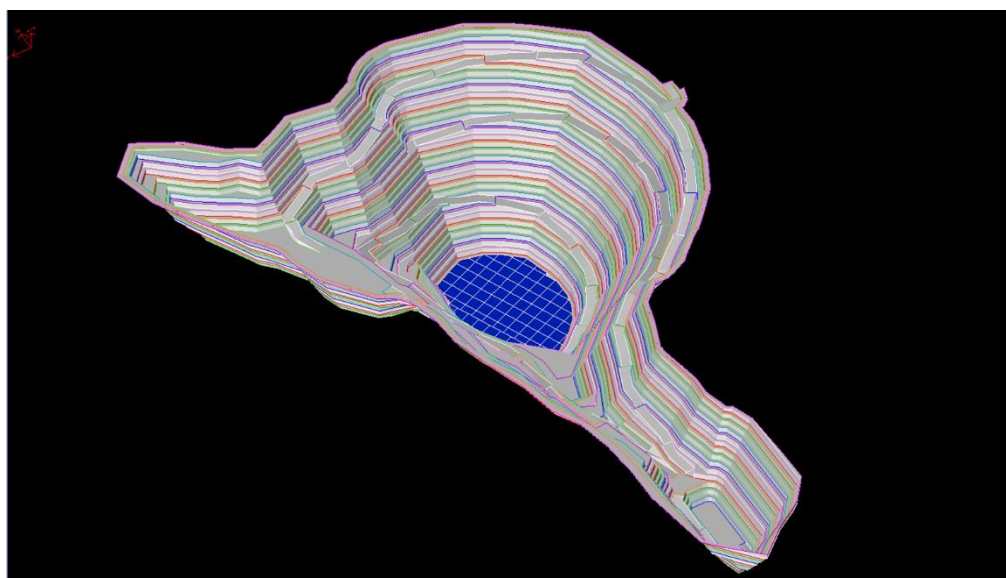


Figure 16-19 Area 4 Year 14

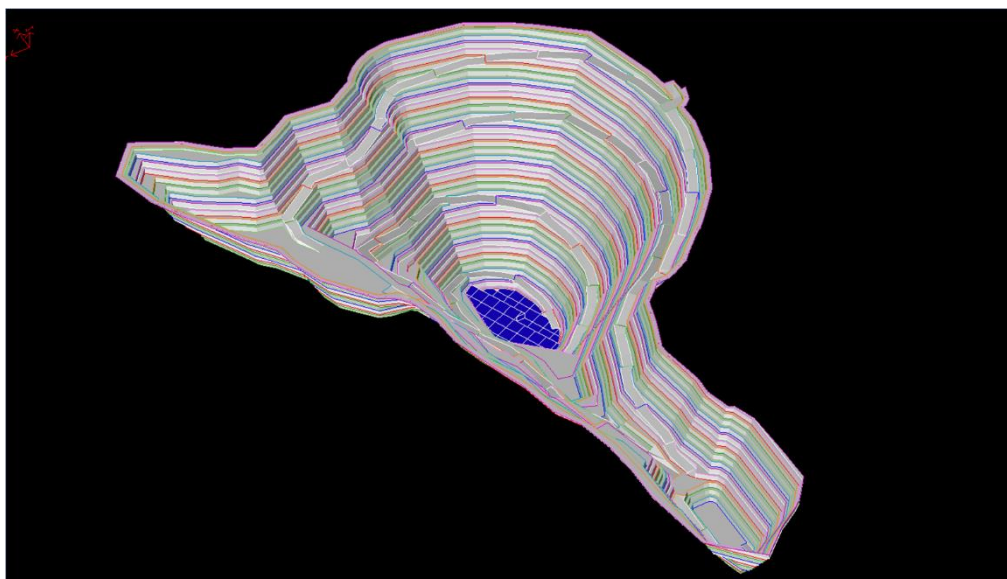


Figure 16-20 Area 4 Year 15

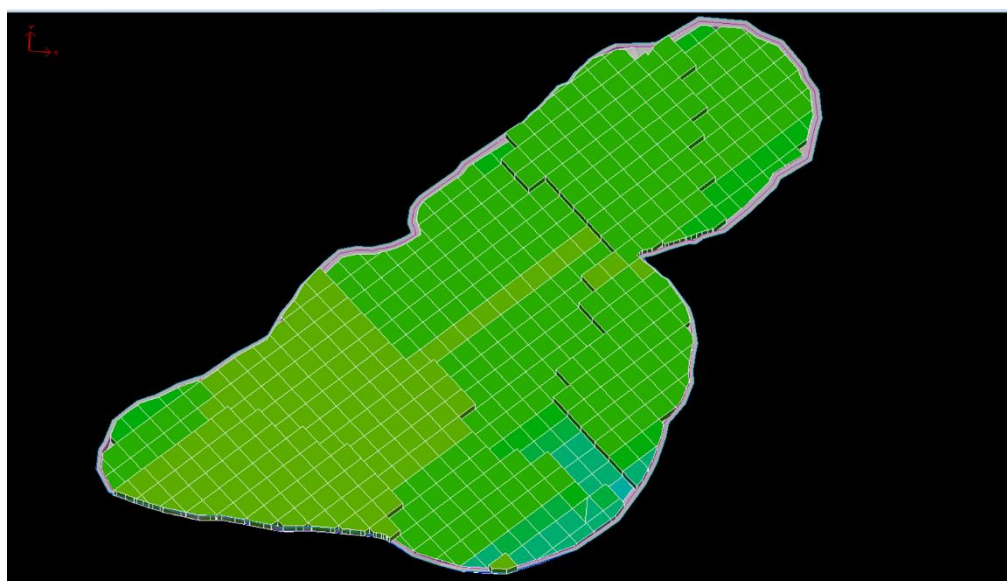


Figure 16-21 Area 2B Year 6

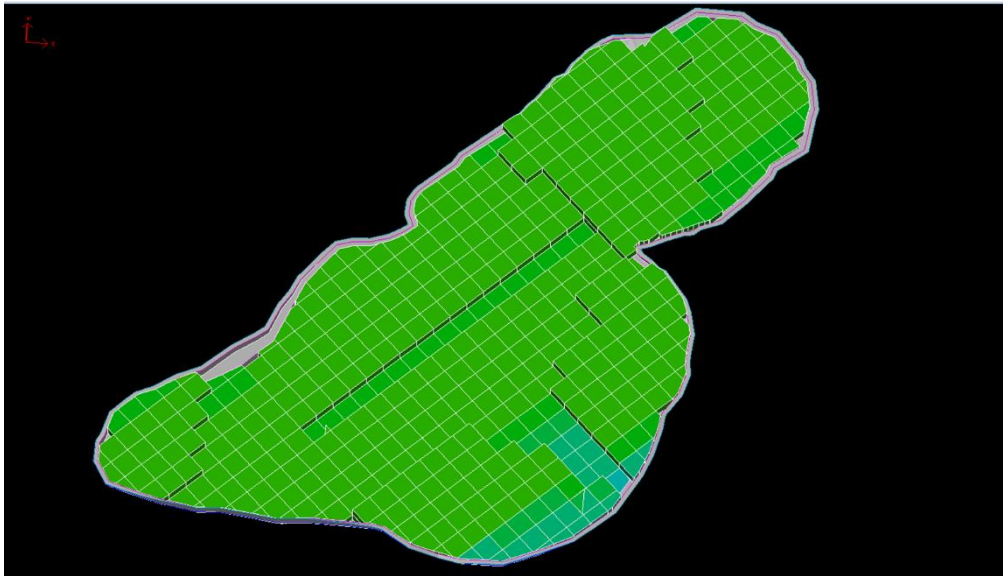


Figure 16-22 Area 2B Year 7



Figure 16-23 Area 2B Year 8

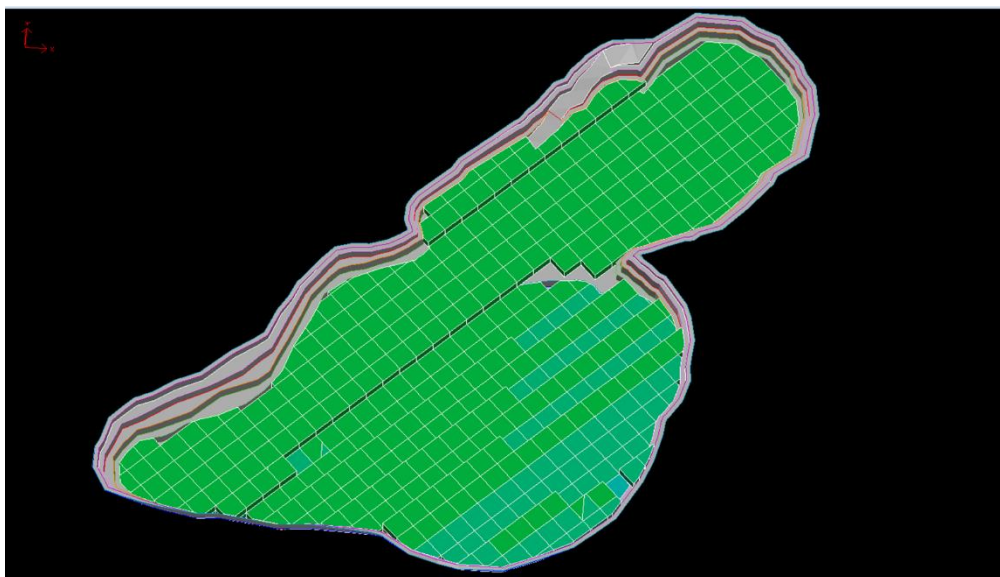


Figure 16-24 Area 2B Year 9



Figure 16-25 Area 2B Year 10

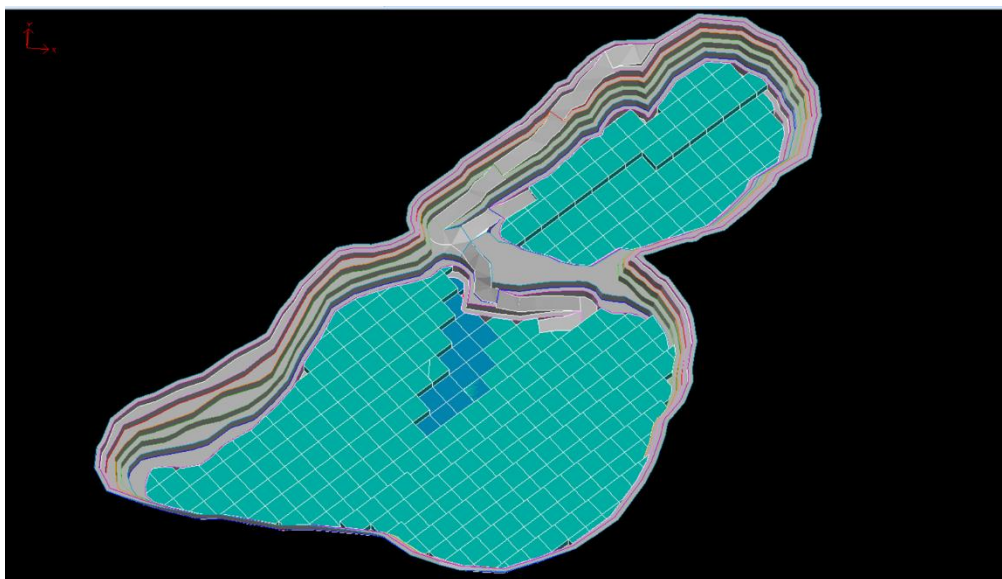


Figure 16-26 Area 2B Year 11

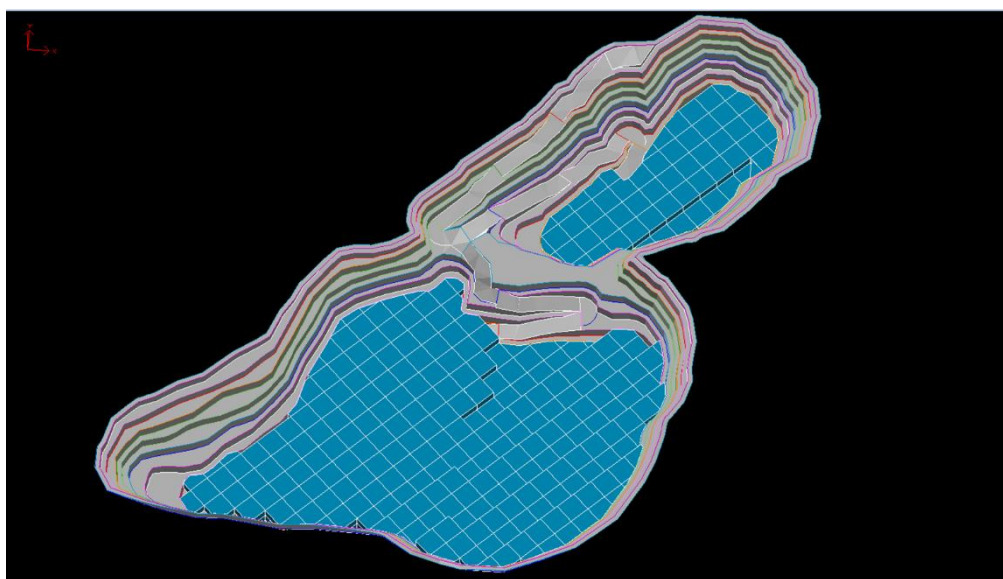


Figure 16-27 Area 2B Year 12

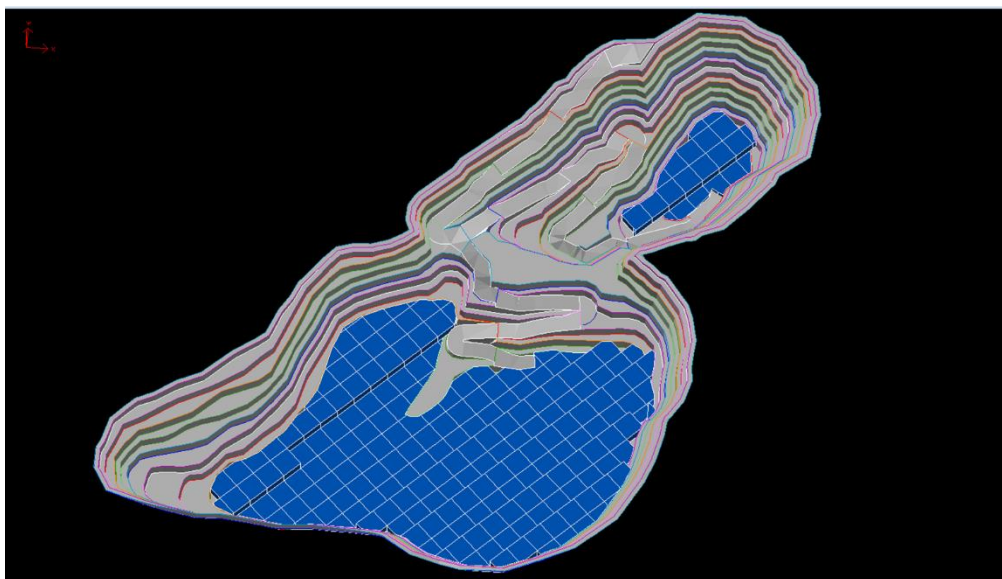


Figure 16-28 Area 2B Year 13

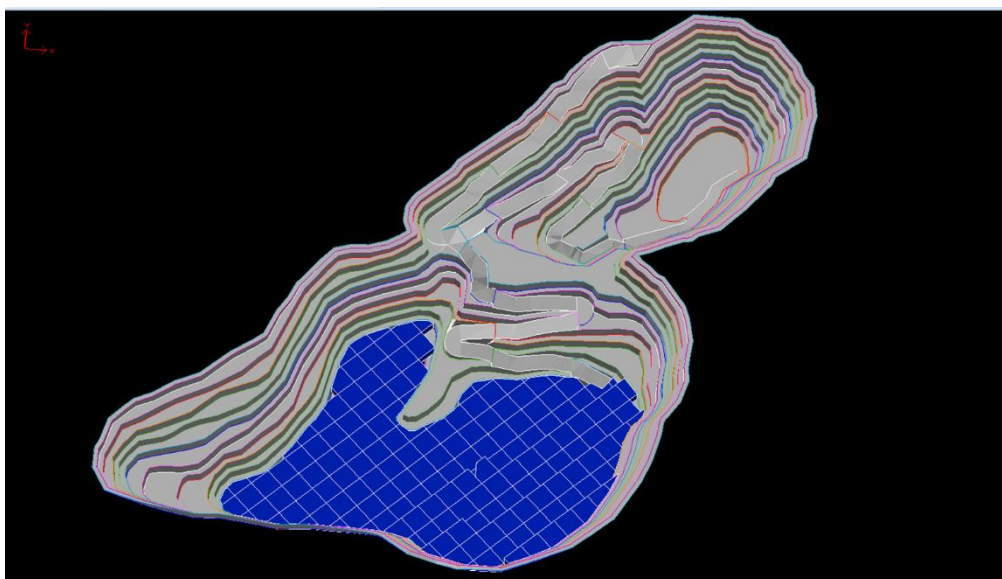


Figure 16-29 Area 2B Year 14

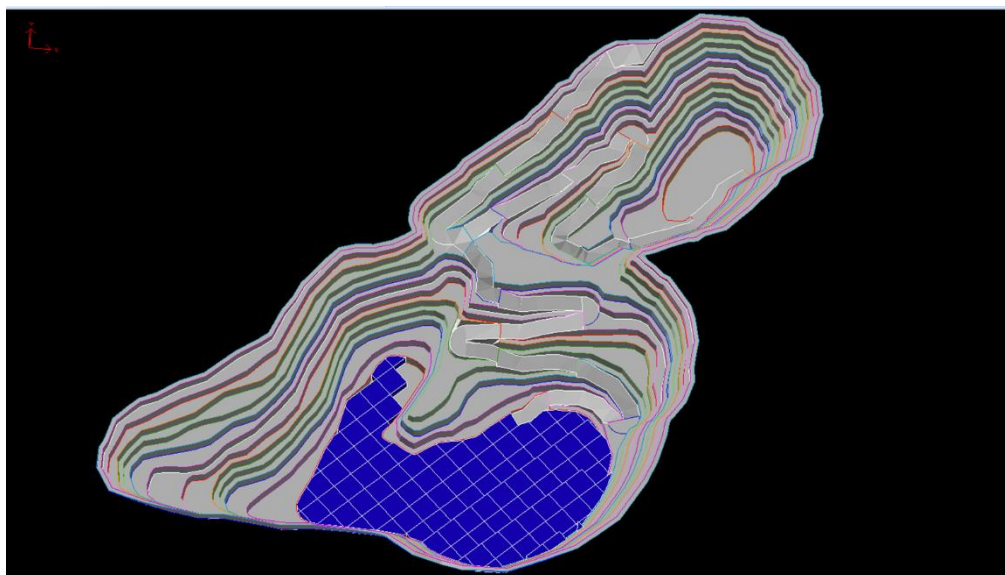


Figure 16-30 Area 2B Year 15

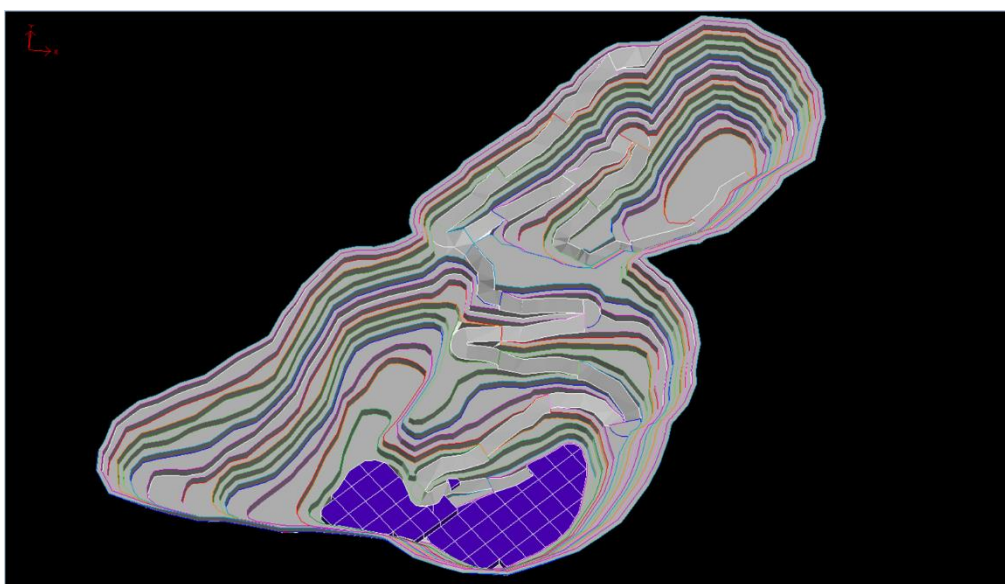


Figure 16-31 Area 2B Year 16

16.13.7 Cost Parameters

16.13.7.1 Haul Distance

The ramp reference points with the bench number are shown in Figure 16-32 for Area 4. Above bench 18 all material south of the slot on each bench in the main east pit is hauled via the ramp marked in orange. All material north of the slot above bench 18 is hauled via the ramp marked in blue. Once below bench 18 all material travels via the blue ramp. The purple numbers show the extension of the blue ramp used for the west pit.

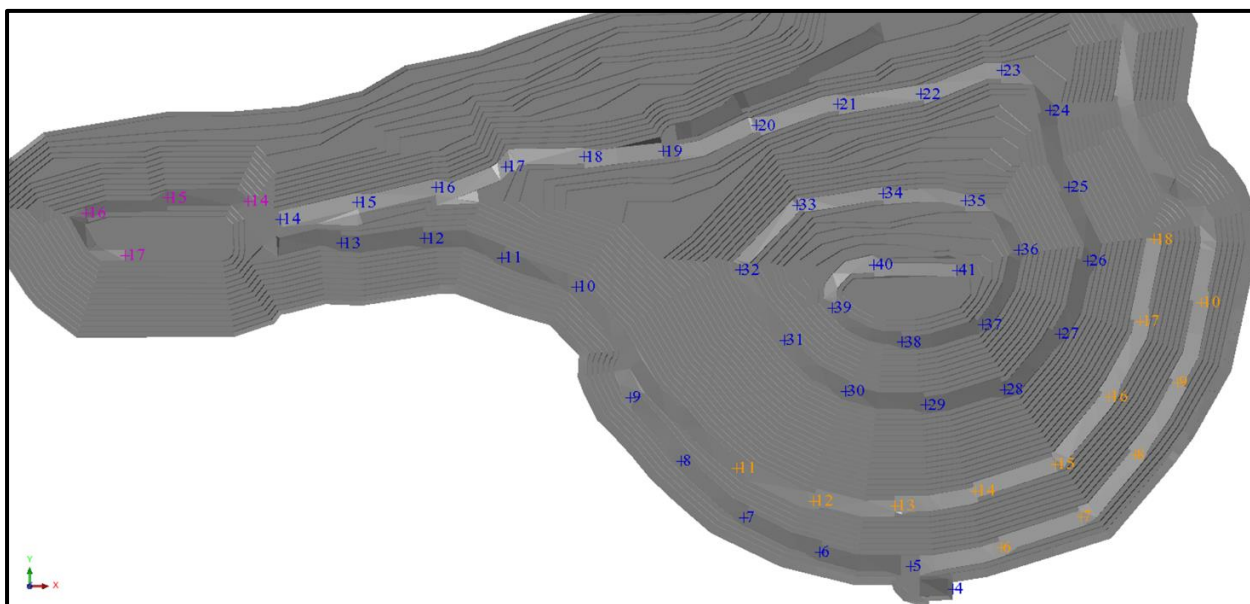


Figure 16-32 Area 4 Haul Reference Points

The ramp reference points with the bench number are shown in Figure 16-33 for Area 2B. All material is hauled from the east and west pits via the applicable ramp. The ramp marked in blue for benches 1 to 5 is common to both pits.

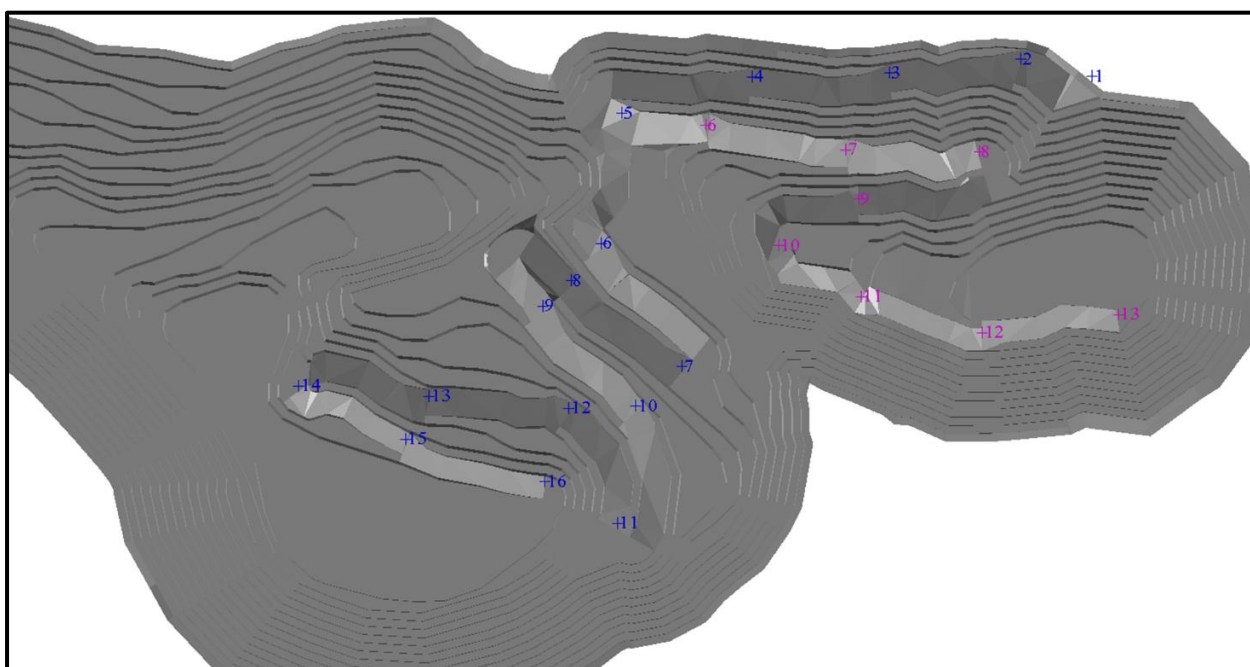


Figure 16-33 Area 2B Haul Reference Points

The coordinates for each of these ramp reference points were imported into an Xpac parameter database for each pit.

In-pit haul distances, both on bench and on ramp were calculated in the Xpac database with reference to these points. The on bench distances were calculated for each block as the sum of the Northing and Easting distance to the applicable reference point. The on ramp distance is the vertical height from the reference point to the ramp exit multiplied by the ramp gradient of 1:10.

These distances, combined with the surface haul distances for the routes shown in Figure 16-3, contribute to the calculation of the truck cycle times that prescribe the total haulage capacity required.

16.13.7.2 Drill and Blast

The number of drill holes, drill metres and explosives required were calculated per block based on the block area and the proportion of each type of material in a block with reference to the parameter database.

Pre-split drill metres and explosives were derived by flagging perimeter blocks on each level and using an assumed 20m face length per block, together with the parameter database values.

The Drill and Blast parameter database is shown in Table 16-23.

Table 16-23 Drill and Blast Parameters

Drill and Blast parameters	Ore	Waste	PreSplit
Burden	4.60	4.60	0.00
Spacing	5.30	5.30	1.40
Bench height	10.00	10.00	10.00
Sub Drill	0.50	0.50	0.00
Powder factor (kg/tonne) PreSplit (kg/m)	0.12	0.12	0.56

16.14 Cost Parameters

The above contents of Section 16 served as inputs to the financial section and is deemed to comply to PEA requirements.

17 RECOVERY METHODS

This study compiled a flowsheet based on best data from testwork at the time of writing for a 5,918 t/d primary processing facility. The final combined rare earth oxide concentrate will be shipped from the port of Walvis Bay, Namibia to rare earth separation processing facilities abroad.

The design of the Lofdal plant is based on the following philosophy:

- A process flowsheet that is simple, but will also be reasonably adaptable to change
- A design that is sensitive to capital and operational costs
- Selection of equipment that is proven
- Ease of maintenance
- Water management and preservation onsite.

17.1 Process Design Basis

The specific area codes as defined by the project work breakdown structure (WBS) are given in Table 17-1. The process block flow diagram is shown in Figure 17-1, with all functional areas included.

Table 17-1 Area Codes According to Project WBS

Area Code	Area Description
B01	Primary and Secondary Crushing
C01	Ball Milling
C02	Magnetic Separation
C03	Flotation
C04	Float Concentrate Thickener
C05	Tails
D01	Acid Mixing
D02	Acid Bake
D03	Water Leach
D04	Impurity Removal
D05	Uranium IX + Precipitation
D06	REE Precipitation
D07	Re-Leach
D08	Silicon Removal
D09	Th Solvent Extraction
D10	Zinc Removal
D11	REE Oxalate Precipitation
D12	Calcination
D13	Final Neutralization
E01	Reagents
F01	Utilities

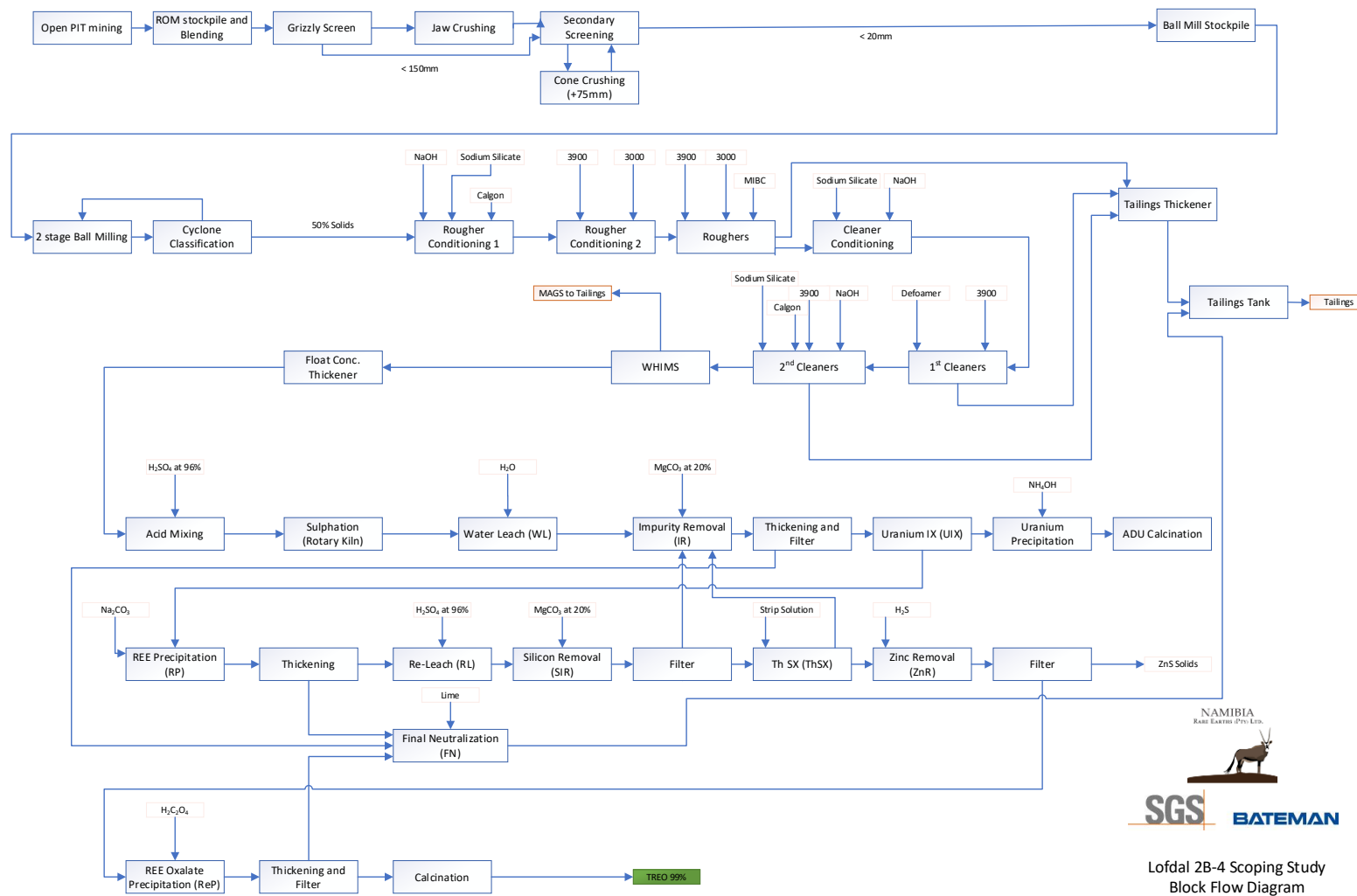


Figure 17-1 Block Flow Diagram Processing Plant

17.2 Process Design Criteria, Summary

Process design criteria (PDC) is compiled to provide guidance in terms of critical process design input, outputs and requirements associated with the global control and operational philosophy. Table 17-2 below presents the major design parameters used in the development of the Mass Balance for process design of a range of operating points. The source of parameters and conditions adopted in the PDC includes:

- (i) Client – Namibia Critical Metals
- (ii) Testwork results performed by Mintek, SGS Lakefield, Geolabs SA, Rados International, Light Deep Earth Laboratories (LDE), IMS Engineering (Steinert), Tomra.
- (iii) Assumptions (by SGS Bateman) based on previous similar projects
- (iv) Recommendations from suppliers (vendors)

Table 17-2 Major Design Parameters

AREA – General

Description	Units	Value	Remarks
Feed Tonnage	tpa	2160000	
Solids Density	SG	2.78	
Feed % Solids	%	98	
Feed Grade	%	0.187	
Plant Availability	%	85	

AREA – Crushing and screening

Description	Units	Value	Remarks
Primary Crushing			
Crushing Availability	%	70	
Crusher	Type	Jaw	
Crusher Feed Method	Type	Grizzly	
Operation	Type	Open Circuit	
Stockpile Reclaim	Type	3 x Apron Feeder	
Product Size P ₁₀₀	mm	100	
Secondary Crushing			
Crusher	Type	Cone	
Feed method	Type	Vibrating Feeder	
Operation	Type	Closed Circuit	
Screen Aperture Size	mm	20	
Product Size P ₁₀₀	mm	20	

AREA– Grinding

Description	Units	Value	Remarks
Ball milling			
Duty		2 stage Ball Milling	

Description	Units	Value	Remarks
Operating mode	Type	closed circuit with scalping and dewatering cyclones	
Grinding Mode	Type	Wet	
Availability	%	85.00	
Mill circuit feed particle size, F ₁₀₀	mm	20.00	
Mill circuit feed particle size, F ₈₀	mm	12.00	
Closed circuit recirculation load (@ mill feed)	%	250.00	
Milling circuit product, P ₈₀	µm	43	
Bond ball mill Work-Index	kWh/t	16.70	
Closed Circuit Classification			
Classifier Type	Type	Flat Bottom Cyclones	
Final Cyclone Overflow P ₈₀	µm	43	
Final Cyclone Overflow % Solids	% solids	50.00	

AREA– Flotation

Description	Units	Value	Remarks
Flotation			
Flotation Conditioning	Type	High Intensity	1,800 rpm
Flotation Conditioning % Solids	%	50	
Flotation Feed Density	SG	1.47	
Overall Float Mass Pull	%	2.9	
Overall Float TREO Recovery	%	64-67	
Collector addition rate	g/t Feed	1690	
Depressant addition rate	g/t Feed	385	
Frother addition rate	g/t Feed	23	
Float Concentrate Thickening			
Thickener Underflow Density	% solids	20.00	
Thickener type	Type	Conventional	
Flocculant addition - current on normal ore	g/t	114.00	
Rise Rate (design)	m/day	1861	

AREA– Magnetic Separation (Iron Removal)

Description	Units	Value	Remarks
Adsorption			
Feed % Solids	%	15.0	
Mass Pull to MAGS	%	1.60	
TREO Recovery to MAGS	%	2.40	Inclusive in Overall Flotation Stage Recovery

AREA– Water Leach and Acid Bake

Description	Units	Value	Remarks
Water Leach			
H ₂ SO ₄ Dosage	kg/t conc.	1250	
Water Leach Solid Density	% solids	20.00	
Number of Tanks	#	3.00	
Total Residence Time	hours	1.00	
% Recovery TREO to Solution	%	96.20	

AREA– Impurity Removal

Description	Units	Value	Remarks
Precipitant		MgCO ₃	
MgCO ₃ Dosage	kg/t conc.	475	
MgCO ₃ Reagent Strength	% (w/w)	20	

AREA– U IX and Precipitation

Description	Units	Value	Remarks
Uranium Ion Exchange			
U IX % Recovery	%	99.9	
U Tenor in Eluate	g/l	5	
Eluate Acid Concentration	g/l	100	
Uranium Precipitation			
Precipitant		NH ₄ OH	
Precipitant Dosage		pH Controlled	
pH		7.3	

AREA– REE Precipitation

Description	Units	Value	Remarks
REE Precipitation			
Precipitant		Na ₂ CO ₃	
Precipitant Dosage	kg/t conc.	110	
Number of Tanks	#	1.0	
Total Residence Time (Required)	minutes	60.0	
Re-seed %	%	350	
Target pH	pH	6.75	
TREO Recovery to solids	%	99.5	

AREA– Re-leach

Description	Units	Value	Remarks
Re-leach			
Pure H ₂ SO ₄ Dosage	kg/t conc.	74.8	
Feed to Re-Leach % Solids	%	50	

Description	Units	Value	Remarks
TREO % Recovery to solution	%	100	

AREA– Silicon Removal

Description	Units	Value	Remarks
Silicon Removal			
Precipitant		MgCO ₃	
MgCO ₃ Dosage	kg/t conc.	23.75	
MgCO ₃ Reagent Strength	%	20	
Number of Tanks	#	1	
Total Residence Time (Required)	minutes	30	

AREA– Th Solvent Extraction

Description	Units	Value	Remarks
Th SX			
A:O Advance Ratio in Extraction	Ratio	5	
A:O Advance Ratio in Scrub	Ratio	0.5	
A:O Advance Ratio in Strip	Ratio	1.25	
Organic % Extractant	%	1	
Organic % Modifier	%	2.5	
Organic % Diluent	%	96.5	

AREA– REE Oxalate Precipitation

Description	Units	Value	Remarks
REE Oxalate Precipitation			
Precipitant		H ₂ C ₂ O ₄	
H ₂ C ₂ O ₄ Dosage	kg/t conc.	43.54	
H ₂ C ₂ O ₄ Reagent Strength	%	100	
Number of Precipitation Tanks	#	1	
Total Residence Time (Required)	minutes	30	
Re-seed %	%	350	
TREO % Recovery to solids	%	100	

AREA– Final Neutralization

Description	Units	Value	Remarks
Final Neutralization			
Ca (OH) ₂ Dosage	kg/t feed	23.3	
Ca (OH) ₂ Slurry Solids Concentration	%	15	
Number of Tanks	#	1	
Total Residence Time	minutes	30	

17.3 Plant Description

17.3.1 Ore Reception and Crushing

ROM material from the mine will be fed into the jaw crusher feed bin at the process plant feed battery limit. An apron feeder will feed a vibrating grizzly screen to remove sub 150 mm material from the feed to the primary Jaw crusher. The Jaw crusher will operate in open circuit producing a $P_{100} = 150$ mm.

Jaw crusher product material will feed to a single deck screen where material that is oversize for the mill circuit can be crushed in close circuit with the secondary cone crusher. This is to ensure a product $P_{100} = 20$ mm from the crushing circuit. The secondary crusher classification screen will screen at the required mill feed size. Any undersize (< 20 mm) from the screen will report directly to the mill circuit as a product stream.

17.3.2 Ball Milling

The ball mill circuit will consist of two mills operating in series. Two mills will be required in order to produce the fine product sizing required for flotation, $P_{80} = 43$ microns. The first of the ball mills will operate in closed circuit with dual cyclone banks, feeding the second ball mill circuit with a 50% solids feed. The second mill will also operate in closed circuit with a single classification cyclone bank, producing the overall circuit product as cyclone overflow at 50% solids. The cyclone overflow will feed forward to flotation conditioning directly.

17.3.3 Flotation

Milled feed is transferred to high intensity conditioning at 50% solids ahead of flotation. The flotation circuit will operate as a single train with a rougher, cleaner and re-cleaner float banks. The re-cleaner concentrate will form the final product feeding the downstream hydrometallurgical sections via a magnetic separation circuit for iron removal. The rougher and cleaner tails will report to tailings and form the basis of the final circuit tails. The flotation sections will be configured as follows:

- Roughers: 6 x 70m³ tank cells
- Cleaners: 4 x 20m³ tank cells
- Re-cleaners: 3 x 20m³ tank cells

A float tailings thickener will be used to de-water the final float tailings to the TSF, which is fed from the final tailings tank. A float concentrate thickener will be fed via the re-cleaner concentrate, producing a thickener underflow product feeding forward to the hydrometallurgical processes.

17.3.4 Magnetic Separation

The magnetic separation circuit, operated as a wet high intensity magnetic separation (WHIMS), functions to reduce the total Fe in the feed to the hydrometallurgical operations. The circuit will operate with a rougher magnetic separator and cleaner magnetic separator.

Rougher mags will report to a cleaning stage from which the final mags and circuit tails will be produced. The non-mags from the rougher and cleaner stages will be combined as feed to the hydrometallurgical sections.

17.3.5 Acid Mixing and Acid Bake

Flotation concentrate, post iron removal by magnetic separation, will report to pre-filtration prior to acid mixing. Filter cake solids will be dried and then mixed with concentrated sulphuric acid prior to feeding the acid bake kiln.

The kiln will operate at 300°C. With a one-hour heating zone and three hours at target reaction temperature. The kiln will be spec'ed in terms of MOC for sulphuric acid at 300°C. The kiln promotes sulphation of the feed material. Final kiln product is then fed forward to a water leach circuit.

17.3.6 Water Leach

Solids from the Acid Bake kiln are re-slurried with water in a three tank leach operation. This is done to selectively dissolve REE from the calcined kiln product. Leach efficiencies are expected to be high, supported by testwork.

17.3.7 Impurity Removal

The water leach slurry is then mixed with magnesium carbonate in an impurity removal stage. The impurity removal stage selectively precipitates most of the dissolved thorium and a vast majority or all of the dissolved iron. Rare earth losses at this stage are expected to be low. The pulp from impurity removal will be thickened and filtered. Solids will report to final neutralization for discard to tailings while the thickener overflow and filter filtrate report forward in the circuit to uranium IX.

17.3.8 Uranium Ion Exchange and Precipitation

Uranium is selectively adsorbed from the impurity removal product solution via a fixed bed ion exchange circuit. The resin used will be strong base type to facilitate uranium adsorption. Uranium recovery to the resin can be considered quantitative with negligible REE losses. Barren liquor proceeds down the circuit, containing the REE component. The resin is eluted with 100 g/l H_2SO_4 , producing an eluate of approximately 5 g/l U_3O_8 equivalent.

Eluate from ion exchange is precipitated using NH_4OH solution under pH control to produce an ADU precipitant product. This can either be dried and shipped as final uranium product or calcined at 750°C to produce a yellow cake product (U_3O_8).

17.3.9 REE Precipitation

Uranium IX barren liquor is contacted with sodium carbonate to produce an impure intermediate rare earth carbonate precipitate. The key precipitate impurities will include thorium, silicon, iron and possibly zinc.

The product from this stage is thickened, with thickener overflow reporting the final neutralization and tailings. Thickener underflow will proceed forward to a re-leach process.

17.3.10 Re-leach

Intermediate rare earth carbonate precipitate is re-leached using H_2SO_4 at $\text{pH} = 1$. Recovery of REE to solution can be considered 100%. Silicon, zinc, and thorium will also co-leach with the REEs. Thus, the product is treated in a few impurity removal stages.

17.3.11 Silicon Removal

Silicon is subsequently removed from re-leach solution via precipitation with Magnesium Carbonate. Silicon removal efficiencies can be considered quantitative. The resulting product is filtered with the filtrate solution proceeding to thorium SX.

17.3.12 Thorium Solvent Extraction

Thorium is selectively extracted via solvent extraction using 1% Primene JMT (Primary Amine), 2.5% Isodecanol (Modifier) and 96.5% Aromatic 150ND (Diluent). This is done via two extraction stages, resulting in 99% thorium extraction to organic. Raffinate containing REEs reports forward to Oxalate precipitation.

Loaded organic is scrubbed with sulphuric acid in a two stage scrubbing process. Spent scrub liquor will be recycled back to extraction feed. Post scrub, the organic will be stripped using HCl in a two stage stripping section. Strip effluent will report to final neutralization for discard to tailings. Stripped organic is recycled back to extraction as fresh feed.

17.3.13 REE Oxalate Precipitation and Calcination

Raffinate from Th SX extraction reports to Oxalate precipitation for final recovery of REE. Precipitation efficiency at this stage is quantitative. Precipitation is facilitated by Oxalic acid ($\text{H}_2\text{C}_2\text{O}_4$). Post precipitation the precipitant is thickened and filtered, producing a solid feed to calcination where the oxalates will be calcined. This produces a mixed rare earth oxide final product.

17.4 Process Plant OpEx

17.4.1 Operating Cost Summary

The following assumptions have been made in estimating the operating costs:

- A mass balance was completed for the circuit to anticipate mass flow through the process. Reagent consumption and required supply has been derived from this balance.
- Preliminary equipment sizing was evaluated from received vendor budget quotes such as to determine motor sizing. Motor sizing was calculated where supplier estimates were not available.
- Budget pricing from the market along with database pricing for major equipment units was used to estimate a capex for the process.
- Reagent pricing was obtained from Protea Chemicals and Florrea. Reagent consumptions were determined via mass balance and based on the data found in the Process Design Criteria (Document No. MS7573-P670-001).
- Acid pricing supplied by Dundee Precious Metals.

- Fuel costs were included for the operating of mobile equipment on site.
- Maintenance costs were estimated as a factor of the mechanical equipment installed costs. This was estimated from the mechanical equipment capex estimation as per SGS Bateman standards and referenced to the Mintek Handbook (March 2002).
- Power estimations from the basic equipment sizing were costed using the supplied power tariff from Independent Power Producing Consortium for solar power.
- Namibian Labour rates provided by Gecko Namibia.
- Costs were determined for typical operating duty at design throughput.
- No contingencies have been included in the operating cost estimates unless otherwise stated.
- Costs are expressed in United States Dollar (USD).
- All costs are in 2021 terms, unless otherwise specified.

The operating cost estimate is summarised in Table 17-3 and shown in Figure 17-2. The estimate includes reagents and consumables, fuel, labour, maintenance materials and power consumption.

Table 17-3 Operating Cost Estimate

Cost Component	Total (\$/a)	Total %
Labour	\$ 5,772,042	8.3
Front End	\$ 734,083	
Hydrometallurgical	\$ 1,489,250	
Maintenance	\$ 3,093,313	4.5
Power	\$ 5,236,333	7.6
Front End	\$ 4,938,962	
Hydrometallurgical	\$ 297,370	
Fuel (Mobile Equipment)	\$ 653,610	0.9
Reagents and Consumables	\$ 54,374,100	78.7
Front End	\$ 26,046,331	
Hydrometallurgical	\$ 28,327,769	
TOTAL	\$69,129,397	100

Total	\$69,129,397
Feed (t/a)	2,160,000
\$/t of ROM Feed	32.00
\$/lb Product	12.6
\$/kg Product	27.9

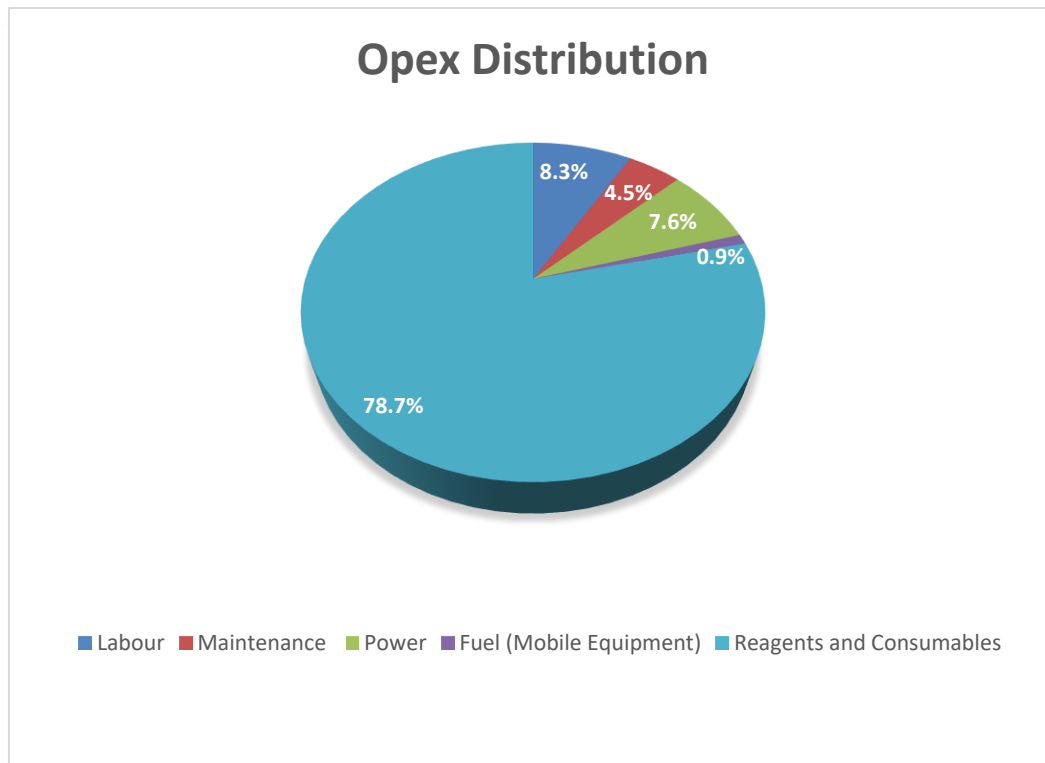


Figure 17-2 Opex Distribution

17.4.2 Basis of Estimate

17.4.2.1 Scope

The operating costs can be categorised as fixed or variable costs.

Fixed costs include:

- Manpower (Labour for plant operation and maintenance only)
- Maintenance and operating supplies

Variable costs include:

- Power
- Reagents, fuel and consumables
- Waste Handling

17.4.2.2 Accuracy

The accuracy for the operating cost estimate is as for the capital cost estimate. The methodology used in preparing the Opex costs in this report was based on the scope, pricing and information available at the time but should be revised as the industry fluctuates, especially in terms of reagent pricing. During commissioning, start-up and ramp-up the unit costs will vary in comparison to when the plant is operating at full capacity.

17.4.2.3 Exchange Rate

Operating costs are base dated July 2021. The estimates are presented in US dollars and the exchange rate used is ZAR 14.6 / USD.

17.4.2.4 Exclusions

The following are excluded from this estimate:

- Mining Costs (Outside of project scope)
- Tailings Handling Costs (Outside of project scope)
- General, Medical and Administration Costs, other than plant and technical/engineering services
- Security costs
- Duties and taxes on exports of products
- Marketing costs
- Depreciation and replacement capital
- Insurance
- In-country corporation tax
- First fill reagents costs (included in capital estimate for Owners Costs)
- No provision for annual increases in salary, services and supplies growth has been allowed
- Product dispatch including handling and cost of transport for products from site to destination. (Outside of project scope)
- Contingency

17.4.3 Fixed Costs

17.4.3.1 Labour

Labour was cost and detailed on the project using supplied rates from Morne du Toit, Gecko CFO (19 Aug 2021).

The following assumptions have applied to the labour cost estimation:

- 30% overheads for total cost to company added

- Rate is based on an average of 45 hours per week converted from annual package
- The type and level of personnel required has been based on estimates for skilled, unskilled, technical, engineering and senior engineering personnel but would require further review by the client
- The manning breakdown is based on an 8 hourly shift rotation and typical SA requirements for process plants inclusive of SHEQ contingents to comply with SA regulations. Client to advise on daily shift applicability to Namibia and adjust accordingly
- The monthly all-inclusive cost to company of personnel would need to be revised based on input from Client
- Includes production and SHEQ personnel only. Assumes Medical, Admin, Finance and Senior HR personnel are included in overheads

The fuel requirements are seen in Table 17-4 below.

Table 17-4 Total Labour Requirements

	Morning	Afternoon	Night	4 th Shift					
Metallurgical Operation Total (All Shifts)	12	12	12	12	48			\$53,194.44	\$638,333.33
Adjustment for Total cost to company (Note 1)									\$734,083.33
Hydrometallurgical Operation Total (All Shifts)	12	12	12	12	48			\$50,416.67	\$605,000.00
Adjustment for Total cost to company (Note 1)									\$695,750.00
Reagents and Utilities Total (All Shifts)	14	14	14	14	56			\$57,500.00	\$690,000.00
Adjustment for Total cost to company (Note 1)									\$793,500.00
Process Plant Technical Total (All Shifts)	52	16	13	10	91			\$257,152.78	\$3,085,833.33
Adjustment for Total cost to company (Note 1)									\$3,548,708.33
Total	90	54	51	48	243			\$418,263.89	\$5,019,166.67
Adjustment for Total cost to company (Note 1)									\$5,772,041.67
Plant Throughput (t/annum)									2,160,000
Operating Cost (\$/t ROM)									\$2.67

17.4.3.2 Maintenance

In all cases, the cost of maintenance supplies is calculated as a factor of the mechanical equipment supply costs excluding piping and valves, electrical and instrumentation based on previous studies for typical uranium hydrometallurgical plants but excludes the maintenance associated with the acid plant. Generally, maintenance materials are considered to be (7 - 13 %) of the mechanical equipment supply cost. An estimate of 10 % has been applied. The maintenance labour component has been allowed for in the annual labour estimate.

17.4.4 Variable Costs

17.4.4.1 Power

The power cost estimate is based on the supplied power tariff structure from infrastructure at a unit price of 62 USD/MW.hr. This cost is assumed to include: -

- Service and Administration charges
- Network Capacity charges
- Network Demand charges
- Ancillary service charges
- Energy charges
- Any other charges that are routinely levied to the client.

Operating power cost for continuous standby power generation (including monthly test-runs) are not included in this cost.

For the proposed flowsheet, basic equipment sizing was completed to determine Absorbed Power (kW) for each unit operation in the process. Where installed power was used a power factor of 0.85 was largely applied.

17.4.4.2 Reagents and Consumables

Reagent consumptions were determined from testwork and supplied as per the Process Mass Balance. The completed mass balance thus allowed for a reagent consumption calculation on a t/h basis.

Reagent prices were obtained from local suppliers as best available. With reagent supply pricing from Protea Chemicals Namibia and Florrea used in the Opex estimate. Acid costs supplied by preliminary MOU from Dundee Precious Metals.

Only major consumables considered. Minor consumables assumed to be 5% of overall annual cost. Minor consumables to include drums, packaging, minor chemicals, office consumables, lab consumables

Crusher liner wear excluded and allowed for in maintenance costing.

All reagent pricing is in July 2021 terms.

Table 17-5 Reagent Costing

Reagents and Consumables	Addition Rate metric t/h	Annual Rate metric t/a	Unit Cost \$/t	Supplier	Annual Cost \$	% of Cost	Total Cost \$
Sodium Silicate (N Type)	0.094	702	693	Florrea	486,233	0.9%	486,233
Calgon	0.017	130	1443	Florrea	186,966	0.3%	186,966
Florrea 3900	0.464	3,456	4543	Florrea	15,699,364	28.9%	15,699,364
Florrea 3000	0.026	194	8043	Florrea	1,563,489	2.9%	1,563,489
MIBC/Pine Oil	0.007	49	2543	Florrea	123,572	0.2%	123,572
Florrea 7411	0.004	27	7543	Florrea	203,651	0.4%	203,651
NaOH	0.029	216	723	Protea	156,090	0.3%	156,090
H ₂ SO ₄	11.154	83,055	118	Protea	9,800,536	18.0%	9,800,536
MgCO ₃	4.191	31,208	266	Protea	8,309,626	15.3%	8,309,626
Na ₂ CO ₃	0.924	6,883	402	Protea	2,769,182	5.1%	2,769,182
H ₂ C ₂ O ₄	0.418	3,115	1193	Protea	3,715,917	6.8%	3,715,917
Magnafloc 10	0.042	312	3222	Protea	1,004,055	1.8%	1,004,055
Ca(OH) ₂	0.000	-	172	Protea	0	0.0%	0
NH ₄ OH	0.048	359	675	Protea	242,133	0.4%	242,133
HCl	0.049	365	585	Protea	213,806	0.4%	213,806
Primene JMT	0.0000035	0.026	7902	Protea	205	0.0003776%	205
Isodecanol	0.0000087	0.065	1452	Protea	94	0.00017%	94
Aromatic 150ND	0.0003367	2.507	115	Protea	287	0.00053%	287
Ball Mill Grinding Media	0.460	3,425.2	1130	GMSA	3,870,431	7.1%	3,870,431
Raw Water	130.8	974,024	0.26	NamWater	253,246	0.5%	253,246
Coal	3.27	24,358	113	Market SA	2,752,479	5.1%	2,752,479
Natural Gas - Acid Bake + REE Oxalate Calciner	7.06	52,569	15.20		798,975	1.5%	798,975
Other Consumables (Drums, packaging, stationary, laboratory, minor water treatment chemicals etc)						4.1%	2,223,760
TOTAL REAGENTS COST							54,374,100

17.4.4.3 Fuel

Diesel requirements and costing has been included for the running of reagent handling vehicles. Fuel costs were included for the operating of mobile equipment on site. A diesel price of 0.91 USD/L was used (Price on 09/08/2021 www.globalpetrolprices.com/Namibia).

Equipment used for the purposes of maintenance are not included as this is covered in the total maintenance costing.

Transportation within plant battery limits not considered. Any external travel requirements beyond plant boundaries not considered.

17.5 Process Plant Capex

17.5.1 Estimating Approach

The Estimating Approach undertaken was to identify and quantify the majority of mechanical equipment items and price these using email budget quotations, in-house database prices and capacity factoring equipment cost from known base cost. The balance of the estimate was factored from mechanical equipment costs or estimated from database records of recent projects of a similar nature.

An allowance has been made for in-country and productivity factor, which takes into consideration locality, weather condition and skills availability in Namibia. This is a provisional allowance which will be firmed up during the next phase of the project with formal quotations from selected contractors.

All the percentage factors used on the capital estimate have been based on SGS Bateman experience and norms on similar projects.

It has been assumed that local construction contractors will be used during construction phase, thus, low factor has been used for contractors' indirect cost.

In addition to the above, it was requested that various changes be made to increase the capacity of certain areas of the facility (or pieces of M&E). These increase throughputs were calculated by the process department and formed the basis for capacity factoring, described in the paragraph below. The original base prices were used, and no escalation was considered from the previous cost estimate submitted in August 2021.

17.5.2 Capacity Factor Estimate

The cost-to-capacity method is an order-of-magnitude cost estimation tool that uses historical costs and capacity in order to develop current cost estimates for an entire facility or a particular piece of machinery or equipment. The cost-to-capacity concept was originally applied in 1947 by Roger Williams Jr. to develop equipment cost estimates; later, in 1950, C.H. Chilton expanded the concept's application to estimate total chemical plant costs.

Cost estimates developed by the cost-to-capacity method can be classified as Class 5 or Class 4 estimates as indicated by the Association for the Advancement of Cost Engineering ("AACE") International. Per AACE International, there are five classes of estimates from 1 through 5. Both Class 5 and Class 4 estimates are

preliminary in nature and are based on limited information, while a Class 1 estimate is highly detailed and based on a fully defined project scope.

The fundamental concept behind the cost-to-capacity method is that the costs of facilities (or pieces of M&E) of similar technology but with different sizes vary nonlinearly. More specifically, cost is a function of size raised to an exponent or scale factor. The governing equation is as follows:

The relationship between cost and capacity is given by the following equation:

$$\frac{\text{Cost 2}}{\text{Cost 1}} = \left(\frac{\text{Capacity 2}}{\text{Capacity 1}} \right)^R$$

The raised scale factor in Equation 1 above accounts for the nonlinear relationship and introduces the concept of economies of scale where, as a facility (or piece of M&E) becomes larger, the incremental cost is reduced for each additional unit of capacity. However, not all facilities (or pieces of M&E) actually experience economies of scale related to costs. A scale factor of less than 1 indicates that economies of scale exist and the incremental cost of the next added unit of capacity will be cheaper than the previous unit of capacity. When the scale factor is greater than 1, economies of scale do not exist; rather, diseconomies of scale exist and the incremental cost becomes more expensive for every added unit of capacity. A scale factor of exactly 1 indicates that a linear relationship exists and there is no change in the incremental cost per unit of added capacity. A scale factor of 1 also indicates that it is just as economically feasible to build two small facilities as one large facility with the same capacity.

17.5.3 Estimate Accuracy

The estimate accuracy for this **Order of Magnitude (OOM) Estimate** is deemed to be within the range of -20% to +35% at a probability of not more than 75% after sufficient contingency is applied.

17.5.4 Estimate Base Date and Escalation

All costs presented are instantaneous at a base date of August 2021 (unchanged from the previous cost estimate submitted in August 2021) and will be subject to contract price adjustment from this date.

17.5.5 Foreign Currency

The estimate is presented in USD currency. The capital estimate was compiled in ZAR. Prices obtained in other currencies have been converted to ZAR using the following rates of exchange (ROE):

Table 17-6 Currency Exchange Rates

Currency Code	Currency	Equivalent in ZAR
ROE Base Date August 1, 2021		
US\$	United State Dollar	14.61
Euro	Euro	17.36

No foreign currency or rate of exchange variations allowance has been made in the estimate and will be for client's scope.

17.5.6 Price Basis

The cost estimate is based on the scope as defined within this document and by the engineering documentation supporting it.

The scope on which the estimate is based on is as follows:

- Mechanical Equipment List Revision D (with changes to plant capacities);

Prices for direct field costs are based on a variety of sources including:

- Budget email quotations for the following mechanical equipment (Prices de-escalated to 2021 pricing for better comparison to previously submitted cost estimate):
 - Primary and Secondary Ball Mill;
 - Flotation Cell Package;
 - Cyclone Package;
- Equipment capacity factored estimate;
- In house database estimate;
- Factored estimate;
- Estimate allowances.

17.5.7 Contracting Strategy

The estimate has been broadly based on the following Contracting Strategy:

- The project will be implemented on an EPCM Re-imbursable contract basis;
- Use will be made, wherever possible, of contractors familiar with the country's location and environments.

17.5.8 Direct Field Cost

17.5.8.1 Introduction

The direct field costs include permanent materials and equipment, freight to site, construction labour and equipment including Preliminary and General costs associated with contractor's supervision, overheads and profits, temporary construction facilities, construction mobile equipment, accommodation of construction labour, and contractor mobilisation and demobilisation.

17.5.8.2 Process Mechanical Equipment

Major mechanical equipment was initially priced using email budget quotations from the market and the balance was capacity factored from a known capacity base to a new plant capacity, this was from similar works executed by SGS Bateman in the past. The previous cost estimate was used as the basis for capacity factoring of the revised scope and equipment as detailed in the Mechanical Equipment List Revision D. The

SMPP P&G's has been included in the price as a percentage of the total install price as per SGS Bateman norms. P&G's cost is the contractors indirect field cost.

The following packages were priced using email budgetary quotations from the market:

- Primary and Secondary Ball Mill;
- Flotation Cell Package;
- Cyclone Package.

All other mechanical equipment cost has been capacity factored from base equipment costs used for the previous cost estimate for the Lofdal REE PEA Project carried out in August 2021.

Equipment installation cost has been calculated by using a factored percentage of equipment supply cost. Contractors indirect field cost has been calculated by using a factored percentage of the erection cost.

17.5.8.3 Bulk Earthworks

The bulk earthworks are priced using preliminary rough approximate layouts. It has been assumed that no piling or blasting is required. An assumption has been made that 50% of the area requires terracing including the ROM tip area. Fencing has been excluded from the estimate. The scope is limited to within the plant area and only 5 km of in-plant road has been allowed for assuming the road is 6m wide, no access road has been allowed for. There has been an allowance made for 2.5 km paved drains including culverts. There is no allowance made for any bridges or river diversions in the estimate. The rates are approximate based on similar projects in Africa.

17.5.8.4 Buildings

All plant buildings were based on the previous PEA study. Data base rates were used from similar projects in Africa. No escalation was applied to the previously submitted cost estimate.

17.5.8.5 Civils

All civil cost has been factored from the priced mechanical equipment list, using SGS Bateman's norms. Preliminary and general cost is included in the capital estimate as a factor of the total install price as per SGS Bateman norms. P&G's cost is the contractors indirect field cost. No geotechnical data available. It has been assumed that ground conditions are suitable for standard concrete equipment support structures, with no piling or special foundations requirements.

The estimate factors are based on the following assumptions:

- An entirely "Greenfield" site;
- A flat site requiring no removal of vegetation and overburden other than 150 mm topsoil;
- No dolomite or sink hole risks were taken into account;
- No soil stabilization;
- A site which is a swamp free area, thus no groundwater will penetrate foundation levels;

- Simple pad foundations (150 kPa/m²);
- No piling or blasting of rock.

17.5.8.6 Structural Steelwork

All supply costs have been factored from the priced mechanical equipment list. The erection cost has been calculated by using a factored percentage of the structural steel supply cost. Preliminary and general cost is included in the capital estimate as a factor of that total install price as per SGS Bateman norms. P&G's cost is the contractors indirect field cost.

17.5.8.7 Plateworks and Tanks

All supply costs were priced using in-house data base prices. The erection cost has been calculated by using a factored percentage of the supply cost. Preliminary and general cost is included in the capital estimate as a factor of the total install price as per SGS Bateman norms. P&G's cost is the contractors indirect field cost.

17.5.8.8 Piping and Valves

All supply costs for in-plant piping and valves have been factored from the priced mechanical equipment list. The erection cost has been calculated by using a factored percentage of the piping and valves supply cost. Preliminary and general cost is included in the capital estimate as a factor of the total install price as per SGS Bateman norms. P&G's cost is the contractors indirect field cost.

17.5.8.9 Electricals

All supply costs for inter-plant electrical equipment complete with bulk supplies (cables, etc.) have been factored from the priced mechanical equipment list. The erection cost has been calculated by using a factored percentage of the electrical supply cost. Preliminary and general cost has been calculated by using a factored percentage of the erection cost as per SGS Bateman norms. P&G's cost is the contractors indirect field cost. It is assumed that required power will be available at plant battery limit, thus no substation required to adjust from high voltage lines has been allowed for in the capital cost estimate.

17.5.8.10 Control and Instrumentation

All supply costs for instruments and bulk supplies have been factored from the priced mechanical equipment list. The erection cost has been calculated by using a factored percentage of the C&I supply cost. Preliminary and general cost has been calculated by using a factored percentage of the erection cost as per SGS Bateman norms. P&G's cost is the contractors indirect field cost.

17.5.8.11 Freight, Duties and Clearing

A percentage allowance has been made on the supply "ex-works" cost. Logistics costs need to be fully investigated and firmed up based on market prices during the next phase of the project.

Import duties and any import handling fees are specifically excluded.

17.5.8.12 Commissioning Spares and Start-up Costs

Provision has been made on a factorised basis for the following Commissioning and Start-up Costs:

- Supply of commissioning spares only;
- Initial first fills of oil and lubricants only;
- Chemical reagents are specifically excluded;
- Vendor assistance on site during construction and commissioning.

17.5.9 Indirect Field Cost

17.5.9.1 Introduction

Indirect field costs are costs related to the project management directly associated with the implementation of the project by the ECPM contractor, including design engineering, procurement, project management and construction management up to and including cold commissioning of the plant.

17.5.9.2 Engineering, Procurement and Construction Management Cost (EPCM)

EPCM costs have been estimated as a factor of the total Direct Field Costs. These factors usually range from between 15% to 18% as per SGS Bateman norms. The indicative amount allows for cold commissioning, but hot commissioning is specifically excluded.

No allowance has been made for training of local personnel, hot commissioning, performance testing and assisting during the defect liability period.

17.5.9.3 External Consultants

External consultants have been excluded.

17.5.10 Other Costs

17.5.10.1 Bonding

An allowance has been made on the estimate to account for bonding costs.

17.5.10.2 Insurance

Insurance Estimate included in the estimate is an allowance for project related risks which are insurable. It is dependent on project variables and project specific circumstances. It typically includes for the following:

- Contractors All Risk on construction and site activities typical cover;

- Third Party Liability insurance typical cover;
- Medical Evacuation and casavac typical cover;
- Marine Cargo and difference in excess typical cover.

The following risks are not allowed for in the estimate and thus excluded due to the specific requirements the client may have. These should be strongly considered in addition to those listed above:

- Delay in Start-up insurance (DSU);
- Project Specific required professional indemnity;
- Advance Loss of Profits (ALOP).

17.5.10.3 Royalties

Royalties and fees associated with project implementation (e.g. technology supplier royalties) were excluded in the capital cost estimate.

17.5.10.4 Escalation

No project forward escalation allowance has been made on the estimate. The Client should make appropriate provision for escalation from base date to project award and through to completion.

17.5.10.5 Project Contingency

Contingency is defined as a special monetary provision in the capital estimate to cover uncertainties or unforeseeable elements of time/cost within in the scope of the project. Contingency provision excludes: escalations, changes in scope of work, catastrophic events, labour strikes and civil unrest, exchange rate variations, etc.

Since no quantitative risk analysis has been done on the capital cost estimate and considering the engineering/scope definition currently available and the estimating methodologies followed, a nominal contingency allowance of 30% has been included in the estimate.

17.5.10.6 Project Growth Allowance

Experience tells us that there is a gap between what is known and identified during the estimating process and what is expected during the execution or construction phase. The addition of growth is intended to allow for the most likely cost in the estimate. For these reasons an allowance for growth on quantities and prices should be included in the estimate.

A quantity growth allowance is a subjective amount added to the material take-off (MTO) based on the degree of engineering completed and a comparison to historical experience of the expected quantity. Growth allowances are generally applied to estimates where materials take-offs (MTO's) are performed.

Estimates are generally based on budget quotations that rarely take into account firm specifications or commercial terms and conditions. A price growth allowance is an amount added to the price of materials

and equipment to account for the cost difference between the budgetary amount quoted now and the expected firm price quotation obtained prior to execution. Pricing growth allowances are generally in the range of 0% to 15% depending on the quality of the pricing information that supports the estimate.

Growth allowances are therefore included in the total summation of the estimate detail to arrive at “most likely” cost for the project. This “most likely” cost is the base estimate upon which the contingency risk analysis process is applied.

Contingency will be influenced if no growth is allowed or vice versa where it should or should not have been included.

17.5.11 Owner's Cost

This estimate specifically excludes owner's costs. These costs shall fall under the client scope of work. Generally, owner's costs are, but not necessarily limited to, the following:

- Owner's project management team and consultants;
- Owner's contingency for changes in scope or additional work;
- Metallurgical Testwork;
- Pre-development costs (cost of study, etc.);
- Land acquisition;
- Insurances, including builder's risk, material damage, delayed start-up, loss of profit and construction work insurance;
- Business systems;
- Loss of production and efficiency resulting from implementation;
- Owner's start-up and commissioning crew;
- Project taxes, permitting and approvals;
- Development fees and approval costs of statutory authorities;
- Cost of any disruption to normal operations;
- Cost of shutdowns;
- Public Liability Insurance;
- Working capital or any additional stay in business capital (sustainability capital);
- Pre-production costs (operator training);
- Workplace health and safety fees;
- Operational readiness;
- Site survey & soils testing;
- IT Infrastructure;
- Communication systems;
- Waste management;
- EIA;

- Mining Cost;
- Tailings storage facility;
- Mining/operational village;
- Mobile and mining equipment;
- Community relations (Social relations);
- Import duties and withholding taxes.

17.5.12 Estimate Exclusions

The following items are specifically excluded from the scope of this estimate:

- Any costs associated with process design changes necessitated by the outcome of geotechnical investigations, site surveys and metallurgical testwork;
- Escalation from the estimate base date to project execution;
- Foreign currency exchange rates variations from the estimate base date;
- Value Added Tax (VAT) and Goods & Services Tax (GST);
- Import Duties;
- Access roads;
- Mining/operational village;
- Construction camps;
- Mobile and mining equipment;
- EPCM contractor support charges and Owner's costs associated with hot commissioning activities;
- Geotechnical studies;
- Provision of landscaping and nursery services;
- Medical staff and practitioners including emergency vehicles during construction;
- Mining and mine closure/rehabilitation costs;
- Laboratory information management systems (LIMS);
- Higher level management systems (MIS, MES or ERP);
- Any cost for a logistical study or resulting increase in capital cost as a result of such a study;
- Costs of EIA permitting and regulatory compliance requirements, together with any costs associated with process design changes necessitated by the outcome of such investigations;
- Any costs associated with statutory requirements, local permits, licensing, royalties and approvals, social, community or environmental requirements;
- Owner's costs;
- Development fees and approval costs of Statutory Authorities;
- Financing and Marketing costs;

- Business system costs;
- Pre-development and operational readiness costs;
- Operational costs (included in Operating Cost Estimate);
- External auditing costs;
- Force Majeure;
- The cost of any disruption to normal operations;
- Testwork and laboratory costs unless specifically included;
- Facilities for disposal of hazardous products generated by operations.

17.5.13 Capital Estimate Summary

Table 17-7 Currency Estimate Summary

MARG/018 /002	SGS BATEMAN PROJECTS				Prepared :	JB	
	ESTIMATE SUMMARY SHEET				Checked :	-	
	REPORT CURRENCY: US\$				Approved :	Price Fix Panel	
Project :					PROJECT Nº :	M7573	
Lofdal Rare Earth Elements (REE)					BASE DATE :	Aug-21	
Plant Type :					REVISION Nº :	04	
Process Paint					REVISION DATE:	2022-09-06	
Client :					ESTIMATE DATE:	2021-08-06	
SGS Geological Services					PRINT DATE:	2022-09-12 15:26	
CODE	DESCRIPTION	A SUPPLY COST	% TOTAL A	B ERECTION COST	TOTAL COST A+B	% TOTAL DFC	% Mechanical
	Direct Field Costs						
	Bulk Earthworks	-	0.00%	7,220,840	7,220,840	6.68%	18.95%
	Civil Works	-	0.00%	5,444,231	5,444,231	5.03%	14.29%
	Buildings Architectural	-	0.00%	3,090,942	3,090,942	2.86%	8.11%
	Structural Steelwork	5,103,967	8.20%	918,714	6,022,681	5.57%	15.80%
	Mechanical Equipment	34,026,445	54.70%	4,083,173	38,109,618	35.24%	
	Piping & Valves	4,083,173	6.56%	2,245,745	6,328,919	5.85%	16.61%
	Overland Piping	1,640,451	2.64%	966,594	2,607,045	2.41%	6.84%
	Electrical	5,103,967	8.20%	1,020,793	6,124,760	5.66%	16.07%
	Instrumentation	3,402,644	5.47%	680,529	4,083,173	3.78%	10.71%
		-		-			
	SMPP P&G's	-	0.00%	9,860,864	9,860,864	9.12%	25.88%
	Bulk Earthworks & Civils P&G's	-	0.00%	6,332,536	6,332,536	5.86%	16.62%
	E&I P&G's	-	0.00%	4,083,173	4,083,173	3.78%	10.71%
		-		-			
	Transportation of Equipment to site - Import Duties Excluded	5,103,967	8.20%	-	5,103,967	4.72%	13.39%
	Commissioning Spares	1,701,322	2.73%	-	1,701,322	1.57%	4.46%
	First fill of lubricants	340,264	0.55%	-	340,264	0.31%	0.89%
	First fill of Chemical Reagents - Excluded	-	0.00%	-	-	0.00%	0.00%
	Vendor assist during Constr & Comm	1,701,322	2.73%	-	1,701,322	1.57%	4.46%
	TOTAL DIRECT FIELD COSTS	62,207,523	100%	45,948,135	108,155,658	100%	
	Home Office & Indirect Field Costs (Factored)						
	EPCM @ 15% of DFC	16,223,349	26.08%	-	16,223,349	15.00%	
	TOTAL H.O. & INDIRECT FIELD COSTS	-	0.00%	-	16,223,349	15.00%	
	TOTAL NET COST	62,207,523	100.00%	45,948,135	124,379,006	115.00%	
	Other Costs					% of TPC	
	Bonds Guarantees etc	145,109	0.23%	-	145,109	0.09%	
	Insurance	2,191,624	3.52%	-	2,191,624	1.33%	
	Contingency 30%	38,014,722	61.11%	-	38,014,722	23.08%	
	TOTAL OTHER COSTS	40,351,455	64.63%	-	40,351,455	24.50%	
	OVERALL PROJECT COST	102,558,978	290.37%	45,948,135	164,730,461	139.50%	
	TOTAL PROJECT COST (US\$)				164,730,461		

17.6 Safety and Risk Assessment

17.6.1 Occupational Health and Safety

The occupational health and safety (OHS) system is a five-stage proactive system:

- Risk and impact assessments;
- First aid and occupational health service facilities;
- Medical services;
- Training;
- Inspection and action.

These stages are described in more detail in the following sections.

17.6.2 Risk and Impact Assessments

Statement of Objectives

To identify and assess OHS risks through a dynamic, formal, structured and holistic process to facilitate effective risk reduction. As NMI has not yet established formal mining safety controls, the South African Mine Health and Safety Act (29) of 1996 has been used as a reference in drawing up standards and systems for mine health and safety for the Project.

General Requirements

There is a systematic approach and standard procedure on all work sites to identify and assess risks and the impact thereof;

- The applicable site OHS practitioner is alerted to planned process activation and modifications. An assessment team is then established for that specific task. All tasks are assessed, incident preventive measures are initiated before the activity starts and the OHS impacts are discussed with all the applicable employees working on that job;
- All applicable parties are involved in the assessment process;
- All applicable assessments are reviewed annually;
- Assessments include all significant activities, covering the full scope of responsibility and accountability of the operations and include the activities of all sub-contractors;
- Employees are competent to evaluate and identify risks related to a specific task / function;
- Assessments are carried out before operation and before changes / modifications are done;
- Normal, abnormal and potential conditions, throughout the process cycle, are taken into consideration when assessing risks;
- Biannual surveys are carried out by appropriately qualified / accredited personnel and if major modifications / new installations are added; steps are taken to identify and control new arising risks.

These surveys include:

A – Health Risk Analyses

A1 - Chemical stress

The survey identifies hazards related to vapours, gases, mist and dust;

- Toxicity of substances is determined;
- Toxicological effects of all substances are identified;
- Absorption factors are included.

A2 - Psychological stress

In areas where extreme concentration is required.

A3 - Physical stress

Noise:

- Noise surveys are conducted in all work areas;
- Noise zones are identified and indicated by means of symbolic signs.

Lighting and vision:

- Light surveys are conducted in all work areas;
- Surveys include all work areas, tasks and conditions;
- Inadequate lighting is assessed and actioned;
- Occupational hazards involving vision and visibility are identified.

Heat and cold extremes:

- Areas above Wet Bulb Globe Temperature (WBGT) index 30°C are identified;
- Cold stress areas are identified;
- Areas likely to inflict cryogenic burns are identified.

Vibration:

- Potential hazards are determined;
- Vibration effects are identified;
- Exposure to vibration is identified.

Ventilation:

- Natural ventilation is sufficient where adequate mechanical ventilation is installed;

- Local exhaust ventilation is adequate;
- All mechanical ventilation systems are on a maintenance program.

Non-ionising radiation:

- Processes utilising non-ionising radiation are identified;
- Health risks pertaining to non-ionising radiation are identified;
- Where applicable a non-ionising program is in place;
- Exposure to excessive sunlight is identified and actioned.

Ionising radiation:

Ionising hazards are identified.

A4 – Biological stress

Occupational hazards involving bacteria, viruses, fungus and parasites are identified.

A5 – Ergonomical stress

- Man / machine activities are assessed;
- Appropriate checklists are used;
- Observations are done while the task is in process;
- Detailed analysis is done on hazardous tasks;
- Impacts are reported and actioned;
- A multi-disciplinary team compiles remedial action plans;
- Health and safety risk tasks are assessed;
- Workstation and workplace designs are ergonomically sound;
- Workflow design and environmental conditions are adequate;
- The posture, actions and movement required for manual handling are not causing a risk;
- The shape, size, weight and nature of objects are assessed beforehand;
- The transportation distances of manual handling are taken into consideration;
- The handlers' age and general state of health is assessed;
- A study of energy consumption demands is carried out where applicable.

A6 – Risk identification and evaluation

- Risk assessments carried out take full cognisance of Material Safety Data Sheet (MSDS) data, survey results, medical information and general complaints;
- Occupational Health experiences are included in assessments of existing processes, new processes and modifications, pre-commissioning, new plant hand over and JSA and WSWP studies.

A7 – Occupational hygiene surveys

- The survey is done by an appropriately qualified hygienist / organisation;
- Surveys are conducted according to legal requirements and methodologies;
- All instrumentation complies with regulated standards;
- Survey reports are circulated to all relevant parties;
- Appropriate laboratories are used to analyse samples.

18 PROJECT INFRASTRUCTURE

The mine, mill processing plant and major mine site related infrastructures will be located at the Lofdal mine site approximately 30 km west of town of Khorixas are described in Sections 18.1 to 18.17.2. The Project infrastructure is designed to support an operation with two (2) open pit mines supplying a 2,000,000 tpa processing plant, operating on a 359 days per year. It has been developed for the most economical operation at this production rate and will require further expansion and development for any increases in throughput. The overall site layout showing location of the open pits, processing plant and waste management is provided on Figure 18-1.

18.1 Summary

The infrastructure required for the Lofdal REE Project will include:

- Mill complex;
- Site development and access;
- Overall water management plan;
- A tailings storage facility (TSF) and associated water management structures;
- Electrical site reticulation and generated power;
- Warehouse, offices, facilities, and other services.

The proposed layout of the Lofdal Project site is shown on Figure 18-1 and the 3D mill plant site layout on Figure 18-2. The block flow diagram processing plant is presented on Figure 17-1. The process plant site layout is presented in Appendix A.

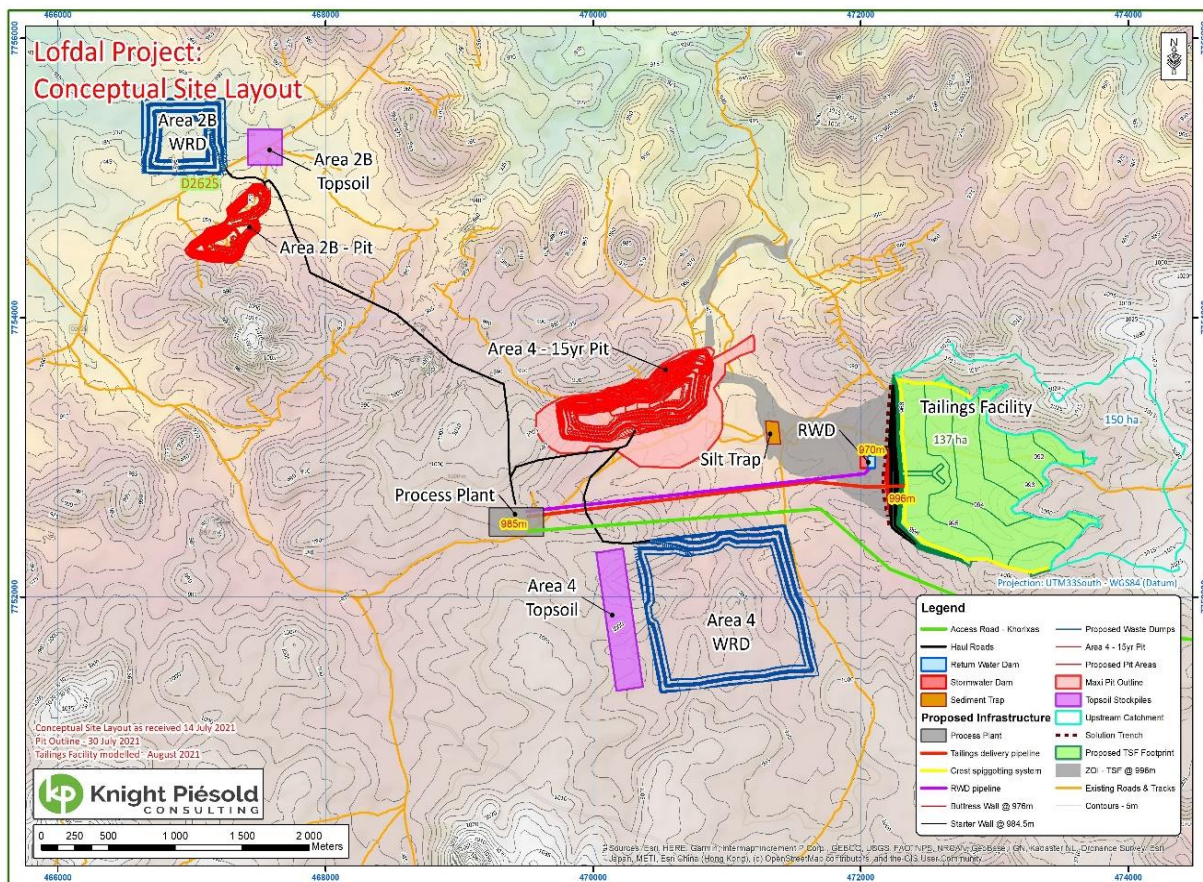


Figure 18-1 Lofdal Project Conceptual Site Layout

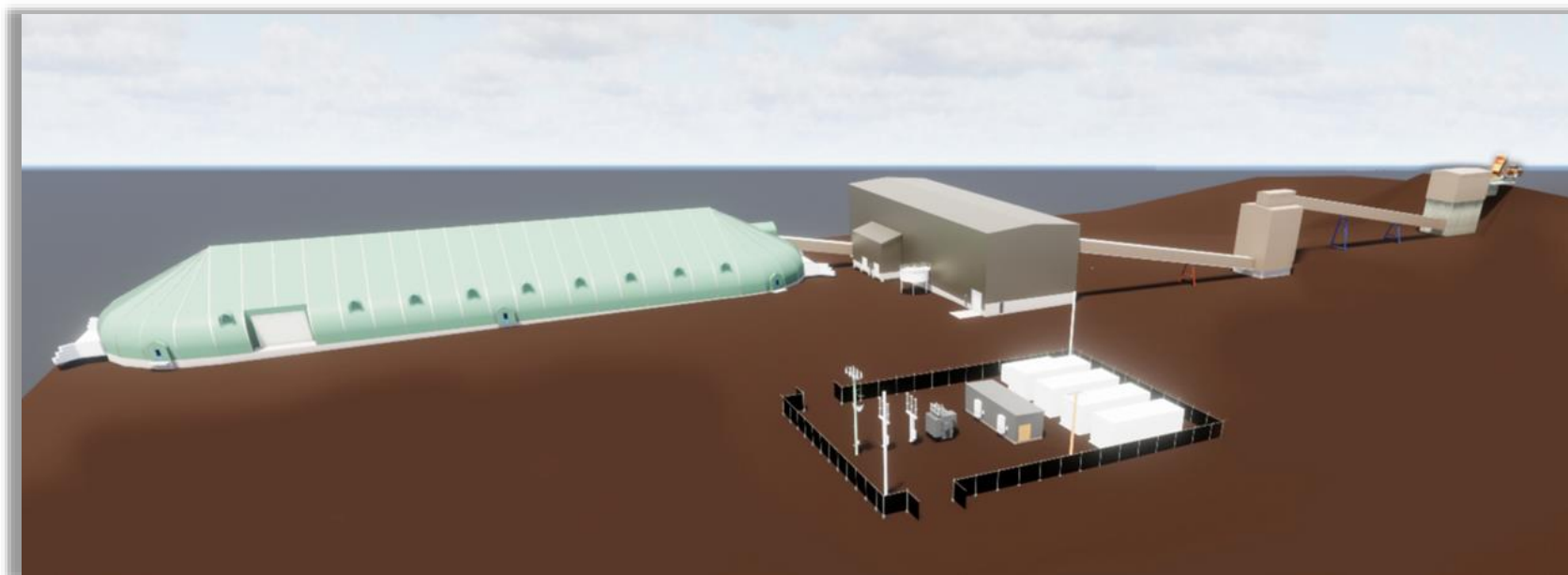


Figure 18-2 3D Mill Complex Layout

18.2 Lofdal General Site Plan

A Site Layout scale plan has been produced for the project: 2021 Site Layout for the PEA represents the overall area where the mining project is to be constructed. The drawing shows the haul roads, open pits, tailings storage facility (TSF), and processing plant with offices, maintenance shop, and related services for waste rock areas, and water treatment, Mill Complex and process plant.

The mill complex site is centrally located between the two open pits, approximately 500 m southwest of A4 pit, 1,200 m southeast of A2B and 2,000 m west of the tailing's facility. It will contain most of the offices, warehouse, maintenance, process plant, and safety office/first aid.

The layout of mill facilities has been optimized to take advantage of topography, and to reduce the earthworks. The entire process plant will cover an area of approximately 271 000 m². The complex will include the primary crusher, ore sorting, grinding via milling, magnetic separation, flotation, E-House, fuel storage, warehouse, office and dry.

18.3 Mill Circuit Controls

All elements of the mill, from the primary crusher through to tailings management, will be monitored and controlled from the mill control room. The control system will be industry proven programmable logic controllers (PLCs) and human machine interfaces (HMIs). A camera system will provide visual monitoring for critical equipment and process transfer points. The rack room adjacent to the control room will house the control system hardware including the networking equipment, virtual machine servers, PLC CPU and camera servers.

Industrial communication protocols such as Ethernet IP, HART for intelligent instruments etc. will be utilized to the greatest extent possible and Ethernet based remote PLC I/O (RIO) cabinets will be strategically placed in the mill to minimize field cabling requirements.

Supervisory staff will have access to the operating graphics remotely for review of plant operation and key performance indicators. Overall plant performance and process data will be historically archived and will be accessible for trending and reporting (shift reports, etc.).

18.4 Site Development and Access

The main access road to the mill complex from the town of Khorixas starts at D2625 turn-off from C39 and it is an 8-kilometer well-constructed gravel district road and regularly maintained by the Roads Authority. The new access road shall be built and starts along D2625 to the TF and A4 pit area and will be a 20 km, two-lane gravel road with a main gate to be manned 24/7.

There are a few minor water courses to cross identified by NMI Inc as areas to consider road construction. The new gravel road will be constructed as per Namibia Roads Authority standards to support two-way traffic and to be utilized to transport materials and supplies.

The haul roads are planned to be constructed on site for transporting ore and waste from the open pits to their designated destinations. Mine haul road and service roads are planned to be constructed to accommodate 30 and 50 nominal tonne trucks carrying ore and waste from the pit to the crusher, and waste to the waste rock area and to the TF. The waste haul road will connect with the aggregate pit and will also serve as the access to the contractor aggregate primary crusher.

18.5 Overall Water Management Plan

Fresh water to the mill complex site will be supplied by a 250 NB SANS 719 steel pipeline from NamWater (Khorixas town) water supply system. The pipeline route in Figure 18-3 starts in Khorixas at an elevation of 950 masl and follows the D2625 northwards for approximately 10 km and then veers west on a small dirt tract for further 20 km. The local highpoint along the route is at 12.4 km mark at an elevation of 1054 masl and the discharge elevation is approximately 960 masl. As per NMI, an alternative source for water (groundwater potential along a major fault zone) is a borehole with a depth of 74 m located 6.2 km south of the A4 Pit. If developed, the line will require 6-inch diameter HDPE pipe, installed overland and potentially be used for additional process as well as potable water supply.

Surface runoff and return water from the TSF and Open pit will be collected in a RW dam located at the west dam of the TSF starter wall, in a natural low area to enable gravity feed where possible and to minimize earthworks. Run-off collected from areas that are not able to gravity feed to the pond will be pumped. The pond will be lined with a 60 mil HDPE liner, and will be equipped with a subdrain system to anticipate groundwater in this area. The pond water will be pumped to either the process plant or the water treatment plant as required. Treated water will be discharged to a 6-inch diameter HDPE pipes.



Figure 18-3 Water Supply Pipeline Route

In general, utilities at the plant site will be buried for vehicular access, while outside the process plant area they will run at grade on gravel pads, with culvert / casing protection at road crossings as required.

Yard utilities comprise potable water, sanitary sewage, and fire main, complete with yard hydrants and building connections. Due to the presence of bedrock, these lines will have a relatively shallow bury for mechanical protection.

Spill containment systems will be provided for the fuel storage facility and the oil-filled transformers in the electrical substation.

Aggregate required for site development can be obtained from the mine site quarry area equipped with a mobile crusher.

A water management plan has been developed and includes the following elements:

The mine's facilities are planned to be designed to minimize the effects on the environment that need to be mitigated to the maximum extent practicable.

Natural runoff is planned to be away from, and around, areas disturbed by the mining and processing activities.

Detention storage is planned to be provided for runoff from disturbed areas to allow suspended particles to settle out. The water will then be discharged under a permit, recycled to the process plant, or pumped to the TMF for re-use. Compliant water may also be used for dust control and progressive reclamation.

Waters that contain, or potentially contain, elevated dissolved metals when precipitation meets mined materials are planned to be collected in water quality control ponds and recycled for reuse in the process plant.

Sufficient water storage is planned to be provided in the TSF and water quality control ponds to prevent discharges during extreme wet periods.

Recycling of all mine waters is planned to be maximized, thereby minimizing the need for make-up from surface and/or groundwater.

Provide treatment of stored waters requiring discharge to meet applicable discharge and receiving water standards, as necessary.

18.5.1 Water Management

18.5.1.1 Climate

The Lofdal Rare Earths Project is located approximately 25 km west of the Khorixas Town, in the Kunene Region of Namibia. The climate in Khorixas can be described as semi-arid to arid, with average summer temperature reaching into 40 degrees Celsius (°C) and winter temperature touching 0 °C. The region is a summer rainfall region with the highest temperature and rainfall depths are recorded from December to May.

The climate in Namibia is highly variable, with extreme drought periods and rainfall events (MET, 2011). Climate change models indicate that Namibia, especially the eastern and southern parts are adversely affected by rising temperatures and the consequences thereof (WBG, 2021).

18.5.1.2 Rainfall and Evaporation

Daily rainfall data was received from the Namibian Meteorological (NMET) services for the Khorixas Station for a record period of 56 years from the year 1955 until 2008 (NMET, 2021). Monthly and daily synthetic rainfall data was also obtained from the Climate Research Unit (CRU) and KNMI database (CRU, 2021) (KNMI, 2021), the records did not correlate well with the actual observed dataset, and it was therefore decided to only use the NMET Khorixas Station record.

The monthly rainfall distribution as obtained from the NMET and the Pan evaporation from the Namibia Department of Water Affairs is summarized in Table 18-1. The mean annual Pan evaporation is 2 850 mm (DWA, 1988) and the mean annual precipitation of 223 mm (NMET, 2021). The months with the highest evaporation are December and January and rainfall are February and March.

Table 18-1 Mean Monthly Rainfall and Evaporation

Monthly Average (mm)	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Rainfall	46	61	60	20	1	0	0	0	2	3	14	16	223
Evaporation	456	428	285	228	143	57	57	86	143	228	342	399	2850

18.5.1.3 Extreme Rainfall Estimation

Daily resolution rainfall data was used to determine the statistical frequency distribution of events. The General-Extreme Value (GEV) resulted in the best fit between the actual observed ranked annual maximum rainfall depths and the various distributions, such as Log-Normal, Log-Pearson and Extreme Value. A Weibull plotting position was used to successfully match the observed data to the distribution. The rainfall depths for different return periods are summarized in Table 18-2 (SANRAL, 2013) (NRA, 2014).

Table 18-2 24-Hour Duration Extreme Rainfall Depths Estimates

Return Periods (years)	2	10	20	50	100	200	500	1000	PMP ¹ (100 000)
Rainfall Depths (mm)	58	90	130	176	197	220	251	277	372

Notes:

1. World Meteorological Organisation (WMO) probable maximum precipitation method could not be applied, due to lack of sub-daily (< 24-hour) rainfall data.
2. 1:100 000-year Return Period rainfall depth derived with the statistical method used as PMP Equivalent.

18.5.1.4 Stormwater Management

A high-level stormwater management assessment was conducted on the surrounding catchments, and they were found to be small in size, as they form the upper reaches of the Huab Catchment. It was found that no storm water diversion system is required upstream of the TSF. It is unlikely that the runoff generated in the catchment adjacent to the toe of the TSF will have an impact on the TSF, as there is sufficient distance between the main watercourse, the RWD and the toe of the TSF.

18.5.1.5 Water Balance

A water balance model was developed, based on the available NMET record and the mine plan. A simplified schematic is shown in Figure 18-4, indicating all the components comprising the water balance, with first order values.

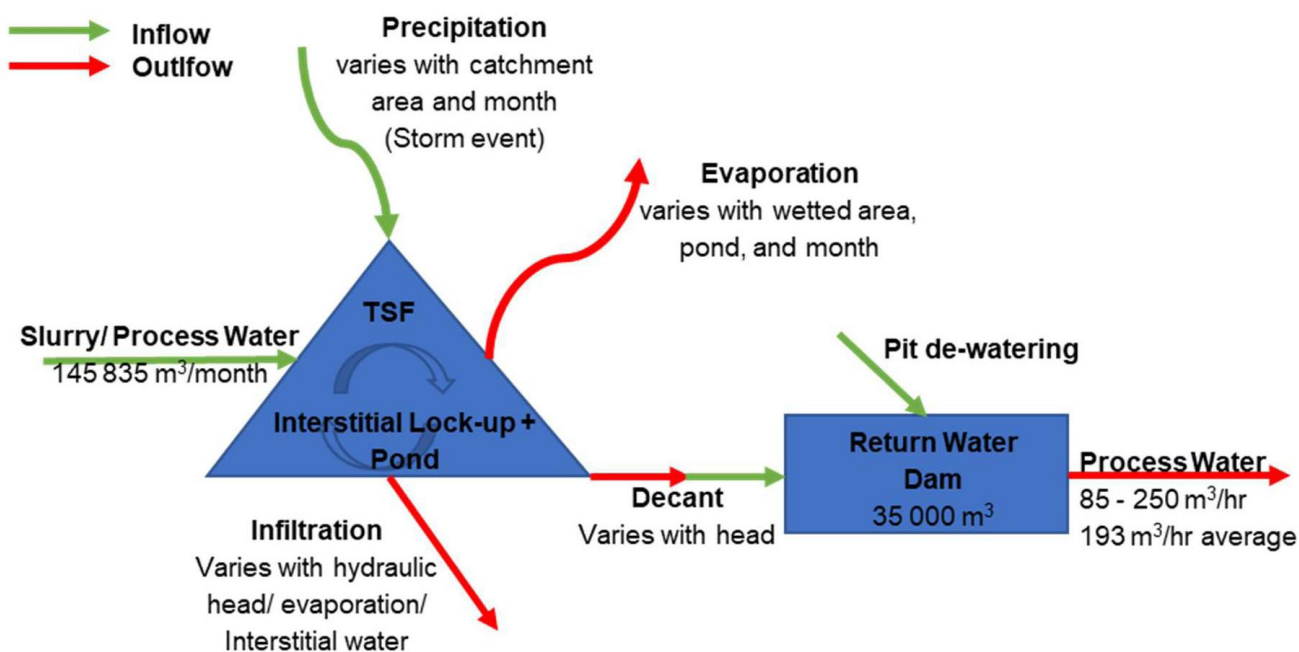


Figure 18-4 Simplified Inflows and Outflows Schematic

The average monthly inflows, outflows and decant volumes over the LOM are presented in Figure 18-5. The return water dam is sized for the worst rainfall month allowing for seven-day storage, with an assumed 8 hours of operation per day.

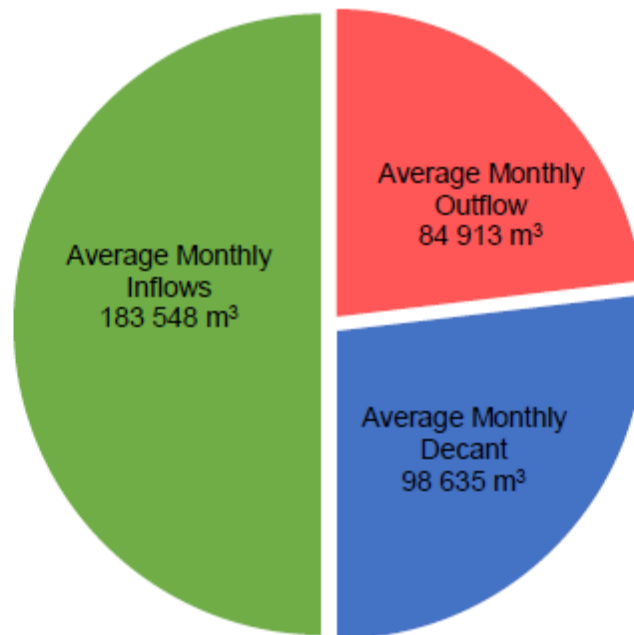


Figure 18-5 Inflow, Outflow and Decant Average Monthly Volumes

18.6 Effluent Treatment

18.6.1 Conceptual Water Management Plan

The conceptual water management plan prepared by Knight Piésold is presented in Figure 18-6 which is expected to minimize surplus water from the tailings management facility that will require treatment and discharge.

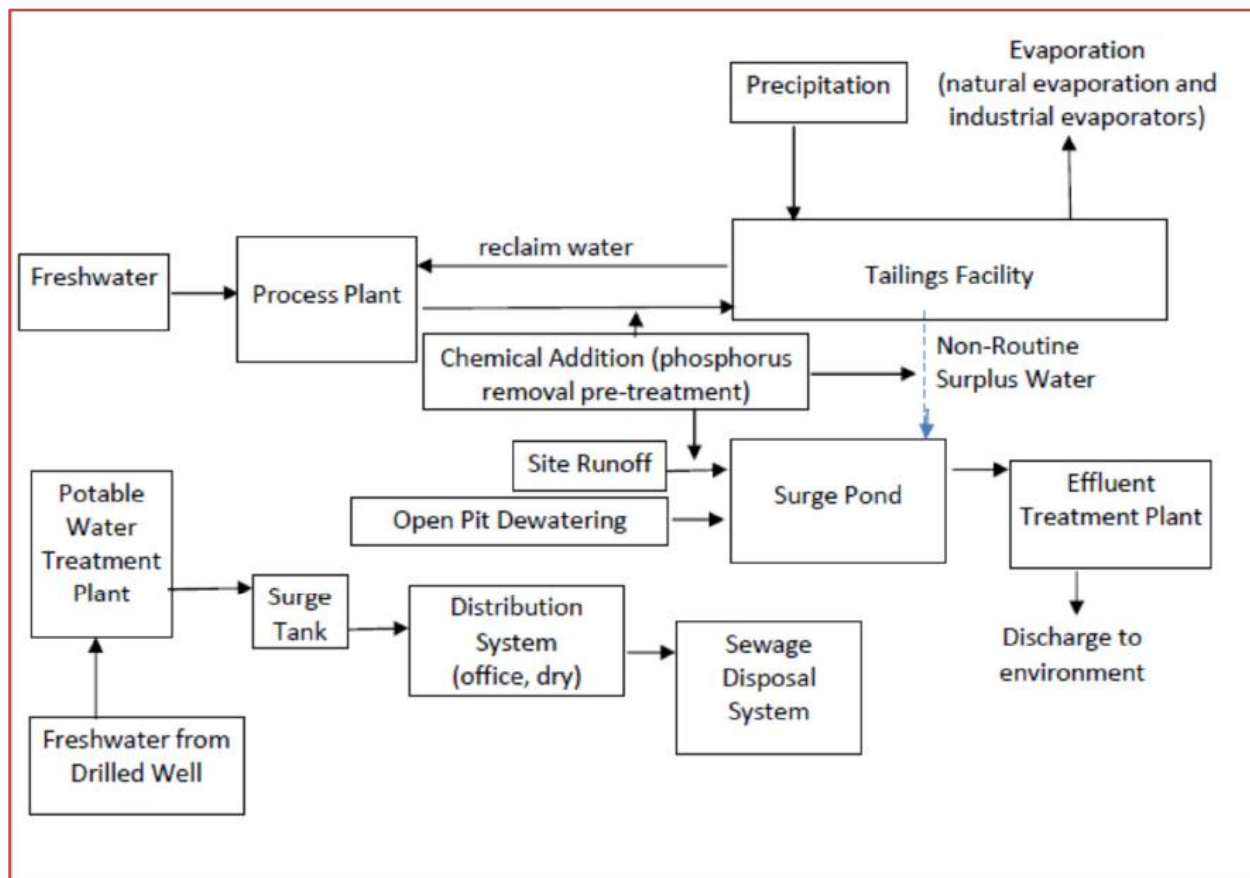


Figure 18-6 Conceptual Water Management Plan

18.6.2 Influent Characterization and Effluent Limits

The following sections summarize the assumptions regarding influent quantity and quality, based on currently available information.

- Influent quantity
- A maximum operating capacity of 6,000 m³/day has been assumed for the purpose of the ETP design.

Assumptions regarding influent quality are as follows:

- Based on benchmarking of open pit operations that use emulsion rather than ANFO and diligently follow good housekeeping practices (e.g. turn off fill hose while moving between blastholes), the concentration of ammonia in open pit dewatering will meet effluent limits without any treatment;
- Based on the geochemical characterization work for the deposit to date, the metals that are anticipated to have elevated concentrations are listed in Table 18-3.

The reagents that will be used in the process plant are the conventional ones listed below:

1. Sodium Silicate (N Type)
2. Calgon
3. Florrea 3900
4. Florrea 3000
5. MIBC/Pine Oil
6. Florrea 7411
7. NaOH
8. H₂SO₄
9. MgCO₃
10. Na₂CO₃
11. H₂C₂O₄
12. Magnafloc 10
13. Ca(OH)₂
14. NH₄OH
15. HCl
16. Primene JMT
17. Isodecanol
18. Aromatic 150ND

It is assumed that the TSF will be operated as a stand-alone system and will generally have a negative water balance due to water loss to tailings solids pore space. Table 18-3 lists parameters of concern with their corresponding effluent limits. It is assumed the wastewater from the surge pond (refer to Figure 18-6) will contain all of the parameters from Table 18-3, with low concentrations of phosphorus, ammonia, as well as oil and grease. It is assumed that wastewater from the TSF will contain all of the parameters from Table 18-3, with higher concentrations of phosphorus from mill reagents and lower concentrations of ammonia, as well as oil and grease.

18.6.3 Effluent Limits

Receiver based effluent limits that have been derived in accordance with Ministry of Environment Conservation policies are listed in Table 18-3.

Table 18-3 Influent Quality and Effluent Limits

Parameter	Units	Influent	Effluent Limit
pH	S.U.	5.0 to 10.5	6.0 to 9.5
TSS	mg/l	100	15
Oil & Grease	mg/l	> - 50	15
Total Ammonia Nitrogen	mg/l	<10	10
Total Phosphorous	mg/l	1.0 to 8.0	1.0
Total Tungsten	mg/L	>2.9	2.9
Total Arsenic	mg/l	5.0 to 11.0	0.5
Total Antimony	mg/l	1.0 to 1.7	1.0
Total Thallium	mg/l	0.8 to 3.0	0.08

Total Uranium	mg/l	0.02 to 0.03	1.5
Total Copper	mg/l	0.1 to 0.6	0.3
Total Nickel	mg/l	>0.5	0.5
Total Zinc	mg/l	>0.5	0.5
Total Lead	mg/l	>0.2	0.2
Radium-226	Bq/L	not determined	0.37
Acute lethality	% mortality in 100% effluent		<50%

18.6.4 Effluent Treatment Plant (ETP) Process

Of the parameters listed in Table 18-3, the following three (3) must be addressed independently.

TSS: Traditional pressure filtration can handle a concentration of 100 mg/l on a periodic basis, but the performance will be encumbered if that level of influent concentration is maintained for any prolonged period of time. Therefore, removal of TSS down to less than 15 mg/l will require the use of a clarifier, which will provide additional benefits for the removal of other contaminants.

Oil and Grease: A large volume of oil in influent will be problematic for the proposed wastewater treatment system. It will be more cost-effective to isolate the source and remove the oil separately if needed, in order to keep the influent oil and grease concentrations to below 10 mg/L, rather than install a complete oil/water separation system for a 6000 m³/day flow.

Phosphorus: Based on available information, the primary source of the phosphorus will be mill reagents, which tend to concentrate in the TSF due to internal recycle between a process plant and TSF. Precautionary reduction and removal of phosphorus, in the phosphate form, should be accomplished by a separate treatment package intercepting the tailings delivery pipeline from the process plant to the TSF. This approach allows for the reduction of phosphorus in a smaller facility that would be deployed as needed based on operating data. The same separate treatment package could also be used to pre-treat the other wastewater streams as they enter the RWD pond if elevated phosphate concentrations are observed.

The remaining parameters listed in Table 18-3 are metals and will be amenable to precipitation by hydroxide, sulfide, oxidation, metal co-precipitation, and/or adsorption provisions in the ETP. The use of a targeted chemical conditioning step upstream of the clarifier allows for the reduction of most of these metals to below the effluent limits. The additional polishing steps downstream of the clarifier could be employed on an as needed basis, based on operating data.

It is assumed there will be no oxygen demand or toxicity from residual process reagents.

18.7 Process Description

The process flowsheet is provided in Figure 18-7 and summarized below.

Wastewater will flow from the surge pond into the first-stage chemical conditioning tank, where specific predetermined chemicals will be added. The tank will be aggressively aerated to foster the co-precipitation of contaminants with metal oxides.

This partially conditioned wastewater will then flow into the second-stage conditioning tank where additional chemicals will be added to enhance the removal of certain more recalcitrant metals. A polymer will also be

added to begin the coagulation process. A final polymer may be added in the effluent from the second-stage conditioning tank to foster flocculation. This conditioned wastewater will flow into the stilling well of a circular mechanical clarifier where liquid solid separation will take place.

In the clarifier, the floc formed in the upstream steps will settle to the bottom where it will be conveyed by a scraper to a center discharge point, from here this settled sludge will be removed for dewatering in a geobag or disposal in a TSF. Filtrate liberated by the geobag dewatering will be returned to the ponds by gravity or pump conveyance. The clear supernatant produced by the settling process will exit the clarifier by displacement and flow into the downstream conditioning/pump tank.

In the pump tank, the pH will be monitored and adjusted to ensure it is within regulatory limits, then the wastewater will be pumped through a final set of pressure filters and discharged into the environment. In the event that certain metals are above discharge limits, the filtrate could be pumped through a set of ion selective media vessels for final polishing by adsorption.

It should be noted that the inclusion of a clarifier is to handle a wastewater flow of 250 m³/hour that could easily have a concentration of TSS in excess of 100 mg/L. Elevated TSS could be a result of site or production issues, but a portion of the TSS will be a result of the chemical conditioning steps required to capture the parameters listed in Table 18-3. The use of a filter as an effluent polishing step is reasonable because the expected concentration of TSS in the effluent from the clarifier is 10 mg/L, so the filters will not be overloaded. The TSF will receive water from precipitation and the recirculating load between the TSF and process plant. The water in the TSF will contain all of the parameters listed in Table 18-3, with the potentially elevated concentration of phosphorus being the most problematic. Any potential flow of surplus water from the TSF will be proportional to precipitation, whereas the flow to and from the TSF and process plant will remain relatively constant. The phosphorus concentration can be dramatically reduced by the addition of a metal salt and a coagulant followed by liquid solid separation by clarification or filtration. The proposed treatment design will provide a pre-treatment container on water reclaim loop returning to the TSF from the process plant. This system will add the chemicals, on an as needed basis, required to precipitate a phosphorus compound that is durable and will settle well in the TSF, which will act as a clarifier. This step will also precipitate and remove some of the stated parameters, but this degree of removal is considered a safety factor and does not enter the design of the subsequent treatment steps. This system could also be used to pre-treat the other wastewater streams as they enter the surge pond if elevated phosphate concentrations are observed.

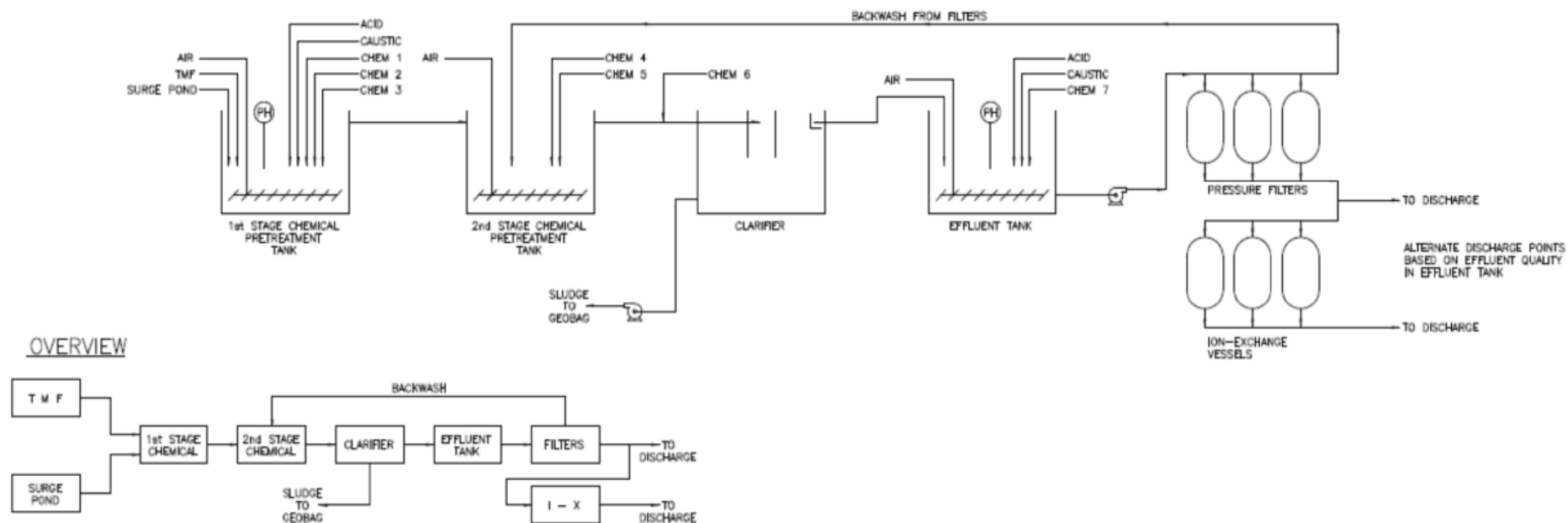


Figure 18-7 ETP Process Flowsheet

The following sections describe the assumptions, model construction, and results of the water balance and water quality modeling.

Waste rock and tailings will be placed in subaerial storage which will result in exposure of these materials to atmospheric conditions. The exposed material surfaces, the fine-grained portions, are susceptible to weathering processes that can lead to the mobilization of constituents through oxidation and dissolution reactions. Water that infiltrates into the waste rock pile and/or runoff that flows along the surface of the tailings pile can interact (come into contact) with material (e.g., waste rock and tailings) surfaces; this water is referred to as “contact” water. Brief descriptions of these site components, and how these components are expected to influence site water quality, are presented below.

18.8 Waste Rock Pile

Waste rock will be stored in two (2) areas and are called Area 2B WRD and Area 4 WRD. The Area 4 WRD adjacent to the TSF. Contact water, generated from water-rock interactions, will contain soluble constituents (i.e., major ions, metals and nitrogen species) from mineral weathering by-products and from residual explosives from blasting, which can persist in the waste rock and are water soluble and provide a source of ammonia and nitrate.

Waste from one of the process streams, referred to as the Dense Media Separation or “DMS” stream, will report to the waste rock areas.

The contact water from the waste rock areas will report to the TSF.

18.9 Tailing Storage Facility and Associated Structures

Tailings are produced as part of ore processing and will be stored in the TSF. The TSF will be a valley impoundment with a 18 m high starter embankment constructed from rockfill. The raising strategy is envisaged to be downstream for the first raise using waste rock from the first 2 years of operation and upstream afterwards to the final elevation. The tailings will be delivered to the TSF via a tailings delivery pipeline and thereafter distributed around the TSF through a main ring pipeline with regularly spaced spigots. A gravity decant-penstock system comprising a starter and final decant will be help control the amount of water on the TSF and has been sized to deliver up to 2.9 m³/s of supernatant water to the RWD based on the preliminary water balance carried out.

Contact water, generated from interaction of runoff water with the tailings, will contain soluble constituents from by-products of tailings oxidation and process water. The contact water from the tailings area will report to the RWD located at the toe of the TSF via a gravity decant/ penstock system. The RWD will have a storage of 40 000 m³ to continuously absorb decant water, consisting of production water and floods, it is recommended to split the RDW into two compartments, for operational purposes. The maximum monthly decanted water volume is 252 000 m³/month (average of 8 300 m³/day). The TSF has been designed to temporarily store storm water from the TSF and the adjacent upstream catchment. In addition to storing return water decanted from the TSF the RWD has been sized to contain seepage water collected from the TSF via a network of underdrains. Provision has been made for a wall front solution trench to intercept seepage through the wall. For this level of study, the underdrains and solution trench are only shown on the accompanying drawings as indicative lines.

18.10 Open Pit

The excavation of mine rock and the development of the open pits will result in the rock face of the pit walls being exposed to atmospheric conditions. The blasting of the rock typically results in a “damaged zone” of rock that consists of shallow fractures that extend into the bedrock from the face of the pit wall. The surfaces of the fractures in the damaged zone are also exposed to atmospheric conditions. Contact water, generated from water-rock interactions (i.e., from direct precipitation, groundwater inflow and runoff from the open pit catchment area) at the pit wall surface, will contain constituents of the exposed rock and explosive residues. Pit-wall contact water will report to the pit sump and affect the quality of the sump water.

The contact water from the open pits will report to the RWD Pond.

18.11 Return Water Dam (RWD) Pond

The RWD Pond was sized to continuously absorb decant water, consisting of supernatant water and environmental design flood. It is envisaged to split the RW pond into two compartments for operational purposes, namely an operational compartment and a stormwater management compartment.

The water quality in the RW will therefore consist of contact water from the TF and open pits, together with natural runoff from the catchment areas.

Excess water from extreme flood event in the RWD will be discharged to the environment in accordance with the Namibian effluent water discharge regulatory/permitting requirements and effluent discharge restrictions (i.e., effluent concentration limits). The surface water receiver of the effluent will need to have an appropriate assimilative capacity to allow for rapid mixing of the effluent, such that Water Quality Objectives are achieved downstream of the outfall. If the effluent has concentrations greater than the effluent concentration limits, and/or the surface water receiver does not have the necessary assimilative capacity, the effluent will need to be treated prior to discharge to the environment.

18.12 Conceptual Tailings Storage Facility (TSF)

Concept Design Assumptions:

- Tailings throughput: Average 1.6 million tonnes per annum (Mtpa), maximum 2.0 Mtpa, at 46% solids content by mass.
- Total storage requirement: 26.7 million tonnes (Mt).
- Life of Mine: 16 years.
- Maximum rate of rise: 2.5 meter per year for upstream raises (m/yr.).
- Tailings geochemistry and classification assumed to require lining system.
- TSF classification (ICMM; UNEP; PRI, 2021): High due to consequence of failure on the pit and mining operation and risk to life.

TSF Geometry

- Starter embankment elevation: 984.5 m.
- TSF final elevation: 999.5 m.

- Total TSF height: 33 m.
- Conceptual freeboard requirement for a 1 in 2,475 years storm event: 1.5 m.
- Overall outer slope: 1 vertical to 4 horizontal (1v:4h) including benches, waste rock buttress and upstream raises.
- Tailings deposition through a ring feed system and includes a pipe laydown access road, decant system, and return water dam.

Quantities:

- Starter wall earth fill volume: 340,000 m³.
- On-going waste rock fill placement for wall raising: 85,000 m³ / year (for 2 years but could be sustained afterwards).
- Embankment face and basin HDPE liner surface area: 500,000 m² in year 1, and 450,000 m² over year 1 and year 2 to complete the basin lining along the valley impoundment.

18.12.1 Design Objectives

The principal objectives of the TSF and related water management infrastructure designs are summarized as follows:

- Provide a facility for permanent and secure storage of thickened slurry tailings during LoM and post closure.
- Preliminary layout of TSF and related access roads plus surface water management structures.
- Control and management of surface water during operations.

The design for the TSF is based on guidelines from accepted local and international standards for mine waste management design, surface water management design and infrastructure design (Canadian Dam Association - CDA, 2014/ 2019; Mining Association of Canada, 2017; Namibia Roads Authority Drainage Manual, 2014a; Namibia Roads Authority Materials Manual, 2014b; the South African National Standards (SANS) 10286; and the Global Industry Standard on Tailings Management (GISTM, 2020).

18.12.2 Design Criteria

KP compiled a design criteria document (refer to the KP *Tailings Storage Facility Conceptual Design* report in Appendix B) and a site visit report (refer to the KP *Initial Site Visit To Lofdal REE Project* report in Appendix C) which was submitted to NMI and the rest of the project team for review and approval. The document has been subsequently amended to indicate the revised mining schedule and TSF storage requirements. The general information availed for the conceptual design of the TSF is presented in Table 18-4.

Table 18-4 Design Criteria-Summary

Item	Design Criteria		Units
1	Topographical Survey	June 2020 topographic survey	
2	Legal Framework	Minerals (Mining and Prospecting) Act of 1992	
		Environmental Management Act of 2007	
		Global Industry Standard on Tailings Management (2020)	
3	Documentation	Preliminary Economic Assessment Report – Lofdal Project, 2014 Updated Environmental Management Plan – Lofdal Project, 2016	
4	Specific Gravity (Solids)	2.78	
5	Particle Size Distribution	P ₈₀ = 35 microns (silts and finer)	
6	Tailings solid content by weight	45.97	%
7	Average settled dry density (Year 1)	1.25	t/cum
8	Average settled dry density (after Year 1)	1.35	t/cum
9	Recoverable grade	0.017	%
10	ROM split	100(ore)	%
11	Average annual throughput to TSF	1.49 (year 1 & 2)	Mtpa
		2.02(year 3 & 12)	Mtpa
		1.19 (year 13 to 15)	Mtpa
12	Design Life	15 and beyond	Years
13	Rate of Rise	2.5 for tailings not impounded behind starter wall (Justified by the fine-grained nature of the tailings)	m/year
14	Storage Capacity Required	26.8	Mt
15	Tailing's chemistry	Tailings are anticipated to be non-acid generating since the ore body does not contain detectable sulphide material and has abundant neutralization potential. Tailings are expected to contain a certain level of radioactive uranium.	
		Considering the lack of geochemistry testing and	
		information it is assumed to classify as hazardous and	
		requires a liner.	
16	Slope Stability	The minimum Factor of Safety for slope stability under normal Operating conditions (local and global stability) will be:	
		• Temporary slopes (during or at end of construction)	1.3
		• Permanent slopes (during operation and post closure)	1.5
		• Post Peak Static Loading Conditions	1.1
		• Pseudo-static	1
17	Overall Outer Side Slope	Assumes 1V:4H (also deemed suitable for closure rehabilitation)	
18	Conceptual Closure Design	The TSF side slopes are to be clad with 750 mm thick layer of waste rock. There will be no bench drains. The penstock(s) will be sealed. The TSF basin is to be covered with 200 mm of topsoil.	

18.12.3 TSF Concept Design

18.12.3.1 Capacity Assessments

18.12.3.1.1 Tailings Throughput

The Lofdal deposit is a low-grade deposit with an anticipated recoverable grade of approximately 0.017%. Initially, the process philosophy included a preliminary ore sorting where 33% of ROM would be sent to a Waste Rock Dump. The remaining 67% would go through the mill plant and ultimately to the TSF. The process has been revised and a conservative assumption was made that the entire 100% fraction of the ROM that ends up at the mill will ultimately end up in the TSF.

A total storage capacity of 26.8 million tonnes (19.9 million m³ at 1.35 t/m³) is therefore required at the TSF over the 16-year, LOM. The previous conceptual TSF design was modelled on an anticipated 15- year LOM. KP further revisited the storage capacity to determine the size of the TSF to final height as well as the size of the starter embankment required to contain tailings in the early years of deposition. The TSF location was maintained the previously selected site by NMI. The storage capacity assessment carried out is documented in subsequent sections.

18.12.3.1.2 Storage Capacity Assessment and Curves

The location of the TSF was maintained as the site selected by NMI. Accordingly, a new facility height was adopted, and a conceptual level layout was developed using Muk3D software. To accommodate the full tailings volumes over the LOM the crest of the outer wall would have to be raised to an approximate elevation of 999.5 masl, which would equate to a total wall height of approximately 33.5 m. A total freeboard of 1.5 m was assumed in accordance with CDA (2019) guidelines. Figure 18-8 shows the proposed TSF layout.

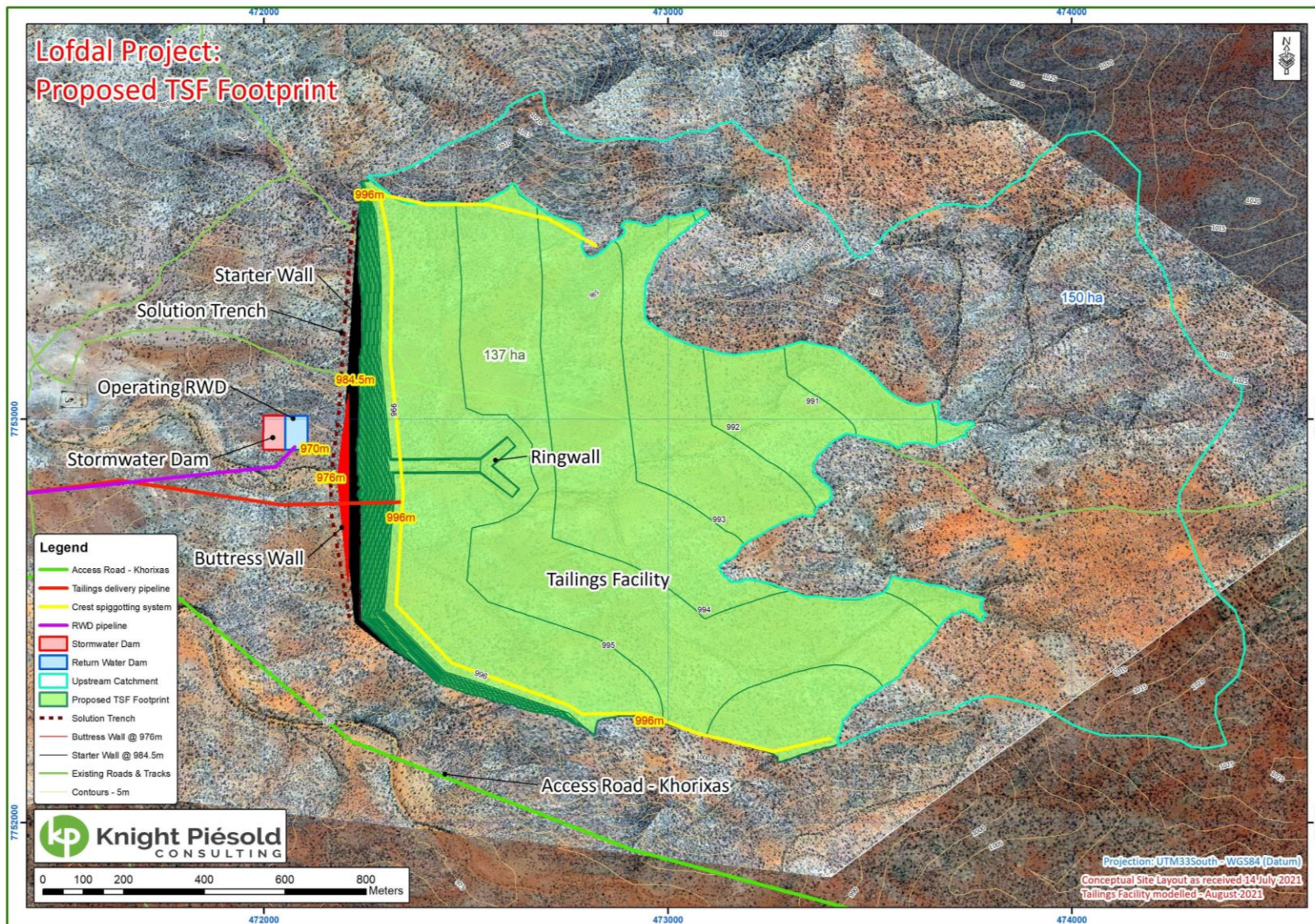


Figure 18-8 Tailings Design Layout

The filling curve for the TSF are presented in Figure 18-9. The tailings storage facility starter wall and final tailings crest levels have been calculated with the stage capacity curves (SCC) developed during the capacity assessment. From the SCC, beach elevations lower than 981 masl result in a rate of rise (RoR) higher than the allowable of 2.5 m/year. From the stage capacity curves it is determined that it would take approximately three (3) years and four (4) months of deposition to reach a rate of rise below 2.5 m/year. Above this elevation upstream wall raising with tailings can safely commence up to a crest elevation of 999.5 masl. At this crest elevation the facility will have an approximate total height of 33.5 m.

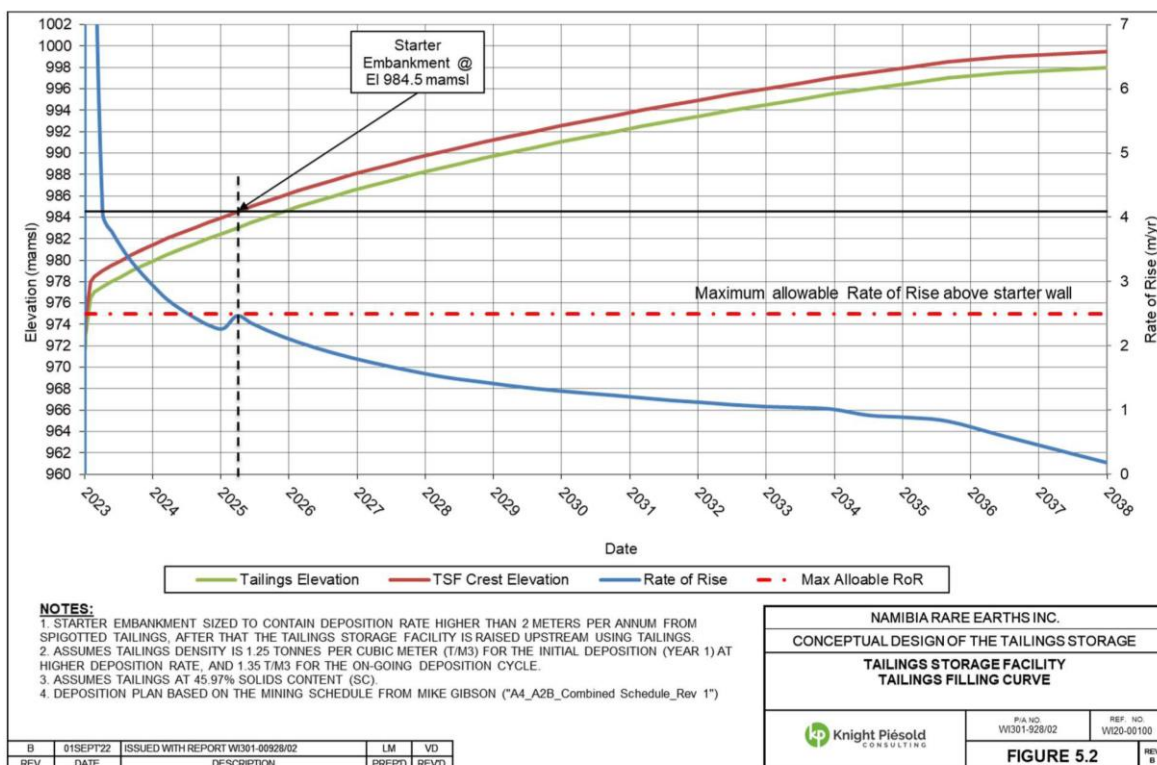


Figure 18-9 TSF Filling Curve and Estimate of Starter Wall Requirement

18.12.4 TSF Components

18.12.4.1 Starter Embankment

The TSF will consist of a starter wall with crest width of 10 m and crest elevation of 984.5 masl. The height of the starter wall will vary from 0 to 19 m high across the valley. The starter wall will be constructed from overburden rockfill borrowed from its footprint and areas surrounding the TSF with an outer wall slope of 1V:2.5H and an inner wall slope of 1V:2H and placed on shallow bedrock.

As soon as sufficient waste rock becomes available from open pit stripping and mining, a waste rock buttress will be constructed on the downstream slope of the starter embankment. The buttress crest should be 10 m wide extend to the height of the starter embankment for a total width of 20m. Further waste rock placement on the downstream face of the self-raising outer should be made in the following years.

18.12.4.2 Self Raising Outer Wall

Above the starter embankment upstream construction with tailings will entail the following:

- Establishment of an appropriate step-in from the previous embankment raise to maintain an overall slope angle of 1V:4H. The outer wall will be raised to El. 999.5 masl, equating to a total height of approximately 33.5 m.
- Construction of consecutive machine built 1 to 1.5 m high tailings lifts around the TSF perimeter.
- Filling of the paddocks in 1 – 1.5 m lifts with tailings via spigots

18.12.4.3 Decant System

At this stage of the study, estimations of the required decant rate have been made with assumptions. It assumes a gravity decant system with a primary intake and secondary (final) intake tower with penstock system conveying water to the RWD. Provision has been made for the primary decant system cost and a secondary one to decant any localized pool that may form due to the double prong shape of the valley the TSF has been placed on. It is recommended in future design state to consider a tradeoff study with a barge decant system in the northeastern valley where the pool is to be located. Current estimates are estimated sufficient for either or option.

18.12.4.4 Return Water Dam

The RWD should have a storage of 35 000 m³ to store operating decant water, and flood/stormwater event requirement for the dam classification. It is envisaged to split the RWD into two compartments, for operational purposes and evaporation mitigation. The maximum monthly decanted water volume is 119 125 m³/month.

18.12.4.5 TSF and RWD Liner

The tailings are assumed to have some amount of radioactive uranium. The TSF basin should be lined considering the possible nature tailings and the lack of information on hydrogeological setting.

The proposed liner system will comprise a 150 mm bedding layer with a 1.5 mm HDPE geomembrane placed on top and covered by a protective A6 bidim or similar. This will help optimize the design and improve costs. An interface layer between the rockfill starter wall and the liner for protection of the liner.

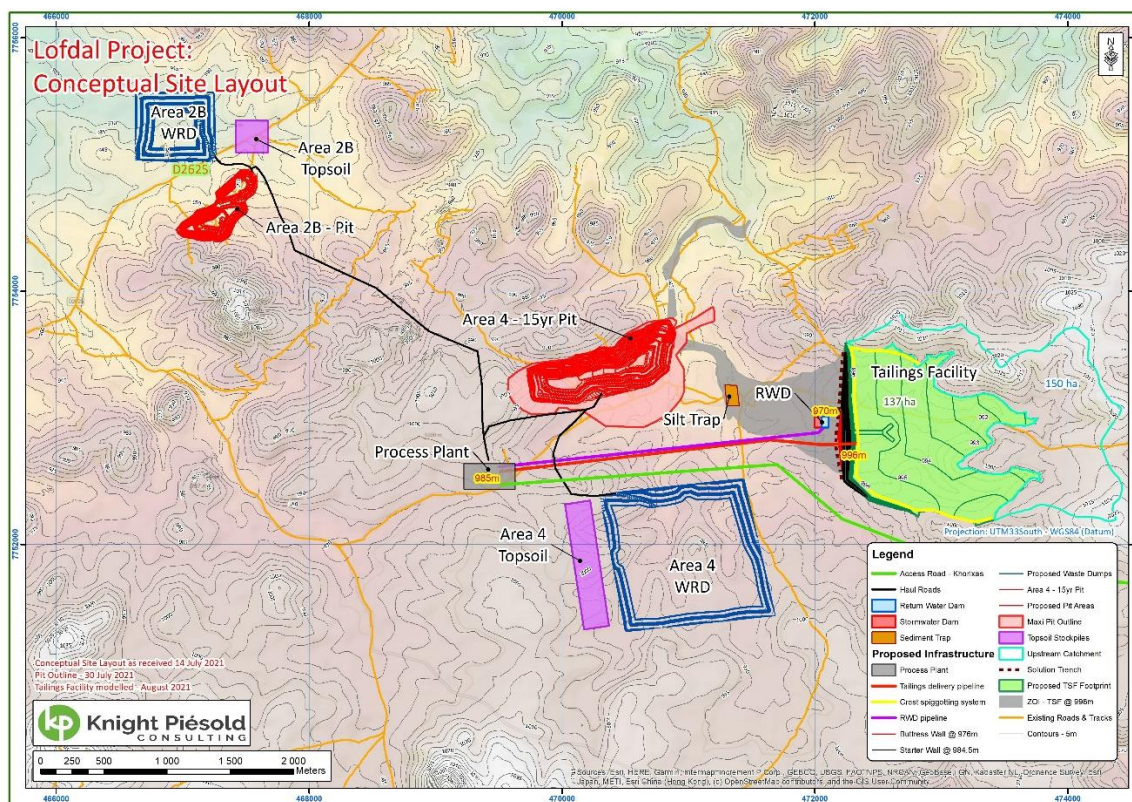
18.12.5 Dam Safety Classification

18.12.5.1 Zone of Influence

The zone of influence is defined as the anticipated area surrounding the TSF that would be adversely affected by the release of tailings to the environment in case of a TSF breach.

A preliminary zone of influence was determined based on guidelines stipulated under SANS 10286:1998 which is a geometric extrapolation of the TSF height fitted to the natural topography. While this doesn't

consider tailings volumes and rheology, it is considered to provide reasonable information to conceptualize the direction of flooding and inundated area following a dam breach. The preliminary zone of influence is illustrated in Figure 18-10.



Notes:

1. Preliminary zone of inundation to be revised in subsequent phases to comply with the GISTM, 2020.

Figure 18-10 TSF Conceptual Zone of Influence

18.12.5.2 Dam Safety Classification – Consequence of Failure

A preliminary Dam Consequence Classification (DCC) for the proposed TSF was developed based on the criteria outline in the 2020 Global Industry Standard on Tailings Management (GISTM, 2020). This criterion is based on the assumption that failure of the facility's outer wall would release a portion of the tailings to the environment in an uncontrolled manner. A hypothetical failure of the TSF could potentially cause incremental losses along the inundation route shown in Figure 18-10. The safety classification is based on the final envisaged tailings storage facility at final wall height of 999.5 m.

The safety/consequence classification is assigned for each individual consequence category outlined in Appendix C of the KP *Tailings Storage Facility Conceptual Design* report (refer to Appendix B of this report), and the overall dam safety classification is High. The classification can be mainly attributed to the possible significant environmental impact of the breached tailings, associated remediation time and costs, and the anticipated disruption to business.

18.12.6 Preliminary Seepage And Stability Analysis

A preliminary assessment of the stability of the dam at the final height was conducted and the Factor of Safety (FOS) targets were in accordance with the CDA (2019) guidelines summarized in the design criteria. The TSF embankment stability was evaluated for both peak static and post-peak static loading conditions. Table 18-5 summarizes the geotechnical parameters for the material properties used in the seepage / slope stability analysis and Figure 18-11 shows a typical section used for the analysis. It assumes a lined embankment and basin, earth fill starter embankment and downstream waste rock zone.

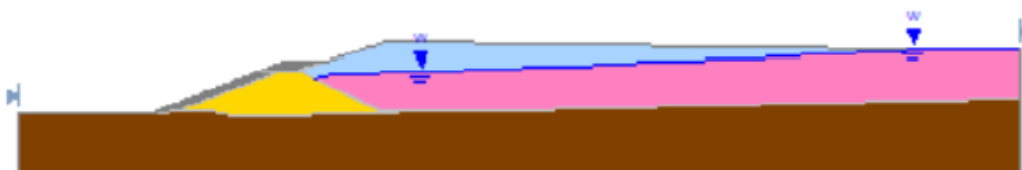


Figure 18-11 Typical Section for Seepage and Stability Analysis

Table 18-5 Seepage/Stability Analysis – Assumed Geotechnical Parameters

Colour	Material	k_s (m/s)	k_{sv}/k_{sh}	Unit Weight (kN/m ³)	ϕ' (°)	Peak undrained ratio	Post-Liq. undrained ratio
	Foundation (sands & gravel) [1]	10^{-4}	1.0	21	35	-	-
	Waste rock [2]	10^{-3}	1.0	21	36	-	-
	Compacted Rockfill [3]	10^{-6}	1.0	21	35	-	-
	Bidim-liner-GCL interface [4]	10^{-12}	impermeable	20	11	-	-
	Unsaturated tailings [5]	10^{-8}	0.8	14	26	-	-
	Saturated tailings [5]	10^{-8}	0.8	14	-	0.20	0.10

NOTES:

1. Typical values assumed from KP previous analysis.
2. Shear strengths based on LEPS (1970)
3. Values assumed from literature (Amini, 2013)
4. GCL = geosynthetic clay liner
5. Material is assumed cohesionless
6. k_s permeability coefficient, k_{sv} vertical direction k_s and k_{sh} horizontal direction k_s
7. ϕ' effective internal friction angle

The analysis results show an elevated phreatic surface which is a result of the liner system and the sloped terrain of the TSF impoundment. Investigations into mitigative measures such as blanket drains at the starter wall crest elevation should be considered. The availability and proximity of waste rock in future allow for further buttressing of the TSF embankment during the self-raising period. This is beneficial for both

stability of the facility and concurrent rehabilitation. The stability analysis model included further buttressing of the upstream self-raised embankment for improvement of FoS.

Results obtained from the coupled seepage/ stability analyses are presented in Table 18-6.

Table 18-6 Conceptual Slope Stability Model – Resulting Factor of Safety

	Static Loading (FoS > 1.5 Required)	Post-peak static Loading (FoS > 1.1 Required)
Result FoS	1.55	1.26
Reference Appendix	D	E

18.13 Stockpiles

Two (2) waste rock dump (WRD) areas will be constructed. One north west of the A2B open pit, Area 2B WRD, and one south, South WRD, of the A4 open pit to store mine rock from the open pit excavations. The rock piles will be built in 15 m lifts to provide an overall safe slope of 35 degrees. The inter-bench slopes will be at the angle of repose of the rock. Details of both WRD specifications are found in Section 16.8.

Collection ditches and contact water collection ponds/sumps will be built at topographical low points around both WRD's perimeters to collect runoff and seepage, which will then be pumped to the RW pond.

A topsoil and overburden stockpile will be established to contain stripped materials from all excavations for the project development. Sedimentation ponds will be built to settle out solids before release to the environment. A perimeter ditch will be constructed at the toe of the topsoil and overburden to keep the material intact.

The stockpiled/reclamation materials will be utilized for rehabilitation applications upon open pit closure.

18.14 Electrical Site Reticulation and Diesel Power Generation

18.14.1 Electrical Load

The predicted electrical demand load is approximately 12MW during open pit mining operations.

This estimated load is based on the current mill process-mechanical load, mill utility load, tailings management facility load, and auxiliary building load, open pit mine load, ancillary loads, and an allowance for future nominal growth / changes of auxiliary loads over time.

18.14.2 Power Generation

Diesel power generation is to be used as the primary power source during construction of the mill and mine site.

A program was recently introduced as part of the PEA for power supply called Renewable Independent Power Producer Programme (REIPPP) which is aimed at bringing additional megawatts onto the company's electricity system through private sector investment in solar, wind, biomass, and small hydro, among others.

In case of the Lofdal Mine with the following assumptions used for the case study:

- 24/7 mining operation with 9.6MW (absorbed), 12.5MW (installed)
- NamPower supply available with 5MW plus 4.6MW Diesel genset capacity
- Location: Khorixas
- Scheduled Power Supply Date: Q1/2025

In the program, a unit called embedded generator which is not directly connected to the transmission system will be connected to an end-customer's electrical equipment.

The renewable energy embedded generator (REEG) is a power plant with a tailor-made design to match the requirements and load curve characteristics of the industrial off taker. Depending on the characteristics the RE power plant, can be extended with storage and/or fossil fuel generators (e.g., HFO, Diesel gensets).

Prior to the start of production, electrical power will be sourced from the PV plant embedded generator with the following parameters:

- 12.4 MWAC PV plant (15.5 MWDC)
- Energy generation: 40 951 kWh/year (based on solar irradiation in Khorixas with 2642 kWh/m²/a)
- years Power Purchase Agreement between IPP and Namibia Critical Metals Inc. (Note: PPA tenor can be discussed and adjusted to client's requirements)
- Tariff: ca. 0.062 USD/kWh (today) with 1% p.a. escalation (Note: the tariff has been calculated based on assumptions and can vary with more detailed project assumptions as well as fine-tuning. Tariff structure can be adjusted to client's requirements)
- Optional: storage –tailor made design required to match load curve and off taker's requirements

There are power generating combinations that had been sourced for the project namely:

- The intent is to install one 1500 kW, 600V diesel generator. The generator will be installed in a substation near the mill for pre-production.
- A main power supply will be PV plant embedded generator and will be augmented by Generator set for back-up.

18.14.3 Main Substation & Site Power Distribution

The main substation will include the following equipment:

- Substation transformer skid 10 MVA 115 kV/5 kV and accessories;
- A substation E-house (modularized, assembled and tested off site) complete with 2000A power distribution center, protective relaying, generator synchronization and load shedding equipment;
- Substation P&C, synchronization and network cabinets
- 4160v switchgear

The power distribution center will distribute power to the site. Power to the mill substation will route through underground trenches, while power to the remote gate house, ancillary buildings and to the open pit mine will route through kV O/H line. The mill substation will be equipped with pad mounted step-down transformers, while power to remote loads will be stepped-down with pole mounted transformers.

18.14.4 Mill Substation

The mill substation will include two 4160/600V 1.5 MVA outdoor oil filled step-down transformers with secondary 600V 2000A power distribution centers feeding motor control centers to provide utilization voltages for mill process and utility equipment.

The power distribution centre and motor control centers will be housed in a E-house (modularized, assembled and tested off site) and will provide power to:

- Crushing, conveying loads;
- Grinding area, including DMS and ball mills;
- Floatation area;
- Reagents, thickening and filter areas;
- Tailings, reclaim water, fresh water and ancillary services.

Plant equipment utilization voltages are provided in Table 18-.

Table 18-7 Power Utilization Voltages

Plant Equipment	Voltages
All motors / VFDs including ball mills	600 volt three-phase
Small drives below 0.5 HP	120 V one-phase
Electrical heaters over 2 kW	600 V three-phase
Electrical heaters up to 1.8 kW	120 V one-phase
Lighting – LED	120 V one-phase
Small power & instrumentation	120 V one-phase
Welding receptacles	600 V three-phase

18.15 Site-Wide Communications

The mine site will employ a site-wide communications system based on a single mode fiber optic backbone. VOIP telephones, intranet/internet access, and control system network connectivity will be integrated into

this fiber backbone so that these systems can be accessible anywhere on site. Broadband internet access will be purchased from a satellite internet service provider. The corporate network (intranet) will be isolated from the control system network via a firewalled DMZ (de-militarized zone) network.

18.16 Warehouse, Offices, Facilities, and Services

Warehouse, offices, facilities, and services will include the following:

- Gate house;
- Four-bay heavy and light vehicle truck shop;
- Truck wash and lube;
- Emergency vehicle and first-aid centre;
- Warehouse/cold storage;
- Assay laboratory;
- Administration office;
- Mine dry;
- Fuel storage and dispensing;
- Process water system;
- Potable water system;
- Sanitary system;
- Water treatment;
- Fire protection;
- Waste management and disposal;
- Auxiliary equipment fleet;
- Emulsion plant and storage areas;
- Sea container;
- Generator set housing; and,
- Network of site access roads.

18.17 REE Process Facility

18.17.1 Location

At the PEA level study management of NMI decided to construct the REE process plant at the proposed processing complex.

18.17.2 Buildings

The REE Process facility will consist of:

- Ore Storage (Stockpile)
 - This area will receive the ore from the mine and stockpile it for feed into the processing plant.
- Process Plant
 - The plant will process the ore to produce the concentrate (a mixed rare earth oxide final product).
 - All reagents required will be stored within the plant facility.
 - All offices, laboratories, lunch/washrooms, and warehouse areas will be located within the plant facility.
- Finished Product Storage Area
 - This area will store the finished REE mixed rare earth oxide final product in bags or drums prior to shipment for further processing.
- Communications
 - Telephone and internet services will be available from local suppliers in the area.

19 MARKET STUDIES AND CONTRACTS

19.1 Introduction to the Rare Earth Elements

19.1.1 Rare Earth Elements: Small Market, Big Necessity

Compared to similarly abundant elements in nature, such as copper, lead, and tin, global annual production of rare earth elements is notably low.

Nevertheless, rare earth elements have become critical enablers of technologies at the heart of clean energy and mobility initiatives worldwide, as well as ubiquitous gadgetry and electronics that continue to pervade modern society.

Rare earth elements are used in small, but often necessary, amounts in hundreds of different technologies, materials, and chemicals worldwide for commercial, industrial, social, medical, and environmental applications.

In just a period of decades, rare earth elements have seeped deeply into the fabric of modern technology and industry and have proven exceptionally challenging to duplicate or replace.

19.1.2 Classification and Terminology

On the Periodic Table of Elements, rare earth elements include the lanthanide series, plus yttrium and scandium (see Figure 19-1).

Yttrium is classified as a rare earth element because of its similar ionic radius to the lanthanides, as well as its similar chemical properties, whereas scandium is classified as a rare earth element because of its tendency to concentrate into many of the same minerals.

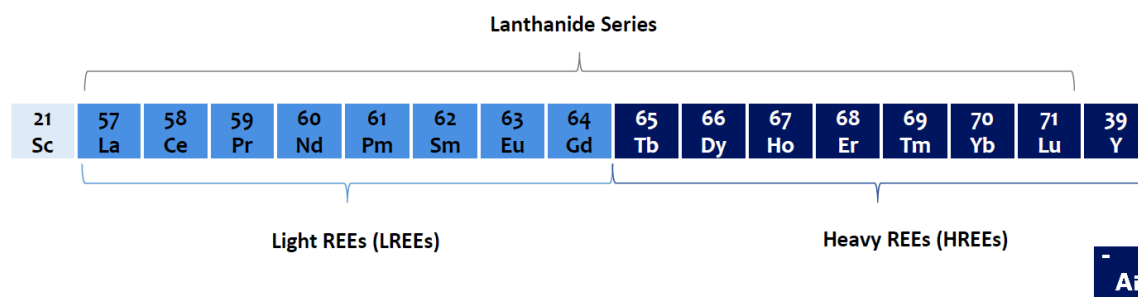


Figure 19-1 Rare Earth Elements include the Lanthanide Series plus Scandium and Yttrium

Rare earth elements are arbitrarily classified as light rare earth elements or oxides (“LREEs” or “LREOs”) or heavy rare earth elements or oxides (“HREEs” or “HREOs”) based on their electron configurations.

By virtue of having a higher crustal abundance, LREOs collectively make up over 90% of the total rare earth oxide (“TREO”) contents of a typical rare earth deposit and thereby also make up the vast majority of the world’s TREO output each year.

Heavy rare earth oxides, on the other hand, such as those found at Namibia Critical Metals’ Lofdal project, are present in the Earth’s crust in substantially lower concentrations than LREOs and as such make up a relatively small portion of the world’s TREO output each year.

19.1.3 Eight End-Use Categories

Rare earth elements are used in hundreds of unique end-uses and applications that collectively fall into one of eight end-use categories: 1.) Battery Alloys, 2.) Catalysts, 3.) Ceramics, Pigments and Glazes, 4.) Glass Polishing Powders and Additives, 5.) Metallurgy and Alloys, 6.) Permanent Magnets, 7.) Phosphors, and 8.) Other End-Uses and Applications (see Table 19-1).

Table 19-1 Rare Earth Applications and End-Uses fall into one of Eight End-Use Categories

End-Use Category	Description
Battery Alloys (La, Ce, Pr, Nd)	Rare earth elements are used to produce anode materials for nickel-metal hydride (“NiMH”) batteries. NiMH batteries are used in hybrid electric vehicles, consumer electronics, cordless shavers, cordless power tools, baby monitors and other applications of rechargeable batteries.
Catalysts (La, Ce)	Rare earth elements, such as cerium and lanthanum, are used in catalytic converters of gasoline- and diesel-powered vehicles, as well as fuel cracking catalysts and additives used by oil refiners to break down crude oil into lighter distillates, such as gasoline, diesel, kerosene and more.
Ceramics, Pigments and Glazes (La, Ce, Pr, Nd, Y)	Rare earth elements are used to produce decorative ceramics, functional ceramics, structural ceramics, bio ceramics and many other types of ceramics used in everything from jet engine coatings to ceramic cutting tools, dental crowns, ceramic capacitors, ceramic tiles, and more.
Glass Polishing Powders and Additives (Ce, La, Er, Gd, Y)	Rare earth elements, such as cerium, are used to polish optical glass, hard disk drive platters, LCD display screens and gemstones, among a long list of applications. Cerium is also used as an additive in UV-filtering glass and container glass, whereas lanthanum, yttrium and gadolinium are used to produce high quality optical glass used in camera lenses, microscopes and telescopes.
Metallurgy and Alloys (La, Ce, Ho, Gd, Y)	Rare earth mischmetal (a mixture of light REE metals) is used during production of some types of steel, as well as ductile iron making. Rare earth elements are also used to produce a variety of different alloys, such as ferro-cerium, ferro-holmium, ferro-gadolinium and a growing list of others.
Permanent Magnets (Nd, Pr, Dy, Tb, Sm)	Rare earth elements are used to produce high-strength permanent magnets that have enabled the production of ubiquitous gadgets and electronics, such as mobile phones and laptops, as well as power dense energy-efficient electric motors and generators used in electric vehicles, wind turbines, energy efficient appliances and hundreds of other applications.
Phosphors (Ce, La, Y, Tb, Eu)	Rare earth elements are used in phosphors for energy efficient lamps, display screens and avionics, and are added to fiat currency in some nations as an anti-counterfeit measure.

Other (La, Ce, Nd, Dy, Tb, Gd, Lu, Tm)	Aside from the above-described end uses and categories, rare earth elements are used in a long list of other end uses and applications, including many in defense, medicine, aerospace, agriculture, high-tech and chemical industries.
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19.1.4 Global Rare Earth Consumption in 2020

By volume, permanent magnets and catalysts were collectively responsible for more than 60% of global TREO consumption in 2020 (see Figure 19-2). However, by value, permanent magnets alone were responsible for over 90% of the total value of global TREO consumption in 2020 (see Figure 19-2) and this share is poised to expand even further as demand for (and prices of) neodymium, praseodymium, dysprosium and terbium continue to rise strongly in the years ahead.

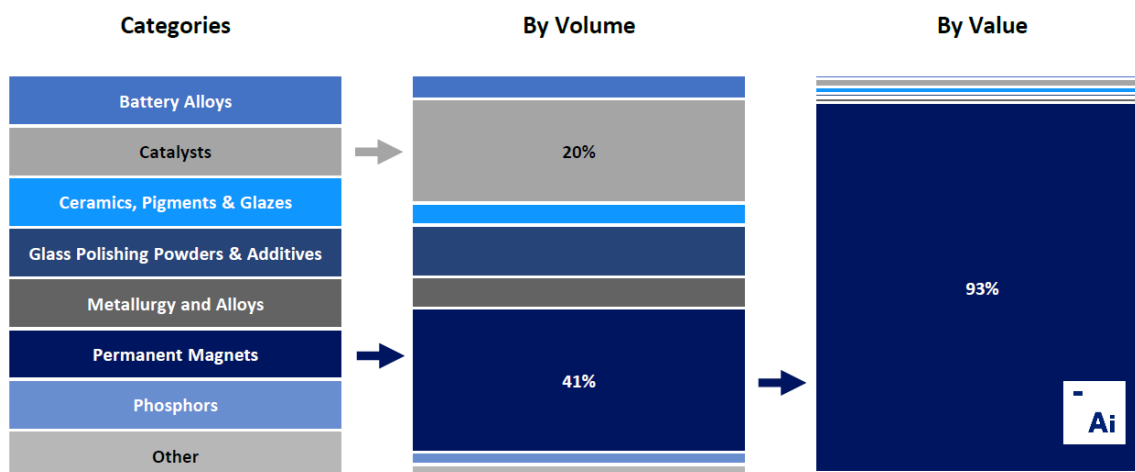
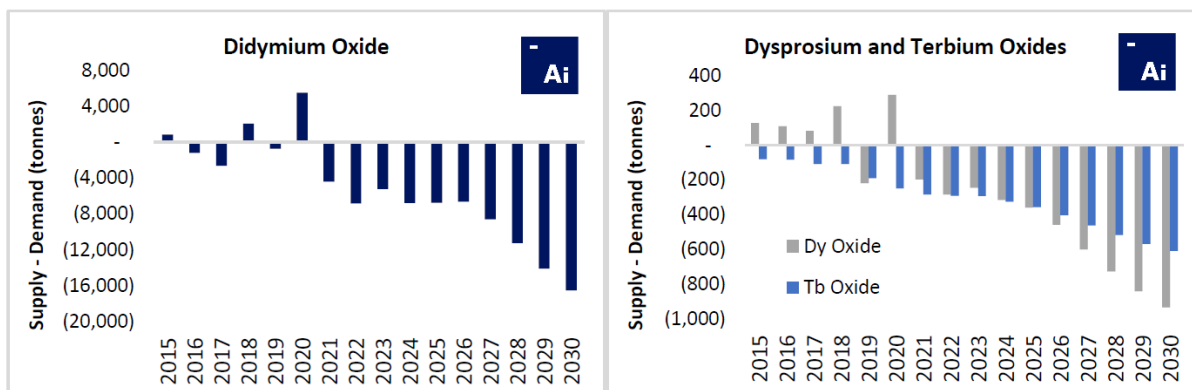


Figure 19-2 Permanent Magnets and Catalysts are the Greatest Rare Earth Demand Drivers

Not only does demand for neodymium, praseodymium, dysprosium and terbium collectively make up the vast majority of global value today, but in the years ahead demand for these four rare earth elements is projected to grow faster than demand for all other rare earth elements, increasingly challenging the ability of the supply-side to keep up.

As shown in Figure 19-3, Adamas Intelligence forecasts that global annual demand for didymium oxide*, dysprosium oxide and terbium oxide (or oxide equivalents) will substantially exceed global annual production from 2021 through 2030, leading to the depletion of historically accumulated inventories and, ultimately, shortages of these critical magnet materials if significant additional sources of supply are not developed.



* Didymium oxide is a compound of 75% Nd oxide and 25% Pr oxide. It is the main rare earth input for production of NdFeB magnets.

Figure 19-3 The Supply-Side will Struggle to Keep Up with Rising Demand for Didymium, Dysprosium and Terbium

19.1.5 Rare Earth Balance Problem

Over the past decade, rare earth producers globally have sacrificially overproduced certain low value rare earth elements, such as cerium (see Figure 19-4 – LHS), in order to keep up with rapidly growing demand for other high value rare earth elements and compounds, such as didymium (see Figure 19-4 – RHS).

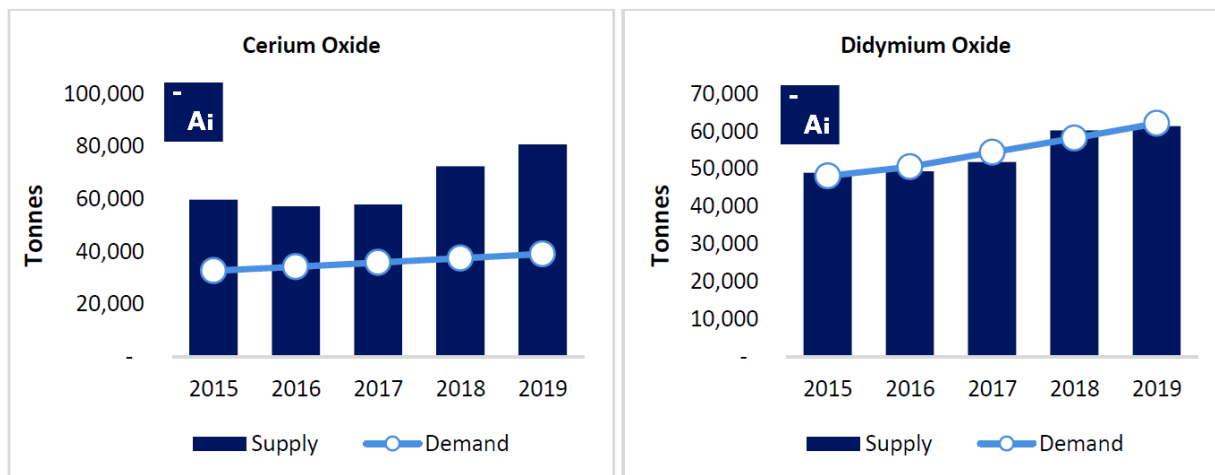


Figure 19-4 Sacrificial Over-Production of Cerium to Satisfy Rapidly Growing Demand for Didymium

Looking ahead, Adamas Intelligence forecasts that ever-increasing demand for rare earth permanent magnets will drive global demand for didymium oxide (or oxide equivalent) to towering new heights in the decade ahead (see Figure 19-5 – RHS), worsening the imbalance between production and demand of other rare earth elements, such as cerium oxide (see Figure 19-5 – LHS) if the supply-side of the industry continues on a path of business-as-usual.

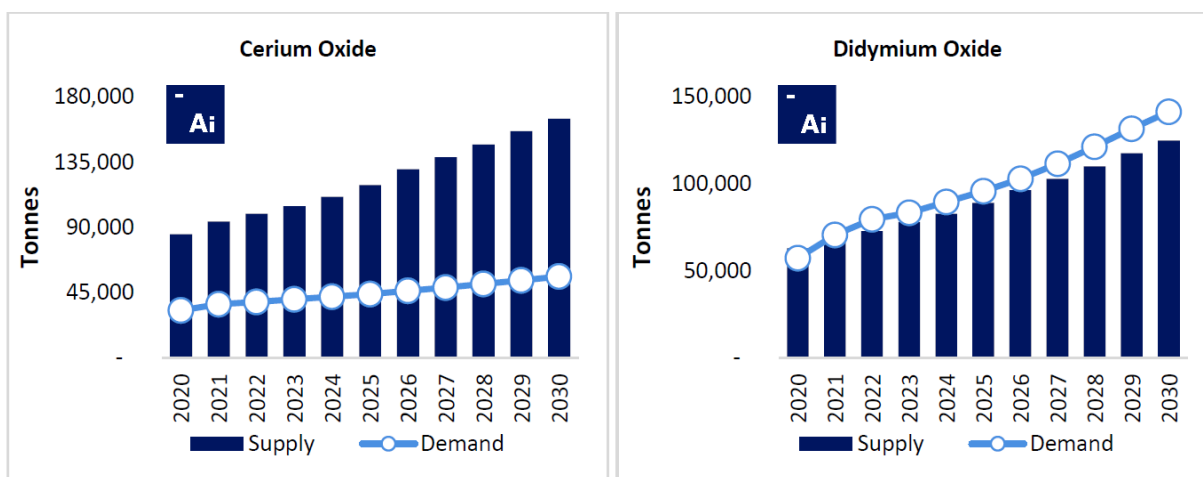


Figure 19-5 Strong Future Demand Growth for Permanent Magnets will Exacerbate the Balance Problem

19.1.6 Forecasted TREO Demand by End-Use Category

Following an estimated 9% pandemic-induced drop in global TREO consumption in 2020, Adamas Intelligence forecasts that global demand for most end-use categories will rebound strongly in 2021 and 2022 and thereafter grow steadily through the end of the decade and beyond (see Figure 19-6).

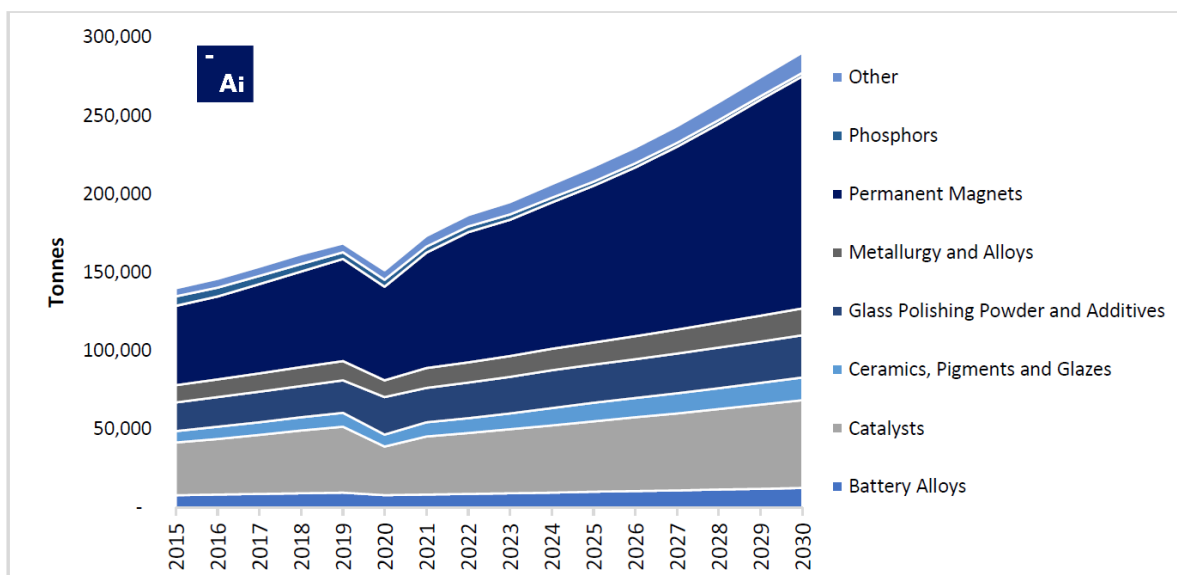
From approximately 62,000 tonnes in 2020, Adamas forecasts that global TREO demand for permanent magnets will increase at a CAGR of 9% to approximately 148,000 tonnes in 2030, driven ever-higher by rapidly growing demand for NdFeB magnets in EV traction motors, wind power generators, consumer appliances and many other end-uses and applications (see Figure 19-6).

Similarly, over the same period Adamas forecasts that global TREO demand for catalysts will increase at a CAGR of 6%, from approximately 31,000 tonnes in 2020 to 56,000 tonnes in 2030, while global TREO demand for ceramics, pigments and glazes increases at a CAGR of 7%, from 7,600 tonnes in 2020 to 14,400 tonnes in 2030 (see Figure 19-6).

Moreover, Adamas forecasts that global TREO demand for metallurgy and alloys will increase at a CAGR of 6%, from 10,500 tonnes in 2020 to 17,200 tonnes in 2030, while global TREO demand for battery alloys increases at a CAGR of 5%, from 7,850 tonnes in 2020 to 12,400 tonnes in 2030 (see Figure 19-6).

Lastly, from 2020 through 2030, Adamas forecasts that global TREO demand for phosphors will decrease at a CAGR of -7%, from 4,300 tonnes to 2,100 tonnes, while global TREO demand for “other” end-uses and applications, including many medical, chemical and defense-related applications, increases at a CAGR of 10%, from 6,000 tonnes to 12,950 tonnes (see Figure 19-6).

In the years ahead, the rapid TREO demand growth expected for permanent magnets will lead the end-use category to continuously absorb market share from incumbent categories. By 2030, Adamas projects that permanent magnets will drive 51% of global TREO demand by volume and over 90% of the market’s value each year.



* Numbers may not compute exactly as described due to rounding.

Figure 19-6 Historical Global Consumption and Forecasted Demand for Rare Earth Oxides by End-Use Category

19.1.7 Forecasted TREO Demand for Permanent Magnets by End-Use Category

Following an estimated 5% drop in global annual TREO consumption for permanent magnets in 2020 to approximately 62,000 tonnes, Adamas Intelligence projects that demand will rebound sharply in 2021 (up 18% year-over-year) and 2022 (up another 13% year-over-year) and thereafter increase at a CAGR of 8% through 2030 to reach 148,000 tonnes on the back of strong demand growth in virtually all magnet-related end-use categories (see Figure 19-7).

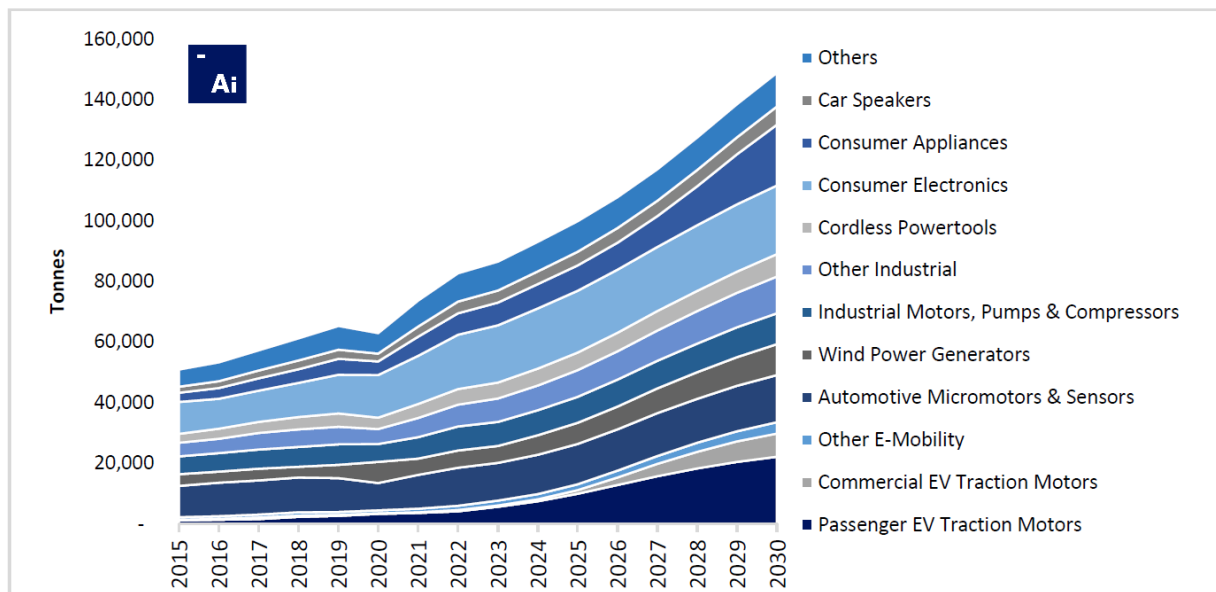
From 2020 through 2030, Adamas expects the greatest demand growth to come from commercial EV traction motors (41% CAGR), passenger EV traction motors (21% CAGR) and “other e-mobility” applications (15% CAGR), including electric bicycles, scooters, mopeds, quadricycles, motorcycles, and low-speed passenger EVs.

Conversely, end-use categories expected to underperform the group average include consumer electronics (5% CAGR), industrial motors, pumps and generators (6% CAGR) and automotive micromotors and sensors (6% CAGR), among others.

Adamas projects that global TREO demand for permanent magnets used in consumer appliances, such as air conditioners and washing machines, will increase at a CAGR of 16% from 2020 through 2030, and TREO demand for permanent magnets used in wind power generators will increase at a CAGR of 4% (see Figure 19-7).

Over the same period, Adamas forecasts that global TREO demand for permanent magnets used in car speakers will increase at a CAGR of 9%, and TREO demand for permanent magnets used in cordless power tools, such as drills, saws, sanders, vacuums, blowers, mowers and trimmers, will increase at a CAGR of 7%.

Lastly, Adamas forecasts that global TREO demand for permanent magnets used in “other industrial” applications, including industrial robots, elevators, escalators and more, will increase at a CAGR of 9% from 2020 through 2030 and demand for all “other” end-uses and applications, including aerospace, defense and medical, will increase at a CAGR of 5% over the same period (see Figure 19-7).



* Numbers may not compute exactly as described due to rounding.

Figure 19-7 Historical Global Consumption and Forecasted TREO Demand for NdFeB Magnets by End-Use Category

19.1.8 EVs to Drive 25% of Global Magnet Earth Oxide Demand by 2030

By 2030, Adamas expects that passenger EV traction motors, commercial EV traction motors and other e-mobility types, such as electric bicycles, motorcycles, and scooters, will collectively be responsible for 23% of total global demand for magnet rare earth oxides. Considering additional uses of NdFeB magnets in EVs, including micromotors, sensors and loudspeakers, Adamas conservatively forecasts that EVs will drive one quarter of total global magnet rare earth oxide demand annually by 2030 (see Figure 19-8 – dark blue bars).

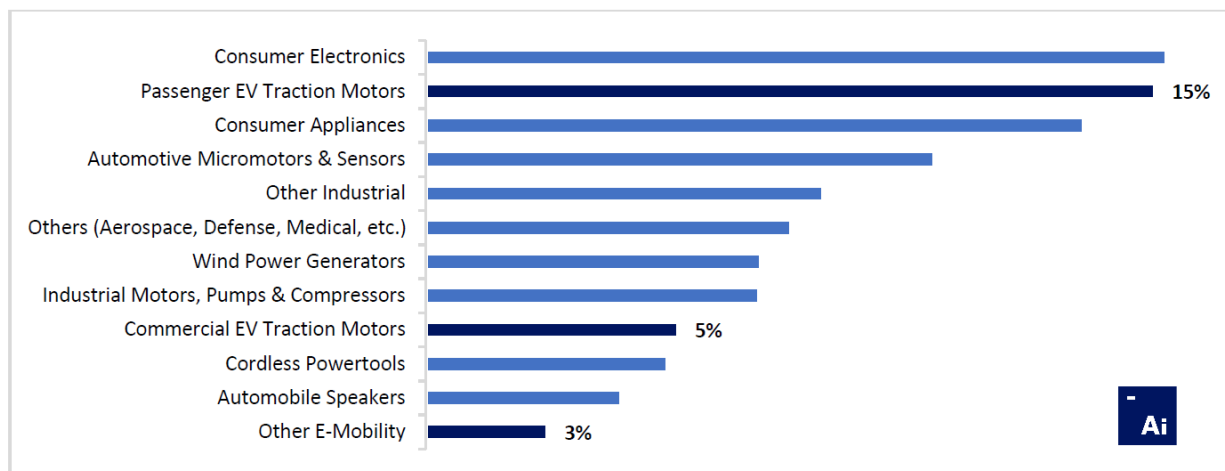


Figure 19-8 Forecasted Breakdown of Magnet Rare Earth Oxide Demand by End-Use Category in 2030

Moreover, given that EV traction motors and generators tend to use high-temperature-performance grades of NdFeB magnets that contain elevated levels of the heavy rare earth elements dysprosium and terbium, Adamas forecasts that EV traction motors, and to a smaller extent micromotors and sensors, will drive nearly half of total global dysprosium and terbium oxide demand annually by 2030 (see Figure 19-9 – dark blue bars).

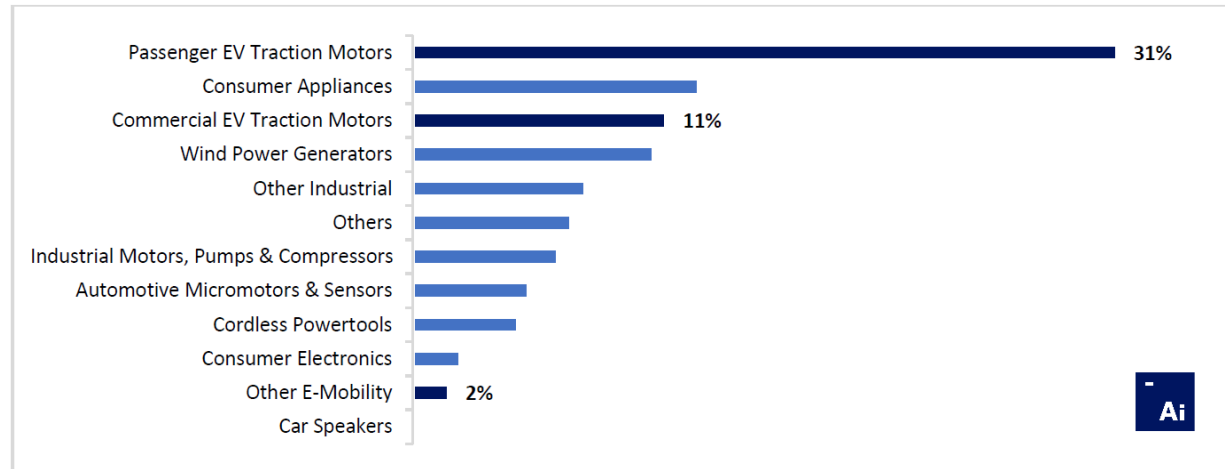


Figure 19-9 Forecasted Breakdown of Dysprosium and Terbium Oxide Demand by End-Use Category in 2030

19.1.9 Forecasted Supply – Demand Balance for NdPr, Dy and Tb to 2030

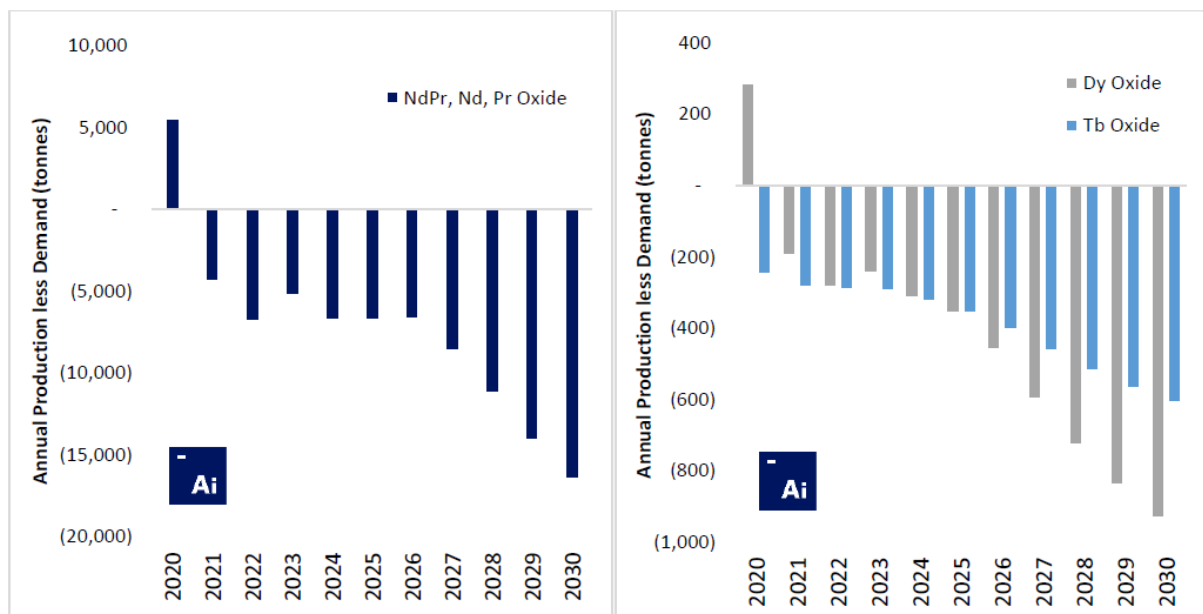
In 2020, stemming from a coronavirus-induced drop in global NdFeB magnet consumption, Adamas Intelligence estimates that global production of didymium, neodymium and praseodymium oxides (and

oxide equivalents), combined, exceeded global consumption by approximately 5,500 tonnes (see Figure 11 – LHS) after a relatively tight year in 2019 that resulted in the partial drawdown of historically accumulated inventories in China.

However, with the ongoing re-opening of key demand markets in Asia, Europe and North America in 2021, Adamas expects demand for didymium, neodymium and praseodymium oxides (and oxide equivalents), combined, to bounce back strongly in 2021, and from 2021 through 2030 forecasts that the global market will consistently underproduce these critical magnet materials (see Figure 19-10) resulting in the depletion of historically accumulated inventories by as early as 2022.

Similarly, in 2020, Adamas estimates that global production of dysprosium oxide (and oxide equivalent) exceeded global demand by almost 300 tonnes after a relatively tight year in 2019 that saw the partial drawdown of historically accumulated inventories in China. However, with the ongoing re-opening of key demand markets in 2021 and limited availability of the alternative heavy rare earth, terbium, in China, Adamas expects demand for dysprosium oxide (and oxide equivalent) to bounce back strongly in 2021, and from 2021 through 2030 forecasts that the global market will consistently underproduce dysprosium oxide (and oxide equivalent) (see Figure 19-10), resulting in the depletion of historically accumulated inventories by as early as 2022.

Conversely, in 2020, Adamas estimates that global consumption of terbium oxide (and oxide equivalent) exceeded global production by a staggering 250 tonnes resulting in the depletion of virtually all historically accumulated inventories in China. In 2021, Adamas expects terbium shortages will translate into strong demand growth for dysprosium oxide (the alternative heavy rare earth of choice for magnet alloy makers) and from 2021 through 2030 forecasts that the global market will consistently underproduce terbium oxide (and oxide equivalent) (see Figure 19-10).



Note: China, the world's only source of separated heavy rare earth oxides, looks to mines in Myanmar for 30-40% of its heavy rare earth feedstock supplies each year. With a coup ongoing in Myanmar through Q1 2021, the future of these critical supplies is at risk, in which case the above-projected shortages would be greater and sooner than expected.

Figure 19-10 Forecasted Global Production – Demand Balance for Magnet Rare Earth Oxides

Table 19-2 below outlines some of the main end-uses and applications of heavy rare earth elements. Despite making up a relatively small share of the global rare earth market overall, heavy rare earth elements have become increasingly critical to a wide range of applications related to electric mobility, clean energy, medicine, defense and other cutting edge high-tech fields.

Table 19-2 Main End-Uses and Applications of Heavy Rare Earth Elements

HREO	Main Applications
Tb	High-temperature-performance permanent magnets for EV traction motors, wind power generators, industrial motors, and other applications. Also used in phosphors for lighting and in specialty alloys.
Dy	High-temperature-performance permanent magnets for EV traction motors, wind power generators, industrial motors, and other applications. Also used in specialty alloys.
Ho	Used as an additive in permanent magnet alloys and as a flux concentrator for high magnetic fields. Also used as a glass and gemstone additive, as a dopant in solid state laser crystals, and as neutron absorbers in nuclear reactors.
Er	Used as an additive in decorative glass, optical glass, and photographic filters, as well as fiber optic amplifiers and laser crystals. Is also used in alloys to decrease hardness and, like holmium, is used as a neutron absorber in nuclear reactors.
Tm	Used in portable X-ray machines, laser crystals and ceramic magnetic materials for microwave equipment. Also used in high-temperature superconductors, arc lighting, personal radiation dosimeters and anti-counterfeiting inks.
Yb	Used in portable X-ray machines, laser crystals and high-stability atomic clocks. Also used in quantum computing, in stress gauges for earthquakes and explosions, and experimentally in solar panels and decoy flares for defense applications.
Lu	Used in positron emission tomography (PET) scanners, cancer therapy and for dating meteorites. Also used for in catalysts for hydrocarbon cracking, in magnetic bubble memory devices and in phosphors for LED lamps and X-rays.
Y	Used in phosphors for lighting and backlighting. Used in a wide array of ceramics and alloys. Used as a host in laser crystals of all kinds, as well as microwave filters for radar systems, polymerization catalysts and optical glass.

19.1.10 Current Market Conditions used for the current PEA

Current market conditions indicate low pricing for lanthanum and cerium products, thus no recovered value was assigned to these products.

Other prices used for this PEA are derived from recent market analysis and other published NI 43-101 complaint resource reports.

Table 19-3 presents the current market prices used for the current report.

Table 19-3 Current Market Prices used for the Current PEA in USD/Kg

Pricing Forecast for REE	Pricing Used for Analysis	Distribution
La ₂ O ₃	\$-	9.2%
Ce ₂ O ₃	\$-	16.0%
Pr ₂ O ₃	\$201.00	1.7%
Nd ₂ O ₃	\$212.00	6.3%
Sm ₂ O ₃	\$5.00	2.2%
Eu ₂ O ₃	\$36.00	1.1%
Gd ₂ O ₃	\$109.00	4.3%
Tb ₂ O ₃	\$2,493.00	0.9%
Dy ₂ O ₃	\$587.00	6.2%
Ho ₂ O ₃	\$290.00	1.3%
Er ₂ O ₃	\$64.00	3.8%
Yb ₂ O ₃	\$20.00	3.5%
Lu ₂ O ₃	\$947.00	0.5%
Y ₂ O ₃	\$17.00	42.4%
Tm ₂ O ₃	\$500.00	0.6%
Average Basket Value	\$103.64	
Realized Basket Price after Treatment Charges	\$91.64	

19.1.11 Wanted: Alternative Sources of Heavy Rare Earth Supply

China is responsible for approximately 85% of total refined rare earth oxide supply globally each year. In the case of heavy rare earth oxides, however, China dominates virtually 100% of global annual supply. In 2020, China's rare earth industry struggled to keep up with rapidly growing demand for magnet rare earth oxides in the second half of the year, fostering conditions that sent prices of terbium, dysprosium, neodymium and didymium oxide to multi-year highs.

With a steady reduction in unregulated production in China since 2015, and a push to preserve domestic heavy rare earth resources in the nation, China's processing and magnet industries have become precariously reliant on neighboring Myanmar for heavy rare earth concentrate supplies in recent years, adding a new layer of complexity and risk to an already untenable situation for heavy rare earths.

Going forward, with the high-temperature-performance ("HTP") segment of the permanent magnet market (i.e., the dysprosium and terbium consuming segment) expected to grow substantially faster than the permanent magnet market as a whole, Adamas Intelligence expects that in the years ahead magnet makers in China will need to decide between exporting to foreign markets or selling domestically because there simply will not be enough HTP alloy available to meet global demand.

In the U.S., for example, the push to fully electrify the nation's vehicle production by 2030 will create new demand for upwards of 20,000 tonnes of NdFeB magnets and 700 tonnes of dysprosium and terbium oxide annually – more than three times planned annual production at Lofdal.

While a number of companies in the U.S., Europe and Japan are actively working to establish alternative mine-to-magnet supply chains, critical to the sustainability and success of all these efforts will be the necessary development of heavy rare earth oxide, metal and alloy supplies to enable domestic production of HTP magnets without reliance on China.

19.1.12 Marketability of Thulium, Ytterbium and Lutetium

Compared to other heavy rare earths, such as dysprosium and yttrium, global demand markets for thulium, ytterbium and lutetium are relatively small and pervasively oversupplied by a handful of producers in China. In the case of thulium oxide, Adamas Intelligence estimates that global consumption each year is less than 50 tonnes. Thulium spot prices are generally unavailable thus Adamas has assigned a value of zero for each year in the forecast period.

In the case of ytterbium oxide, Adamas Intelligence estimates that current global consumption amounts to approximately 500 tonnes annually, of which 70% to 80% is consumed in China and 20% to 30% outside of China.

Despite ample supplies in China, Adamas believes that end-users outside of China would welcome an alternative source of ytterbium supply, thus it is reasonable that Namibia Critical Metals could sell 75% to 100% of its ytterbium output annually.

In the case of lutetium oxide, Adamas Intelligence estimates that current global consumption amounts to approximately 100 tonnes annually, of which 70% to 80% is consumed in China and 20% to 30% outside of China.

Despite plentiful supplies in China, Adamas believes that, as with ytterbium, end-users outside of China would welcome an alternative source of lutetium supply, thus it is reasonable that Namibia Critical Metals could sell 75% to 100% of its ytterbium output annually.

19.1.13 Conclusions

- From 2021 through 2030, Adamas forecasts that the global rare earth industry will consistently underproduce neodymium, praseodymium, dysprosium and terbium oxides (or oxide equivalents), resulting in the depletion of historically accumulated inventories and, ultimately, shortages of these critical magnet materials if supply is not increased beyond levels currently anticipated.
- The Lofdal project offers strong economic exposure to the rare earth permanent magnet sector, which is the fastest-growing end-use category and most in need of additional rare earth supplies, according to Adamas Intelligence.
- The Lofdal project also offers strong economic exposure to the heavy rare earth sector, which is currently 100% dominated by China and neighboring Myanmar and is in dire need of alternative sources of supply.
- From a marketing, logistics and economic standpoint, the high proportion of valuable magnet rare earth elements in Lofdal's prospective TREO production means that a future mine (with separation) could generate approximately 80% of its revenues from just 11% of its production volume.
- From 2021 through 2030, Adamas Intelligence envisions three distinct scenarios unfolding for rare earth oxide prices.
- Scenario 1 considers a future in which current sentiment- and speculation-driven prices persist in the near-term with minimal downward impact on prices from supply increases in China, Australia/Malaysia, the U.S. and elsewhere.
- Scenario 2 considers a future in which near-term supply increases drag rare earth oxide prices down moderately, albeit prices remain substantially higher than in 2020 resulting from ongoing Myanmar-related uncertainty and speculation.
- Lastly, Scenario 3 considers a future in which near-term supply increases, coupled with reduced uncertainty and speculation related to Myanmar, substantially undermine rare earth oxide prices.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

William van Breugel P.Eng. (SGS) has relied upon SLR Consulting (Namibia) (Pty) Ltd (SLR), who completed an independent analysis of permitting and environmental requirements for Item 20.

20.1 Introduction

Namibia Critical Metals Inc. (NMI) was originally named Namibia Rare Earths Incorporated (NRE), a wholly owned subsidiary company of Namibia Rare Earths (Pty) Ltd). NMI holds an Exclusive Prospecting License (EPL) 3400, approximately 25 km north-west of the town of Khorixas in the Kunene Region of Namibia. EPL 3400 includes the farm Lofdal and falls within the Huab and Doro !Nawas Conservancy areas (refer to Figure 20-1).

Exploration activities commenced in 2006. In 2016 NMI started the process to obtain an Environmental Clearance Certificate (ECC) for an open-pit mine and processing plant (and the associated access road to the mine) for the production of a concentrate consisting mainly of 'rare earths. The principal rare earth mineral of economic interest is xenotime. In addition to the of the mining infrastructure there was also the development of additional electricity and water supply infrastructure.

NMI appointed SLR Consulting (Namibia) (Pty) Ltd (SLR) to undertake the Environmental Impact Assessment (and related processes) for the following:

- Environmental Impact Assessment Report for the Proposed Lofdal Rare Earths Mining Project (SLR, 2016a);
- Environmental Impact Assessment Report for the Water Supply Pipeline of the Proposed Lofdal Rare Earths Mining Project (SLR, 2016b); and
- Summary of the Environmental Impact Assessment Report and Specialist Studies for the Lofdal Mining Project (SLR, 2016c).

NMI is now proposing to approximately double the production rate of the mine and processing plant to 10 440 000 tons/annum, this will require a Preliminary Economic Assessment (PEA) to be submitted to the relevant authority. SLR has been appointed by NMI to summarise the 2016 EIA as regulated by the Ministry of Environment and Tourism (MET) in terms of the Environmental Management Act 7, of 2007.

20.1.1 Project Motivation

The motivation for the original Lofdal Rare Earths Mining Project (Lofdal Project) is economic and strategic in nature. The Lofdal deposit has one of the highest levels of heavy rare earth enrichment in the world. The project has the potential to benefit the country, society, and surrounding communities both directly and indirectly. Direct economic benefits will be derived from wages, taxes, and profits. Indirect economic benefits will be derived from the procurement of goods and services and the increased spending power of employees through the creation of new jobs at the mine.

Figure 20-1 below shows the Mining Licence (ML) and EPL 3400.

20.1.2 Status of Authorisations

The Environmental Clearance Certificate (ECC) was approved on December 8, 2017 for both the Mining License and the linear infrastructure by the Ministry of Environment and Tourism (MET). The clearance was

deemed valid for a period of 3 years from the date of issue. This clearance was issued on the basis of the SLR Consulting EIA and EMP Report of 2016. This report is a summary of the EIA and EMP. Currently NMI is in the process of renewing the ECCs.

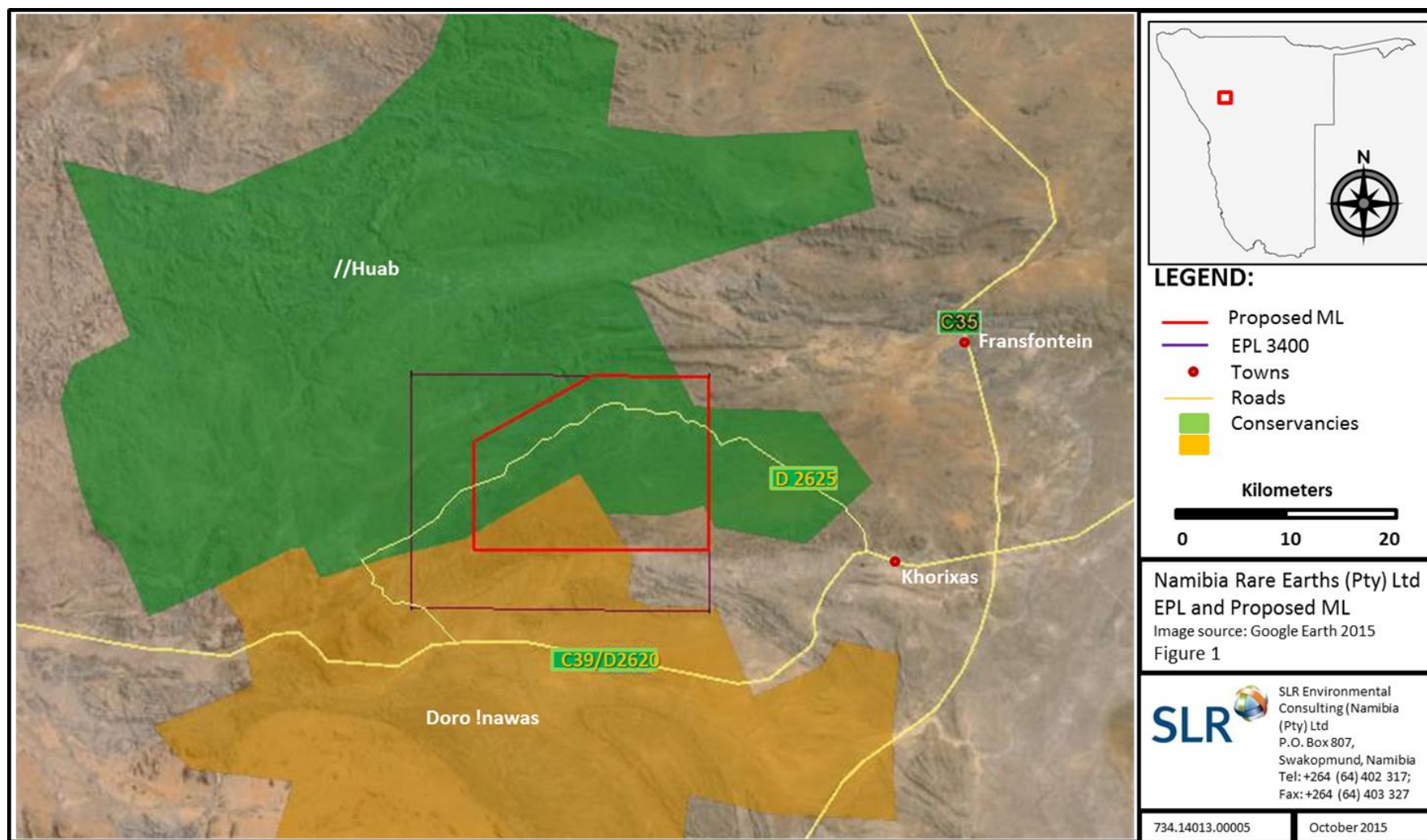


Figure 20-1 Locality Map Indicating Regional and Local Setting of the EPL and the Mining License (ML)

20.2 Legal Framework

EIAs are regulated by the Ministry of Environment and Tourism (MET) in terms of the Environmental Management Act (EMA) (No 7 of 2007). The regulation identifies listed activities which may detrimentally affect the environment. The listed activities trigger the requirement for an EIA and issuance of an Environmental Clearance Certificate (ECC) from the Ministry of Environment and Tourism (MET) prior to any activities commencing. The following activities in the regulations were identified as part of the original Lofdal Project:

- Energy generation, transmission, and storage activities;
- Waste management, treatment, handling, and disposal activities;
- Mining and quarrying activities;
- Forestry activities;
- Water resource development;
- Hazardous substance treatment, handling and storage; and
- Infrastructure

20.2.1 Environmental Impact Assessment Process

NMI applied for a ML from the Ministry of Mines and Energy (MME) before the development of the mine and processing plant. MME required an EIA to be undertaken so that an ECC could be obtained. EIAs are regulated by the Ministry of Environment and Tourism (MET) in terms of the EMA. The EIA process for all the Lofdal project components were conducted in parallel, however two separate EIA Reports were developed for each of these. The EIA approach included the following:

- The scoping process was conducted to identify the environmental issues associated with the project and to define the terms of reference for the required specialist studies and the EIA;
- Specialist studies were commissioned in accordance with the relevant terms of reference;
- The specialists were selected on the basis of their expertise and knowledge of the project area;
- The EIA report was compiled on the basis of the findings of the specialist studies;
- The Lofdal Environmental Management Programme (EMP) was elaborated on the mitigation objectives and included additional actions that were described in the EIA report; and
- A project specific public participation process was conducted. As part of this process the regulatory authorities and interested and affected parties (I&APs) were given the opportunity to attend information sharing meetings, submit questions and comments to the project team, and review the background information document and scoping report. All questions and comments that were raised by the authorities and I&APs were included and addressed in the EIA report.

20.3 Project Overview

The project was comprised of the development of an open-pit mine and processing plant at Lofdal that produces a concentrate consisting mainly of 'rare earths'. At a high level the two different main site

infrastructure layout options (referred to as 'option 1' and 'option 2') shown in Figure 20-2 and Figure 20-3 respectively were considered and assessed in the EIA process.

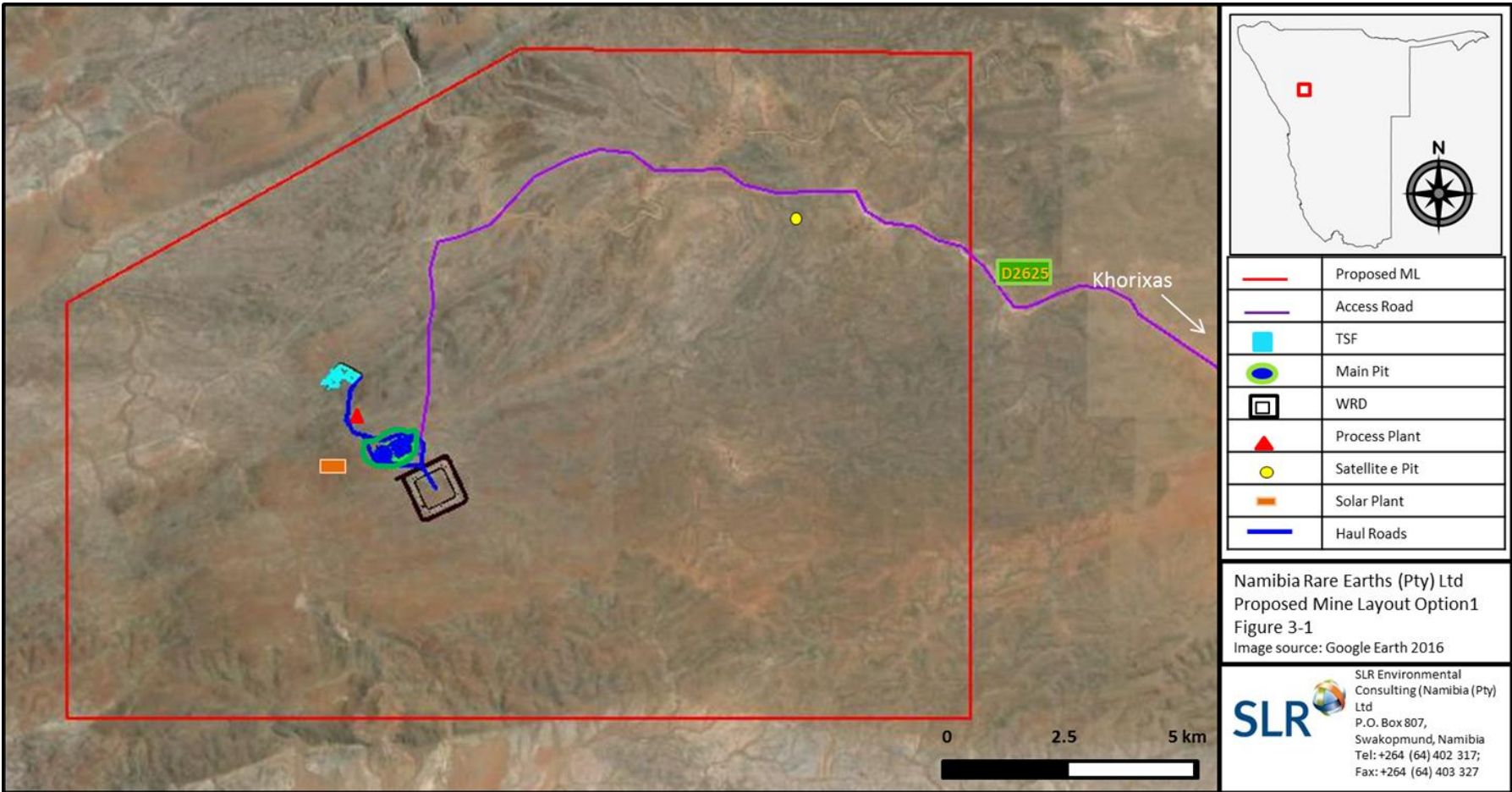


Figure 20-2 Lofdal Site Layout Option 1 (SLR EIA Report, 2016)

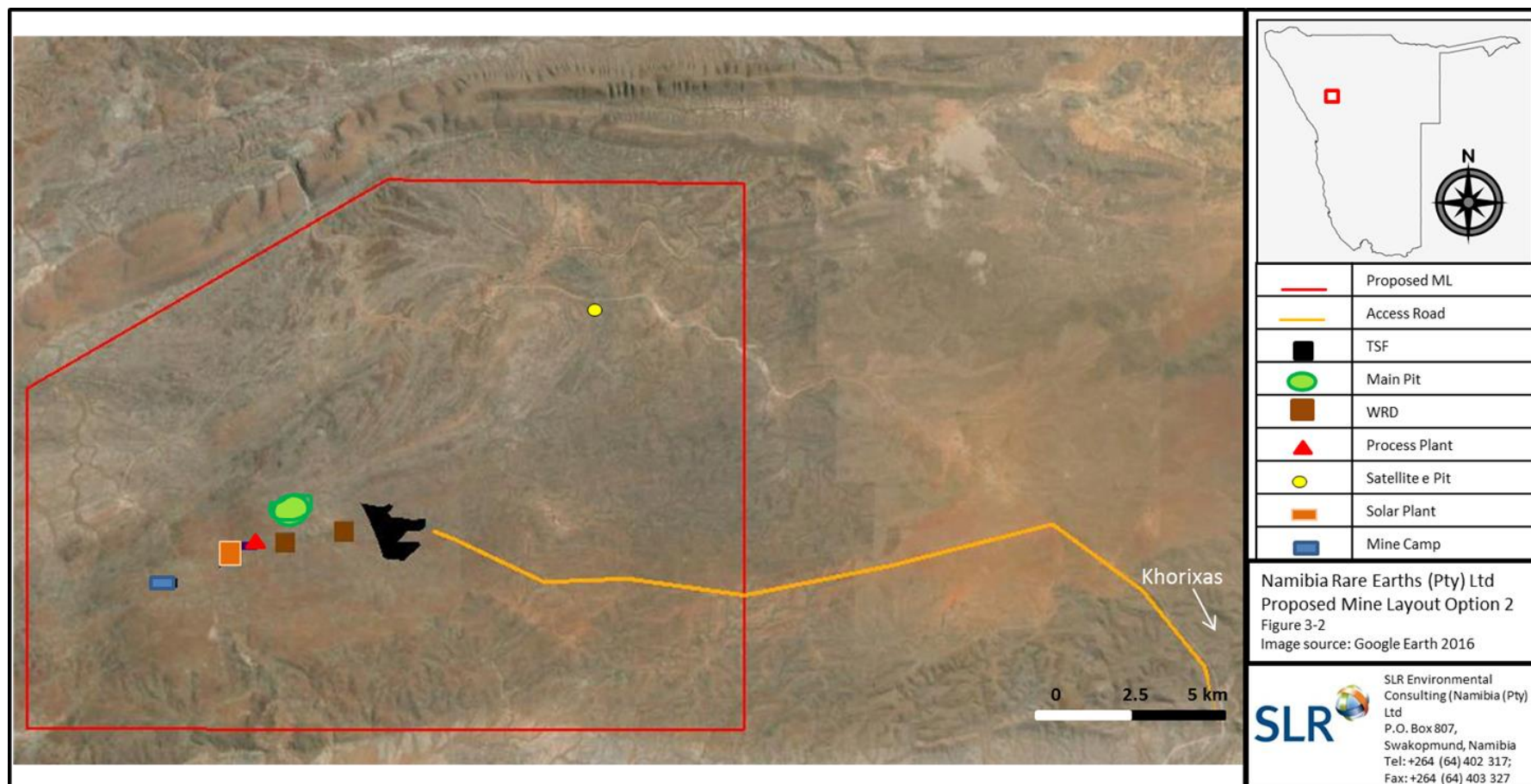


Figure 20-3 Lofdal Site Layout Option 2 (Preferred Layout from an Environmental Impacts Perspective) (SLR EIA Report, 2016)

Table 20-1 was included in EIA (SLR EIA Report, 2016) providing a summary of data for the original project.

Table 20-1 Project Data Summary that Provides Perspective on the Scale of the Project

Group	Specific	Details
Mining	Target mineral	Rare Earths - mixed rare earth oxide final product
	Mineable area	Main pit: ± 0.4 km ² (40 hectares) Smaller satellite pits up to 10 km away within the ML area may be developed in the future
	Depth of the minerals below surface	Occur at surface and will be mined to a depth of 200 m
	Rate	10,440,000 tons per year (total) 840,000 tons per year to be sent to the crusher
	Life of mine	Current resource allows for approximately 7 years; however, this could be extended by developing additional resources
	Extend of areas required for infrastructure	The processing plant, pit, tailings storage facility (TSF), waste rock dump (WRD) and solar plant (and other onsite infrastructure) would cover about 20 km ² (2,000 ha).
Mine residues	Waste rock	Waste rock (i.e. mine material that does not contain rare earths to be processed) will be stockpiled on a WRD. Two options for the WRD are being considered. The total extent of the WRD (whether it is one or two dumps) will be approximately 1,000 m x 1,000 m and 50 m high.
Processing	Plant	Mill Process Plant
	Rate	70,000 tons per month ore received (the plant design capacity would be 75,000 tons per month). Produce 16,000 tons per year concentrate (after the Gangue Acid Leach (GAL) stage).
Processing residues	Tailings	About 54% of the mined ore after leaching will be waste product (i.e. tailings) to be disposed of at a TSF. Two different locations for the TSF are considered. Depending the chosen alternative location, the size of the TSF (i.e. waste storage area) will be between 342,000 m ² and 1,159,000 m ² as a result of the topographical difference of the two sites.
Resource use	Water demand	Approximately 1,000,000 m ³ per year
	Power demand	± 21,178,700 kWh per annum
Employment	Staff: construction	Approximately 300 to 340
	Staff: operational	Approximately 226
	Operating times	24 hours a day, 7 days a week

A conceptual diagram of the proposed Lofdal mining process (i.e. the main facilities) is provided in Figure 20-4. This was originally included in the Mining Project EIA Report.

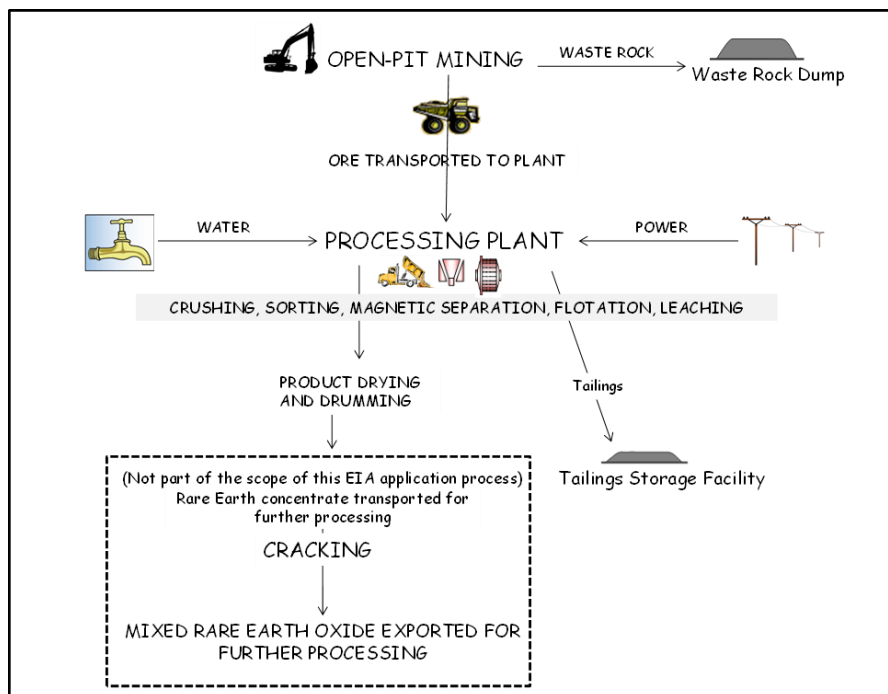


Figure 20-4 Conceptual Diagram of the Mining Process (SLR EIA Report, 2016)

20.4 Environmental Impact Assessment Findings

Various baseline studies and / or relevant site monitoring activities have been conducted at the Lofdal site from the beginning of 2015 (i.e. before the commencement of the formal EIA application and scoping process) until submission of the EIA Report in June 2016. The baseline studies were detailed in the full EIA documents compiled for the mining operations as well as the electricity and water pipeline infrastructure.

This section provides a high-level overview of the different environmental aspects and impacts that were detailed in the EIA (SLR Consulting, 2016). The conceptual discussion of the management/mitigation measures, for each aspect, is provided and detailed in the EMP for each aspect of the project, namely the Mining Project, the Pipeline and the Powerline.

20.4.1 Topography

The Lofdal Mine is in a fairly remote area, with people (with livestock) staying on farms in relatively close proximity to the Project Area. The general area is also (infrequently) visited by third parties for trophy hunting and other tourist activities. It is therefore possible that curious third parties can access the site from various directions per foot; or existing tracks previously used for explorations activities in the area. In the unmitigated scenario, it was stated that it was possible that excavations and infrastructure presented a risk to unaccompanied third parties during all phases. The detailed mitigation measures focused on infrastructure safety and on limiting access to third parties and animals reducing the probability of the impact occurring.

20.4.2 Soil – Potential Loss Of Soil Resources From Pollution and/or Physical Disturbance

The area is characterised by lithic leptosols. Lithic leptosols (lithic refers to very thin or shallow soils) are represented by thin layers of coarse-textured soils often containing much gravel. They are characterised by a low water holding capacity, often resulting in high rates of runoff. However, soils are a significant component of most ecosystems, even if only rudimentary developed. As an ecological driver, soil is the medium in which most vegetation grows and a range of vertebrates and invertebrates exist. In the context of mining, soil is even more significant if one considers that mining is a temporary land use where rehabilitation (using soil) is the key to re-establishing post closure land capability that will support post closure land uses.

In the unmitigated scenario it was stated that there are a number of activities that would disturb and potentially damage the soils through physical disturbance and/or pollution. Key measures to reduce the impacts that were included in the report include the following:

- Limit the disturbance of soils to what is absolutely necessary both in terms of site clearing and in terms of ongoing project development and use of vehicles.
- Pollution prevention through basic infrastructure design and through education and training of workers.
- Implementation of procedures to enable fast reaction to contain and remediate spills.
- Stripping, storing, and maintaining soils for re-use during the rehabilitation and restoration process.

20.4.3 Blasting

Blasting is associated with the following pathways that can injure third parties and/or damage structures: fly rock, vibration, and air blasts. Given the remote setting of the mine the probability of any blast related impacts is low. This assumed that residents/households staying at Oas Post 3 would be relocated.

It was stated that the issue would require mitigation because the consequences associated with this impact type are potentially significant.

The blast design, implementation and monitoring included in the reports put forward the following measures:

- Fly rock is contained within a maximum of 500m of the blast site;
- Prior to each blast the blast area will be cleared of third parties to a safe distance determined by appropriate legislation and safe working procedures. Prior to each blast an audible warning will be sounded;
- Ground vibration at the closest third-party structures is less than 12mm/s peak particle velocity;
- Air blast at the closest third-party structures is less than 115dB; and
- All registered complaints will be documented, investigated and efforts made to address the area of concern where possible.

20.4.4 Air Quality

It was stated that there were a number of activities in all phases that would have the potential to pollute the air. In the construction and decommissioning phases these activities were considered to be temporary in

nature. The operational phase would present more long-term activities and the closure phase would present final landforms that may have the potential to pollute the air through long term wind erosion.

The air quality specialist used dispersion modelling to predict spatial air quality concentrations for the operational phase. Dispersion modelling for the construction and decommissioning phases were considered to be unrepresentative of the actual activities that will result in dust and gaseous emissions, due to the overly conservative emission rate calculation. Both mine layout options were considered as well as the access road. It was noted that the sensitive receptors (i.e. residents/households staying at Oas Post 3 refer to Figure 20-5) who were close to the proposed mining site (in the location of the proposed WRD) were included as part of this assessment.

The below figure was available In the Mining Project EIA Report.

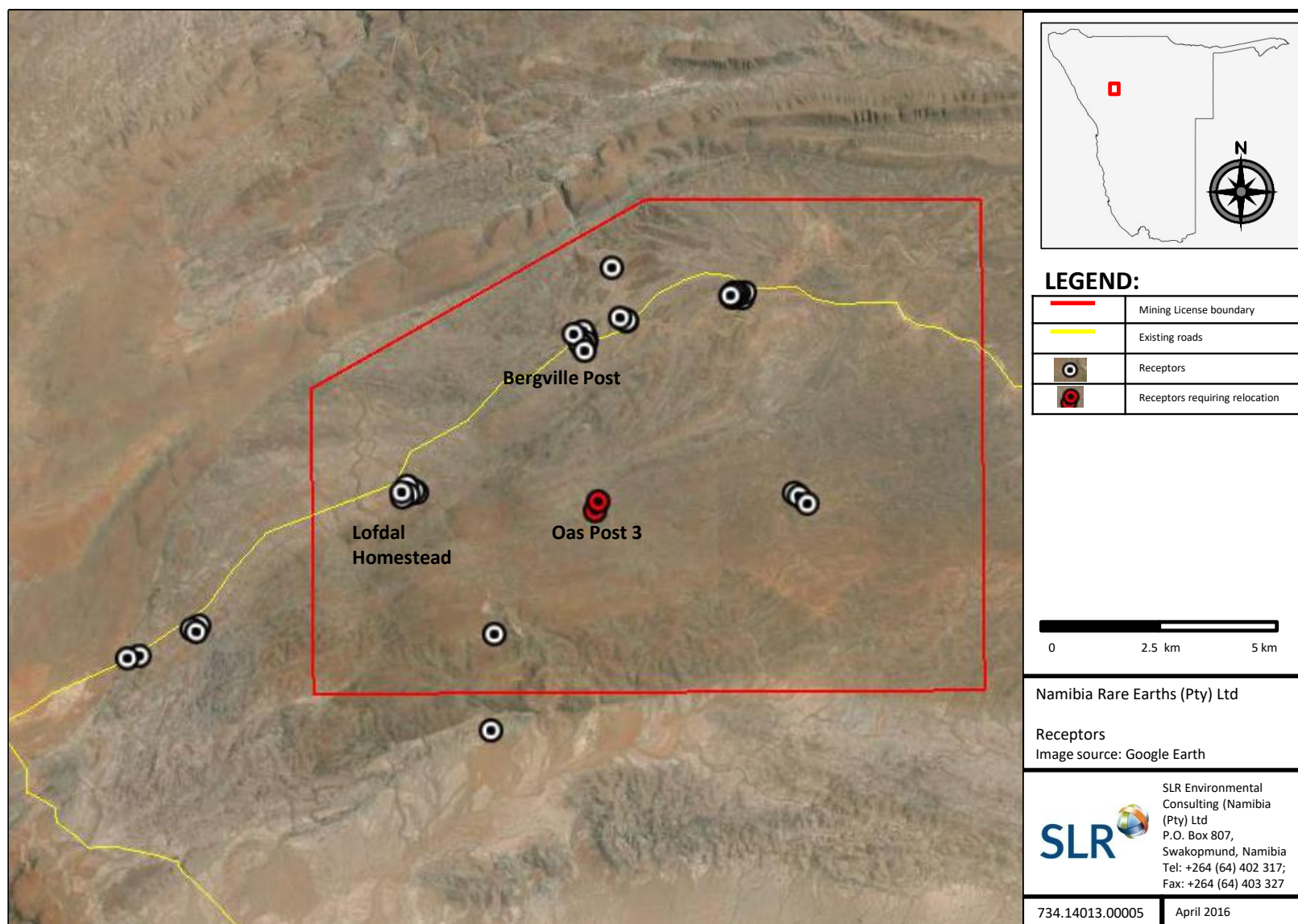


Figure 20-5 Summary of the Lofdal Project's Closest Sensitive Receptors)

The air quality assessment found that the proposed Lofdal Project was likely to result in exceedances of the selected criteria for PM_{2.5}, PM₁₀, NO₂ and diesel particulate matter (DPM) at sensitive receptors within the mining lease area. With design mitigation measures in place ambient pollutant concentrations as a result of the Lofdal Project operations would be reduced significantly, only affecting the sensitive receptors located near the proposed mining operations (i.e. residents/households staying at Oas Post 3). Cumulatively, for the unmitigated scenario, the potential existed that PM_{2.5} and PM₁₀ concentrations would be in exceedance of the selected criteria outside the project boundary and at the sensitive receptors. With the design and/ or additional mitigation measures in place, the cumulative impacts were proposed to be within acceptable levels.

It is suggested that this would be confirmed through continuous PM₁₀ and PM_{2.5} sampling at strategic locations.

SLR's confidence level is **moderate** for this significance rating.

The reports stated that the project was likely to result in exceedances of the selected criteria for PM_{2.5}, PM₁₀, NO₂ and DPM at nearby sensitive receptors. With regard to cumulative impacts, there was the possibility of exceedances of the selected criteria outside the project boundary and at the sensitive receptors due to already elevated background concentrations in the area.

It was recommended that the proposed management and mitigation measures set out in the EMP would be implemented over and above the requirements for the design mitigation. The recommendations included the following:

- Water sprays with chemical suppressants on unpaved road surfaces (on-site roads and public access roads used by mining vehicles);
- Water sprays at the screen and materials handling points;
- Continuation of dust fall and ambient PM₁₀ and PM_{2.5} monitoring at newly identified locations;
- Passive NO₂ sampling campaigns; and
- Continuation of meteorological recordings.

With the potential for elevated baseline PM₁₀ and PM_{2.5} concentrations, mainly due to natural sources resulting in a “saturated” airshed within the study area, it was recommended that the mining operations be allowed to contribute an additional 25% of the selected ambient air quality criteria. This equates to a daily PM_{2.5} addition of 9 µg/m³ and PM₁₀ of 19 µg/m³, with an annual addition of 4 µg/m³ for PM_{2.5} and 8 µg/m³ for PM₁₀. This would require establishing background ambient concentrations for PM₁₀ and PM_{2.5} through at least one year of sampled data. Once background concentrations have been established, the additional allowable contributions should be added, and those limits adopted as the internal performance targets for the LMP.

20.4.5 Archaeology

The Project was considered to have a low archaeological sensitivity with no known archaeological sites that would be affected by the Project. However, dense vegetation and difficulty of access on foot meant that the survey by the archaeologist was not exhaustive. It was therefore possible that some archaeological remains were not identified and would be revealed in the course of exploration work and future mining activity. Such remains were likely to comprise stone encampment and/or burial features, possible rock art, and some traces of colonial settlement. If such remains were to be found it is also possible that burial sites would be found in association with settlement remains.

The significance of this potential impact was medium in the unmitigated scenario and low in the mitigated scenario. SLR's confidence level was moderate to high for this significance rating.

The field survey report indicated that the proposed project was not expected to have any archaeological impact. It was recommended that contractors working on the site were to be made aware that under the National Heritage Act any items protected under the definition of heritage found in the course of development should be reported to the National Heritage Council.

20.4.6 Biodiversity

The following natural features were vital aspects of the ecosystem in the study area and on the project site. If impacts to these aspects could be minimised or avoided, the structure and function of the ecosystem might be maintained even though the 'natural character' may decline significantly.

- The study area is located in the escarpment zone of high diversity and endemism. This automatically increases the importance of all aspects related to the occurrence of endemic or protected species;
- Large mammals such as elephant, are definitely present, and rhino, are potentially present. Other large mammals for which the area is suitable are giraffe and Hartmann's mountain zebra. These game species are attractive for ecotourism and hunting and are thus a potential income for local communities;
- High diversity habitats such as the Rocky Hills, appear to support a high diversity of plant and animal species, and are home to specialised rock dwelling mammals and reptiles;
- Perennial and ephemeral springs and artificial water points are present;
- Large watercourses and drainage lines, provide resources and movement corridors for mammals and birds;
- Higher elevations and denser and taller trees are attractive to raptors.

With reference to the combined biodiversity sensitivity map (Figure 20-6), the proposed activities and infrastructure were positioned in the least sensitive biodiversity areas. The exception was the proximity of the process plant and TSF, as part of the site layout option 1, to the Very Sensitive Rocky Outcrops habitat (the large red patch in Figure 20-6) as well as a service road and tailings pipeline within its footprint, and the location of the single WRD option 1 relative to a large drainage Line.

The unmitigated impact significance was high which could be reduced depending on the success of the mitigation measures. The site layout option 2 (including the process plant, TSF, WRD and solar plant locations) was recommended to avoid the sensitive biodiversity features and therefore formed part of the proposed mitigation measures to reduce the impact severity.

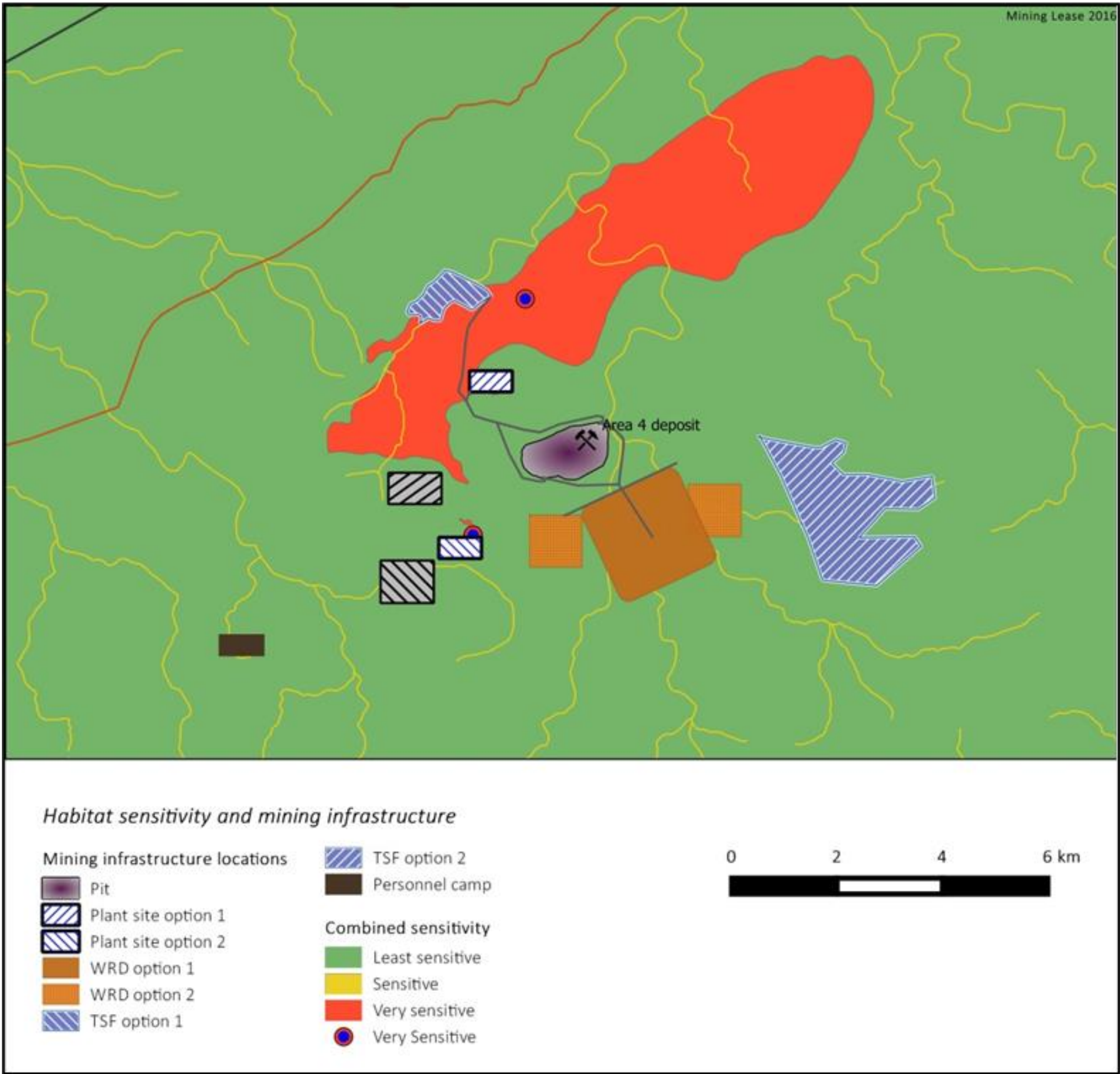


Figure 20-6 Habitat Sensitivity and Lofdal Mining Infrastructure (source from Mining Project EIA Report)

Water is a key ecological driver, an element that is important for the functioning of habitats and related ecosystems. In the context of the Lofdal Mining project the issues of concern were the de-watering of the mine pit, the introduction of the proposed TSF as a potentially large water body and the potential for the blocking or deviation of water flow caused by the placement of project infrastructure in particular the WRD(s) and the TSF. The groundwater assessment concluded that dewatering the pits would only influence the deeper bedrock aquifer because the bedrock aquifer is not hydraulically linked to the surface and near surface water resources. It follows that pit dewatering was not expected to impact on either the springs or the vegetation growth.

The local hydrology is controlled by the watershed located just south of the mine infrastructure, so that all the drainage is in a northerly direction towards the Sout River. The drainage lines were generally poorly defined channels, and only flowed very infrequently. Periodic surface water run-off is understood to be a key ecological driver for vegetation, vertebrates, and invertebrates within and downstream of the project site. The surface water run-off promotes the downstream dispersion of seeds and nutrients.

Both project layout options were located in an area with small catchments with no significant contribution to downstream runoff. The option 1 WRD was situated across larger drainage lines and the option 2 WRDs formed part of the proposed mitigation to reduce the impact significance. Further mitigation measures included suitable stormwater management measures, such that clean water is collected and routed around the mine infrastructure and dirty water is collected and stored on site, to prevent possible polluted water flowing downstream.

In the unmanaged scenario, it was assessed that biodiversity could be further disturbed in the following ways:

- Economic land uses around the Lofdal Project included, amongst others, wildlife use for trophy hunting.
- Species that could be affected included elephant, which are known to avoid high density of humans and high levels of human activities. There is much anecdotal evidence that elephants do adapt to human presence, as witnessed by the frequent occurrence of human-elephant conflict in Namibia, but in general, research has shown that there is a critical density and level of activities where they start avoiding the area as a whole. This may be true for most of the larger species, although it is unlikely that the smaller antelopes would be affected any more than that which had already occurred through hunting over many years. Elephants were not expected to leave the greater area surrounding the Lofdal Project.
- Animals may fall into the pit.
- Animals may be killed on the access road and haul roads.
- Illegal collection of vegetation, vertebrate, and invertebrate species.

In the mitigated scenario, many of these disturbances were thought to be prevented or mitigated to an acceptable levels.

- The significance of the physical destruction of biodiversity potential impact was high in the unmitigated scenario and medium in the mitigated scenario. SLR's confidence level was high for this significance rating.
- The significance of the water resource as an ecological driver potential impact was high in the unmitigated scenario. In the mitigated scenario, the significance reduces to low. SLR's confidence level was moderate for this significance rating.

- The significance of the general biodiversity disturbance potential impact was medium to high in the unmitigated scenario. In the mitigated scenario, the significance was reduced to low to medium. SLR's confidence level was moderate to high for this significance rating.

20.4.7 Groundwater

20.4.7.1 Issue: Dewatering

Dewatering activities from the pit were identified as having the potential to reduce the local groundwater level. The following potential impacts on the hydrogeology were considered in the assessment:

- Lowering of groundwater levels and reduction in aquifer supply;
- Declining yield in existing 3rd party boreholes due to declines in water levels; and
- Impacts on groundwater dependent ecosystems and natural springs.

Dewatering is mainly an operational issue, but it could start towards the end of the construction phase and it could continue for part of the decommissioning phase during the Life of Mine (LOM) of seven years.

Assuming a low and high aquifers storage scenario the radius of influence ('ROI') was predicted due to dewatering over the 7 years. Dewatering the pits would lower the existing ground water levels to a radius of approximately 1.5 km from the mine pit in the high storage scenario and approximately 2.5 km in the low storage scenario during the LOM. After closure the open pit would form a natural groundwater sink as a pit lake with an influence on the groundwater levels of maximum ROI of 4 km in the high storage scenario and a maximum ROI of 8 km in the low storage scenario after 100 years post closure.

Only in the low storage scenario with the greatest ROI of a maximum of 8 km (worst case scenario) after 100 years post closure, could it possibly have an influence on 3rd party boreholes with a drawdown of a maximum of 5 m. However, the residents/households staying at Oas Post 3 located in close proximity to the proposed WRD would be affected for all scenarios. This confirmed the requirement for these specific households to be relocated and an alternate water supply provided to them in their new location. There were no anticipated impacts on the natural springs that occur in areas situated above the bedrock aquifer.

Taking the above mentioned into consideration (assuming the residents/households staying at Oas Post 3 will be relocated), the ROM has no or only little influence (max. 5 m) on the drawdown in 3rd party boreholes or downstream users so the severity was considered to be low in the unmitigated and mitigated scenarios.

The overall significance was rated as medium in the unmitigated scenario reduced to low in the mitigated scenario. SLR's confidence level was moderate to high for this significance rating.

20.4.7.2 Issue: Contamination of Groundwater - Radiological and Non-Radiological

There were a number of potential sources to pollute groundwater in all mine phases. In the construction and decommissioning phases these potential pollution sources were considered to be temporary and diffuse in nature, usually existing only for a short term (weeks to months) but related potential pollution could be long term. The operational phase would present more long-term potential sources and the closure phase would present final landforms that could have the potential to pollute water resources through long term seepage and/or run-off.

The groundwater specialist identified that the most significant pollution sources (also after mine closure) were the mineralised waste facilities, i.e., the WRD(s) and TSF. The assessment considered both layout options specifically relating to these facilities.

Two types of pollution sources were broadly considered. The one type was diffuse pollution which included ad-hoc spills and discharges of polluting substances. The other type was point source pollution which included more long-term pollution associated with longer term sources such as the mineralised waste facilities (i.e. WRD(s) and TSF). From the geochemical analysis it was found that the WRD(s) and the TSF were likely to have the following contaminant components of concern: arsenic (As³⁺), iron (Fe²⁺) and fluoride (F⁻). Also, the final pH of waste rocks in the leachate was analysed to be above 9. The potential for acidic leachate is low because the waste rocks and tailings were predominantly non-potentially acid generating (Non-PAG). The uranium and thorium concentrations were below guideline values for leachates emanating from both waste rock and tailings. Compared to the baseline concentrations of radionuclides, there was a small possible increase of uranium and thorium concentrations which would have no impact on the classification of water quality.

The spatial scale of the potential seepage impact was directly related to the spatial scale of the dispersion of any ground water pollution that in turn has the potential to impact on human health. The potential leachate emanating from the WRD(s) for both options and for option 2 TSF were forecast to be captured by the mine pit (pit lake). The plumes emanating from option 1 TSF migrate towards the northeast while the concentrations decrease to less than 5% of the initial source concentration within a maximum distance of approximately 400 m and 600 m after 100 years and 200 years post closure, respectively. The groundwater specialist concluded that no impact on water quality in 3rd party boreholes would occur for 200 years post mine closure, assuming that the residents/households staying at Oas Post 3 would be relocated.

Summarizing the above assessment, the overall significance was medium in the unmitigated case and low in the mitigated case, because of measures to both limit the amount of seepage from the TSF and to prevent its movement. SLR's confidence level is moderate to high for this significance rating. From the groundwater perspective both options are viable and no fatal flaws were predicted. However, Option 2 was more favourable in terms of a long-term legacy after mine closure.

20.4.8 Surface Water

There were a number of pollution sources in all project phases that had the potential to pollute surface water, particularly in the unmitigated scenario. In the construction and decommissioning phases these potential pollution sources were temporary in nature, usually existing from a few weeks to a few months. Although these sources could be temporary, the potential pollution could be long term. The operational phase presented more long-term potential sources and the closure phase presented final landforms that could have the potential to contaminate surface water through long term seepage and/or run-off.

20.4.8.1 Issue: Altering Drainage Patterns

The identified impacts on ecology associated with altering surface water drainage were addressed in Section 20.4.6. In this regard, the key issue was the loss of surface water flow volume as an important ecological driver. It should be noted that there were no known surface water users in the vicinity of the proposed Lofdal Mine, so that any changes to surface water quantity would not have an impact on the local community, who rely on groundwater.

20.4.8.2 Issue: Pollution of Surface Water – Radiological and Non-Radiological

Rainfall in the area is generally from storms producing short cloud bursts of low to high intensity over rocky areas, resulting in significant surface run-off. In lower lying sandy areas around the project site, infiltration would be proportionately higher reducing surface runoff volumes.

Surface water flow occurs infrequently and for short durations after rainfall events. In the unmitigated scenario, surface water may collect contaminants (hydrocarbons, salts, chemicals/reagents as well as radionuclides) from numerous sources. Water quality could be affected by stormwater runoff from the TSF and WRD infrastructure as well as any contact runoff generated in other areas of the mine. With a suitable stormwater management plan, all contact water would be routed to storage areas to settle out the sediment load. This could then be reused as process make up water or for dust suppression. At elevated pollution concentrations these contaminants could be harmful to humans and the environment either directly or indirectly. It should be noted that there are no known surface water users in the vicinity of the proposed Lofdal Mine, so that any changes to surface water quality would not have an impact on the local community, who rely on groundwater. The mine could have a minor negative effect on the surface water resources downstream of the site, with small decreases in downstream runoff and a possible small deterioration in the surface water quality. Any small deterioration in water quality would be diluted by additional inflow from the tributaries lower in the catchment, such that by the time the floodwaters reached the main Huab River it is unlikely that any pollution would be measurable.

Taking the above into consideration (relevant to both site layouts), the significance of this potential impact was high in the unmitigated scenario. In the mitigated scenario, the significance was reduced to low because of the reduction in severity, duration, and probability. SLR's confidence level was moderate for this significance rating. However, it is important to note that mitigation measures are based on the principle of containing dirty water and diverting clean water as far as possible, so the drainage scheme would ensure that clean surface water flow was directed around structures such as the waste rock dumps and tailings disposal facility.

From the impact assessment the surface water conclusions were that the mine could have a small negative impact on the surface water quality and a negligible reduction in the surface water volumes, however neither of these were identified as a significant risk to the local or larger environment, as there were no identified surface water users in the vicinity of the mine who could be affected by any changes to the current surface water situation.

20.4.9 Noise

Noise sensitive receptors (NSRs) generally include places of residence and areas where members of the public may be affected by noise generated by mining/industrial activities. The nearest town to the Lofdal Project, Khorixas, lies over 25 km to the east and is unlikely to be affected. More likely NSRs that were identified in the project area included scattered farmsteads/homesteads. There were a range of construction, operation and decommissioning activities that had the potential to generate noise and cause related disturbance and nuisance. This section focuses on the human related noise impacts. There was no material difference between the significance of noise impacts associated with the two layout options and the assessment below applied to both options.

The assessment assumed that the group of residents/households staying at Oas Post 3 who resided within the Lofdal Protect Area (in the location of the proposed WRD) would be relocated. The significance of operational phase impacts was therefore based on simulated impacts at NSRs outside active mining area.

Predicted noise impacts were all within the relevant evaluation criteria at sensitive receptor sites. This does not mean that third parties would not hear the mine related activities, but the related impacts were expected

to be low. Limited mitigation was therefore proposed, but confirmatory monitoring was recommended to be undertaken early in the operating phase to verify the model predictions.

The significance of noise impacts on noise sensitive receptors outside active mining areas and along the access route were considered to be low in the unmitigated and mitigated scenarios.

In the quantification of noise emissions and simulation of noise levels as a result of the proposed Lofdal Project, it was calculated that ambient noise evaluation criteria for human receptors would only be exceeded in very close proximity to activities and at the closest on-site NSR.

From a noise perspective it was recommended that the project could proceed provided that the management and mitigation measures were implemented as part of the conditions of environmental authorization to ensure minimal impacts on the surrounding environment.

20.4.10 Socio-Economic

20.4.10.1 Issue: Impacts on the National, Regional and Local Economy

The positive economic impacts expected during the construction and operational phases that could bring about substantial improvement (i.e. positive impacts – indicated with a “+” in the table below) to the regional and national economy are as follows.

Direct economic impacts: Mining and processing operations at Lofdal generating direct value added include: wages and salaries of direct employees inclusive of all employees’ benefits – pensions, medical benefits, etc; tax revenue from royalties; mining tax; import taxes; shareholders tax; taxes on services and consumer goods; Social Security Commission contributions; Khorixas Municipality rates and taxes for any services provided to the mine and employees; interest and amortization paid for finance; and profits earned by mine.

Indirect/induced impacts: Value added due to mine purchases for production (inputs) including the backward supply chain; accounting; insurance and scientific services purchased in Namibia; spending of salaries and wages of mine employees; contractors and other service providers’ employees; increased demand for and supply of locally produced food, goods and services, business opportunities & local employment; increased links to mainstream economy through improved roads and communications.

In the unmitigated scenario, the significance of this potential impact was high positive. In the mitigated scenario, the significance was further increased. SLR’s confidence level was high for this significance rating.

20.4.10.2 Issue: Potential Economic Impacts on Local Non-Mining Livelihoods

The specialist found that it was unlikely that the mine would have a significant negative impact on either of the two conservancies as the area it occupies is small compared to the size of both conservancies. It was more likely to bring positive business opportunities which could then be enhanced.

The most serious impact is that the households (i.e. Oas Post 3) living close to the proposed mining operations would need to be relocated as some of the land was proposed for mine site infrastructure and there would be increased levels of dust, noise, and operational disturbance. Some other farmers may also be economically affected as they would no longer be able to graze their livestock on the land that would be occupied by the mine’s infrastructure. Any physical displacement or loss of livelihood (by the loss of access to their means of livelihood or assets that lead to loss of income) is classified as a high negative impact in the unmitigated scenario but would reduce to medium with mitigation.

NMI was strongly advised to follow international best practice regarding project affected people with the benchmark being the International Finance Corporation's Performance Standard 5: Land Acquisition and Involuntary Resettlement. It aims to ensure that people who are physically or economically displaced as a result of a project end up no worse off and preferably, better off than they were before the project was undertaken. It refers to both physical displacement (relocation or loss of shelter) and to economic displacement (loss of assets or access to assets) as a result of the project related land acquisition and/or restrictions on land use.

Taking the above into consideration, the significance of this potential impact was high in the unmitigated scenario, and medium in the mitigated scenario. The significance is SLR's confidence level was high for this significance rating.

20.4.10.3 Issue: Job Creation and Skills Development

Approximately 300 to 340 jobs would be created during the construction phase of the project and 226 people during operations. These jobs will be created in a region where 39% of the people are classified as poor, i.e. they spend less than N\$4,535.52 per adult equivalent consumption per year on basic needs. The Khorixas, Kamanjab and Outjo Constituencies have poverty levels of 18-20% while people in the Epupa Constituency are the poorest of all the 107 constituencies in the country with 69% of the people ranked as poor and more than half are ranked as severely poor. Therefore, jobs are desperately needed in this region as unemployment levels are high. Skills acquisition is possible. However, the Social Impact Assessment assumed that many of the available jobs would be taken up by Namibians from other regions as the education levels within this region are not high enough to meet much of the mine's requirements. The EIA stated that raising education levels and acquiring skills would be a critical enhancement measure to ensure the constituency and region benefits from the project. The severity was therefore regarded as high positive and could be further enhanced.

The impact significance of these positive impacts was high. SLR's confidence level was high for this significance rating.

20.4.10.4 Issue: In-Migration Impacts

A major project such as a mine is known to attract people into an area even prior to construction, and during construction and operations. Once the mine prepares to close, different sets of impacts still associated with this influx can occur as the local economy shrinks. The impacts of in-migration affect all phases of the life of mine.

The influx of job seekers and workers are likely to bring:

- an increased demand for basic local services – land, water, power;
- the full range of housing options – rental, informal shacks, low income, middle to high income and a concomitant increase in prices, affecting the local population the most;
- an increased demand for health care services;
- social ills of which sexually transmitted diseases such as HIV and AIDS is the highest risk;
- an increase in crime; and
- an increased pressure for more school places and classrooms if migrants bring their families with them.

However, in-migration should not be viewed as only negative. A World Bank study concluded that the most effective route to poverty reduction and economic growth is to encourage the highest possible urban population density as long as rural migrants into towns are given intensive investment and infrastructure development by governments (World Bank, 2009). Migrants are often the movers and shakers in a community and they set up small businesses which add to the local economy.

The specialist concluded that, in-migration would probably lead to substantial deterioration of basic services in neighbouring settlements which would need careful management and considerable investment to turn the situation into a positive scenario. Taking the above into consideration, the severity in the unmitigated scenario was high and could be reduced to medium with mitigation, also taking into consideration the positive impacts associated with in-migration.

In the unmitigated scenario, the significance of this potential impact was high. In the mitigated scenario, the significance remains high due to a high consequence.

20.4.10.5 Issue: Impact Relating to Traffic

In the broadest sense, the activities associated with the project generates traffic in all phases. Two access road options were assessed. The highest levels of traffic would be likely to be experienced during the construction phase which was relatively short term. Levels over the operational life of the mine would be lower and traffic in the decommissioning phase would reduce to the extent that production related traffic ceases. In the closure phase traffic would be limited to just occasional trips for monitoring and aftercare activities. This section in the EIA qualitatively examined the traffic impacts on the existing road to the mine.

Traffic impacts associated with the predicted additional vehicle numbers using the C39 and D2625 included:

- the high speed of the traffic using the road and the potential for road traffic or pedestrian accidents;
- loose gravel can lead to cracked windscreens; and
- the presence of animals and the risk of collision.

From site observations as part of the EIA, the traffic numbers on the above-mentioned roads were low and the additional Lofdal traffic as a result of the project (in all phases) were also predicted to be relatively low. However, the predicted increases in traffic associated with the Lofdal project had the potential to add to the abovementioned issues and lead to additional road accidents. The existing D2625 road is narrow in certain sections and specifically where the road crosses bigger drainage lines (e.g. Sout River), it poses safety risks. It was stated that additional mine traffic on these sections could potentially pose additional safety risk.

The access route option 2 is also preferred to access route option 1 due to the fact the mine traffic would be diverted off the D-route much faster and avoid the above mentioned “dangerous sections”.

In the unmitigated scenario, the potential for injury and death to road users gave this a high severity. With mitigation, the potential accident rate associated with the Lofdal Mine development could be reduced and therefore the severity was reduced to medium - high.

The significance of this potential impact in the unmitigated scenario was high. In the mitigated scenario, the significance was reduced to medium. SLR's confidence level was moderate for this significance rating.

20.4.10.6 Issue: Mine Decommissioning and Closure

Possible reasons for mine closure are varied but the result is a loss various benefits identified in above sections. There will be reduced demand for municipal services in Khorixas, reduced business turnover of suppliers, service industries and retail businesses, reduced government revenue, loss of contributions to pension and social security funds, and bad debts are likely to occur. Some skilled people may find new employment. Low and semi-skilled members of the workforce may struggle to find new employment in the area unless other mines are developed, and the region has diversified its economy.

The significance of the impacts was high both in the unmitigated and mitigated scenario because the consequence and probability of the impacts occurring were high. SLR's confidence level was high for this significance rating.

The project's positive benefits were deemed to outweigh the negative impacts which could be managed if all stakeholders work together. Government would need to provide upfront investment to the local area in improving education and basic services in preparation for a potential influx of between 3,000 – 5,000 job-seekers and their followers. It was stated in the EIA that once the mine became operational the government would reap benefits from these investments and the mine could target its corporate social investments to continue such improvements. The mine could contribute to the three overarching goals of the Government's Fourth National Development Plan 2012/13 to 2016/17 (NDP4) which aims to achieve high and sustained economic growth, employment creation, and increased income equality.

20.4.11 Visual

Visual impacts may be caused by activities and infrastructure in all mine phases. The more significant activities and infrastructure are associated with the operational, decommissioning and closure phases when the mineralized waste facilities (WRD(s) and TSF) are in place and the open pit has been developed.

The scenic quality; the expected receptor sensitivity to landscape change and the distance of the proposed landscape modification from the key receptor points were considered as part of the visual assessment. Potential visual impacts related to loss of landscape character of the site and surrounds, in a conservancy area that has a strong wilderness sense of place without any existing dominating man made infrastructure. The specialist drew the following conclusions:

Without mitigation the potential cumulative degradation of the surrounding landscapes was considered high for layout option 1. For option 2, it was considered medium, due to the following changes in layout:

- The location of the processing plant off the rocky outcrop in a less prominent location;
- The splitting up of the WRD into two heaps does reduce the massing effect and reduces hydrological integrity impacts. However, the placement of the WRDs at slightly higher elevations would marginally increase the mine zone of visual influence in comparison to WRD Option 1;
- The TSF is placed in a well topographically screened location not on a drainage channel and away from the rocky outcrops section that has higher visual resources; and
- The location of the proposed PV site further to the south in a well topographically screened location.

However, with mitigation the severity was considered Medium for both options.

The significance of the visual impact for option 1 was high in both the mitigated and unmitigated scenarios. For option 2, the probability was high in the unmitigated scenario but medium in the mitigated scenario.

SLR's confidence level was moderate to high for this significance rating.

Potential cumulative visual impacts associated with this mine and infrastructure options are mainly related to the degradation of surrounding landscape character from possible ad-hoc development attracted to the area, as well as informal settlements associated with migrant labour. By ensuring that adequate planning takes place to limit un-authorised development and with the proper implementation of the social impact assessment recommendations regarding the management of labour, the extent of the cumulative impacts could be reduced.

20.4.12 Radiation

For the purpose of this assessment, a 'critical group' is defined as a group of members of the public which is reasonably homogeneous with respect to its exposure from a specific radiation source and through a given exposure pathway, and who are representative of individuals receiving the highest exposure dose along a given exposure pathway from the source under consideration.

A hypothetical critical group of persons living under 'worst-case' exposed conditions were considered. It was assumed that the mining option that results in the largest on-site footprint, which is option 2, would be realised. The postulated hypothetical group consists of adults who live within the most exposed public area, at a location some 5 km north of the proposed mining license site, on the public access road D2625 (i.e. Bergville Post).

However, there was one particular group of residents (residents/households staying at Oas Post 3) who resided closer to the proposed mining site (in the location of the proposed WRD) than the members of the hypothetical group. The radiation impact assessment assumed that these residents would be relocate prior to the commencement of mine construction activities.

20.4.12.1 Issue: Impacts Associated with Direct Exposure to External Gamma Radiation

In the context of the natural environment, radiation can occur from natural sources such as cosmic and terrestrial radiation. In the context of a mine, external gamma radiation typically originates from mineralised substances (ore stockpiles, the pit area and immediate surrounding areas, waste rock dumps, tailings storage facilities and ore concentrate) and non-mineralised radioactive contaminated waste.

It was assumed that the on-site exposure from external gamma radiation resulted in the maximum permissible exposure dose to members of staff, i.e. 20mSv.a⁻¹ (such an on-site dose from gamma radiation from on-site sources is considered to be extremely unlikely), then such gamma radiation would imply a theoretical exposure dose of less than 0.1 µSv.a⁻¹ at a location which is some 5 km away, provided it is not further shielded (which is usually the case). Therefore, the incremental gamma exposure dose contribution from such on-site radiation to receptors belonging to the critical group is a factor 100 less than 10 µSv.a⁻¹, which is a common measure for a trivial dose contribution. Even in cases where potential receptors stay considerably closer to the proposed mining site than is assumed for the members of the hypothetical group, public exposure to gamma radiation originating from activities on the mining site can be ignored in most cases.

The significance rating was low in both the unmitigated and mitigated scenarios. SLR's confidence level is high for this significance rating.

20.4.12.2 Issue: Radiological Impacts Associated with the Atmospheric Pathway

Given that this radiological pathway assessment is closely linked to the outcomes of the potential pollution assessments for air, it should be read in conjunction with Section 7.8 of the EIA as the information relevant to the air quality assessment was not repeated within this report.

Both components of the atmospheric pathway may cause internal exposure doses of members of the critical group as a result of the inhalation and/or ingestion of source material by the receptor(s). The dominant physical process underpinning the atmospheric pathway included atmospheric dispersion and deposition. Deposition may lead to direct or indirect ingestion of the source material by a receptor. As there is some limited production of food undertaken in the greater project area, for example in the form of subsistence agricultural activities in which a percentage of the total food intake is locally produced, the intake of contaminated food stuff could theoretically occur.

It is noted that in practice the location of the hypothetical critical group is some 5 km from the proposed project area, and that other persons even further from the mine would have significantly reduced potential dust fallout rates.

The potentially significant components of the atmospheric pathway were assessed separately as detailed in the sections below.

20.4.12.3 Assessment of impact– exposure from Radon Progeny

Atmospheric concentrations of radon isotopes are based on estimates of radon exhalation rates from the ore body, pit, unsurfaced roads, ore stockpiles, waste rock dumps, and tailings storage areas. However, the largest radon sources are due to the TSF, WRD and main pit, contributing some 92% to the total mining related radon exhalations.

The modelled annual average incremental atmospheric radon Rn222 concentration, in Bq.m⁻³, based on the radon source terms determined by the radiation specialist are highest in locations close-by the largest sources of radon, including the TSF, WRDs and pit area. The predicted average atmospheric radon concentrations are incremental, meaning that they are only attributable to radon sources from mining operations, over and above the natural radon sources that exist in the area. The average incremental radon concentration decreases rapidly as the distance from the main mining-related sources increases. At the location where members of the hypothetical critical group of members of the public are assumed to reside, the modelled average radon concentration attributable to mining infrastructure is no longer quantifiable. This implies that Rn222 from mining operations does not pose a risk as the incremental contribution to the ambient atmospheric concentration is negligible. In addition, as a result of the much shorter half-life of Rn220, there is no credible pathway for this radioactive gas to reach the location of the critical group and was therefore considered to be of no consequence for this group.

The significance rating was low in both the unmitigated and mitigated scenarios. SLR's confidence level was high for this significance rating.

20.4.12.4 Assessment of impact– exposure from radioactive dust

This section assessed the potential contribution due to radioactive atmospheric dust. A critical input parameter needed for the computation of an inhalation dose is the annual average atmospheric PM10 concentration at the point(s) of interest (refer to section 7.8 for the assessment of PM10 dust impacts).

Assuming that members of the critical group live permanently at the hypothetical location as defined in the radiation impact assessment, such persons would receive an exposure dose of between approximately 2 and 5 $\mu\text{Sv.a}^{-1}$ (depending on whether dust mitigation is implemented at the project or not), as a result of the inhalation of radioactive dust from mining activities at the mining site as the worst-case unmitigated scenario. Therefore, the incremental exposure dose that third parties staying at the hypothetical location or other members of the public living in homesteads in the greater Lofdal Project area would potentially be exposed to are therefore below the trivial incremental dose of 10 $\mu\text{Sv.a}^{-1}$, and therefore were of no further significance. Even if potential future receptors would stay considerably closer to the proposed mining site than was assumed for the members of the hypothetical group, exposure to radioactive dust originating from activities on the mining site were considered to remain insignificant and could be ignored in most cases.

The significance rating was low in both the unmitigated and mitigated scenarios. SLR's confidence level was high for this significance rating.

20.4.12.5 Issue: Radiological Impacts Associated with the Aquatic Pathway

Given that this radiological pathway assessment was closely linked to the outcomes of the potential pollution assessments for groundwater and surface water. The radiation specialist concluded that the aquatic pathway is unlikely to constitute a significant pathway for human and environmental exposure to ionising radiation provided that on-site engineering and process controls are in place. As a result, this pathway was excluded from the further quantification of potential exposure doses of members of the critical group.

The radiation relevant impact assessment did not provide direct guidance in regard to which of the two infrastructure options were to be used as result in trivial public dose contributions. Based on the results presented in this study, neither of the two infrastructure options were disqualified due to a fatal flaw. However, as a general principle, most potential impacts from the public and/or environmental exposure to radiation originating from operations at Lofdal could be minimised if the environmental footprint of all mining related infrastructure was kept as small as was reasonably achievable.

20.5 Environmental Impact Statement and Conclusion

A tabulated summary of the potential impacts is presented in Table 20-2 below and summarised as follows:

The significance of impacts associated with the project varied from high positive to high negative. It was stated that mitigation of the potential negative impacts could be achieved by committing to apply the findings of the assessment and related mitigation objectives and actions as presented in the EMP.

Two potential negative impacts were thought to remain as high negative residual impacts even with mitigation. These related to socio-economic impacts regarding in-migration and mine decommissioning and closure.

The site layout option 2 (including the process plant, TSF, WRD and solar plant locations) was recommended to avoid the sensitive biodiversity features. Implementing the option 2 layout formed part of the proposed mitigation to reduce the impact significance both from a biodiversity and visual impacts perspective.

Implementing the option 1 TSF and process plant and associated connection roads and pipelines, biodiversity and visual impacts would have high significant impacts, even with mitigation.

The optimal layout of specifically the WRDs needed to be determined (during the detail design phase of the project), taking into consideration that both these WRDs are situated across drainage lines. Various recommendations and management and mitigation requirements were stipulated in the EMP in this regard.

The impacts from the mining activities on the residents at Oas Post 3 were very significant and their health and safety would be compromised. Residents (residents/households staying at Oas Post 3) who currently resided close to the proposed mining site (in the location of the proposed WRD) were advised to be relocated prior to the commencement of mine construction activities. The impacts associated with relocating these specific residents were also assessed and actions required for their relocation presented in the EMP.

The table below can be found 1 in the Mining Project EIA Report.

Table 20-2 Summary of Potential Impacts Associated with the Mining Project

Section	Potential impact	Significance of the Impact (the ratings were negative unless otherwise specified)	
		Unmitigated	Mitigated
Topography	Injury to people from hazardous excavations and infrastructure.	H	M
Soils and land capability	Loss of soil resources from pollution	H	L
	Loss of soil resources from physical disturbance	M	M
Biodiversity	Direct physical destruction of biodiversity and their habitats, particularly sensitive and restricted habitats from clearing land and placing infrastructure.	H	M
	Loss of biodiversity from the reduction of water resources as an ecological driver.	M-H	L
	General disturbance of biodiversity through a range of aspects including dust, noise, vibration, pollution, lighting, firewood collection, poaching, and vehicle movement.	M-H	L-M
Radiological	Impacts associated with direct exposure to external gamma radiation	L	L
	Radiological impacts relating to third party's health and safety associated with the Atmospheric pathway (radon progeny and radioactive dust)	L	L
Surface water	Pollution of surface water	H	L
Groundwater	Pit dewatering	M	L
	Groundwater contamination.	M	L
Air quality	Air pollution – PM _{2.5}	M-H	L
	Air pollution – PM ₁₀	M-H	L
	Air pollution – Dust fall	M	L
	Air pollution – SO ₂	M	L
	Air pollution – NO ₂	M-H	L
	Air pollution – CO	M-H	L

Section	Potential impact	Significance of the Impact (the ratings were negative unless otherwise specified)	
		Unmitigated	Mitigated
	Air pollution – DPM	M-H	L
	Air pollution – VOC	M	L
Noise	Noise pollution in the context of noise sensitive developments.	L	L
Blasting impacts	Blast injury to third parties or damage to structures.	M	M-L
Archaeology	Damage to archaeological sites.	M	L
Visual impacts	Visual impact from key viewpoints.	H	M
Socio-economic impacts	Economic impact including the positive impacts on local, regional and national economies	H ⁺	H ⁺
	Economic impacts on local non-mining livelihoods	H	M
	Job creation and skills development	H ⁺	H ⁺
	In migration and community health and safety	H	H
	Traffic impacts	H	M
	Mine decommissioning and closure	H	H

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

21.1.1 Basis of Capital Cost Estimates

The accuracy of the capital costs estimates is considered to be $\pm 40\%$.

The pricing is based on an used is an engineering, procurement and construction management (EPCM) model. Costs are estimated using a work breakdown structure (WBS) which permits summing of subordinate costs for tasks, materials, etc., into successively higher level “parent” tasks, materials, etc.

Cost build-ups have generally been done in ZAR and converted to USD at an exchange rate of ZAR 16.07: USD 1.00.

No provision has been made for cost escalation or exchange rate fluctuations.

Capital cost estimates for the identified disciplines have been estimated as per the following:

Mining

Mining contractors provided quotes for a turn key operation, with no initial capital outlay by Namibia Critical Metals. Contractor capital recovery is reflected in the equipment operating hourly rates.

Site Power

Site power will be provided by a combination of grid power supplied by Nampower and solar installations. Grid power by Nampower would be available during the first year of operation and thus those capital costs are placed in Year 1 sustaining capital.

Capital costs for the solar installation are captured in monthly rentals based on power provided per MW-hr. This is shown under G&A in the operating cost estimate. There is no initial capital outlay for the solar installation.

Earthworks

Limited bulk earthworks and site clearing for the process plant have been allowed for in the CAPEX estimate. This allowance includes provision for the complete earthworks, including the ROM ramp feeding into the crusher feed bin.

Civil Works

Civil works costs for the process plant were factorised as a percentage of the mechanical equipment supply. The factorisation value varies according to the WBS area.

External Infrastructure

The following surface facilities are included with the process plant estimate:

- Site Water Supply

- Non Process Site Buildings
- Access Roads and Transportation

Table 21-1 presents the total capital costs for each area.

Structural Steel Supply and Erection

Structural steel costs for the mill site process plant were factorised from mechanical equipment costs for the relevant sections based on in-house experience for similar projects.

Platework and Lining

All platework and lining costs for the process plant are calculated as a percentage of the mechanical equipment costs for the relevant sections based on in-house experience for similar projects.

Mechanical Equipment

The PFDs and MELs for the process plant were used as the basis for the compilation of a costing register. Prices were obtained from the existing SGS Bateman data base, or requested verbally from vendors if not available inhouse. Installation costs for mechanical equipment were factorised from mechanical equipment supply costs for the relevant sections based on in-house experience for similar projects.

Allowances have been made for vendor installation, supervision and commissioning.

Piping and Valves

The process plant piping and valves cost estimates were factorised as a percentage of the mechanical supply cost. The factor is based on SGS Bateman historical data

Electrical, Control and Instrumentation

The cost of the process plant electrical and control and instrumentation (C&I) items were factorised at 40% from mechanical equipment costs for the relevant sections based on in-house experience for similar projects.

Transportation

Transportation costs for the process plant mechanical and electrical equipment supply, material supply, etc. were included and based on an average of 5% of the supply price.

Preliminary and General (P&Gs)

P&Gs costs for the process plant have been calculated as a percentage of the sum of the direct costs and included in the capital cost.

EPCM, Spares, Consumables, First Fills and Camp Allowance

An allowance for EPCM, spares, consumables, first fills and the construction camp are included in the indirect capital cost allowance. See section 21.1.4.

21.1.2 Summary of Capital Requirements

The total capital costs for the Project are estimated at USD \$207.08 M and include direct capital costs for mill site process plant, tailings storage facility; sustaining capital for the mill site process plant, TSF closure costs; indirect costs and contingency. Indirect costs, including EPCM, first fills, spares and a camp allowance have been estimated at 30% of Direct and Indirect plant capital costs. TSF construction has a contingency of 15% as much of the construction material will be sourced from the pit stripping. The summary of the initial direct capital and total capital costs are as follows.

Table 21-1 Total Capital Cost Summary

Description	Total Cost (USD)
Direct Mill Site Process Plant	\$117.58 M
Direct Tailings Storage Facility	\$13.63 M
Subtotal Initial Direct Capital Costs	\$131.21 M
Sustaining Capital Processing	\$6.01 M
Sustaining Capital Tailings Storage Facility	\$5.43 M
Mine Closure Costs	\$5.00 M
Indirect Costs	\$18.56 M
Contingency	\$40.87 M
Total Capital Cost Estimate	\$207.08 M

21.1.3 Process Plant Initial Capital Cost Estimate

The Mill Site process plant and in-plant infrastructure capital cost estimate for the Lofdal Project has been prepared to a PEA level of accuracy of +40% and according to the scope as detailed in Table 21-2.

Table 21-2 Initial Site Process Plant Capital Cost Estimate

Description	Sum of Total (million USD)
Process Plant Direct Capital Costs	
Bulk Earthworks	\$7.221 M
Civil Works	\$5.444 M
Buildings Architectural	\$3.091 M
Structural Steelwork	\$6.023 M
Mechanical Equipment	\$38.110 M
Piping & Valves	\$6.329 M
Overland Piping	\$2.607 M
Electrical	\$6.125 M
Instrumentation	\$4.083 M
SMPP P&G's	\$9.861 M
Bulk Earthworks & Civils P&G's	\$6.333 M
E&I P&G's	\$4.083 M
Transportation of Equipment to site	\$5.104 M
Commissioning Spares	\$1.701 M
First fill of lubricants	\$0.340 M
Vendor assistance during Construction & Commissioning	\$1.701 M
Sub-Total Process Plant Direct Costs	\$108.156 M
Non Process Infrastructure	
Site Water	\$1.123 M
Site Buildings	\$0.717 M
Access and Transportation	\$7.581 M
Total Non Process Infrastructure	\$9.422 M
Total Plant Site Direct Capital	\$117.577 M
Process Plant Indirect Costs	
EPCM Estimate (factored at 15% of Direct Costs)	\$16.223 M
Bonds Guarantees etc	\$0.145 M
Insurance	\$2.192 M
Process Plant Contingency	\$38.015 M
Sub-Total Plant Indirect Costs	\$56.575 M
Process Plant and Site Total Initial Capital Costs	\$174.152 M

*Note: Amounts have been rounded for presentation. Rounding may result in inaccurate summing.

Sustaining capital costs for the plant and site are estimated at \$6.01M, including site power from NAMPower.

21.1.4 Tailings Storage Facility Initial Cost

The estimated initial capital cost of the TSF is \$15.673M USD (refer to Table 21-3 for details). The estimate includes site investigation, dam construction cost, a high pressure centrifugal pump, piping, and construction personnel. The ongoing cost of raising the dam is included in the cost of stripping waste.

Aggregates have been identified on site and there is no shortage of waste rock for use in the TSF dam. Once the TSF starter dam is in place, the cost of raising the dam will be minimal because waste rock can be directed to the dam to raise the dam level. The waste stockpile and tailings facility are about the same distance from the mine so there will be no incremental cost to using waste rock from the mine to start and raise the TSF dam.

Table 21-3 Capital Cost Estimate for the Tailings Storage Facility

Description	Capital Cost (Before start of Deposition)	Capital Cost (At the end of Year 1)	Capital Cost (At the end of Year 2)
Mobilization and Demobilization	\$3.029 M	\$.207 M	\$.220 M
Earthworks	\$5.068 M	\$.213 M	\$.213 M
Geosynthetics	\$4.370 M	\$2.067 M	\$2.204 M
Pipeworks and Appurtenances	\$.404 M		\$.063 M
Geotechnical Instrumentation	\$.054 M		
Infrastructure	\$.199 M		
Engineering	\$.505 M	\$.124 M	\$.121 M
Sub Total	\$13.628 M	\$2.611 M	\$2.821 M
Contingency (15%)	\$2.044 M	\$.392 M	\$.423 M
Grand Total	\$15.673 M	\$3.003 M	\$3.244 M

21.1.5 Site Closure Costs

An allowance of \$5.0M USD is made for site closure.

21.1.6 Indirect Allowance for Initial Capital Costs

Indirect capital costs, calculated as a percentage of all initial direct capital costs, are added to initial direct capital costs to cover the cost of EPCM, spares, consumables and first fills.

EPCM costs include project management, detailed engineering, procurement and construction management, and insurance costs during construction for general all risks, construction, professional indemnity, political risk and advance loss of profits insurances. The indirect capital cost allowance also covers the cost of spares, consumables, and first fills for the mine site and process plant. Spares costs

cover operating, strategic and commissioning spares for mechanical and electrical equipment. Allowances were made for first fills only. Indirect capital costs were included as 30% of total initial direct capital costs.

21.2 Indirect Allowance for Initial Capital Costs

21.2.1 Basis of Operating Cost Estimate

The accuracy of the operating costs estimates is considered to be $\pm 40\%$.

Cost build-ups have generally been done in ZAR or NAD and converted to USD. Exchange rates used are ZAR 16.07: USD 1 and NAD 15.99: USD 1.

No provision has been made for cost escalation or exchange rate fluctuations.

21.2.2 Summary of Total Operating Cost Estimate

The overall operating costs for the Mill Site process plant, mining operation and G&A are in Table 21-4. This table shows the overall operating costs per tonne of ROM material mined and per kg of TREO produced.

Table 21-4 Total Operating Cost Summary

Description	Cost per Tonne Processed (USD/t)	Cost Per kg TREO Recovered (USD/t)
Mining	\$16.25	\$14.32
Mill Site Process Plant	\$32.00	\$28.21
General and Administration	\$1.41	\$1.25
Royalties	\$5.20	\$4.58
Total Operating Cost	\$54.86	\$48.36

21.2.3 Process Plant Operating Cost Estimate

General Information

Process operating costs are built up from estimated reagent consumption based on metallurgical testwork, power consumption and labour costs. Plant Maintenance is estimated at 10% of capital cost of installed capital equipment. Plant labour also includes administration (separate from G&A in Section 21.2.5).

Overall operating costs are provided in Table 21-5 below:

Table 21-5 Process Operating Cost Build Ups

	Total
Cost Component	(\$/a)
Labour	\$5,772,042
Front End	\$734,083
Hydrometallurgical	\$1,489,250
Maintenance	\$3,093,313
Power	\$5,236,333
Front End	\$4,938,962
Hydrometallurgical	\$297,370
Fuel (Mobile Equipment)	\$645,010
Reagents and Consumables	\$54,374,117
Front End	\$26,046,331
Hydrometallurgical	\$28,327,786
TOTAL	\$69,120,815
Feed (t/a)	2,160,000
\$/t of ROM Feed	32.00
\$/kg Product	27.9

Details for the following areas are provided below:

- Crushing
- Milling
- Flotation, WHIMS and Tailings
- Hydrometallurgy
- Utilities and Reagents

Crushing

	Total
Cost Component	(\$/a)
Labour	\$264,500
Maintenance	\$320,072
Power	\$166,680
Fuel (Mobile Equipment)	\$64,501
Reagents and Consumables	\$0
TOTAL	\$815,753
Solids Feed (t/a)	2,160,000
\$/t of Section Feed	0.38

Milling

	Total
Cost Component	(\$/a)
Labour	\$195,500
Maintenance	\$1,003,128
Power	\$4,213,382
Fuel (Mobile Equipment)	\$64,501
Reagents and Consumables	\$3,870,431
TOTAL	\$9,346,942
Solids Feed (t/a)	2,160,000
\$/t of Section Feed	4.33

Flotation + WHIMS + Tails

	Total
Cost Component	(\$/a)
Labour	\$274,083
Maintenance	\$433,144
Power	\$558,900
Fuel (Mobile Equipment)	\$64,501
Reagents and Consumables	\$21,171,845
TOTAL	\$22,502,473
Solids Feed (t/a)	2,160,000
\$/t of Section Feed	10.42

Hydrometallurgy

	Total
Cost Component	(\$/a)
Labour	\$695,750
Maintenance	\$862,404
Power	\$98,645
Fuel (Mobile Equipment)	\$64,501
Reagents and Consumables	\$26,854,835
TOTAL	\$28,576,135
Solids Feed (t/a)	62,572
\$/t of Section Feed	456.70

Utilities and Reagents

	Total
Cost Component	(\$/a)
Labour	\$4,342,208
Maintenance	\$474,566
Power	\$198,725
Fuel (Mobile Equipment)	\$387,006
Reagents and Consumables	\$2,477,007
TOTAL	\$7,879,512
Solids Feed (t/a)	2,160,000
\$/t of Section Feed	3.65

Plant Labour

Provision has been made for process plant overhead cost in the labour cost calculation. The overhead cost was based on 5% for normal dayshift workers and at 7.5% for shift workers and operators. The overhead cost is indicated in the staffing table below.

The labour rates used for the process plant OPEX estimate have been based on labour rates from Gecko.

Table 21-6 Process Plant Staff Complement

Area	Position	Number	Annual Cost
Crushing	Crushing and Screening Plant Supervisor	4	\$75.0 K
	Stockpile Unit operator	4	\$60.0 K
	Secondary Crusher Unit operator	4	\$60.0 K
	General labour	4	\$35.0 K
Milling	Section Plant Supervisor	4	\$75.0 K
	Ball Mill 1 Unit operator	4	\$60.0 K
	General labour	4	\$35.0 K
Magnetic Separation	Rougher Unit operator	4	\$60.0 K
Flotation	Rougher Unit operator	4	\$60.0 K
	General labour	4	\$35.0 K
Thickener	Thickener Unit operator	4	\$48.3 K
Tails	General labour	4	\$35.0 K
Acid Mixing	Section Plant Supervisor	4	\$75.0 K
	General labour	4	\$35.0 K
Acid Bake	General Section operator (Senior)	4	\$60.0 K
	General labour	4	\$35.0 K
Impurity Removal	General Section operator (Senior)	4	\$60.0 K
Uranium IX and Precipitation	General Section operator (Senior)	4	\$60.0 K
	General labour	4	\$35.0 K
REE Precipitation	General Section operator	4	\$41.7 K

Area	Position	Number	Annual Cost
Re-Leach	Unit operator	4	\$48.3 K
Solvent Extraction	General Section operator (Senior)	4	\$60.0 K
REE Oxalate Precipitation	General Section operator (Senior)	4	\$60.0 K
TREO Calcination	General labour	4	\$35.0 K
Process Reagents	Reagents Plant supervisor	8	\$150.0 K
	Reagents Plant general operator	16	\$193.3 K
	Reagents Plant general labour	16	\$140.0 K
Plant Services and Utilities	Compressed Air Facility supervisor	4	\$75.0 K
	Compressed Air Facility Operator	4	\$48.3 K
	Compressed Air Facility labor	4	\$35.0 K
	Fire Protection Facility Operator	4	\$48.3 K
Plant Management, Technical Support	Plant Manager	1	\$129.2 K
	Senior Metallurgist	2	\$258.3 K
	Plant Metallurgist	3	\$212.5 K
	Shift Foreman	8	\$566.7 K
	Chief Safety Officer	1	\$70.8 K
	SHEQ Officer	1	\$45.8 K
	Training Officer	1	\$29.2 K
	Control Room Operators	16	\$193.3 K
	Metallurgical Accountant	1	\$70.8 K
	Plant Operators (Leave relief)	4	\$60.0 K
	Plant Attendant (Leave Relief)	4	\$35.0 K
	Plant Resident Engineer	1	\$129.2 K
	Engineering Foreman (Mech & Elect)	4	\$283.3 K
	Snr Technician	2	\$141.7 K
	Instrument Technicians	4	\$150.0 K
	Instrument Mechanics	2	\$75.0 K
	Electricians	6	\$175.0 K
	Boilermakers	6	\$75.0 K
	Fitters	6	\$75.0 K
	Rigger	1	\$12.5 K
	Rigger Aids	2	\$24.2 K
	Crane Driver	2	\$30.0 K
	Mobile Equipment Driver	4	\$48.3 K
	Onsite rubber liner	1	\$18.8 K
	Onsite rubber lining crew	1	\$12.1 K
	Chemist	1	\$70.8 K
	Analysts	2	\$58.3 K
	Sample prep	4	\$35.0 K
Total Plant Personnel		243	\$5.02 M

Power and Water Cost

The main energy source for the various mill site process plant operations is electrical power. The power cost is a combination of grid power supplied by NAMPower and solar power provided by Agrekko.

An allowance was made for the cost to purchase raw water from the local water authorities. This is indicated in the Table 21-7.

Table 21-7 Mill Site Process Plant Energy and Water Cost

Description	Unit	Unit Cost (USD)	Price Basis	Information Source
Power Price	kWh	\$0.062	Namibia	NAMPower / Agrekko
Raw Water	m ³	\$0.95	Namibia	Estimate

Process Plant Power Consumption

Process plant power consumption and costs are presented in Table 21-8.

Table 21-8 Annual Process Consumption

Area		Shift	Total hours	Hourly rate		
	MW	hrs/day	hrs/a	\$/MWh	\$/a	\$/t
Entire plant	11.43			62.00		
Primary & Secondary Crushing	0.438	16.80	6132	62.00	166,680	0.0772
Ore Sorting	0.000	16.80	6132	62.00	0	0.0000
Tertiary Crushing	0.000	16.80	6132	62.00	0	0.0000
Milling	9.127	20.40	7446	62.00	4,213,382	1.9506
Magnetic Separation	0.014	20.40	7446	62.00	6,569	0.0030
Flotation	0.788	20.40	7446	62.00	363,690	0.1684
Tailings	0.409	20.40	7446	62.00	188,641	0.0873
Hydrometallurgical Operations	0.213	20.40	7446	62.00	98,498	0.0456
Hydrometallurgical Batch Operations	0.002	3.00	1095	62.00	147	0.0001
Reagents	0.079	20.40	7446	62.00	36,523	0.0169
Reagents Batch	0.014	3.00	1095	62.00	926	0.0004
Utilities	0.349	20.40	7446	62.00	161,276	0.0747
Total					5,236,333	2.42

Consumables Cost for Mill Site Process Plant

The process plant requirements will consume different types of consumables, ranging from crusher liners, grinding media, flotation reagents and HCL.

The sources for the cost of the different consumables were SGS Bateman's data base, Client information and estimates based on the current OPEX. Total Annual costs for consumables used for the OPEX estimate for the mill site process plant are presented in Table 21-9.

Table 21-9 Consumable Cost for Mill Site Process Plant

Process Reagents and Consumables	Supplier	Annual Cost
Sodium Silicate (N Type)	Florrea	\$486.2 K
Calgon	Florrea	\$187.0 K
Florrea 3900	Florrea	\$15699.4 K
Florrea 3000	Florrea	\$1563.5 K
MIBC/Pine Oil	Florrea	\$123.6 K
Florrea 7411	Florrea	\$203.7 K
NaOH	Protea	\$156.1 K
H ₂ SO ₄	Protea	\$9800.5 K
MgCO ₃	Protea	\$8309.6 K
Na ₂ CO ₃	Protea	\$2769.2 K
H ₂ C ₂ O ₄	Protea	\$3715.9 K
Magnafloc 10	Protea	\$1004.1 K
NH ₄ OH	Protea	\$242.1 K
HCl	Protea	\$213.8 K
Primene JMT	Protea	\$.2 K
Isodecanol	Protea	\$.1 K
Aromatic 150ND	Protea	\$.3 K
Ball Mill Grinding Media	GMSA	\$3870.4 K
Raw Water	NamWater	\$253.2 K
Coal	Market SA	\$2752.5 K
Natural Gas - Acid Bake + REE Oxalate Calciner		\$799.0 K
Other Consumables (Drums, packaging, stationary, laboratory, minor water treatment chemicals etc)	\$2223.8 K	
Total Reagents Cost		\$54.37 M

21.2.4 Mine Operating Cost Estimate

The mine operating costs average \$19.16M per year for direct mining costs and \$10.08M per year for indirect mining costs.

Direct mining costs are built up based on:

- Equipment and labour for loading, hauling, drilling and blasting
- Contractor equipment rental rates
- Contractor capital recovery on the rentals
- Fuel consumption based on 5,600 operating hours per year
- Factors for maintenance, ground engaging tools, lubricants and tire consumption

Indirect mining costs are based on:

- Equipment and labour for pit support equipment
- Contractor equipment rental rates
- Contractor capital recovery on the rentals
- Fuel Consumption based on factored hours for indirect equipment (varies between 1,800 to 5,600 hours per year, depending on use and type)
- Factors for maintenance, lubricants and tire consumption

Labour rates include factors for benefits, sick leave, and statutory requirements.

Average annual values are presented in Table 21-10.

Table 21-10 Consumable Cost for Mill Site Process Plant

Direct Mining Costs	Per Year	Per Tonne Mined
Load / Haul	\$14.39 M	\$1.24
Drill / Blast	\$2.57 M	\$0.22
Direct Personnel (Operators)	\$2.20 M	\$0.19
Indirect Mining Costs		
Equipment Operation	\$9.01 M	\$0.78
Indirect Personnel	\$1.07 M	\$0.09
Average Cost Per Year	\$29.24 M	\$2.51

Equipment totals by year are presented in Table 21-11.

Annual operating costs by year are presented in Table 21-12.

Table 21-11 Equipment Fleet by Year

Equipment Fleet	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Direct Mining Fleet																
115t Hydraulic Excavator	2	4	4	4	3	3	3	3	3	3	2	2	1	1	1	1
30t Hydrualic Excavator	1	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
30t Articulated Truck	4	6	7	8	7	7	7	8	8	8	9	9	8	8	5	1
60t Rigid Frane Truck	7	15	16	19	14	14	14	15	16	12	11	7	5	4	2	1
Drill Rigs	2	3	3	3	3	3	3	3	3	2	2	2	1	1	1	1
Indirect Mining Fleet																
20 t grader	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
25 000l water tankers;	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
25 000l Diesel bowzers	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
55t Track dozers	2	3	4	4	4	4	4	4	4	4	4	4	4	4	4	4
30t Wheel Dozer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
8t Class TLB	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Service truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Hiab for a pump truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Pumps	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Tyre handler	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lighting plants	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
LDV	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18
Taxis	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Drills	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Wheel loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
	58	72	76	80	73	73	73	75	76	72	71	67	63	62	56	51

Table 21-12 Direct and Indirect Mine Operating Costs by Year

Equipment	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
30t Articulated Truck	\$.99M	\$1.66M	\$1.77M	\$2.04M	\$1.84M	\$2.01M	\$2.02M	\$2.17M	\$2.24M	\$2.25M	\$2.40M	\$2.46M	\$2.15M	\$2.23M	\$1.38M	\$.19M
60t Rigid Frane Truck	\$3.31M	\$7.38M	\$7.87M	\$9.67M	\$6.84M	\$6.95M	\$6.99M	\$7.50M	\$8.07M	\$5.85M	\$5.16M	\$3.18M	\$2.27M	\$1.54M	\$.92M	\$.17M
30t Hydrualic Excavator	\$.20M	\$.33M	\$.35M	\$.39M	\$.33M	\$.34M	\$.33M	\$.35M	\$.34M	\$.33M	\$.33M	\$.32M	\$.27M	\$.26M	\$.16M	\$.02M
115t Hydraulic Excavator	\$.93M	\$2.27M	\$2.23M	\$2.50M	\$1.83M	\$1.80M	\$1.92M	\$1.91M	\$1.88M	\$1.53M	\$1.27M	\$.83M	\$.60M	\$.44M	\$.32M	\$.05M
Direct Fuel Cost	\$2.98M	\$6.44M	\$6.77M	\$8.11M	\$5.96M	\$6.08M	\$6.16M	\$6.53M	\$6.88M	\$5.36M	\$4.87M	\$3.48M	\$2.66M	\$2.17M	\$1.34M	\$.22M
Drilling Direct	\$.90M	\$2.03M	\$2.02M	\$2.26M	\$1.70M	\$1.71M	\$1.78M	\$1.77M	\$1.74M	\$1.47M	\$1.28M	\$.96M	\$.73M	\$.61M	\$.40M	\$.06M
Charge and Blast	\$.72M	\$1.62M	\$1.62M	\$1.81M	\$1.36M	\$1.35M	\$1.42M	\$1.43M	\$1.41M	\$1.19M	\$1.04M	\$.78M	\$.59M	\$.49M	\$.33M	\$.05M
Direct Personnel	\$1.63M	\$2.55M	\$2.65M	\$2.86M	\$2.49M	\$2.49M	\$2.49M	\$2.59M	\$2.65M	\$2.23M	\$2.17M	\$1.96M	\$1.54M	\$1.49M	\$1.16M	\$.89M
Total Direct Mining	\$11.66M	\$24.27M	\$25.28M	\$29.64M	\$22.36M	\$22.74M	\$23.11M	\$24.25M	\$25.21M	\$20.22M	\$18.51M	\$13.98M	\$10.80M	\$9.23M	\$6.01M	\$1.66M
Cost Per Tonne Mined	\$1.58	\$1.47	\$1.54	\$1.61	\$1.61	\$1.65	\$1.60	\$1.67	\$1.76	\$1.67	\$1.76	\$1.79	\$1.82	\$1.86	\$1.82	\$3.16
20 t grader	\$.62M	\$.60M	\$.60M	\$.60M	\$.60M	\$.60M	\$.60M	\$.60M	\$.60M	\$.60M	\$.60M	\$.60M	\$.60M	\$.60M	\$.60M	\$.60M
25 000l water tankers;	\$.54M	\$.52M	\$.52M	\$.52M	\$.52M	\$.52M	\$.52M	\$.52M	\$.52M	\$.52M	\$.52M	\$.52M	\$.52M	\$.52M	\$.52M	\$.52M
25 000l Diesel bowsers	\$.27M	\$.26M	\$.26M	\$.26M	\$.26M	\$.26M	\$.26M	\$.26M	\$.26M	\$.26M	\$.26M	\$.26M	\$.26M	\$.26M	\$.26M	\$.26M
55t Track dozers	\$1.63M	\$2.38M	\$3.17M	\$3.17M	\$3.17M	\$3.17M	\$3.17M	\$3.17M	\$3.17M	\$3.17M	\$3.17M	\$3.17M	\$3.17M	\$3.17M	\$3.17M	\$3.17M
30t Wheel Dozer	\$.65M	\$.63M	\$.63M	\$.63M	\$.63M	\$.63M	\$.63M	\$.63M	\$.63M	\$.63M	\$.63M	\$.63M	\$.63M	\$.63M	\$.63M	\$.63M
8t Class TLB	\$.13M	\$.13M	\$.13M	\$.13M	\$.13M	\$.13M	\$.13M	\$.13M	\$.13M	\$.13M	\$.13M	\$.13M	\$.13M	\$.13M	\$.13M	\$.13M
Service truck	\$.19M	\$.18M	\$.18M	\$.18M	\$.18M	\$.18M	\$.18M	\$.18M	\$.18M	\$.18M	\$.18M	\$.18M	\$.18M	\$.18M	\$.18M	\$.18M
Hiab for a pump truck	\$.10M	\$.09M	\$.09M	\$.09M	\$.09M	\$.09M	\$.09M	\$.09M	\$.09M	\$.09M	\$.09M	\$.09M	\$.09M	\$.09M	\$.09M	\$.09M
Pumps	\$.41M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M
Tyre handler	\$.08M	\$.08M	\$.08M	\$.08M	\$.08M	\$.08M	\$.08M	\$.08M	\$.08M	\$.08M	\$.08M	\$.08M	\$.08M	\$.08M	\$.08M	\$.08M
Lighting plants	\$.11M	\$.11M	\$.11M	\$.11M	\$.11M	\$.11M	\$.11M	\$.11M	\$.11M	\$.11M	\$.11M	\$.11M	\$.11M	\$.11M	\$.11M	\$.11M
LDV	\$.41M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M	\$.40M
Drills	\$1.39M	\$1.70M	\$2.46M	\$2.29M	\$2.29M	\$2.29M	\$2.29M	\$2.29M	\$2.29M	\$2.29M	\$2.29M	\$2.29M	\$2.29M	\$2.29M	\$2.29M	\$2.29M
Wheel loader	\$.40M	\$.39M	\$.39M	\$.39M	\$.39M	\$.39M	\$.39M	\$.39M	\$.39M	\$.39M	\$.39M	\$.39M	\$.39M	\$.39M	\$.39M	\$.39M
Indirect Personnel	\$.94M	\$.99M	\$1.09M	\$1.09M	\$1.09M	\$1.09M	\$1.09M	\$1.09M	\$1.09M	\$1.09M	\$1.09M	\$1.09M	\$1.09M	\$1.09M	\$1.09M	\$1.09M

Total Indirects	\$7.87M	\$8.86M	\$10.51M	\$10.34M	\$10.34M	\$10.34M	\$10.34M	\$10.34M	\$10.34M	\$10.34M	\$10.34M	\$10.34M	\$10.34M	\$10.34M	\$4.13M	\$2.07M
Indirect Cost Per Tonne	\$1.07	\$0.54	\$0.64	\$0.56	\$0.74	\$0.75	\$0.72	\$0.71	\$0.72	\$0.85	\$0.98	\$1.32	\$1.74	\$2.08	\$1.25	\$3.93
Total Cost Per Tonne Mined	\$2.65	\$2.01	\$2.17	\$2.17	\$2.35	\$2.40	\$2.32	\$2.38	\$2.48	\$2.53	\$2.75	\$3.11	\$3.56	\$3.93	\$3.08	\$7.09

Mine Pit Workforce

The pits will operate year round, 24 hours per day, based on 2 – 12 hour shifts per day.

The majority of the mine workforce will be contractor personnel, with the company providing management and technical support (engineers, geologists, surveyors). Company provided labour is captured under General and Administration.

The contractor workforce by year is summarized in Table 21-13.

Table 21-13 Contract Mining Personnel

	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Contractor Personnel	95	163	172	191	158	158	158	167	172	148	143	124	100	95	66	42
Operators	67	130	139	158	125	125	125	134	139	120	115	96	77	72	43	19
Loads Control	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Pump Attendants	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Blasters	10	15	15	15	15	15	15	15	15	10	10	10	5	5	5	5
Contract Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Production Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Production Supervisors	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Technical	42	44	48	48	48	48	48	48	48	48	48	48	48	48	48	48
Maint Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maint Supervisor	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Data Capturers	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Tool Storeman	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
NQF 3 Artisans Lv	10	11	13	13	13	13	13	13	13	13	13	13	13	13	13	13
NQF 2 Artisans Lv	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Assistant Artisans	10	11	13	13	13	13	13	13	13	13	13	13	13	13	13	13
Other Work Shop Labour	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10
OEM Mechanics	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Washery Staff	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Safety Officer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total Pit Contractor Personnel	140	210	223	242	209	209	209	218	223	199	194	175	151	146	117	93

21.2.5 General and Administration Expenses

General and Administration expenses are comprised of the following components:

- Company management and technical support
- Personnel transport
- Licencing fees
- Waste management
- Insurance
- Warehousing
- Water Supply
- Photovoltaic Rental (Solar)
- Site Bussing

Processing management and overheads are captured in the process administration costs.

Company personnel are shown in Table 21-14.

Table 21-14 Company Personnel

Position	Quantity	Total Annual Salary (USD)
NMI General Manager	1	\$131.3 K
Manager Mining	1	\$116.3 K
Manager Administration	1	\$116.3 K
HR Superintendent	1	\$75.0 K
HR Assistant	1	\$33.8 K
Finance Superintendent	1	\$75.0 K
Accounts Payable	1	\$33.8 K
Accounts Receivable	1	\$33.8 K
Purchaser	1	\$63.8 K
Warehouse Supervisor	1	\$63.8 K
Shipper / receiver	2	\$67.5 K
Warehouse Floor Staff	4	\$37.5 K
Manager Technical Services	1	\$116.3 K
Mine Planning Engineer	1	\$63.8 K
Mine Surveyors (2)	2	\$97.6 K
Pit Geologists (2)	2	\$127.6 K
Geotechnicians	4	\$135.1 K
Manager HSE	1	\$116.3 K
Environment Compliance Officers	1	\$63.8 K
Environment Technicians	2	\$67.5 K
First Aid / Medical Staff	4	\$195.1 K
Fire / Mine Rescue Response	4	\$195.1 K
Total Annual Cost		\$2.03 M

Total G&A components are summarized in Table 21-15

Table 21-15 G&A Components

Mine G&A Overheads	
Levies	\$37.3 K
Licence fees	\$11.2 K
Drill & Blast overheads	\$485.4 K
Waste Management	\$37.3 K
Other	\$74.7 K
Insurance	\$9.0 K
Site Survey	\$11.2 K
Component transport	\$37.3 K
Workshop Operating cost	\$14.9 K
Component & parts stock	\$3.7 K
Site Office	\$13.4 K
Total Mine Overheads	\$735.5 K
General Site Overheads	
Personnel	\$2016.2 K
Water Supply	\$94.4 K
Site Transportation	\$33.6 K
PV Power Supply Rental	\$173.0 K
Total General Overheads	\$2317.2 K
Total G&A per Year	\$3052.7 K
G&A Per Processed Tonne	\$1.41

21.2.6 Exclusions to Capital and Operating Cost Estimates

The following items have been specifically excluded from the project capital and operating cost estimates:

- EPCM contractor's margin or mark-up
- Civil piling or blasting;
- Storm water handling;
- Any work outside of the described battery limits including any off-site facilities, roads and buildings;
- Site cranes for Operations;
- Owners' costs during the construction phase;
- Price escalation and inflation;
- Foreign exchange fluctuations;
- Cost of financing;
- Land acquisition costs;
- Sunk costs;

- Additional studies or testwork prior to project implementation; including any required metallurgical testwork; environmental impact assessment (EIA) studies, or geotechnical studies
- Any impact on scope of work as a result of metallurgical, EIA or geotechnical studies;
- All value added tax (VAT), import duties, surcharges, levies, government duties or any other statutory taxation (including withholding tax), other than the statutory income tax for mining companies operating in Namibia which is included;
- All royalties, commissions, lease payments, rentals and other payments to landowners, title holders, mineral rights holders, surface right holders, and/or any other third parties other than the Namibia and Guerrero royalties
- Provision for working capital
- Sales and marketing expenses for finished product

22 ECONOMIC ANALYSIS

22.1 Introduction

The economic evaluation of the Lofdal Rare Earths Project as presented in this PEA and prepared jointly by NMI and SGS assumes the project will be 100% equity financed.

This economic evaluation uses parameters relevant as of August 2021, with updates for October 2022 under conditions likely to be applicable to project development and operation and analyses the sensitivity of the project to changes in the key project parameters.

Mining and treatment data, capital cost estimates and operating cost estimates have been put into a base case financial model to calculate the internal rate of return (IRR) and net present value (NPV) based on calculated project after tax cash flows. The scope of the financial model has been restricted to the project level and as such, the effects of interest charges and financing have been excluded.

The model includes sensitivity analyses to demonstrate the effect of variations in key parameters on the economic returns from the project.

Estimated project returns and the key financial statistics are summarized and discussed in this section and are supported by tables and charts. A summary of the financial model results for the project is included in Table 22-1.

22.2 Basis of Economic Analysis

The analysis has been conducted on a pre-debt financing basis. Escalation and inflation have been excluded.

The currency adopted for the analysis is the USD.

In calculating the returns from the project, the following fundamental assumptions have been made:

- The operating life of the project will be approximately 16 years;
- The design throughput for the project is 2,160,000 t/a,
- The economic returns are assessed at the project level on a pre-financing basis;
- The evaluation includes a 24-month project development period prior to the commencement of production, a total of approximately 16 years;
- The exchange rate used to convert ZAR into USD is 16.07 to 1. The exchange rate used to convert NAD into USD is 15.99 to 1.

All assumptions made as part of the economic evaluation are detailed in Section 22.5.

22.3 Summary of Results

The key project statistics for the life of the project are summarized in Table 22-1 below.

Table 22-1 Key Financial Statistics

ITEM	UNIT	BASE CASE
PROJECT ECONOMICS		
NPV at a Discount Rate of 5% (after-tax)	USD M	\$390,979,000
Internal Rate of Return (after-tax)	%	34%
Payback Period from Start of Production	y	3.2
PRODUCTION		
ROM Produced (over life of mine)	t	26,836,000
TREO Produced (over life of mine)	t	30,442
REVENUE		
Average TREO Price (over life of mine) ¹	USD/kg	\$91.64
Total Revenue (undiscounted, after royalties)	USD M	\$1,110
OPERATING COSTS		
Average Unit Operating Costs per Tonne Mined		
Mining	USD/t Processed	\$16.25
Mill Site Process Plant	USD/t Processed	\$32.00
General and Administration	USD/t Processed	\$1.41
Royalties	USD/t Processed	\$5.20
Total Operating Costs	USD/t Processed	\$54.86
ITEM	UNIT	BASE CASE
OPERATING COSTS		
Average Unit Operating Costs per kg TREO Produced		
Mining	USD/kg Produced	\$14.32
Mill Site Process Plant	USD/kg Produced	\$28.21
General and Administration	USD/kg Produced	\$1.25
Royalties	USD/kg Produced	\$4.58
Total Operating Costs	USD/kg Produced	\$48.36
CAPITAL COSTS		
Process Plant – Direct	USD	\$117.28 M
Mill Plant – Sustaining Capital	USD	\$6.01 M
Tailings Facility – Direct	USD	\$13.63 M
Tailings Facility – Sustaining Capital	USD	\$5.43 M
TSF Closure Costs	USD	\$5, M
Indirect (EPCM, First Fills, Spares and Camp Allowance) at 30% of total initial direct capital costs	USD	\$18.56 M
Contingency	USD	\$40.87 M
Total Capital Cost	USD	\$207.08 M

¹ Treatment charges of \$12 per kg TREO are included in the net average price TREO price and are subtracted from gross revenue in the financial model.

22.4 Project Economics

Based on the extraction of 2,160,000 t/a of ROM feed from the mine, the project is anticipated to yield a pre-tax IRR of 34% with a pre-tax NPV, at a discount rate of 5% of USD \$632.7M, and an after-tax IRR of 28% with an after-tax NPV, at a discount rate of 5%, of USD \$391 M. Cumulative cash flows are USD \$1,110 M pre-tax and USD \$698 M after-tax over the sixteen year LOM.

The project is expected to pay back initial capital in 3.2 years after production starts.

Figure 22-1 shows the cumulative after-tax cash flow over the total project life.

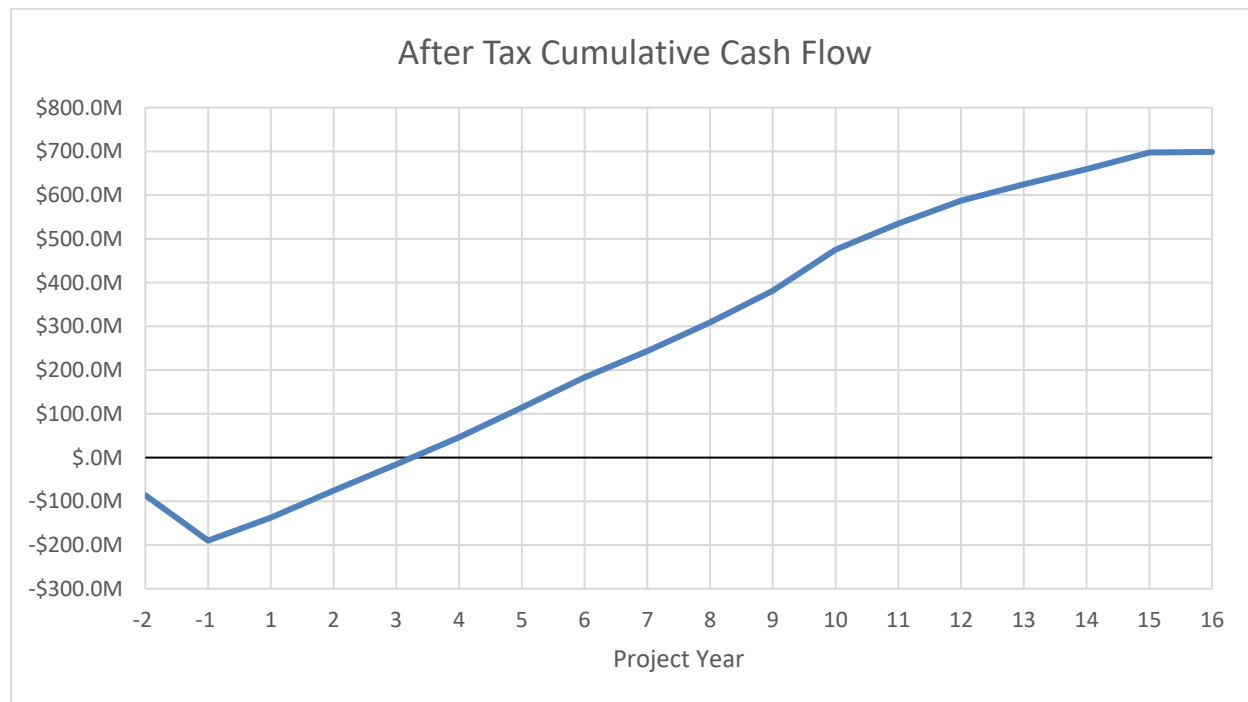


Figure 22-1 Cumulative Cash Flow (After Tax)

22.5 Clarification and Assumptions

Economic analysis has been carried out on the basis detailed within this section.

22.5.1 Analysis Period

The period of analysis is from the commencement of Process plant construction during quarter one, Year - 1 until the end of mining through to the last sale of product in Year 16.

22.5.2 Revenue

The calculation of production tonnages is based on the mining and treatment schedules. The recovery factors for TREO are based on available testwork. The compounded LREO recovery over both the flotation

and hydrometallurgical sections of the processing facility is 53% and the HREO is 64% with the combined TREO recovery of 60% for the revenue calculation in the financial evaluation.

The basket prices used for this economic evaluation are presented in Table 22-2 in \$USD/kg.

Table 22-2 Basket Prices used for Economic Evaluation

Pricing Forecast for REE	Pricing Used for Analysis	Distribution
La ₂ O ₃	\$-	9.2%
Ce ₂ O ₃	\$-	16.0%
Pr ₂ O ₃	\$201.00	1.7%
Nd ₂ O ₃	\$212.00	6.3%
Sm ₂ O ₃	\$5.00	2.2%
Eu ₂ O ₃	\$36.00	1.1%
Gd ₂ O ₃	\$109.00	4.3%
Tb ₂ O ₃	\$2,493.00	0.9%
Dy ₂ O ₃	\$587.00	6.2%
Ho ₂ O ₃	\$290.00	1.3%
Er ₂ O ₃	\$64.00	3.8%
Yb ₂ O ₃	\$20.00	3.5%
Lu ₂ O ₃	\$947.00	0.5%
Y ₂ O ₃	\$17.00	42.4%
Tm ₂ O ₃	\$500.00	0.6%
Average Basket Value	\$103.64	
Realized Basket Price after Treatment Charges	\$91.64	

22.5.3 Operating Costs

Operating costs have been estimated as per the following functional headings:

- Mining;
- Mill Site Process Plant;
- General and Administration.

A full description of operating costs can be found in Section 21 of the Report.

22.5.4 Capital Costs

Capital costs include the direct capital costs the process plant, non process infrastructure and tailings storage facility; sustaining capital for the process plant; mine and tailings facility closure costs; indirect costs (including EPCM, first fills, spares and a construction camp allowance) and a contingency.

A full description of capital costs can be found in Section 21.1 of the Report.

22.5.5 Funding

The financial model for the project is presented on the basis of pre-financing cash flows and as such excludes the impact of both debt funding and equity finance.

22.5.6 After Tax Free Cash Flow

After-tax free cash flow is calculated by deducting operating costs, royalties, taxes and sustaining capital expenditures from revenue.

22.5.7 Net Present Value

After-tax NPVs are calculated from the annual free cash flows. The financial model is capable of applying a range of discount factors. The use of various discount rates in the base case and sensitivity analysis of this report should not be taken as an endorsement of those discount rates as appropriate rates of return for this project.

22.5.8 After Tax Internal Rate of Return

The after-tax IRR is calculated from the annual after-tax free cash flows.

22.5.9 Payback Period

The payback period is identified as the period in which the cumulative undiscounted (after-tax free) cash flow becomes positive, having paid back the development costs.

22.6 Financial Model

A summary of the financial model results is presented below. The base case financial analysis is presented at a 5% discount rate.

Table 22-3 Summary Financial Results

Pre-Tax NPV @5%	\$632.7 M
Pre-Tax IRR	34%
Pre-Tax Net Cash Flow	\$1110.4 M
After Tax NPV @5%	\$391.0 M
After Tax IRR @5%	28%
After Tax Net Cash Flow	\$698.7 M

Table 22-4 Financial Model

	Year																	
Mining	- 2	- 1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Resource Tonnes			1199.9k	1553.6k	1708.7k	2000.k	2000.k	2000.k	1992.5k	1976.5k	1978.5k	1951.8k	1933.1k	1955.7k	1801.8k	1707.1k	946.6k	129.8k
Stockpile Tonnes			589.k	1314.4k	1332.2k	1394.3k	875.1k	974.9k	916.7k	1072.1k	1030.9k	919.8k	926.k	864.1k	555.5k	599.7k	449.6k	79.3k
Waste Tonnes			5570.4k	13647.6k	13418.8k	15027.3k	11014.3k	10822.4k	11537.5k	11486.4k	11325.6k	9225.4k	7648.8k	5011.2k	3581.8k	2668.1k	1902.3k	317.5k
Total Tonnes			7359.2k	16515.6k	16459.7k	18421.7k	13889.4k	13797.3k	14446.8k	14535.1k	14335.k	12097.k	10507.9k	7830.9k	5939.2k	4975.k	3298.5k	526.6k
Strip Ratio W:O			3.11	4.76	4.41	4.43	3.83	3.64	3.97	3.77	3.76	3.21	2.68	1.78	1.52	1.16	1.36	1.52
Processing																		
Tonnes Processed			1199.9k	1553.6k	1708.7k	2000.k	2000.k	2000.k	1992.5k	1976.5k	1978.5k	1951.8k	1933.1k	1955.7k	1801.8k	1707.1k	946.6k	129.8k
Grades																		
LREO Grades			0.078%	0.072%	0.072%	0.071%	0.070%	0.067%	0.069%	0.072%	0.081%	0.092%	0.070%	0.067%	0.080%	0.083%	0.103%	0.179%
HREO Grades			0.113%	0.115%	0.103%	0.112%	0.124%	0.128%	0.116%	0.122%	0.129%	0.153%	0.114%	0.102%	0.084%	0.082%	0.114%	0.085%
TREO Grades			0.192%	0.186%	0.175%	0.182%	0.195%	0.194%	0.184%	0.193%	0.210%	0.245%	0.184%	0.169%	0.164%	0.165%	0.217%	0.263%
Elements Recovered (kg)																		
Light Elements Recovered (kg)			498.3k	590.k	650.8k	747.3k	744.3k	707.1k	725.1k	752.3k	850.k	952.1k	713.6k	690.2k	764.2k	749.k	517.3k	123.k
Ce ₂ O ₃			228.7k	270.4k	298.1k	343.2k	340.9k	321.6k	325.k	338.8k	386.3k	435.6k	320.1k	307.8k	349.3k	340.3k	225.k	50.8k
La ₂ O ₃			136.9k	158.2k	172.9k	194.7k	189.5k	181.9k	182.2k	194.1k	218.9k	249.9k	178.9k	173.5k	202.1k	200.6k	139.9k	32.9k
Nd ₂ O ₃			84.k	101.2k	112.5k	130.9k	132.1k	124.6k	133.4k	134.k	150.8k	163.5k	131.8k	128.2k	133.2k	130.9k	93.8k	24.3k
Pr ₂ O ₃			23.7k	28.1k	31.k	35.9k	35.9k	34.k	34.8k	35.6k	40.4k	45.3k	34.6k	33.2k	36.3k	35.3k	23.5k	5.5k
Sm ₂ O ₃			25.k	32.1k	36.3k	42.7k	46.k	45.k	49.7k	49.9k	53.6k	57.7k	48.3k	47.6k	43.4k	41.9k	35.1k	9.5k
Heavy Elements Recovered (kg)			871.k	1139.5k	1130.3k	1427.5k	1592.1k	1635.7k	1474.2k	1539.5k	1636.k	1909.2k	1413.9k	1277.4k	969.k	893.3k	688.8k	70.2k
Dy ₂ O ₃			84.4k	110.5k	109.6k	137.2k	153.4k	156.7k	141.5k	149.7k	157.6k	179.6k	136.1k	123.8k	95.4k	88.7k	69.k	7.9k
Er ₂ O ₃			52.5k	68.9k	66.9k	85.k	93.1k	95.2k	85.1k	88.8k	96.k	110.8k	82.6k	75.k	56.5k	53.8k	42.k	3.2k
Eu ₂ O ₃			12.7k	16.4k	17.8k	21.k	23.5k	23.9k	24.3k	25.4k	26.5k	28.4k	23.2k	22.4k	19.k	18.1k	15.6k	3.8k
Gd ₂ O ₃			54.2k	70.6k	75.1k	90.3k	102.5k	104.1k	100.1k	105.1k	108.7k	119.3k	93.6k	87.4k	72.6k	67.6k	56.2k	11.4k
Ho ₂ O ₃			18.1k	23.7k	23.3k	29.3k	32.3k	33.2k	29.8k	30.9k	33.1k	38.4k	28.8k	25.9k	19.7k	18.2k	13.9k	1.3k
Lu ₂ O ₃			7.k	9.3k	8.8k	11.4k	12.1k	12.4k	11.3k	11.4k	12.8k	14.7k	11.1k	10.1k	7.5k	7.3k	5.7k	.4k
Tb ₂ O ₃			12.4k	16.1k	16.6k	20.4k	23.k	23.3k	21.5k	22.6k	23.4k	26.5k	20.2k	18.4k	14.6k	13.6k	10.9k	1.7k
Tm ₂ O ₃			7.9k	10.4k	10.k	12.8k	13.8k	14.3k	12.9k	13.2k	14.5k	16.9k	12.4k	11.1k	8.3k	8.k	6.3k	.4k
Y ₂ O ₃			573.9k	750.4k	742.2k	942.1k	1054.6k	1086.9k	970.1k	1012.8k	1074.8k	1273.5k	930.1k	833.6k	623.6k	568.2k	430.2k	37.6k
Yb ₂ O ₃			48.k	63.2k	60.2k	77.9k	83.8k	85.8k	77.7k	79.6k	88.7k	101.2k	75.7k	69.7k	51.6k	49.7k	39.1k	2.6k
Total TREO's Recovered (kg)			1369.4k	1729.6k	1781.1k	2174.7k	2336.5k	2342.8k	2199.3k	2291.8k	2486.k	2861.2k	2127.5k	1967.7k	1733.2k	1642.3k	1206.1k	193.2k
Gross Revenue																		
Light Elements			\$22.7M	\$27.3M	\$30.3M	\$35.2M	\$35.5M	\$33.5M	\$35.5M	\$35.8M	\$40.4M	\$44.1M	\$35.1M	\$34.1M	\$35.7M	\$35.0M	\$24.8M	\$6.3M
Ce ₂ O ₃			\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M
La ₂ O ₃			\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M
Nd ₂ O ₃			\$17.8M	\$21.5M	\$23.9M	\$27.7M	\$28.0M	\$26.4M	\$28.3M	\$28.4M	\$32.0M	\$34.7M	\$27.9M	\$27.2M	\$28.2M	\$27.7M	\$19.9M	\$5.2M
Pr ₂ O ₃			\$4.8M	\$5.7M	\$6.2M	\$7.2M	\$7.2M	\$6.8M	\$7.0M	\$7.2M	\$8.1M	\$9.1M	\$6.9M	\$6.7M	\$7.3M	\$7.1M	\$4.7M	\$1.1M
Sm ₂ O ₃			\$.1M	\$.2M	\$.2M	\$.2M	\$.2M	\$.2M	\$.2M	\$.2M	\$.3M	\$.3M	\$.2M	\$.2M	\$.2M	\$.2M	\$.2M	\$.0M
Heavy Elements			\$116.6M	\$152.6M	\$152.6M	\$190.6M	\$212.7M	\$217.0M	\$197.7M	\$207.4M	\$218.7M	\$249.7M	\$189.0M	\$171.9M	\$133.3M	\$124.3M	\$97.6M	\$12.0M
Dy ₂ O ₃			\$49.6M	\$64.9M	\$64.3M	\$80.5M	\$90.0M	\$92.0M	\$83.0M	\$87.8M	\$92.5M	\$105.4M	\$79.9M	\$72.6M	\$56.0M	\$52.1M	\$40.5M	\$4.7M
Er ₂ O ₃			\$3.4M	\$4.4M	\$4.3M	\$5.4M	\$6.0M	\$6.1M	\$5.4M	\$5.7M	\$6.1M	\$7.1M	\$5.3M	\$4.8M	\$3.6M	\$3.4M	\$2.7M	\$.2M
Eu ₂ O ₃			\$.5M	\$.6M	\$.6M	\$.8M	\$.8M	\$.9M	\$.9M	\$.9M	\$1.0M	\$1.0M	\$.8M	\$.8M	\$.7M	\$.7M	\$.6M	\$.1M
Gd ₂ O ₃			\$5.9M	\$7.7M	\$8.2M	\$9.8M	\$11.2M	\$11.3M	\$10.9M	\$11.5M	\$11.8M	\$13.0M	\$10.2M	\$9.5M	\$7.9M	\$7.4M	\$6.1M	\$1.2M
Ho ₂ O ₃			\$5.2M	\$6.9M	\$6.7M	\$8.5M	\$9.4M	\$9.6M	\$8.6M	\$9.0M	\$9.6M	\$11.1M	\$8.4M	\$7.5M	\$5.7M	\$5.3M	\$4.0M	\$4M
Lu ₂ O ₃			\$6.6M	\$8.8M	\$8.3M	\$10.8M	\$11.5M	\$11.8M	\$10.7M	\$10.8M	\$12.1M	\$13.9M	\$10.5M	\$9.6M	\$7.1M	\$6.9M	\$5.4M	\$.3M
Tb ₂ O ₃			\$30.9M	\$40.2M	\$41.3M	\$50.8M	\$57.4M	\$58.1M	\$53.5M	\$56.4M	\$58.3M	\$66.0M	\$50.4M	\$46.0M	\$36.5M	\$33.9M	\$27.0M	\$4.1M
Tm ₂ O ₃			\$3.9M	\$5.2M	\$5.0M	\$6.4M	\$6.9M	\$7.1M	\$6.4M	\$6.6M	\$7.2M	\$8.4M	\$6.2M	\$5.6M	\$4.2M	\$4.0M	\$3.1M	\$.2M
Y ₂ O ₃			\$9.8M	\$12.8M	\$12.6M	\$16.0M	\$17.9M	\$18.5M	\$16.5M	\$17.2M	\$18.3M	\$21.6M	\$15.8M	\$14.2M	\$10.6M	\$9.7M	\$7.3M	\$.6M

Yb ₂ O ₃			\$1.0M	\$1.3M	\$1.2M	\$1.6M	\$1.7M	\$1.7M	\$1.6M	\$1.6M	\$1.8M	\$2.0M	\$1.5M	\$1.4M	\$1.0M	\$1.0M	\$.8M	\$.1M
Final Separation Costs			\$16.4M	\$20.8M	\$21.4M	\$26.1M	\$28.0M	\$28.1M	\$26.4M	\$27.5M	\$29.8M	\$34.3M	\$25.5M	\$23.6M	\$20.8M	\$19.7M	\$14.5M	\$2.3M
Total Gross Revenue			\$122.9M	\$159.2M	\$161.5M	\$199.7M	\$220.1M	\$222.4M	\$206.8M	\$215.7M	\$229.2M	\$259.4M	\$198.6M	\$182.4M	\$148.3M	\$139.6M	\$107.9M	\$16.0M
TREO Revenue Per KG (Basket)			\$89.76	\$92.02	\$90.68	\$91.83	\$94.22	\$94.93	\$94.03	\$94.12	\$92.19	\$90.67	\$93.34	\$92.70	\$85.54	\$85.02	\$89.47	\$82.67
Costs																		
Direct Mining			\$11.7M	\$24.3M	\$25.3M	\$29.6M	\$22.4M	\$22.7M	\$23.1M	\$24.2M	\$25.2M	\$20.2M	\$18.5M	\$14.0M	\$10.8M	\$9.2M	\$6.0M	\$1.7M
Indirect Mining			\$7.9M	\$8.9M	\$10.5M	\$10.3M	\$10.3M	\$10.3M	\$10.3M	\$10.3M	\$10.3M	\$10.3M	\$10.3M	\$10.3M	\$10.3M	\$10.3M	\$4.1M	\$2.1M
Total Mining Costs			\$19.5M	\$33.1M	\$35.8M	\$40.0M	\$32.7M	\$33.1M	\$33.4M	\$34.6M	\$35.5M	\$30.6M	\$28.8M	\$24.3M	\$21.1M	\$19.6M	\$10.1M	\$3.7M
Mining Cost Per Tonne			\$ 2.65	\$ 2.01	\$ 2.17	\$ 2.17	\$ 2.35	\$ 2.40	\$ 2.32	\$ 2.38	\$ 2.48	\$ 2.53	\$ 2.75	\$ 3.11	\$ 3.56	\$ 3.93	\$ 3.08	\$ 7.09
Processing																		
Direct Processing Costs			\$38.4M	\$49.7M	\$54.7M	\$64.0M	\$64.0M	\$64.0M	\$63.8M	\$63.2M	\$63.3M	\$62.5M	\$61.9M	\$62.6M	\$57.7M	\$54.6M	\$30.3M	\$4.2M
Total Processing Costs			\$38.4M	\$49.7M	\$54.7M	\$64.0M	\$64.0M	\$64.0M	\$63.8M	\$63.2M	\$63.3M	\$62.5M	\$61.9M	\$62.6M	\$57.7M	\$54.6M	\$30.3M	\$4.2M
General & Administration			\$1.7M	\$2.2M	\$2.4M	\$2.8M	\$2.8M	\$2.8M	\$2.8M	\$2.8M	\$2.8M	\$2.8M	\$2.7M	\$2.8M	\$2.5M	\$2.4M	\$1.3M	\$.2M
Royalties																		
Namibian Government Royalty			\$3.7M	\$4.8M	\$4.8M	\$6.0M	\$6.6M	\$6.7M	\$6.2M	\$6.5M	\$6.9M	\$7.8M	\$6.0M	\$5.5M	\$4.4M	\$4.2M	\$3.2M	\$.5M
Land Owner's Royalty			\$2.5M	\$3.2M	\$3.2M	\$4.0M	\$4.4M	\$4.4M	\$4.1M	\$4.3M	\$4.6M	\$5.2M	\$4.0M	\$3.6M	\$3.0M	\$2.8M	\$2.2M	\$.3M
Total Operating Costs			\$65.8M	\$93.0M	\$101.0M	\$116.8M	\$110.5M	\$111.0M	\$110.4M	\$111.4M	\$113.1M	\$108.7M	\$103.4M	\$98.8M	\$88.8M	\$83.6M	\$47.2M	\$8.9M
Net Revenue			\$57.1M	\$66.2M	\$60.5M	\$82.9M	\$109.6M	\$111.4M	\$96.4M	\$104.3M	\$116.1M	\$150.7M	\$95.2M	\$83.6M	\$59.5M	\$56.0M	\$60.7M	\$7.1M
Cumulative Revenue			\$57.1M	\$123.3M	\$183.9M	\$266.8M	\$376.4M	\$487.8M	\$584.2M	\$688.5M	\$804.6M	\$955.2M	\$1050.4M	\$1134.1M	\$1193.6M	\$1249.6M	\$1310.4M	\$1317.5M
Capital Schedule																		
Total Capital Schedule	\$86.1M	\$103.7M	\$4.7M	\$4.1M	\$.8M	\$.8M	\$.9M	\$.8M	\$.0M	\$.0M	\$.0M	\$.1M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$5.0M
Pre-Tax Cash Flow	-\$86.1M	-\$103.7M	\$52.4M	\$62.1M	\$59.7M	\$82.1M	\$108.7M	\$110.6M	\$96.4M	\$104.3M	\$116.1M	\$150.6M	\$95.2M	\$83.6M	\$59.5M	\$56.0M	\$60.7M	\$2.1M
Cumulative Cash Flow	-\$86.1M	-\$189.8M	-\$137.4M	-\$75.3M	-\$15.6M	\$66.5M	\$175.2M	\$285.8M	\$382.2M	\$486.5M	\$602.6M	\$753.2M	\$848.4M	\$932.0M	\$991.5M	\$1047.5M	\$1108.3M	\$1110.4M
Namibian Taxes	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$20.3M	\$40.8M	\$41.5M	\$36.2M	\$39.1M	\$43.5M	\$56.5M	\$35.7M	\$31.4M	\$22.3M	\$21.0M	\$22.8M	\$.8M
Net Revenue After Tax			\$57.1M	\$66.2M	\$60.5M	\$62.7M	\$68.9M	\$69.9M	\$60.3M	\$65.2M	\$72.5M	\$94.2M	\$59.5M	\$52.3M	\$37.2M	\$35.0M	\$38.0M	\$6.3M
After Tax Cash Flow	-\$86.1M	-\$103.7M	\$52.4M	\$62.1M	\$59.7M	\$61.9M	\$67.9M	\$69.1M	\$60.3M	\$65.2M	\$72.5M	\$94.1M	\$59.5M	\$52.3M	\$37.2M	\$35.0M	\$38.0M	\$1.3M
Cumulative Cash Flow After Tax	-\$86.1M	-\$189.8M	-\$137.4M	-\$75.3M	-\$15.6M	\$46.3M	\$114.2M	\$183.3M	\$243.6M	\$308.8M	\$381.3M	\$475.4M	\$534.9M	\$587.2M	\$624.4M	\$659.4M	\$697.4M	\$698.7M
Operating Income	\$.0M	\$.0M	\$57.1M	\$66.2M	\$60.5M	\$82.9M	\$109.6M	\$111.4M	\$96.4M	\$104.3M	\$116.1M	\$150.7M	\$95.2M	\$83.6M	\$59.5M	\$56.0M	\$60.7M	\$7.1M
Development Capex	-\$86.1M	-\$103.7M	-\$4.7M	-\$4.1M	-\$.8M	-\$.8M	-\$.9M	-\$.8M	\$.0M	\$.0M	\$.0M	-\$.1M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	-\$5.0M
Taxable Income and capital costs current year	-\$86.1M	-\$103.7M	\$52.4M	\$62.1M	\$59.7M	\$82.1M	\$108.7M	\$110.6M	\$96.4M	\$104.3M	\$116.1M	\$150.6M	\$95.2M	\$83.6M	\$59.5M	\$56.0M	\$60.7M	\$2.1M
Carry forward	-\$12.5M	-\$98.6M	-\$202.3M	-\$149.9M	-\$87.8M	-\$28.1M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M
Taxable income after carry foward	-\$98.6M	-\$202.3M	-\$149.9M	-\$87.8M	-\$28.1M	\$54.0M	\$108.7M	\$110.6M	\$96.4M	\$104.3M	\$116.1M	\$150.6M	\$95.2M	\$83.6M	\$59.5M	\$56.0M	\$60.7M	\$2.1M
Carry forward of exploration expense	-\$12.5M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M
Capital Allowance for development expense (3 years)	-\$28.7M	-\$63.3M	-\$64.9M	-\$37.5M	-\$3.2M	-\$1.9M	-\$.8M	-\$.8M	-\$.6M	-\$.3M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	-\$1.7M
Taxable income after carry forward and capital allowance	-\$41.2M	-\$63.3M	-\$7.7M	\$28.7M	\$57.3M	\$81.0M	\$108.8M	\$110.5M	\$95.9M	\$104.0M	\$116.1M	\$150.6M	\$95.2M	\$83.6M	\$59.5M	\$56.0M	\$60.7M	\$5.4M
Tax Rate		37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%
Tax	\$.0M	\$.0M	\$.0M	\$.0M	\$.0M	\$20.3M	\$40.8M	\$41.5M	\$36.2M	\$39.1M	\$43.5M	\$56.5M	\$35.7M	\$31.4M	\$22.3M	\$21.0M	\$22.8M	\$.8M

22.7 Sensitivity Analysis

For the purposes of the PEA, the evaluation is based on 100% of the Project cash flows before distribution of profits to equity owners. Economic sensitivities are presented for various scenarios:

- Discount rates of 5%, 7%, 8%, 9% and 10%
- Sensitivity ranges for operating and capital costs between +/- 40% of base case values
- Sensitivity ranges for TREO recoveries from 43% to 74%
- Sensitivity ranges for revenues (Basket Pricing) of -25% to +20%

Table 22-5 to Table 22-8 present the various sensitivity results. Figure 22-2 shows the Sensitivity Graph for the various ranges.

Table 22-5 After Tax NPV at Range of Operating Costs

Discount	60%	70%	80%	90%	100%	110%	120%	130%	140%
5%	\$625.1M	\$566.7M	\$508.2M	\$449.8M	\$391.0M	\$332.2M	\$273.1M	\$213.7M	\$153.9M
7%	\$509.0M	\$459.4M	\$409.9M	\$360.3M	\$310.3M	\$260.3M	\$210.0M	\$159.3M	\$108.1M
8%	\$459.9M	\$414.1M	\$368.3M	\$322.5M	\$276.2M	\$230.0M	\$183.3M	\$136.4M	\$88.8M
9%	\$415.8M	\$373.4M	\$331.0M	\$288.6M	\$245.7M	\$202.8M	\$159.5M	\$115.9M	\$71.7M
10%	\$376.1M	\$336.8M	\$297.5M	\$258.1M	\$218.3M	\$178.4M	\$138.1M	\$97.5M	\$56.3M

Table 22-6 After Tax NPV at Range of Capital Costs

	\$ 124.2	\$ 145.0	\$ 165.7	\$ 186.4	\$207.1M	\$227.8	\$ 248.5	\$ 269.2	\$ 289.9
Discount	60%	70%	80%	90%	100%	110%	120%	130%	140%
5%	\$475.0M	\$452.3M	\$430.8M	\$410.3M	\$391.0M	\$372.8M	\$355.7M	\$339.7M	\$324.5M
7%	\$378.0M	\$359.6M	\$342.2M	\$325.8M	\$310.3M	\$295.8M	\$282.3M	\$269.8M	\$257.9M
8%	\$337.2M	\$320.5M	\$304.9M	\$290.2M	\$276.2M	\$263.3M	\$251.3M	\$240.2M	\$229.7M
9%	\$300.6M	\$285.6M	\$271.5M	\$258.2M	\$245.7M	\$234.1M	\$223.4M	\$213.6M	\$204.2M
10%	\$267.9M	\$254.2M	\$241.5M	\$229.5M	\$218.3M	\$207.9M	\$198.4M	\$189.6M	\$181.3M

Table 22-7 After Tax NPV at Basket Price Levels

Discount	\$70	\$75	\$80	\$85	\$92	\$95	\$100	\$105	\$110
5%	\$141.0M	\$199.3M	\$257.2M	\$314.8M	\$391.0M	\$429.5M	\$486.7M	\$548.7M	\$600.7M
7%	\$97.8M	\$147.5M	\$196.8M	\$245.8M	\$310.3M	\$343.0M	\$391.4M	\$443.9M	\$487.8M
8%	\$79.6M	\$125.7M	\$171.3M	\$216.6M	\$276.2M	\$306.4M	\$351.2M	\$399.5M	\$440.1M
9%	\$63.4M	\$106.2M	\$148.5M	\$190.5M	\$245.7M	\$273.7M	\$315.0M	\$359.8M	\$397.3M
10%	\$48.9M	\$88.7M	\$128.0M	\$167.0M	\$218.3M	\$244.2M	\$282.6M	\$324.0M	\$358.8M

Table 22-8 After Tax NPV at Varying Recovery Ranges

Discount	43%	48%	53%	57%	59%	61%	64%	69%	74%
5%	\$100.8M	\$192.5M	\$283.1M	\$355.1M	\$391.0M	\$426.9M	\$480.7M	\$570.0M	\$659.3M
7%	\$63.5M	\$141.7M	\$218.8M	\$279.8M	\$310.3M	\$340.8M	\$386.3M	\$461.8M	\$537.4M
8%	\$47.8M	\$120.3M	\$191.7M	\$248.1M	\$276.2M	\$304.4M	\$346.5M	\$416.1M	\$485.8M
9%	\$33.8M	\$101.2M	\$167.4M	\$219.6M	\$245.7M	\$271.8M	\$310.7M	\$375.1M	\$439.5M
10%	\$21.4M	\$84.1M	\$145.6M	\$194.1M	\$218.3M	\$242.4M	\$278.5M	\$338.2M	\$397.9M

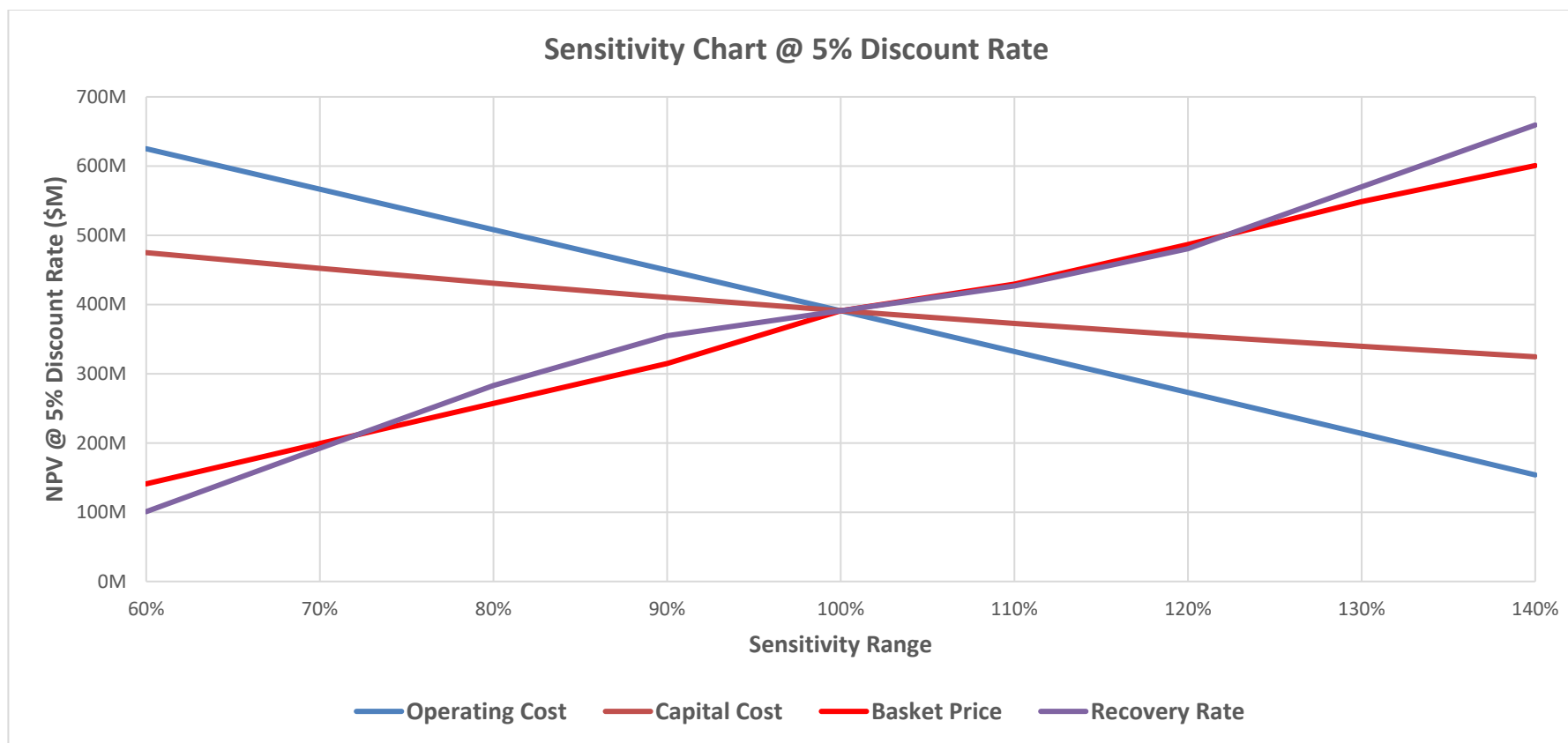


Figure 22-2 Sensitivity graph for the Various Grades

23 ADJACENT PROPERTIES

There are no adjacent properties of relevance to this report.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information available that is necessary to make the current technical report understandable and not misleading.

25 INTERPRETATION AND CONCLUSIONS

25.1 Mineral Resource Estimate

On behalf of NMI, MSA has completed a Mineral Resource estimate for Area 4 and Area 2B of the Lofdal Heavy Rare Earths 2B-4 Project.

The Mineral Resource was reported as Measured, Indicated and Inferred Mineral Resources as shown in Table 25-1 for Area 4 and Table 25-2 for Area 2B. The Mineral Resource was estimated using The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019) and is reported in accordance with the 2014 CIM Definition Standards, which have been incorporated by reference into National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).

In the QP's opinion, the Mineral Resources reported herein at the selected cut-off grade have "reasonable prospects for eventual economic extraction", taking into consideration mining and processing assumptions (refer to 14.11). The Mineral Resource was reported from within a Whittle optimised pit shell at a cut-off grade of 0.10% TREO.

Table 25-1 Area 4, Measured, Indicated and Inferred Mineral Resource Estimates above 0.1% TREO cut-off grade – May 12, 2021

Category	Tonnes (Mt)	TREO* %	HREO** %	LREO*** %	Dy ₂ O ₃ ppm	TREO (Kt)
Measured	5.93	0.21	0.14	0.07	138	12.71
Indicated	36.63	0.16	0.08	0.08	82	59.97
Measured & indicated	42.57	0.17	0.09	0.08	90	72.68
Inferred	6.09	0.17	0.07	0.09	72	10.12

Notes:

- (1) All tabulated data have been rounded and as a result minor computational errors may occur.
- (2) Mineral Resources, which are not Mineral Reserves, have no demonstrated economic viability.
- (3) Quantities reported are the total quantities for the project regardless of ownership.
- (4) *TREO = Total Rare Earth Oxides and includes Y₂O₃
- (5) **HREO = Heavy Rare Earth Oxides and includes Y₂O₃
- (6) ***LREO = Light Rare Earth Oxides
- (7) Mt = Million tonnes, kt = Thousand tonnes.

Table 25-2 Area 2B, Indicated and Inferred Mineral Resource Estimates above 0.1% TREO cut-off grade – May 12, 2021

Category	Tonnes (Mt)	TREO* %	HREO** %	LREO*** %	Dy ₂ O ₃ ppm	TREO (kt)
Indicated	2.20	0.19	0.10	0.09	104	4.27
Inferred	2.58	0.19	0.09	0.09	92	4.80

Notes:

- (1) All tabulated data have been rounded and as a result minor computational errors may occur.
- (2) Mineral Resources, which are not Mineral Reserves, have no demonstrated economic viability.
- (3) Quantities reported are the total quantities for the project regardless of ownership.
- (4) *TREO = Total Rare Earth Oxides and includes Y₂O₃
- (5) **HREO = Heavy Rare Earth Oxides and includes Y₂O₃
- (6) ***LREO = Light Rare Earth Oxides
- (7) Mt = Million tonnes, kt = Thousand tonnes.

The Area 4 Mineral Resource Estimate has increased significantly from the previous estimate of July 31, 2012, due to extensive step out drilling down dip and along strike from the previous mineral resource area. The Area 2B Mineral Resource Estimate is the first estimate for this Area and was the results of infill and step out drilling from the sparse grid drilled previously.

25.2 Capital and Operating Costs

The PEA includes open pits A4 and A2B. Each pit has a dedicated waste dump. Open pit production is delivered to a central mill complex on the property. The mill complex produces a mixed rare earth oxide final product.

Table 25-3 summarizes the estimated operating costs developed for the PEA.

Table 25-3 Total Operating Cost Summary

Description	Cost per Tonne Processed (USD/t)	Cost Per kg TREO Recovered (USD/t)
Mining	\$16.25	\$14.32
Mill Site Process Plant	\$32.00	\$28.21
General and Administration	\$1.41	\$1.25
Royalties	\$5.20	\$4.58
Total Operating Cost	\$54.86	\$48.36

The plans and costs presented in this PEA indicates a robust project with favourable economics. The mine plan and Total Rare Earth Element recoveries honour the mineral resource.

Financial analysis yields positive economic returns for the project with an Initial Direct Capital Costs US\$ 131.21 million and US\$ 75.87 million of sustaining Capital investment including mine closure cost, indirect and contingencies. Table 25-4 shows the capital requirements for the Lofdal Heavy Rare Earth Project “2B-4” Project.

Table 25-4 Total Capital Cost Summary

Description	Total Cost (USD)
Direct Mill Site Process Plant	\$117.58 M
Direct Tailings Storage Facility	\$13.63 M
Subtotal Initial Direct Capital Costs	\$131.21 M
Sustaining Capital Processing	\$6.01 M
Sustaining Capital Tailings Storage Facility	\$5.43 M
Mine Closure Costs	\$5.00 M
Indirect Costs	\$18.56 M
Contingency	\$40.87 M
Total Capital Cost Estimate	\$207.08 M

Based on the results of the PEA (Table 25-5), the following conclusions can be made:

- The Lofdal Heavy Rare Earths 2B-4 Project has significant potential to provide positive and robust returns.
- Additional exploration and expansion of the current resource, including the potential conversion of Inferred resources to Measured and Indicated.

Table 25-5 Summary Financial Results

Pre-Tax NPV @5%	\$632.7 M
Pre-Tax IRR	34%
Pre-Tax Net Cash Flow	\$1110.4 M
After Tax NPV @5%	\$391.0 M
After Tax IRR @5%	28%
After Tax Net Cash Flow	\$698.7 M

25.3 Opportunities

The PEA demonstrates that the Lofdal Heavy Rare Earths 2B-4 Project has the potential to be technically and economically viable. The Project is technically uncomplicated because of the near surface nature of the deposit and relatively simple access. Several opportunities for the project are available to further enhance the project:

- Underground resource at A4 and A2B pits;
- Expansion at each pit with additional resources;
- Optimize the overall production plan (mine/mill/chemical plant).

25.4 Risks

There are risks that have been identified within the recommendations.

- Main risks to project success would be:
- Changes in environmental regulations;
- Pilot plant testing should be performed to confirm flowsheet and lock cycle testing results;
- Chemical Plant flowsheet and pilot plant;
- The potential for additional area requirement for Tailings Storage facility;
- Availability of skilled labour during the construction phase.

26 RECOMMENDATIONS

It is recommended that NMI complete the following:

Mineral Resources

- Undertake an infill drilling programme in the portions of Inferred Mineral Resource falling within the Mineral Resource pit-shells to ensure that all Mineral Resources that have the potential to be converted to Mineral Reserves are in the Indicated category.
- The project area has potential to increase the Mineral Resources. Should additional Mineral Resources be required to support the project, further exploration would be warranted.

Mining

- No geotechnical study has yet been undertaken. This should be done if the project advances to PFS as this will have a material impact on the stripping ratio.
- A preliminary hydrological study should be commissioned to validate the assumption that there is no water related issues at depth.

Mineral Processing

- Recommendations for future opportunities as well as possible trade off studies for crushing circuit, comminution circuit optimization and future testwork.
- The current crushing circuit was designed to facilitate product sizing for sorter feed. With the sorting circuit removed from the PEA study there is an opportunity to further optimize the crushing circuit prior to milling, which may reduce mill sizing and power consumption by increasing reduction ratios across the crushing circuit prior to ball milling.
- The current milling circuit can be explored in further detail to optimize mill sizing and classification selection. Currently the circuit consists of 2 ball mills in series, both closed with cyclones. This was proposed to minimize mill sizing as a preference over open circuit milling. A more detailed look at the mill circuit in tandem to the classification regime may result in some additional optimization over the circuit.
- The next phase of study for the Lofdal project can introduce circuit optimization, variability, and clarification testwork regimes. These will provide further clarity to the circuit design. Some of these suggested campaigns are as follows:
 - Comminution variability testwork
 - Flotation locked cycle testwork
 - Variability testwork
 - Hydrometallurgical optimization testwork
 - Solvent Extraction testwork
 - Uranium and REE Precipitation testwork
 - Acid Neutralization testwork
 - Solid Liquid Separation testwork (Thickening and Filtration)
 - Geochemistry/tailings testwork
 - Piloting

Water Supply

- As per NMI, an alternative source for water (groundwater potential along a major fault zone) is a borehole with a depth of 74 m located 6.2 km south of the A4 Pit. Development of the water supply with quality/quantity test. Assessment of the groundwater potential in the area around this borehole is highly recommended.

Electrical Supply

- Complete details for the program called Renewable Independent Power Producer Programme (REIPPP) which is aimed at bringing additional megawatts onto the company's electricity system through private sector investment in solar, wind, biomass, and small hydro, among others.

Mine Access Road

- For the access road construction, topographic survey of the road alignment is essential to accurately quantify the cut and fill volumes including a geotechnical assessment.

REE Pricing

- The marketing study and basket price should be updated to reflect current market conditions when the pre-feasibility study is conducted.

Economic Analysis

- This PEA provides suitable economics to progress to the next stage of project development via a Pre-Feasibility study, with updated costs.

Tailings Storage Facility

- Complete at pre-feasibility level and alternative assessment for the TSF location in consideration of overall risk to the operation, waste, and water management practice. This should consider alternative technology and practice to determine the optimum waste and water management system for the project.
- Consider opportunities of cost reduction in the capital expenditures for the TSF through revised deposition strategy, staging and scheduling. There might be an opportunity to use pre-stripping material for construction material and to schedule non acid generating rock earlier for the containment walls.
- Initiate field and laboratory geotechnical testwork including foundation and tailings physical and geochemical characterisation during the PFS studies for the preferred site and tailings technology to advance the TSF design and confirm lining system requirements, as well as potential local borrow material.
- Develop the site water balance, freshwater requirements, and overall water management strategy for the project in consideration of the tailings management system.
- Develop a detailed dam breach analysis for the design dam at pre-feasibility or feasibility level to assess the risks to the downstream users, confirm dam classification and emergency response requirements

Overall

- Carry out a six-month PFS to further develop the engineering design of the plant and recognise value engineering where possible.
- Revisit the capital cost estimates in general for possible savings due to optimising the cost estimates from $\pm 50\%$ to $\pm 10\%$ (PFS Level).
- The budget for the recommendations provided is designed to collect the data required to complete a pre-feasibility study is estimated at \$USD 3.6 with the details provided in the following Table 26-1.

Table 26-1 Budget for Future Work

Tasks	Estimated Cost (US\$)
Geology (Major Upgrade Drilling Program)	1,000,000
Water Supply (Alternative Source)	50,000
Geotechnical and Hydrogeology	340,000
TSF Optimization/Studies	350,000
Environmental, Permitting and Community Relations	50,000
Mineral Processing (Metallurgy and Hydrometallurgy)	266,400
Mine Access PFS (Road Studies)	70,000
Engineering and Prefeasibility	1,500,000
Total	3,626,400

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28 DATE AND SIGNATURE PAGE

This report titled “NI 43-101 TECHNICAL REPORT ON THE LOFDAL HEAVY RARE EARTH PROJECT 2B-4 PRELIMINARY ECONOMIC ASSESSMENT (PEA), NAMIBIA” November 14th, 2022 (the “Technical Report”) for Namibia Critical Metals Inc. was prepared and signed by the following authors:

The effective date of the report is October 3, 2022.

The date of the report is November 14th, 2022.

Signed by:

"Original Signed and Sealed"

Qualified Person
Micheal Archer, Sr. Metallurgist
November 14th, 2022

Company
SGS Canada Inc.

Signed by:

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Qualified Person
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November 14th, 2022

Company
SGS Canada Inc.

Signed by:

"Original Signed and Sealed"

Qualified Person
Johnny Canosa, P.Eng.
November 14th, 2022

Company
SGS Canada Inc.

Signed by:

"Original Signed and Sealed"

Qualified Person
Veronique Daigle, Pr Eng.
November 14th, 2022

Company
Knight Piésold Consulting (Pty) Ltd.

Signed by:

"Original Signed and Sealed"

Qualified Person
Johann Hager, Pr. Eng.
November 14th, 2022

Company
BBMC (Pty) Ltd

Signed by:

"Original Signed and Sealed"

Qualified Person
Scott Swinden, P. Geo.
November 14th, 2022

Company
Swinden Geoscience Consultants Ltd

Signed by:

"Original Signed and Sealed"

Qualified Person
Jeremy Charles Witley, Pr. Sci. Nat
November 14th, 2022

Company
The MSA Group (Pty) Ltd.

29 CERTIFICATES OF QUALIFIED PERSONS

QP CERTIFICATE – MICHEAL ARCHER

To accompany the report entitled: NI 43-101 TECHNICAL REPORT ON THE PRELIMINARY ECONOMIC ASSESSMENT (PEA) Lofdal Rare Earths 2B-4 Project, Namibia dated November 11, 2022, and with an effective date of October 3, 2022.

I, Micheal Archer, P. Eng. of Peterborough, hereby certify that:

- a) I am a Senior Metallurgist for SGS Canada Inc, with an office located at 185 Concession Street, Lakefield, Ontario, Canada.
- b) I graduated from the University of Waterloo in 2007 (BASc. Chemical Engineering). I am a member of good standing of Professional Engineers Ontario (License #100185475). I have worked as a metallurgist for over 15 years since my graduation from university. My hydrometallurgical testing experience includes a variety of critical metals recovery projects (Li, Ni, Co, graphite, REE) from properties located globally. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- c) I have not conducted a site visit of the property.
- d) I am an author of this report and responsible for sections 1.5, 1.9 and 13. I have reviewed these sections and accept professional responsibility for these sections of this technical report.
- e) I am independent of Namibia Critical Metals as defined in Section 1.5 of National Instrument 43-101.
- f) I have had no prior involvement with the subject property.
- g) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
- h) As at the effective date of the technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- i) I have read National Instrument 43-101, Form 43-101F1 and confirm that this technical report has been prepared in accordance therewith.

Signed and dated this 11th day of November 2022 at Peterborough, Ontario.

"Original Signed and Sealed"

Micheal Archer, P.Eng.

QP CERTIFICATE – DANIEL MILLAR

To accompany the report entitled: NI 43-101 TECHNICAL REPORT ON THE PRELIMINARY ECONOMIC ASSESSMENT (PEA) Lofdal Rare Earth 2B-4 Project, Namibia dated November 14, 2022, and with an effective date of October 3, 2022.

I, Daniel Millar, Pr. Eng. of Johannesburg, hereby certify that:

I am a Principal Process Engineer for SGS Bateman SA., with an office located at 8 Greenstone Place, Edenvale, Johannesburg, 1609, South Africa.

- a) I am a Principal Process Engineering with SGS Bateman, with over 11 years' experience in the metallurgical industry, including process engineering roles in both operations and project environments. My experience covers mineral processing, hydrometallurgical processes and electrometallurgical processes. I have been involved in process design, study work, technology development and plant design for a number of uranium, gold, base metal and rare earth projects. I am a registered professional engineer (Pr.Eng.) with the Engineering Council of South Africa (ECSA-00040151). I hold a BSc. Chemical Engineering degree from the University of the Witwatersrand and I have contributed to the work presented in item 17 of this Technical Report. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- b) I have not conducted a site visit of the property.
- c) I am an author of this report and responsible for section 17. I have reviewed this section and accept professional responsibility for this section of this technical report.
- d) I am independent of Namibia Critical Metals as defined in Section 1.5 of National Instrument 43-101.
- e) I have had no prior involvement with the subject property.
- f) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
- g) As at the effective date of the technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- h) I have read National Instrument 43-101, Form 43-101F1 and confirm that this technical report has been prepared in accordance therewith.

Signed and dated this 14th day of November 2022 at Johannesburg, South Africa.

"Original Signed and Sealed"

Daniel Millar, Pr.Eng.

QP CERTIFICATE – WILLIAM VAN BREUGEL, P.Eng.

To accompany the report entitled: NI 43-101 TECHNICAL REPORT ON THE PRELIMINARY ECONOMIC ASSESSMENT (PEA) Lofdal Rare Earths 2B-4 Project, Namibia dated November 14, 2022, and with an effective date of October 3, 2022.

I, William van Breugel, P. Eng. of Saskatoon, hereby certify that:

- a) I am an Associate Mining Engineer for SGS Canada Inc, with an office located at 931 Greaves Crescent, Saskatoon, Saskatchewan, Canada.
- b) I graduated from the University of Waterloo in 1990 (BaSc (Hons). Geological Engineering). I am a member of good standing of the Association of Professional Engineers and Geoscientists of Saskatchewan (License #22452). I have worked as a mining engineer for over 31 years since my graduation from university. I have worked on precious metals, base metals, industrial commodities and diamond projects including mine operations and property evaluations. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- c) I have not conducted a site visit of the property.
- d) I am an author of this report and responsible for sections 1.11, 1.12, 1.14, 1.15, 1.16, 19, 20, 21, 22, 25.2. I have reviewed these sections and accept professional responsibility for these sections of this technical report.
- e) I am independent of Namibia Critical Metals as defined in Section 1.5 of National Instrument 43-101.
- f) I have had no prior involvement with the subject property.
- g) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
- h) As at the effective date of the technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- i) I have read National Instrument 43-101, Form 43-101F1 and confirm that this technical report has been prepared in accordance therewith.

Signed and dated this 14th day of November 2022 at Christopher Lake, Saskatchewan.

"Original Signed and Sealed"

William van Breugel, P.Eng.

QP CERTIFICATE – JOHNNY CANOSA, P. Eng.

To accompany the report entitled: NI 43-101 TECHNICAL REPORT ON THE PRELIMINARY ECONOMIC ASSESSMENT (PEA) Lofdal Rare Earths 2B-4 Project, Namibia dated November 14, 2022, and with an effective date of October 3, 2022.

I, Johnny Canosa, P. Eng. of Alberta, and Ontario, hereby certify that:

- a) I am a Senior Mining Engineer for SGS Canada Inc, with an office located at 3260 Production Way, Burnaby, BC V5A 4W4, Canada.
- b) I am a graduate of Bachelor of Science in Mining Engineering from Saint Louis University, Baguio City, Benguet, Philippines with diploma issue date on March 23, 1980. I am a member in good standing of the Association of Professional Engineers of Ontario (licence # 100509964) and the Association of Professional and Geoscientist of Alberta (licence #93946). My relevant experience includes more than 20 years of experience in mine engineering, mine planning and mining operation, including mine optimization, projects, open pit planning and scheduling, and mining consultancy.
- c) I have not conducted a site visit of the property.
- d) I am an author of this report and responsible for the preparation of sections 3, 18.1 to 18.8, 18.10, 18.13 to 18.17, and co-responsible for the preparation of sections 1, 2, and 26. I have reviewed these sections and accept professional responsibility for these sections of this technical report.
- e) I am independent of Namibia Critical Metals as defined in Section 1.5 of National Instrument 43-101.
- f) I have had no prior involvement with the subject property.
- g) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
- h) As at the effective date of the technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- i) I have read National Instrument 43-101, Form 43-101F1 and confirm that this technical report has been prepared in accordance therewith.

Signed and dated this 14th day of November 2022 at Surrey, British Columbia, Canada.

"Original Signed and Sealed"

Johnny Canosa, P.Eng.

QP CERTIFICATE – VERONIQUE DAIGLE, Pr. Eng.

To accompany the report entitled: NI 43-101 TECHNICAL REPORT ON THE PRELIMINARY ECONOMIC ASSESSMENT (PEA) Lofdal Rare Earths 2B-4 Project, Namibia dated November 14, 2022, and with an effective date of October 3, 2022.

I, Veronique Daigle, hereby certify that:

- a) I am employed as Lead Engineer and a Director at Knight Piésold Consulting (Pty) Ltd (registration 2008:0657), at 11 Nelson Mandela, Klein Windhoek, Windhoek, Namibia.
- b) I graduated with a Civil Engineering Degree (Cooperative Program) in 2006 from the Université de Sherbrooke, in the Province of Québec, Canada.
- c) I am a member in good standing with the Engineering Council of Namibia, and Registered as Professional Engineer (license number PE2017-19) since 2017.
- d) My Relevant Experience includes 15 years of continuous experience in tailings, geotechnical engineering and water management employed at Knight Piésold. I am a member in good standing with the South African Committee on Large Dams, the Canadian Dam Association, and the Ordre des Ingenieurs du Quebec, Canada.
- e) I have not conducted a site visit of the property.
- f) I am a co-author of this report and responsible for the preparation of Sections 18.9, 18.11 and 18.12. I have reviewed these sections and accept professional responsibility for these sections of this technical report.
- g) I am independent of the issuer, Namibia Critical Metals, as defined in Section 1.5 of National Instrument 43-101.
- h) I have had no prior involvement with the subject property.
- i) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
- j) As at the effective date of the technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- k) I have read National Instrument 43-101, Form 43-101F1 and confirm that this technical report has been prepared in accordance therewith.

Signed and dated this 14th day of November 2022 at Windhoek, Namibia.

"Original Signed and Sealed"

*Veronique Daigle, Pr. Eng., Director at Knight Piésold
Consulting (Pty) Ltd*

QP CERTIFICATE – JOHANN HAGER, Pr Eng

To accompany the report entitled: NI 43-101 TECHNICAL REPORT ON THE PRELIMINARY ECONOMIC ASSESSMENT (PEA) Lofdal Rare Earths 2B-4 Project, Namibia dated November 14, 2022, and with an effective date of October 3, 2022.

I, Johann Hager, (Pr Eng) of Pretoria, South Africa hereby certify that:

- a) I am a Principal Mining Engineer for BBMC (Pty) Ltd, - Bergbau Beratung Mining Consulting - with an office at 261 Hans Strijdom Ave, Lyttelton 0157, Pretoria, South Africa.
- b) I am a graduate of the University of Pretoria (South Africa) with the following degrees:
 - BSc (Mining Engineering), University of Pretoria
 - BEng (HONS) Mining, University of Pretoria
 - MEng (Mining), University of Pretoria – No 78095222, dated 2014/09/05
- c) I am a member of good standing of the Association of Registered Professional Engineers of South Africa, ECSA No 20050209 and an associate member of The Southern African Institute of Mining and Metallurgy – No 20224
- d) My relevant experience includes more than 40 years of experience in mine engineering, mine planning and mining operation, including mine optimization, projects, open pit planning and scheduling, and mining consultancy.
- e) I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- f) I have not personally inspected the Property.
- g) I am an author of this report and responsible for sections 1.8 and 16. I have reviewed these sections and accept professional responsibility for these sections of this technical report.
- h) I am independent of Namibia Rare Earths Inc. as defined in Section 1.5 of National Instrument 43-101.
- i) I have had no prior involvement with the subject property.
- j) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
- k) As at the effective date of the technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- l) I have read National Instrument 43-101, Form 43-101F1 and confirm that this technical report has been prepared in accordance therewith.

Signed and dated this 14th day of November 2022, Pretoria, South Africa

"Original Signed and Sealed"

Johann Hager (Pr Eng), 20050209 – ECSA, RSA

QP CERTIFICATE – SCOTT SWINDEN, P. Geo.

This Certificate of Qualified Person has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects, Part 8.1.

I, Scott Swinden, do hereby certify that:

1. I am a Principal Geoscience Consultant of:
Swinden Geoscience Consultants Ltd
224 Main Street
Wolfville, Nova Scotia
B4P 1C4
Canada
2. This certificate applies to the technical report titled “NI 43-101 Technical Report on the Preliminary Economic Assessment (PEA) Lofdal Rare Earths 2B-4 Project, Namibia dated November 14, 2022, and with an effective date of October 3, 2022.”
3. I graduated with a BSc (Hons) degree in Geology from Dalhousie University in 1970. In addition, I obtained a Master of Science degree in Geology from Memorial University of Newfoundland and Labrador in 1975 and Ph.D. in Earth Sciences from Memorial University of Newfoundland and Labrador in 1988.
4. I am a registered Professional Geologist (P.Geo.) with the Association of Professional Engineers and Geoscientists of Nova Scotia.
5. I have worked as a geologist for a total of 49 years, including exploration for base and precious metals and rare metals (including the rare earth elements), as a research scientist for provincial and federal geological surveys, and as manager and executive for provincial geological surveys. In the latter role, I was responsible for mineral resources programs (including programs focussed on Rare Earth Elements) throughout Newfoundland and Labrador from 1988 to 1996. From 2010 to the present, I have specialized in the study and exploration of Rare Earth Deposits. I explored for REE in various parts of Namibia. I was part of the discovery team at the Lofdal deposit and led the geological mapping and geological modelling of the deposit. I have worked at a number of other REE prospects including Swartbooisdrift (Kunene Region), Epembe (Ta-Nb-REE in carbonatite), Marinkas Quellen). I have taught mineral deposits including the geology of Rare Earth Deposits to 4th year undergraduates at Dalhousie and Acadia Universities. I have recently served as a member of the Expert Council for the HiTech AlkCarb project, a European Union project investigating geomodels for REE deposits. I believe my experience in operating and managing REE exploration projects, studying REE deposits and teaching about REE deposits, make me a Qualified Person for a REE project.
6. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a “qualified person” for the purposes of NI 43-101.
7. I visited the Lofdal Property for 14 days from 29 May to 11 June 2014, for 1 day on July 31, 2017, and for 1 day on April 25, 2018.

8. I am responsible for the preparation of items 4 to 11 and co-responsible for, the preparation of items 1, 2 and 24 to 27 of the Technical Report.
9. I have been involved with the property that is the subject of the Technical Report as a paid, independent geoscience consultant since July, 2010. All prior involvement is with the current issuer at various stages of the project for which remuneration has been received as an independent consultant that is not contingent on the outcome of the study work and did not involve management of the project.
10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
11. I am independent of the issuer according to the definition of independence described in section 1.5 of National Instrument 43-101.
12. I have read National Instrument 43-101 and Form 43-101F1 and, as of the date of this certificate, to the best of my knowledge, information and belief, those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated 14 November 2022.

"Original Signed and Sealed"

Signature of Qualified Person

Scott Swinden
Principal Geoscience Consultant
Swinden Geoscience Consultants Ltd

QP CERTIFICATE – JEREMY CHARLES WITLEY, Pr. Sci. Nat

I, Jeremy Charles Witley do hereby certify that:

1. I am a Principal Mineral Resource Consultant of:
The MSA Group (Pty) Ltd
Henley House, Greenacres Office Park,
Victory Park, Randburg, 2195
South Africa.
2. This certificate applies to the technical report titled “Lofdal Heavy Rare Earths Project 2B-4 Preliminary Economic Assessment (PEA) Namibia” effective October 3, 2022 and dated November 14, 2022 (the “Technical Report”).
3. I graduated with a BSc (Hons) degree in Mining Geology from the University of Leicester in 1988. In addition, I obtained a Master of Science degree in Engineering from the University of Witwatersrand in 2015.
4. I am a registered Professional Natural Scientist (Geological Science) with the South African Council for Natural Scientific Professions (SACNASP) and a Fellow of the Geological Society of South Africa.
5. I have worked as a geologist for a total of 34 years. I have worked in a number of roles, including senior management, in mine geology, exploration projects and Mineral Resource management. I have conducted Mineral Resource estimates, audits and reviews for a wide range of commodities and styles of mineralization including complex mixed distribution multi element deposits. Specific REE experience includes deposits in Burundi, Mauritania, South Africa and Malawi, as well as the Lofdal deposit in Namibia.
6. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a “Qualified Person” for the purposes of NI 43-101.
7. I visited the Lofdal Property for 3 days from 28 to 30 October 2020 and again on 10 November 2022.
8. I am responsible for the preparation of Items 1.6, 12, 14 and 25.1 of the Technical Report and co-responsible for Items 2, 24 and 27.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
11. I am independent of the issuer according to the definition of independence described in section 1.5 of National Instrument 43-101.
12. I have read National Instrument 43-101 and Form 43-101F1 and, as of the date of this certificate, to the best of my knowledge, information and belief, those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 14th day of November 2022.

“signed and stamped”
(Jeremy Charles Witley, Pr. Sci. Nat)

30 APPENDICES

Appendix A:
Process Plant Site Layout
(see page 404)

Appendix B:
Tailings Storage Facility Conceptual Design report from Knight Piésold
(see page 406)

Appendix C:
Initial Site Visit To Lofdal REE Project report from Knight Piésold
(see page 447)

Appendix D:
Technical Memorandum from Light Deep Earth (LDE)
(see page 455)

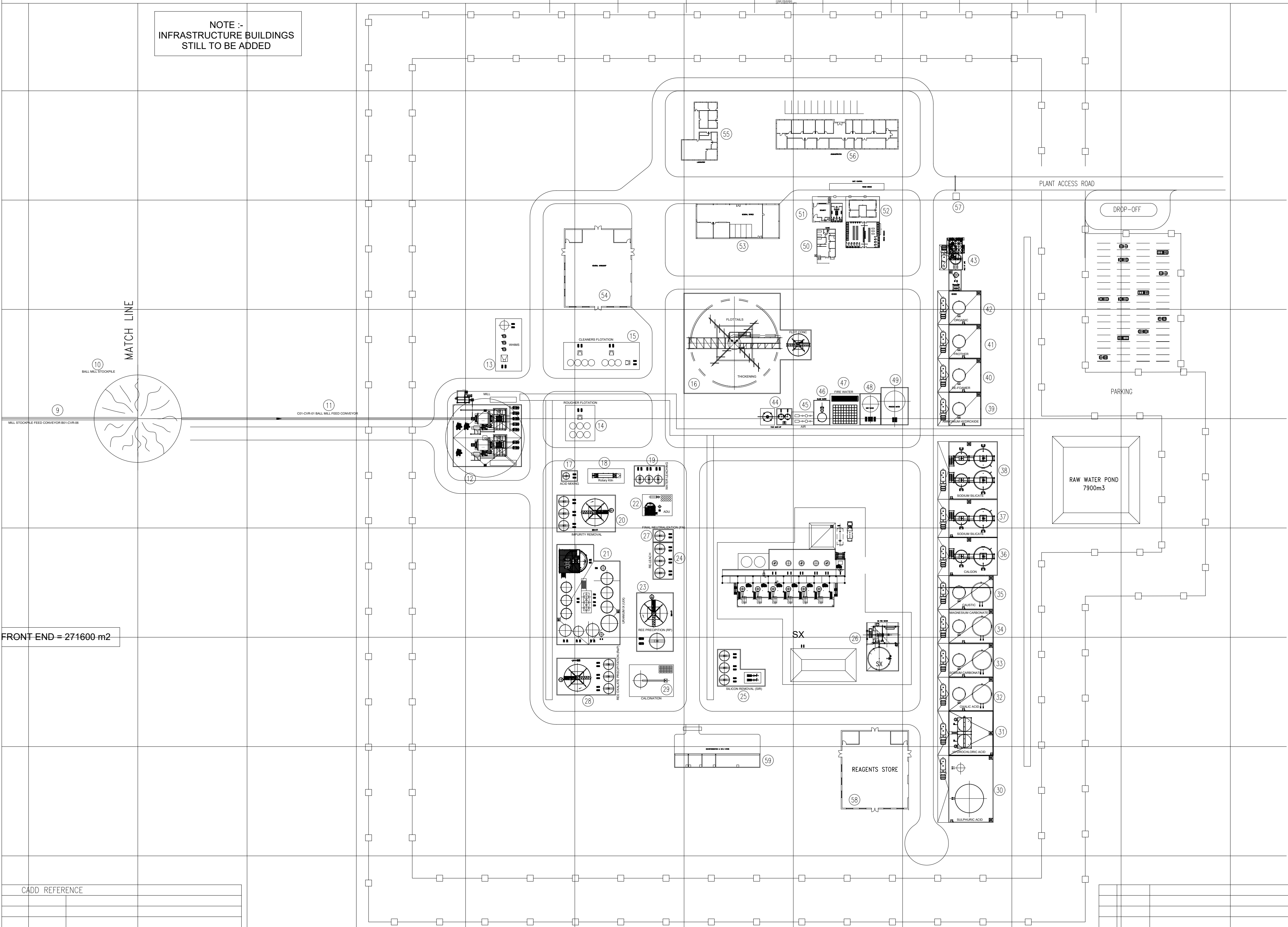
Appendix E:
HLS and Bond Testing on Lofdal Samples report from Geolabs Global Pty Ltd
(see page 479)

Appendix F:
An Investigation into Hydrometallurgical Recovery of Rare Earth Elements from the Lofdal Deposit
prepared for Namibia Critical Metals and dated January 20, 2022 from SGS Canada Inc.
(see page 501)

Appendix G:
An Investigation into Hydrometallurgical Recovery of Rare Earth Elements from the Lofdal Deposit
prepared for Namibia Critical Metals and dated October 7, 2022 from SGS Canada Inc.
(see page 658)

Appendix H:
An Investigation into Metallurgical Testwork on Samples from The Lofdal Heavy Rare Earth Project
prepared for Namibia Critical Metals from SGS Canada Inc.
(see page 692)

Appendix A:
Process Plant Site Layout



No.	DESCRIPTION
1.	TIP AND PRIMARY CRUSHING
2.	SACRIFICIAL CONVEYOR B01-CVR-01
3.	TRANSFER TOWER AND BELT MAGNET
4.	CRUSHER DUEL PRODUCT CONVEYOR B01-CVR-02
5.	SCREENING BUILDING
6.	CONE CRUSHER RE-CIRCULATION CONVEYOR No. 1 B01-CVR-03
7.	SECONDARY CRUSHING BUILDING (CONE CRUSHING)
8.	CONE CRUSHER RE-CIRCULATION CONVEYOR No. 2 B01-CVR-04
9.	MILL STOCKPILE FEED CONVEYOR B01-CVR-06
10	BALL MILL STOCKPILE B02-STK-02
11	BALL MILL 1 FEED CONVEYOR C01-CVR-01
12.	MILL AREA
13	WHIMS
14.	ROUGHER FLOTATION
15.	CLEANER FLOTATION
16.	FLOTATION THICKENERS
17.	ACID MIXING
18.	SULPHATION (Rotary Kiln)
19.	WATER LEACH (WL)
20.	IMPURITY REMOVAL
21.	URANIUM (IX)
22.	ADU
23.	REE PRECIPITATION
24.	RE-LEACH (RL)
25.	SILICON REMOVAL
26.	TH SX (THSX)
27.	FINAL NEUTRALIZATION (FN)
28.	REE OXALATE PRECIPITATION (ReP)
29.	CALCINATION AND PACKING REAGENTS
30.	SULPHURIC ACID
31.	HYDROCHLORIC ACID
32.	OXALIC ACID
33.	SODIUM CARBONATE
34.	MAGNESIUM CARBONATE
35.	CAUSTIC
36.	CALCON
37.	SODIUM SILICATE
38.	COLLECTOR
39.	AMMONIUM HYDROXIDE
40.	DE-FOAMER
41.	FROTHER
42.	ORGANIC
43.	MILK OF LIME
	SERVICES
44.	FLOCCULANT MAKE-UP
45.	PLANT AND INSTRUMENT AIR
46.	GLAND SEAL WATER
47.	FIRE WATER
48.	RAW WATER
49.	PROCESS WATER
	INFRASTRUCTURE
50.	CLINIC
51.	SECURITY
52.	CHANGE HOUSE
53.	GENERAL STORE
54.	GENERAL WORKSHOP
55.	LABORATORY
56.	ADMIN OFFICE
57.	GATE HOUSE AND BOOM
58.	REAGENTS STORE
59.	DE-CONTAMINATION AND CAT 3 STORE

NOTE :-
GRID IS 50M SQUIRE

CADD REFERENCE	
DWG NO.	TITLE
	REFERENCE DRAWINGS

[illegible]

Appendix B:
Tailings Storage Facility Conceptual Design report from Knight Piésold

Prepared for
SGS Bateman (Pty) Ltd
Woodmead North Office Park
54 Maxwell Drive
Waterfall, Gauteng
South Africa, 2191

Prepared by
Knight Piésold Consulting (Pty) Ltd.
11 Nelson Mandela
Klein Windhoek
Windhoek, Namibia
PO Box 80682, Eros

RI301-00928/01-A

LOFDAL RARE EARTHS PROJECT

TAILINGS STORAGE FACILITY CONCEPTUAL DESIGN REPORT

Rev	Description	Date
0	Issued in Final	21 October, 2022

EXECUTIVE SUMMARY

Knight Piésold Consulting (Pty) Ltd. was appointed by SGS Bateman to update the conceptual design for the Tailings Storage Facility (TSF) for the Lofdal heavy rare earth project in Namibia. The Lofdal project is developed in a joint venture between Namibia Critical Metals Inc (NCMI) and Japan Oil, Gas and Metals National Corporation (JOGMEC), and comprises two open pit mine, processing facility, waste rock dumps and TSF. The project is located in the northwest Kunene region of Namibia near Khorixas. NCMI and JOGMEC are completing a Preliminary Economic Assessment and 43-101 Technical Instrument for the project for which the conceptual design will be used.

The proposed TSF is a valley impoundment with an earthfill and waste rock embankment storing thickened slurry tailings on the eastern side of the site infrastructures layout and main open pit. The site was identified as a preferred location by Namibia Critical Metals for the conceptual design. The TSF comprises an earthfill starter embankment with liner system over the embankment face and basin, and a ring type deposition system with spigots spaced evenly around the main embankment and sides of the TSF. The raising strategy is envisaged to be downstream for the first raise using waste rock from the first 2 years of operation and upstream afterwards to the final elevation. The decant water strategy was assumed gravity decant with a starter and final decant.

The conceptual design, risk and opportunities and recommendations for future development of the Lofdal Rare Earths Project mining residue strategy is summarized as follows:

Concept Design Assumptions:

- Tailings throughput: Average 1.6 million tonnes per annum (Mtpa), maximum 2.0 Mtpa, at 46% solids content by mass.
- Total storage requirement: 26.7 million tonnes (Mt).
- Life of Mine: 16 years.
- Maximum rate of rise: 2.5 meter per year for upstream raises (m/yr.).
- Tailings geochemistry and classification assumed to require lining system.
- TSF classification (ICMM; UNEP; PRI, 2021): High due to consequence of failure on the pit and mining operation and risk to life.

TSF Geometry

- Starter embankment elevation: 984.5 m.
- TSF final elevation: 999.5 m.
- Total TSF height: 33 m.
- Conceptual freeboard requirement for a 1 in 2,475 years storm event: 1.5 m.
- Overall outer slope: 1 vertical to 4 horizontal (1v:4h) including benches, waste rock buttress and upstream raises.
- Tailings deposition through a ring feed system and includes a pipe laydown access road, decant system, and return water dam.

Quantities and cost estimates:

- Starter wall earthfill volume: 340,000 m³.
- On-going waste rock fill placement for wall raising: 85,000 m³ / year (for 2 years but could be sustained afterwards).

- Embankment face and basin HDPE liner surface area: 500,000 m² in year 1, and 450,000 m² over year 1 and year 2 to complete the basin lining along the valley impoundment.
- CAPEX – Year 0: 230 million Namibian Dollars (MNAD); Year 1: 50 MNAD; Year 2: 50 MNAD. On going sustainable capital for full lining: 50 MNAD. This is equivalent to approximately 14 NAD per tonne of tailings.

Risk and Opportunities:

- Risk of lack of storage should the tailings storage requirement increases.
- Opportunity to reduce the starter embankment size and use more waste rock as a raising strategy during the life of mine.
- Opportunity to consider filtered tailings storage to minimize freshwater requirement, achieve better storage density and possibly combine waste and tailings storage strategy. There is an opportunity to limit the extent of the liner requirement through a more compact stacking system using filtered tailings. Line facilities for filtered tailings must give consideration to drainage and must avoid developing pore pressure in the stack.

Recommendations:

- Complete at pre-feasibility level and alternative assessment for the TSF location in consideration of overall risk to the operation, waste, and water management practice. This should consider alternative technology and practice to determine the optimum waste and water management system for the project.
- Consider opportunities of cost reduction in the capital expenditures for the TSF through revised deposition strategy, staging and my scheduling. There might be an opportunity to use pre-stripping material for construction material and to schedule non acid generating rock earlier for the containment walls.
- Initiate field and laboratory geotechnical test work including foundation and tailings physical and geochemical characterisation during the PFS studies for the preferred site and tailings technology to advance the TSF design and confirm lining system requirements, as well as potential local borrow material.
- Develop the site water balance, freshwater requirements, and overall water management strategy for the project in consideration of the tailings management system.
- Develop a detailed dam breach analysis for the design dam at pre-feasibility or feasibility level to assess the risks to the downstream users, confirm dam classification and emergency response requirements.

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APPENDICES

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Appendix B	Concept Drawings
Appendix C	GISTM Consequence Classification Table
Appendix D	Stability Analysis Results - Peak Strength
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ABBREVIATIONS

CAPEX	Capital Expenditure
CDA	Canadian Dam Association
DCC	Dam Consequence Classification
EMP	Environmental Management Plan
FOS	Factor of Safety
GISTM	Global Industry Standard on Tailings Management
KP	Knight Piésold Consulting (Pty) Ltd.
LoM	Life of Mine
NMET	Namibian Meteorological Services
NRE	Namibia Rare Earths Inc.
OPEX	Operating Expenditure
PEA	Preliminary economic assessment
RoM	Run of Mine
RoR	Rare of Rise
RWD	Return Water Dam
SANS	South Africa National Standard
tpa	Tonnes per annum
tpd	Tonnes per day
SSC	Stage Capacity Curves
TSF	Tailings Storage Facility
WMO	World Meteorological Organization
WRD	Waste Rock Dump
ZOI	Zone of Influence

1.0 INTRODUCTION

1.1 PROJECT DESCRIPTION

Namibia Rare Earths Inc. (NRE) intends to develop the Lofdal Rare Earths Project located approximately 25 km NW of the town of Khorixas in the Kunene Region in Namibia (see Figure 1.1). The Lofdal project is developed in a joint venture between Namibia Critical Metals Inc (NCMI) and Japan Oil, Gas and Metals National Corporation (JOGMEC). The key project components consist of two proposed open pit mines (Pit Area 2B and Pit Area 4B) with their respective waste rock dumps (WRD), the plant site, a possible solar power plant, a topsoil stockpile area adjacent to the Area 4B WRD, several access and haul roads, Run of Mine (RoM) Stockpile Area and a valley confined Tailings Storage Facility (TSF) to store thickened slurry tailings.

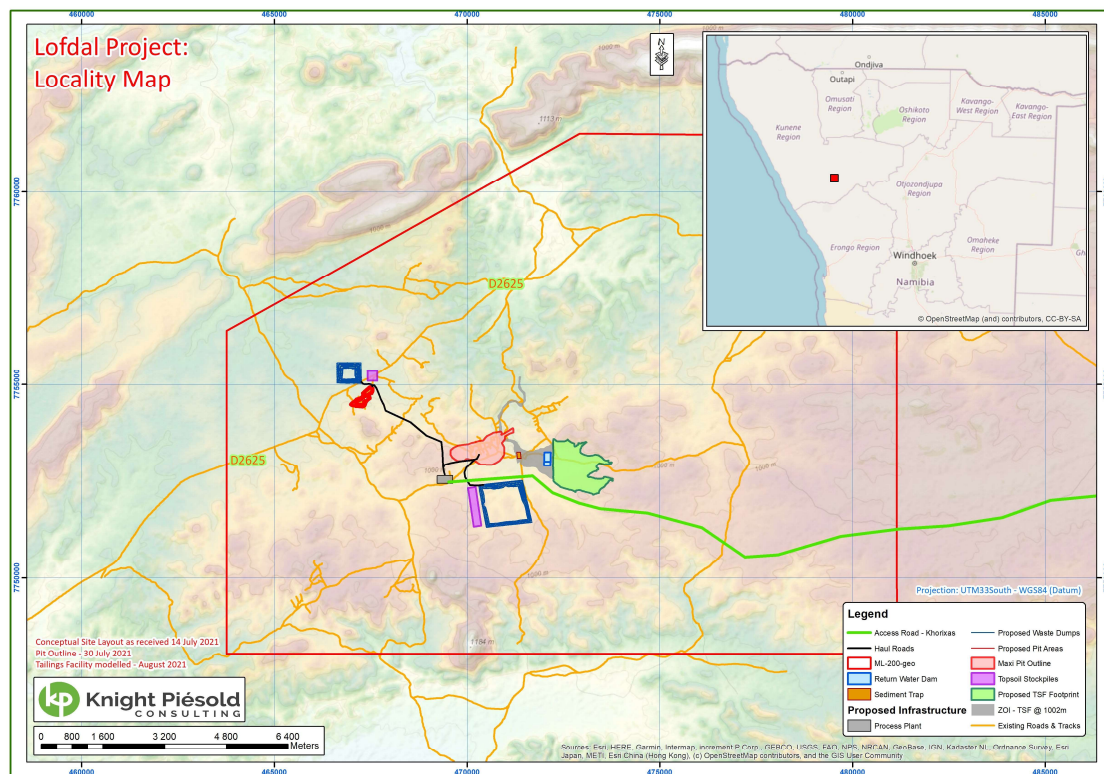


Figure 1.1 Project location

A Preliminary Economic Assessment (PEA) was previously conducted in 2014 and included conceptual designs for the TSF. Since then, however, production rates, the life of mine (LoM) and location of the TSF site have changed significantly, which in turn necessitated a need to update the 2014 TSF conceptual design to ensure alignment with the new LoM, production volumes and the newly selected site for the TSF. NRE appointed Knight Piésold Consulting (Pty) Ltd (KP) to update the 2014 conceptual design for the TSF. The project aims to extract approximately 26.8 Mt of resource, of which approximately 67% would be subjected to metallurgical processing to produce Rare Earth Elements (REE) concentrate over an anticipated 16-year Life of Mine (LoM).

1.2 PROJECT SCOPE

The scope of work for this conceptual study performed by KP can be summarized as follows:

- Conceptual design of the TSF including capacity modelling and development of stage capacity curves for the preselected TSF area.
- Conduct basic hydrological assessment for the TSF site location and a basic water balance for design of the decant system to deliver water from the TSF to the Return Water Dam (RWD).
- Determine preliminary safety classification and identify the zone of influence (ZOI) at final height for the TSF.
- Conceptual design of RWD.
- Preliminary stability and seepage analysis for the final TSF height.
- Conceptual drawings of the TSF and associated infrastructure showing the overall layout plan, some typical sections, and details.
- Compile a priced BoQ in Namibian dollars based on local rates, with an accuracy level of $\pm 40\%$.
- Compile a concept design report.

Note that other TSF infrastructure such as roads, stormwater diversion ditches, collection ditches, sediment ponds, and fencing will only be shown as indicative lines on drawings with no design work at this stage.

2.0 DESIGN OBJECTIVE AND CRITERIA

2.1 DESIGN OBJECTIVE

The principal objectives of the TSF and related water management infrastructure designs are summarized as follows:

- Provide a facility for permanent and secure storage of thickened slurry tailings during LoM and post closure.
- Preliminary layout of TSF and related access roads plus surface water management structures.
- Control and management of surface water during operations.

The design for the TSF is based on guidelines from accepted local and international standards for mine waste management design, surface water management design and infrastructure design (Canadian Dam Association - CDA, 2014/ 2019; Mining Association of Canada, 2017; Namibia Roads Authority Drainage Manual, 2014a; Namibia Roads Authority Materials Manual, 2014b; the South African National Standards (SANS) 10286; and the Global Industry Standard on Tailings Management (GISTM, 2020).

2.2 DESIGN CRITERIA

KP compiled a design criteria document (attached – Appendix A) which was submitted to NRE and the rest of the project team for review and approval. The document has been subsequently amended to indicate the revised mining schedule and TSF storage requirements. Table 2.1 presents the general information availed for the conceptual design of the TSF.

Table 2.1 Design criteria – summary

Item	Design Criteria
1. Topographical Survey	June 2020 topographic survey
2. Legal Framework	Minerals (Mining and Prospecting) Act of 1992
	Environmental Management Act of 2007
	Global Industry Standard on Tailings Management (2020)
3. Documentation	Preliminary Economic Assessment Report – Lofdal Project, 2014
	Updated Environmental Management Plan – Lofdal Project, 2016
4. Specific Gravity (Solids)	2.78
5. Particle Size Distribution	P80 = 35 microns (silts and finer)
6. Tailings solid content by weight	45.97 %

Item	Design Criteria
7. Average settled dry density (Year 1)	1.25 t/m ³
8. Average settled dry density (after Year 1)	1.35 t/m ³
9. Recoverable grade	0.017%
10. ROM split	100% (resource)
11. Average annual throughput to TSF	- 1.49 Mtpa (year 1 & 2) - 2.02 Mtpa (year 3 & 12) - 1.19 Mtpa (year 13 to 15)
12. Design Life	15 Years and Beyond
13. Rate of Rise	2.5 m/year for tailings not impounded behind starter wall (justified by the fine-grained nature of the tailings)
14. Storage Capacity Required	26.8 Mt
15. Tailing's chemistry	Tailings are anticipated to be non-acid generating since the resource body does not contain detectable sulphide material and has abundant neutralization potential. Tailings are expected to contain a certain level of radioactive uranium. Considering the lack of geochemistry testing and information it is assumed to classify as hazardous and requires a liner.
16. Slope Stability	The minimum Factor of Safety for slope stability under normal Operating conditions (local and global stability) will be: <ul style="list-style-type: none"> • Temporary slopes (during or at end of construction): 1.3 • Permanent slopes (during operation and post closure): 1.5 • Post Peak Static Loading Conditions: 1.1 • Pseudo-static: 1.0
17. Overall Outer Side Slope	Assumes 1V:4H (also deemed suitable for closure rehabilitation)
18. Conceptual Closure Design	The TSF side slopes are to be clad with 750 mm thick layer of waste rock. There will be no bench drains. The penstock(s) will be sealed. The TSF basin is to be covered with 200 mm of topsoil.

3.0 TSF CONCEPT DESIGN

3.1 CAPACITY ASSESSMENTS

3.1.1 TAILINGS THROUGHPUT

The Lofdal deposit is a low-grade deposit with an anticipated recoverable grade of approximately 0.017%. Initially, the process philosophy included a preliminary resource sorting where 33% of ROM would be sent to a Waste Rock Dump. The remaining 67% would go through the mill plant and ultimately to the TSF. The process has been revised and a conservative assumption was made that the entire 100% fraction of the ROM that ends up at the mill will ultimately end up in the TSF.

A total storage capacity of 26.8 million tonnes (19.9 million m³ at 1.35 t/m³) is therefore required at the TSF over the 16-year, LOM. The previous conceptual TSF design was modelled on an anticipated 15-year LOM. KP further revisited the storage capacity to determine the size of the TSF to final height as well as the size of the starter embankment required to contain tailings in the early years of deposition. The TSF location was maintained the previously selected site by NRE. The storage capacity assessment carried out is document in subsequent sections.

3.1.2 STORAGE CAPACITY ASSESSMENT AND CURVES

The location of the TSF was maintained as the site selected by NRE. Accordingly, a new facility height was adopted, and a conceptual level layout was developed using Muk3D software. To accommodate the full tailings volumes over the LOM the crest of the outer wall would have to be raised to an approximate elevation of 999.5 mamsl, which would equate to a total wall height of approximately 33.5 m. A total freeboard of 1.5 m was assumed in accordance with CDA (2019) guidelines. Figure 3.1 shows the proposed TSF layout.

The filling curve for the TSF are presented in Figure 3.2. The tailings storage facility starter wall and final tailings crest levels have been calculated with the stage capacity curves (SCC) developed during the capacity assessment. From the SCC, beach elevations lower than 981 mamsl result in a rate of rise (RoR) higher than the allowable of 2.5 m/year. From the stage capacity curves it is determined that it would take approximately three (3) years and four (4) months of deposition to reach a rate of rise below 2.5 m/year. Above this elevation upstream wall raising with tailings can safely commence up to a crest elevation of 999.5 mamsl. At this crest elevation the facility will have an approximate total height of 33.5 m.

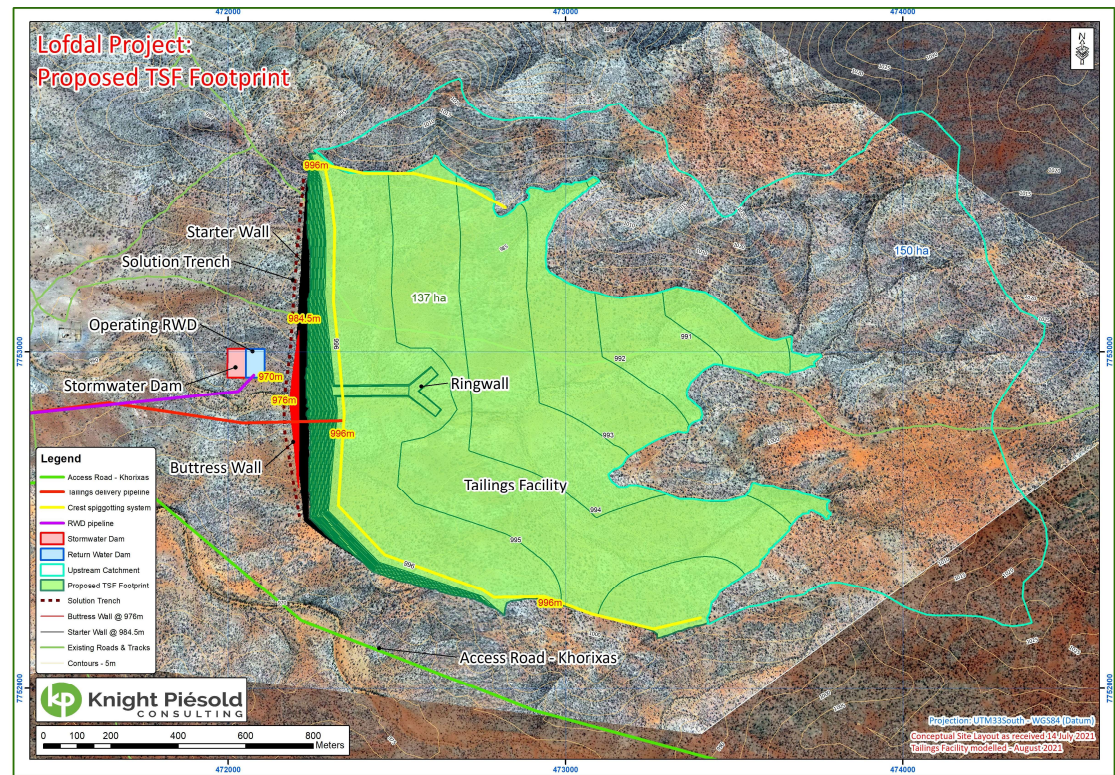


Figure 3.1 Conceptual layout of TSF

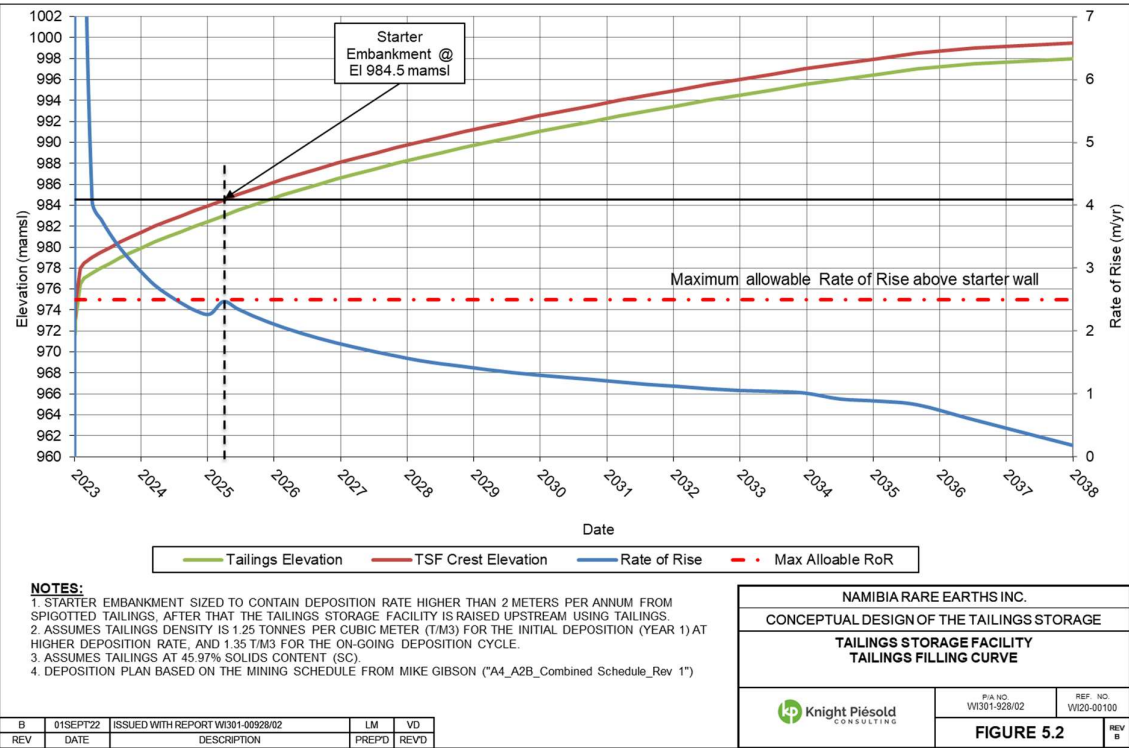


Figure 3.2 TSF filling curve and estimate of starter wall requirement

3.2 TSF COMPONENTS

3.2.1 STARTER EMBANKMENT

The TSF will consist of a starter wall with crest width of 10 m and crest elevation of 984.5 mamsl. The height of the starter wall will vary from 0 to 19 m high across the valley. The starter wall will be constructed from overburden rockfill borrowed from its footprint and areas surrounding the TSF with an outer wall slope of 1V:2.5H and an inner wall slope of 1V:2H and placed on shallow bedrock.

As soon as sufficient waste rock becomes available from open pit stripping and mining, a waste rock buttress will be constructed on the downstream slope of the starter embankment. The buttress crest should be 10 m wide extend to the height of the starter embankment. Further waste rock placement on the downstream face of the self raising outer should be made in the following years.

3.2.2 SELF RAISING OUTER WALL

Above the starter embankment upstream construction with tailings will entail the following:

- Establishment of an appropriate step-in from the previous embankment raise to maintain an overall slope angle of 1V:4H. The outer wall will be raised to El. 999.5 mamsl, equating to a total height of approximately 33.5 m.
- Construction of consecutive machine built 1 to 1.5 m high tailings lifts around the TSF perimeter.
- Filling of the paddocks in 1 – 1.5 m lifts with tailings via spigots

3.2.3 DESCANT SYSTEM

At this stage of the study, estimations of the required decant rate have been made with assumptions. It assumes a gravity decant system with a primary intake and secondary (final) intake tower with penstock system conveying water to the RWD. Provision has been made for the primary decant system cost and a secondary one to decant any localised pool that may form due to the double prong shape of the valley the TSF has been placed on. It is recommended in future design state to consider a trade off study with a barge decant system in the northeastern valley where the pool is to be located. Current estimates are estimated sufficient for either or option.

3.2.4 RETURN WATER DAM

The RWD should have a storage of 35 000 m³ to store decant water, and flood/stormwater event requirement for the dam classification. It is envisaged to split the RWD into two compartments, for operational purposes and evaporation mitigation. The maximum monthly decanted water volume is 119 125 m³/month.

3.2.5 TSF AND RWD LINER

The tailings are assumed to have some amount of radioactive uranium. The TSF basin should be lined considering the possible nature tailings and the lack of information on hydrogeological setting.

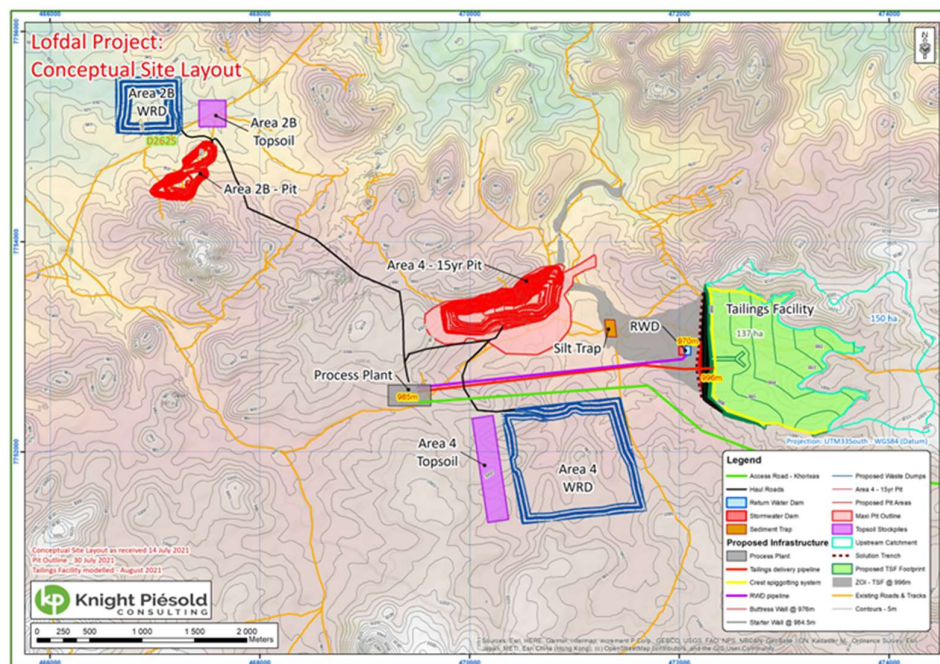
The proposed liner system will comprise a 150 mm bedding layer with a 1.5 mm HDPE geomembrane placed on top and covered by a protective A6 bidim or similar. This will help optimise the design and improve costs. An interface layer between the rockfill starter wall and the liner for protection of the liner.

3.3 DAM SAFETY CLASSIFICATION

3.3.1 ZONE OF INFLUENCE

The zone of influence is defined as the anticipated area surrounding the TSF that would be adversely affected by the release of tailings to the environment in case of a TSF breach.

A preliminary zone of influence was determined based on guidelines stipulated under SANS 10286:1998 which is a geometric extrapolation of the TSF height fitted to the natural topography. While this doesn't consider tailings volumes and rheology, it is considered to provide reasonable information to conceptualise the direction of flooding and inundated area following a dam breach. The preliminary zone of influence is illustrated in Figure 3.3.



NOTES:

1. PRELIMINARY ZONE OF INUNDATION TO BE REVISED IN SUBSEQUENT PHASES TO COMPLY WITH THE GISTM, 2020.

Figure 3.3 TSF conceptual zone of influence

3.3.2 DAM SAFETY CLASSIFICATION – CONSEQUENCE OF FAILURE

A preliminary Dam Consequence Classification (DCC) for the proposed TSF was developed based on the criteria outline in the 2020 Global Industry Standard on Tailings Management (GISTM, 2020). This criteria is based on the assumption that failure of the facility's outer wall would release a portion of the tailings to the environment in an uncontrolled manner. A hypothetical failure of the TSF could potentially cause incremental losses along the inundation route shown in Figure 3.3. The safety classification is based on the final envisaged tailings storage facility at final wall height of 999.5 m.

The safety/consequence classification is assigned for each individual consequence category outlined in Appendix C, and the overall dam safety classification is High. The classification can be mainly

attributed to the possible significant environmental impact of the breached tailings, associated remediation time and costs, and the anticipated disruption to business.

3.4 PRELIMINARY SEEPAGE AND STABILITY ANALYSIS

A preliminary assessment of the stability of the dam at the final height was conducted and the Factor of Safety (FOS) targets were in accordance with the CDA (2019) guidelines summarized in the design criteria. The TSF embankment stability was evaluated for both peak static and post-peak static loading conditions. Table 3.1 summarises the geotechnical parameters for the material properties used in the seepage / slope stability analysis and Figure 3.4 shows a typical section used for the analysis. It assumes a lined embankment and basin, earthfill starter embankment and downstream waste rock zone.

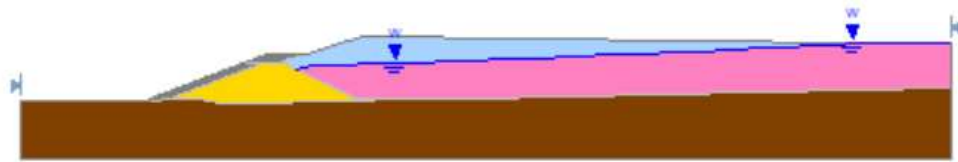


Figure 3.4 Typical section for seepage and stability analysis

Table 3.1 Seepage / stability analysis - assumed geotechnical parameters

Colour	Material	k_s (m/s)	k_{sv} / k_{sh}	Unit Weight (kN/m ³)	ϕ' (°)	Peak undrained ratio	Post-Liq. undrained ratio
	Foundation (sands & gravel) [1]	10^{-4}	1.0	21	35	-	-
	Waste rock [2]	10^{-3}	1.0	21	36	-	-
	Compacted Rockfill [3]	10^{-6}	1.0	21	35	-	-
	Bidim-liner-GCL interface [4]	10^{-12}	impermeable	20	11	-	-
	Unsaturated tailings [5]	10^{-8}	0.8	14	26	-	-
	Saturated tailings [5]	10^{-8}	0.8	14	-	0.20	0.10

NOTES:

1. TYPICAL VALUES ASSUMED FROM KP PREVIOUS ANALYSIS
2. SHEAR STRENGTHS BASED ON LEPS (1970)
3. VALUES ASSUMED FROM LITERATURE (AMINI, 2013)
4. GCL = GEOSYNTHETIC CLAY LINER
5. MATERIAL IS ASSUMED COHESIONLESS
6. k_s PERMEABILITY COEFFICIENT, k_{sv} VERTICAL DIRECTION OF k_s AND k_{sh} HORIZONTAL DIRECTION OF k_s
7. ϕ' EFFECTIVE INTERNAL FRICTION ANGLE

The analysis results show an elevated phreatic surface which is a result of the liner system and the sloped terrain of the TSF impoundment. Investigations into mitigative measures such as blanket drains at the starter wall crest elevation should be considered. The availability and proximity of waste rock in future allow for further buttressing of the TSF embankment during the self raising period. This is beneficial for both stability of the facility and concurrent rehabilitation. The stability analysis model included further buttressing of the upstream self raised embankment for improvement of FoS.

Results obtained from the coupled seepage/ stability analyses are presented in Table 3.2.

Table 3.2 Conceptual slope stability model – resulting factor of safety

	Static Loading (FoS > 1.5)	Post-peak static Loading (FoS > 1.1)
Result FoS	1.54	1.13
Reference Appendix	D	E

4.0 WATER MANAGEMENT

4.1 CLIMATE

The Lofdal Rare Earths Project is located approximately 25 km west of the Khorixas Town, in the Kunene Region of Namibia. The climate in Khorixas can be described as semi-arid to arid, with average summer temperature reaching into 40 degrees Celsius (°C) and winter temperature touching 0 °C. The region is a summer rainfall region with the highest temperature and rainfall depths are recorded from December to May.

The climate in Namibia is highly variable, with extreme drought periods and rainfall events (MET, 2011). Climate change models indicate that Namibia, especially the eastern and southern parts are adversely affected by rising temperatures and the consequences thereof (WBG, 2021).

4.2 RAINFALL AND EVAPORATION

Daily rainfall data was received from the Namibian Meteorological (NMET) services for the Khorixas Station for a record period of 56 years from the year 1955 until 2008 (NMET, 2021). Monthly and daily synthetic rainfall data was also obtained from the Climate Research Unit (CRU) and KNMI database (CRU, 2021) (KNMI, 2021), the records did not correlate well with the actual observed dataset, and it was therefore decided to only use the NMET Khorixas Station record.

The monthly rainfall distribution as obtained from the NMET and the Pan evaporation from the Namibia Department of Water Affairs is summarised in Table 4.1. The mean annual Pan evaporation is 2 850 mm (DWA, 1988) and the mean annual precipitation of 223 mm (NMET, 2021). The months with the highest evaporation are December and January and rainfall are February and March.

Table 4.1 Mean monthly rainfall and evaporation

Monthly Average (mm)	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Rainfall	46	61	60	20	1	0	0	0	2	3	14	16	223
Evaporation	456	428	285	228	143	57	57	86	143	228	342	399	2850

4.3 EXTREME RAINFALL EVENT ESTIMATION

Daily resolution rainfall data was used to determine the statistical frequency distribution of events. The General-Extreme Value (GEV) resulted in the best fit between the actual observed ranked annual maximum rainfall depths and the various distributions, such as Log-Normal, Log-Pearson and Extreme Value. A Weibull plotting position was used to successfully match the observed data to the distribution. The rainfall depths for different return periods are summarised in Table 4.2 (SANRAL, 2013) (NRA, 2014).

Table 4.2 24-hour duration extreme rainfall depths estimates

Return Periods (years)	2	10	20	50	100	200	500	100	PMP ¹ (100 000)
Rainfall Depths (mm)	58	90	130	176	197	220	251	277	372

NOTES:

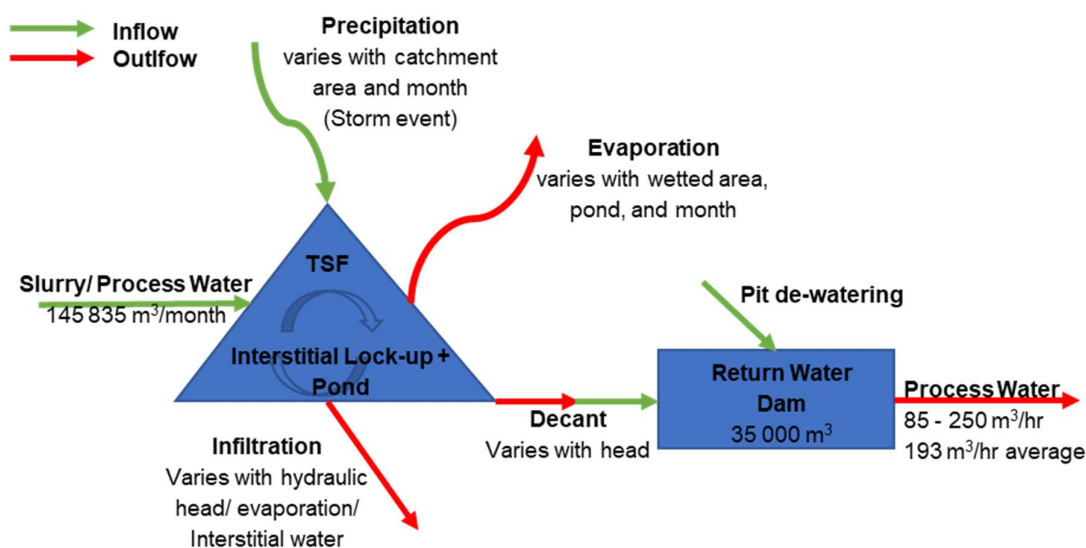
1. WORLD METEOROLOGICAL ORGANISATION (WMO) PROBABLE MAXIMUM PRECIPITATION METHOD COULD NOT BE APPLIED, DUE TO LACK OF SUB-DAILY (< 24-HOUR) RAINFALL DATA
2. 1:100 000-YEAR RETURN PERIOD RAINFALL DEPTH DERIVED WITH THE STATISTICAL METHOD USED AS PMP EQUIVALENT

4.4 STORMWATER MANAGEMENT

A high-level stormwater management assessment was conducted on the surrounding catchments, and they were found to be small in size, as they form the upper reaches of the Huab Catchment. It was found that no storm water diversion system is required upstream of the TSF. It is unlikely that the runoff generated in the catchment adjacent to the toe of the TSF will have an impact on the TSF, as there is sufficient distance between the main watercourse, the RWD and the toe of the TSF.

4.5 WATER BALANCE

A water balance model was developed, based on the available NMET record and the mine plan. A simplified schematic is shown in Figure 4.1, indicating all the components comprising the water balance, with first order values.



The average monthly inflows, outflows and decant volumes over the LOM are presented in Figure 4.1. The return water dam is sized for the worst rainfall month allowing for seven-day storage, with an assumed 8 hours of operation per day.

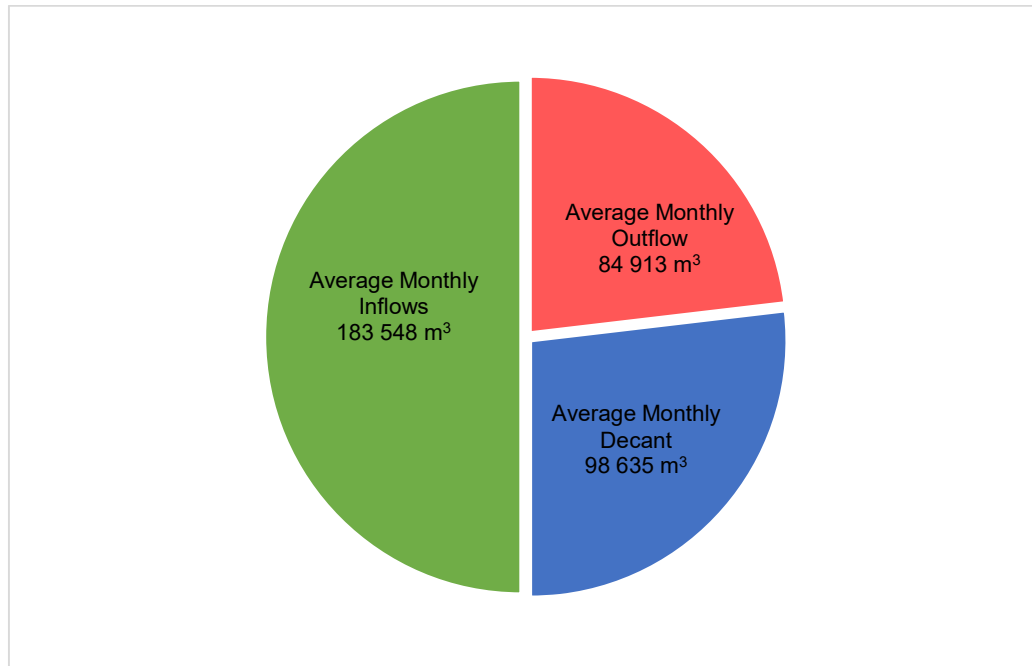


Figure 4.2 Inflow, outflow and decant average monthly volumes

5.0 COST ESTIMATE

A bill of quantities was generated from the drawings presented under Appendix B. The cost estimate is based on the calculated quantities, estimated P&G's and rates that were obtained from recent similar projects in other parts of Namibia. The bill of quantities is included as Appendix F.

A concept level costing exercise was undertaken for the capital expenditure and the construction of the waste rock buttress at the end of Years 1,2 and 3. This included the lining of the TSF for each phase. The phased installation of the floor liner for the TSF will help reduce initial CAPEX costs and eliminate the need for protection against UV light over a large surface area. The cost estimates are summarized in Table 5.1 below. These include an assumed 30% for P&G's, 15% contingency and 5% for engineering and permitting support in the initial CAPEX costs.

Table 5.1 Summary of preliminary estimates of initial CAPEX and sustaining Capital

Description	Capital Cost (Before start of Deposition) (NAD)	Capital Cost (At the end of Year 1) (NAD)	Capital Cost (At the end of Year 2) (NAD)	Sustaining Capital (Ongoing Lining/Raising) – Year 3-15 (NAD)
Mobilization and Demobilization	48,426,100.00	3,305,300.00	3,523,910.00	3,076,090
Earthworks	81,043,900.00	3,407,100.00	3,407,100.00	2,788,000
Geosynthetics	69,869,500.00	33,053,000.00	35,239,100.00	30,760,900
Pipe works and Appurtenance	6,462,000.00	-	1,000,000.00	-
Geotechnical Instrumentation	860,000.00	-	-	-
Infrastructure	3,185,000.00	-	-	-
Engineering	8,071,000.00	1,988,270.00	1,938,150.50	1,691,849
SUB TOTAL	217,917,500.00	41,753,670.00	5,108,260.50	38,316,839
CONTINGENCY (15%)	32,687,625.00	6,263,050.50	6,766,239.08	5,747,526
GRAND TOTAL	50,605,125.00	48,016,721.00	1,874,500.00	44,064,366

NOTES:

1. THE RATES USED ARE BASED ON KP EXPERIENCE FROM OTHER SIMILAR PROJECTS
2. NOT TO NET PRESENT VALUE, NO CONSIDERATION TO INDEXATION/INFLATION WHICH IS TO BE ADDED IN FINANCIAL MODEL BY OTHERS
3. CAPEX ASSUMES LINING INSTALLATION IN YEAR 0 IS TO 978.5 m ELEVATION, IN YEAR 1 TO 981.5 m ELEVATION, AND YEAR 2 TO 984 m ELEVATION WHICH IS THE TOTAL HEIGHT OF THE STARTER EMBANKMENT. SUSTAINING CAPITAL IS FOR BASIN LINING DURING THE ON-GOING OPERATION

6.0 RISK AND RECOMMENDATIONS

Bellow is a list of information gaps and risks to the current concept level design of the TSF:

- There is uncertainty of the geotechnical and hydrogeological conditions of the selected TSF site. The sources of starter embankment material and clay that form part of the liner system should be further investigated.
- The uncertainty in tailings geotechnical parameters present a risk on the geotechnical stability and drainage behaviour of the TSF.
- Uncertainty in levels of residual uranium in the tailings presents a risk on radiation exposure of personnel at the TSF as well as on potential pollution of groundwater.
- The uneven sub-grade terrain of the site posses a challenge for installation of the liner system at the TSF site.

The below is a list of recommendations for the feasibility design phase:

- Further Geochemical testing both static and dynamic should be conducted in the next phase of the study to confirm the tailings properties.
- Geotechnical and hydrogeological investigations are recommended to establish groundwater conditions as well as the extent of locally available bedding quality material.
- Further detailing of the starter wall embankment should consider an engineered interface layer between the proposed compacted rock fill and the proposed liner system.
- An alternative assessment of the TSF site location, technology, and deposition method should be conducted. The feasibility of filter stack deposition should be investigated.
- Consider optimising the TSF geometry and technology to reduce risks and capital cost requirements. There is an opportunity to further integrate the mining plan and containment walls of the TSF.
- Conduct site-specific geotechnical investigation over sites identified in the alternative assessment study to characterize founding conditions. Investigations should involve engineering (shear strength, consolidation, permeability) testing of tailings and in situ soils.
- Update the dam classification based on new geotechnical information for the tailings and the refinement of the zone of influence.
- Detailed stability analyses will need to be completed for the feasibility study for both drained and undrained loading conditions in accordance with the requirements of the GISTM (2020).
- It is recommended that ground water studies are undertaken / updated not only for the TSF, but for the wider pit and plant areas.


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8.0 CERTIFICATION

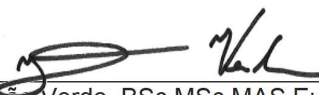
This report was prepared and reviewed by the undersigned.

Prepared:



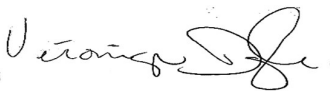
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Director

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APPENDIX A

Design Criteria

TABLE 1

NAMIBIA RARE EARTHS INC.
LOFDAL RARE EARTHS PROJECT - NAMIBIA

CONCEPTUAL DESIGN OF TAILINGS STORAGE FACILITY AND WATER MANAGEMENT STRUCTURES
DESIGN CRITERIA

Item No.	Item	Design Criteria	Reference/Approval
1.0 GENERAL			
1.1	Language and Units	• Documents to be written in English	NRE
		• Metric units, unless otherwise noted	NRE
		• Costs in \$NAD and \$USD. \$NAD to \$USD at an approximate exchange rate of 15:1	NRE, SGS
		• Coordinate System: LO22-15, Zone 33S	NRE
1.2	Codes, Standards, and Guidelines	• Namibian Roads Authority Materials Manual (1st Edition)	NRA, 2014
		• Namibian Roads Authority Drainage Manual (1st Edition)	NRA, 2014
		• Minerals (Prospecting and Mining) Act of 1992, Republic of Namibia	MA, 1992
		• Namibian Water Act 11 of 2013, Republic of Namibia	NWA, 2013
		• Global Industry Standard on Tailings Management (GISTM)	GISTM, 2020
		• Tailings Management - Good practice guide	ICMM, 2021
		• South African National Standard - Stormwater Disposal	SANS, 10400-NBR
		• South African National Standard - Code of Practice for Mine Residue Deposits	SANS, 10286
		• South African National Standard - Management of Mine Residue	SANS 10286
		• Technical Bulletin - Application of Dam Safety Guidelines to Mining Dams (2019 Edition)	Canadian Dam Association (CDA), 2014 & 2019
1.3	Site Information	• A Guide to Management of Tailings Facilities	Mining Association of Canada, 2019
		• Location: approximately 25 km NW of Khorihax, Kunene Region of north Western Namibia	KP
		• Site Coordinates: 20°19'14.15" S, 14°44'29.63" E	KP
		• Site Elevation: El. 971 mamsl to 1003 mamsl above mean sea level within the TSF area	KP
2.0	HYDROLOGY	• Site Area: Approximately 21 035 ha (210 350 000 m ²)	NRE
		• License Details: Exclusive Prospecting License (EPL) 3400	NRE
2.1	Meteorological Parameters	• Catchment: Huab-Ugab Basin	NRE, 2014
		• Maximum Annual Temperature: 40°C	NRE, 2014 & KP (calculated)
		• Mean Annual Precipitation: 150 - 200 mm	NRE, 2014
		• Mean Annual Evaporation: 2850 mm	TR 102 (Adamson, 1980)
		• 24 Hour Return Period Rainfall Events:	KP, Calculated
		o 1 in 10 year, 24 hour storm event: 74 mm	Namibia MET-Khorixas
		o 1 in 50 year, 24 hour storm event: 113 mm	
		o 1 in 100 year, 24 hour storm event: 130 mm	
		o 1 in 200 year, 24 hour storm event: 149 mm	
		o 1 in 1,000 year, 24 hour storm event: 197 mm	
		o 1 in 10,000 year, 24 hour storm event: 277 mm	
2.2	TSF Catchment	• Catchment Area: 110.1 ha (1 101 000 m ²) upstream of TSF	KP (estimated)
		• Catchment Area: 177.3 ha (1 773 000 m ²), upstream of the starter embankment	KP (estimated)
		• Catchment upstream of the TSF is relatively small, and of steep slopes. Hence, the SCS method is deemed appropriate for flood hydrology assessment.	KP (recommended)
2.3	Runoff Coefficients	• Tailings Beach: 70% of runoff over dry beach, and 90% of runoff over wet beach. Rational Method Cv=0.72 - 0.8	KP (assumed)
		• Upstream catchment: 75% of runoff	KP (assumed)
2.5	Stormwater Management Method	• Alternative Rational Method used for estimating peak flow/ SCS Method over the TSF	NRA, 2014 / SANRAL, 2013

TABLE 1

NAMIBIA RARE EARTHS INC.
LOFDAL RARE EARTHS PROJECT - NAMIBIA

CONCEPTUAL DESIGN OF TAILINGS STORAGE FACILITY AND WATER MANAGEMENT STRUCTURES
DESIGN CRITERIA

Item No.	Item	Design Criteria	Reference/Approval
2.6	Runoff Management System	<ul style="list-style-type: none"> Diversion Ditch to divert runoff from catchment area on the southern side of the TSF. In next phase of study should use the 1 in 50 year, 24 hour storm event to size the Diversion Ditch, as per GN704 Mine Water Regulations. 	DWS, 1998
3.0 TAILINGS STORAGE FACILITY			
3.1	Type and Function	<ul style="list-style-type: none"> Valley impoundment, earthfill/rockfill starter embankment enlarged with waste rock downstream and upstream construction for final raises. Permanent and secure storage of tailings solids and management of process water and runoff during the 15 years LoM and post-closure. 	KP
3.2	Mining Throughput	<ul style="list-style-type: none"> Mining Throughput <ul style="list-style-type: none"> Life of Mine: 15 Years Total ROM Tonnage: 26.8 Million tonne (Mt) 	GENET (Mike Gibson), 2021 GENET (Mike Gibson), 2021 GENET (Mike Gibson), 2021
3.3	Plant Throughput	<ul style="list-style-type: none"> Plant (Plant El: 985 - 993 mamsl) <ul style="list-style-type: none"> Life of Mine: 16 Years Plant utilization: 85% or 20.4 h for 365 days ROM split: 100% (slurry tailings) Neutralizing tailings with a higher water content to be added to tailings after thickener Plant designed for 2.16 Mt capacity Throughput from plant to TSF: <ul style="list-style-type: none"> - 26.8 Mt over the LoM Solids content by mass: 45.97% 	SGS-RSA (Daniel Millar), 2021 SGS-RSA (Daniel Millar), 2021 SGS-RSA (Daniel Millar), 2022 SGS-RSA (Daniel Millar), 2021, KP (calculated) SGS-RSA (Daniel Millar), 2021
3.4	Tailings slurry delivery and return water pipeline	<ul style="list-style-type: none"> Tailings delivery pipeline to final height (Plant to TSF) <ul style="list-style-type: none"> Conceptual Pipe Type and Size: standard HDPE 220 mm/ 250 mm (inside/ outside diameter) Overall length: 2 900 m Crest spigotting pipeline <ul style="list-style-type: none"> Conceptual Pipe Type and Size: standard HDPE 145mm/ 180 mm (inside/ outside diameter) Overall length Required to control pool position: 2 800 m Static head difference (Plant-TSF): <ul style="list-style-type: none"> Stage 1 (to El. 981 mamsl): 9 m (drop) At final height (to El. 1001.5 mamsl): 11.5 m (rise) Return Water Pipeline (RWD to Plant) <ul style="list-style-type: none"> Conceptual Pipe Type and Size: HDPE pipe, ID/OD to be determined. Overall length: 3 000 m 	SGS KP, Measured SGS KP, Measured KP, Measured KP, Measured SGS
3.5	Dam Classification	<ul style="list-style-type: none"> High, based on 1-100 population at risk and potential impact on the pit area from preliminary Zone of Influence. 	GISTM, 2020
3.6	Tailings Characteristics	<ul style="list-style-type: none"> Solids SG: 2.78 Grain Size: P80 = 0.035mm (silts and finer) Atterberg Limits: Liquid limit of 24%, plastic limit of 21% and a plasticity index of 3% Material Classification: Silts and finer Tailings Settled Dry Density at Start-up: 1.25 t/m³ Avg. Tailings Settled Dry Density: 1.35 t/m³ Max. Tailings Settled Dry Density: 1.47 t/m³ (actual lab results) Shear Strength: <ul style="list-style-type: none"> Peak strength is a friction angle: 26° Peak undrained strength: 0.20 Post-liquefaction undrained strength: 0.10 Hydraulic conductivity: 10⁻⁸ m/s 	SGS-RSA (Daniel Millar), 2021 SGS-RSA (Daniel Millar), 2021 SGS-RSA (Daniel Millar), 2021 KP, assumed KP, assumed SGS-RSA (Daniel Millar), 2021 KP Assumed due to very fine grading, TBC in next phase of study KP Assumed, TBC in next phase of study KP Assumed, TBC in next phase of study KP Assumed due to very fine grading, TBC in next phase of study

TABLE 1

NAMIBIA RARE EARTHS INC.
LOFDAL RARE EARTHS PROJECT - NAMIBIA

CONCEPTUAL DESIGN OF TAILINGS STORAGE FACILITY AND WATER MANAGEMENT STRUCTURES
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Item No.	Item	Design Criteria	Reference/Approval
		<ul style="list-style-type: none"> Tailings Chemistry: Tailings are non-acid generating with non-detectable sulphide and have neutralization potential. Tailings may contain some radioactive uranium. 	SGS Geological, 2021 (Liu Jing & Barbara Bowman)
3.7	Operational Criteria	<ul style="list-style-type: none"> Tailings Production Period: (15 years) Allowable Rate of Rise: 2.5 m/year Starter embankment, of conventional compacted earthworks, to be constructed before mining starts, and to be buttressed with waste rock during deposition. Flood management: All storm water, from storm event on TSF and from upstream catchment, will be contained on the TSF. Supernatant water will be decanted to return water dam and recycled for process water. 	<p>Mike Gibson, 2021</p> <p>KP (assumed)</p> <p>KP (recommended)</p> <p>NRE, KP</p>
3.8	Freeboard	<ul style="list-style-type: none"> Storm Management Strategy: To temporarily contain runoff from the Environmental Design Storm (EDF) and safely convey the Inflow Design Flood (IDF). Conveyance via penstocks to Return Water Dam. EDF: 1 in 200 year, 24 hour duration storm event IDF: Annual exceedance probability: 1/2,475 years 24 hrs event = 309 mm Total Freeboard Required = 1.2 m (TBC in next phase of studies) 	<p>CDA, 2019</p> <p>CDA, 2019</p> <p>GISTM, 2020</p> <p>ANCOLD Best Practice</p>
3.9	Seismic Design Criteria	<ul style="list-style-type: none"> Design Earthquake: TBC in next phase of studies Annual exceedance probability: 1 in 2475 year event. Peak Magnitude experienced in the area: 5.6 Peak Ground Acceleration: TBC 	<p>TBC in next phase of study</p> <p>GISTM, 2020 for High dam classification</p> <p>NRE, 2014</p> <p>TBC in next phase of study</p>
3.10	Stability Requirements	<ul style="list-style-type: none"> Static (Operations and Post-closure - long-term): Min FoS = 1.5 Static/Seismic Post-Liquefaction: Min FoS = 1.1 (residual strength) TSF final crest El. 999.5 mamsl 	<p>CDA, 2019</p> <p>CDA, 2019</p> <p>KP (calculated)</p>
3.11	Outer Wall Geometry	<ul style="list-style-type: none"> Starter Embankment (Sand and Gravel waste rockfill) <ul style="list-style-type: none"> o Crest El.: 981 mamsl sufficient to contain Year 1 tailings) o Downstream Slope: 1V:2.5H o Upstream Slope Angle: 1V:2H o Embankment Height: 15 m o Embankment Crest Width: 12 m Ultimate Outer Embankment <ul style="list-style-type: none"> o Inter Bench Height: 5 - 7m o Inter Bench Slope Angle: 1V:2H o Bench Width: 7 m external, 4 m internal if required o Overall Embankment Slope Angle: Approx. 1V:4H 	<p>TBC after GI</p> <p>KP (TBC after model revision)</p> <p>KP (assumed)</p> <p>KP (assumed)</p> <p>KP (revised model)</p> <p>KP (recommended)</p>
3.12	Rate of Rise	<ul style="list-style-type: none"> Maximum of 2.5 m/year for tailings not impounded behind starter wall 	KP (TBC after model revision)
3.13	Site Topography	<ul style="list-style-type: none"> There is an elevation drop of about 10 m between the plant site and the TSF starter wall 	KP (TBC after model revision)
3.14	TSF cross section	<ul style="list-style-type: none"> Starter Embankment: Sand, gravel and waste rock fill Embankment Raises: Upstream construction; tailings impounded behind starter embankment in years 1 to 2, and thereafter self raise with tailings Minimum Allowable Beach Length: 300 m 	<p>KP (TBC after model revision)</p> <p>KP (TBC after model revision)</p> <p>KP (TBC after model revision)</p> <p>KP, Estimated</p> <p>KP (assumed due to very fine grading)</p> <p>KP (measured)</p> <p>KP</p> <p>KP</p> <p>KP (estimated)</p>

TABLE 1

NAMIBIA RARE EARTHS INC.
LOFDAL RARE EARTHS PROJECT - NAMIBIA

CONCEPTUAL DESIGN OF TAILINGS STORAGE FACILITY AND WATER MANAGEMENT STRUCTURES
DESIGN CRITERIA

Item No.	Item	Design Criteria	Reference/Approval
3.15	Foundation	Founded on residual gravel that closely reflects the composition of the underlying amphibolites/ pegmatite/ gneisses	NRE, 2014 & KP. TBC after GI
		• bedrock. This residuum is typically less than 1 m thick on the high ground, but locally thickens in the dry valleys. Outcrop is widespread throughout the area	
		• Sandy and gravelly material readily available in streams within the project area, and could be used for liner bedding, drainage, and protection layers	NRE, 2014 & KP
		• Starter embankment foundation prepared by cutting 1 m into sandy-gravelly foundation	KP (recommended)
		• The TSF basin and upstream slope of starter embankment will be lined with a single liner system to prevent and minimize seepage. The system will be installed on the prepared foundation and along the upstream slope face of the TSF, and anchored in 1 m deep trench and will consist of the following components:	KP (recommended)
		o 150 mm thick bedding layer of clay liner over the prepared foundation to prevent liner puncture	
		o Layer of 1.5 mm HDPE geomembrane over the clay liner	
3.16	Seepage/ Drainage Management	• Protective layer of non-woven geotextile (e.g., A6 Bidim or equivalent) over the geomembrane	
		• Underdrainage system to be installed within the TSF basin to capture seepage. Underdrains will tie to an outlet pipeline which will deliver captured seepage to the RWD.	KP (recommended)
		• A decant penstock system will be required to convey supernatant and runoff (from storm event on TSF and upstream catchment) to the Return Water Dam.	KP (recommended)
		• Underdrainage system to be installed underneath the starter embankment and within the TSF basin to capture seepage. Underdrains will tie to an outlet pipeline which will deliver captured seepage to the RWD.	KP (recommended)
3.17	Tailings Delivery	• A seepage collection trench to be constructed along the downstream toe of the starter wall/ buttress to collect seepage and convey to the RWD via a silt trap	KP (recommended)
		• Tailings will be conveyed as thickened slurry (with 50% solids by mass) from the Plant to the TSF via pipeline and a centrifugal slurry pumping system. Tailings thickened to optimize water recovery as water in the area is scarce	NRE, 2014
		• Tailings delivery method: Cyclone deposition	KP (recommended)
		• Tailings delivery pipeline design for a throughput of 374 tph, designed by 3rd party	SGS-RSA, 2021; KP, calculated
3.18	Water Reclaim System	• Tailings Slurry Solids Content: 50% by mass	SGS-RSA (Daniel Millar), 2021
		Overview: Supernatant water to be decanted to Return Water Dam and thereafter pumped to Plant for re-use in process to reduce fresh water make-up requirements. Decant system must be designed/ sized to deliver process water, water from storm event on TSF and water from storm event from upstream catchment.	NRE, KP
3.19	Runoff Management	Overview: Construct sediment traps downstream of TSF to intercept runoff from southern side of TSF. All runoff from upstream catchment to be contained on the TSF and decanted to RWD together with process water and stormwater on TSF. RWD located downstream of starter wall at lowest point.	KP (recommended)
3.20	Dust Suppression	• The tailings are generally fine-grained (silts and clay size), and are thus susceptible to dust generation. Progressive waste rock cladding for dust suppression on outer wall slopes as part of closure plan	KP (recommended)
4.0 RETURN WATER DAM			
4.1	Function	• Collection of supernatant, runoff, and seepage from the TSF. Collected water reclaimed to the Plant via floating pump barge and pipeline.	KP
4.2	Embankment Section	• RWD excavated into foundation at a slope of 1V:3H	KP (recommended)
		• 1.5 m high berms; Berm slopes are also 1V:3H	KP (recommended)
		• RWD will be lined with a geosynthetic lining system to prevent seepage. The system will be installed on the prepared foundation and along the upstream berm face of the RWD, and anchored in 1 m deep trench and will consist of the following components:	KP (recommended)
		o 50 mm thick bedding layer of silty sand over the foundation materials to prevent liner puncture	
		o Layer of 1.5 mm HDPE geomembrane over the bedding layer	
4.3	Freeboard	o Layer of non-woven geotextile over the geomembrane	Can be excluded if RWD will not need to be desilted
		o 300 mm thick erosion protection layer over the geomembrane to prevent liner puncture	Can be excluded if RWD will not need to be desilted
		• Storm Management Strategy: Return Water Dam to temporarily contain runoff from the EDF. IDF conveyance via spillway into the Diversion Ditch.	KP, NRE
4.3	Freeboard	• EDF: 1 in 200 year, 24 hour duration storm event	CDA, 2019
		• IDF: 1/3 of 1/10 000 -> PMP = 217mm	CDA, 2019. Confirm magnitude during design

TABLE 1

NAMIBIA RARE EARTHS INC.
LOFDAL RARE EARTHS PROJECT - NAMIBIA

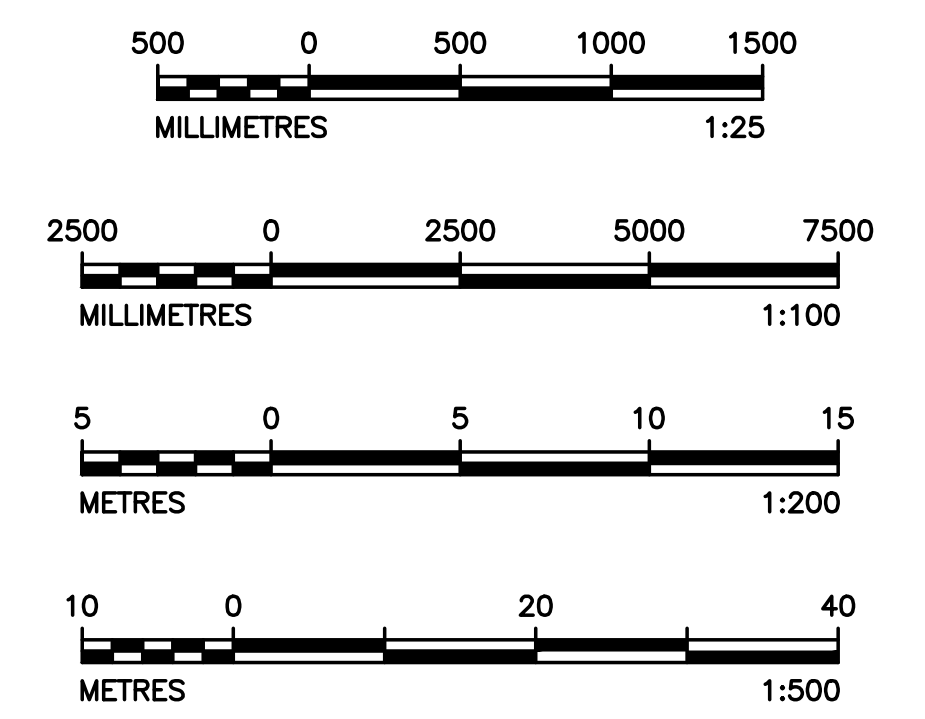
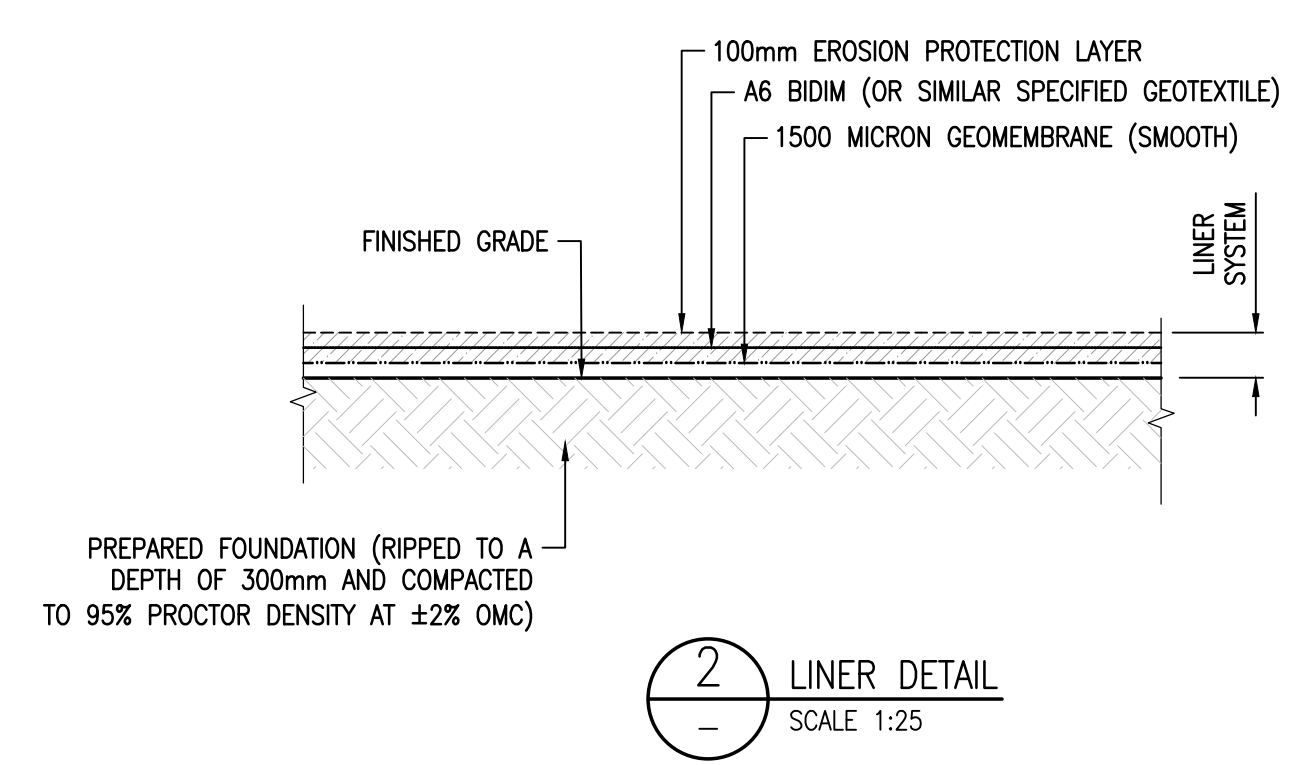
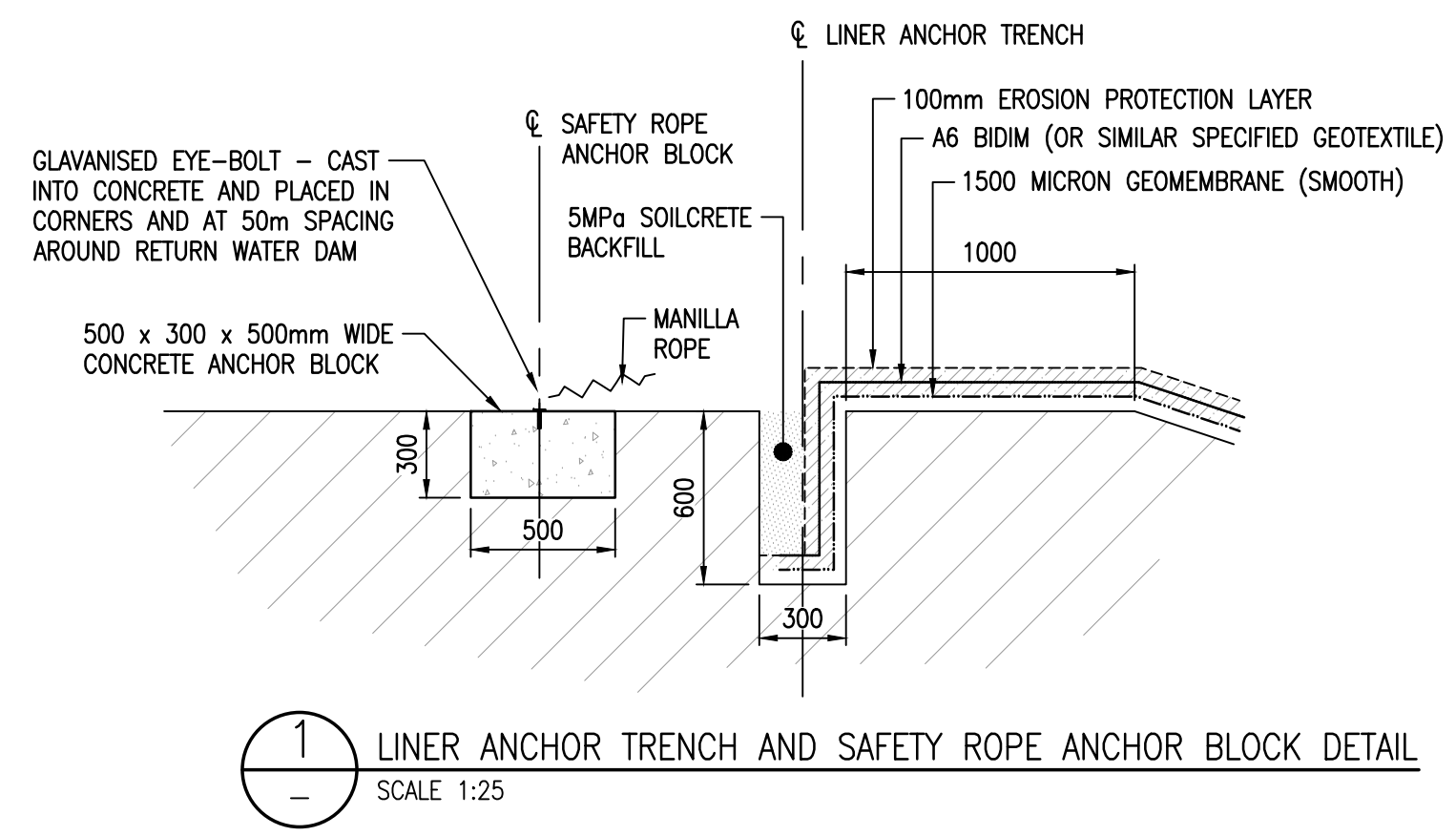
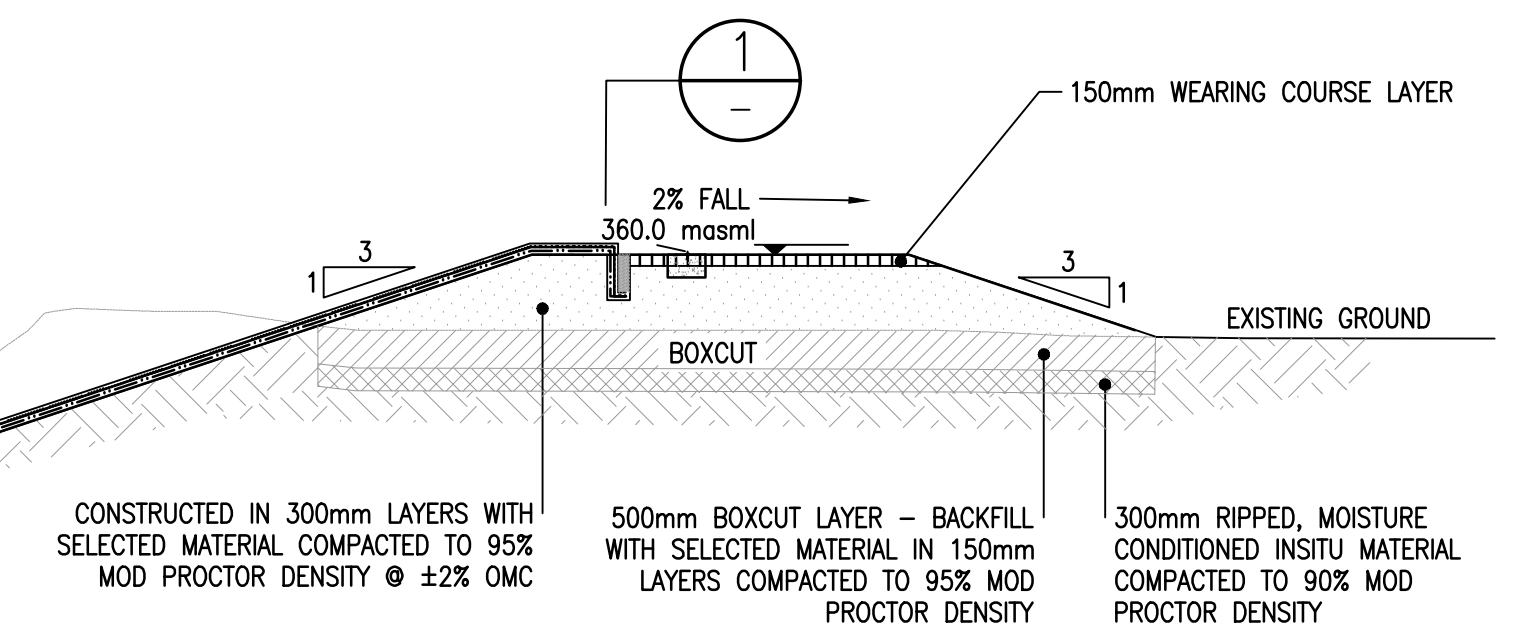
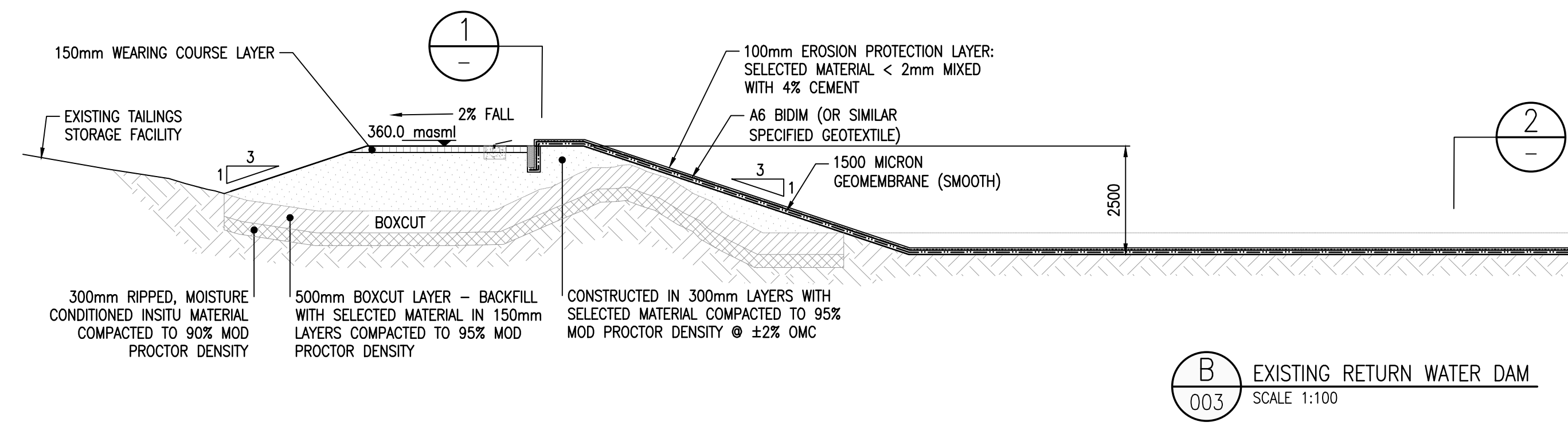
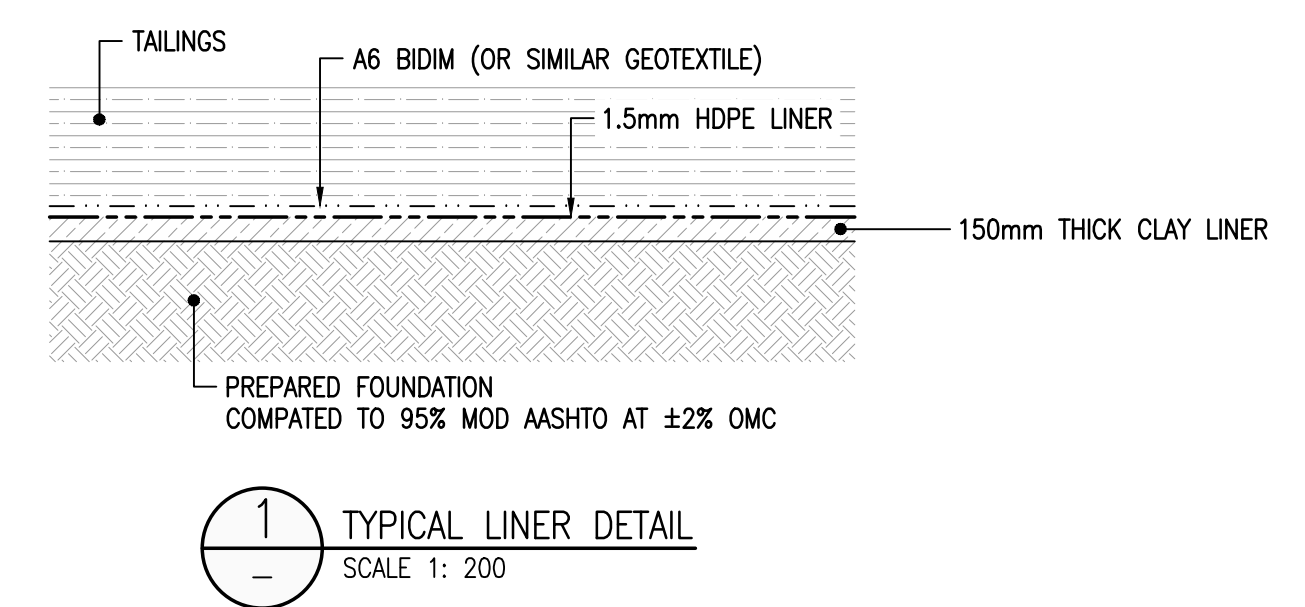
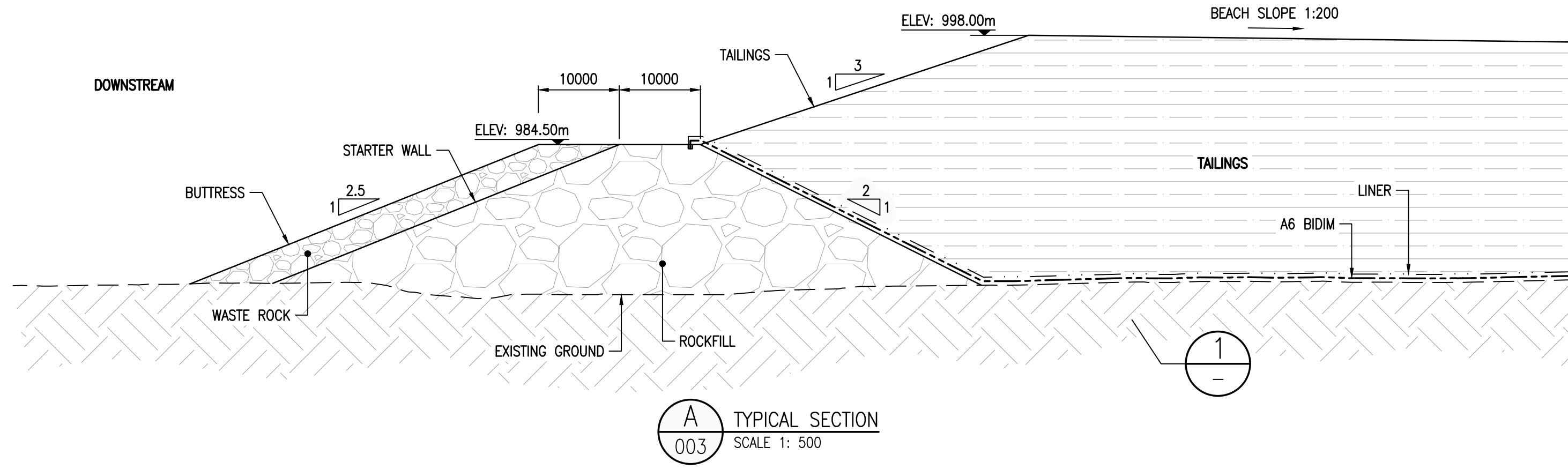
CONCEPTUAL DESIGN OF TAILINGS STORAGE FACILITY AND WATER MANAGEMENT STRUCTURES
DESIGN CRITERIA


Item No.	Item	Design Criteria	Reference/Approval
		• Wet Freeboard Requirement: 0.5 m	KP, Assumed
5.0 INSTRUMENTATION AND MONITORING			
5.1	General	• Standpipe and vibrating wire piezometers to be installed at different times during Life of Facility to measure long-term phreatic surface in foundation, embankments, and tailings	KP
		• Survey monuments to measure potential surface movement of embankment	KP
		• Monitoring of surface erosion and progressive reclamation measures	KP
		• SCPTu testing at different times during Life of Facility to confirm in situ tailings shear strength, consolidation and drainage properties	KP
		• Monitoring of surface water quality in all basins	KP

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APPENDIX B

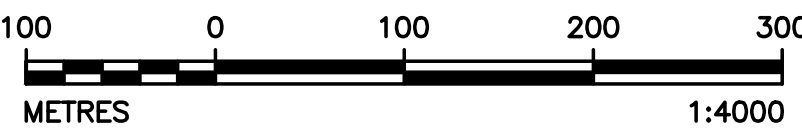
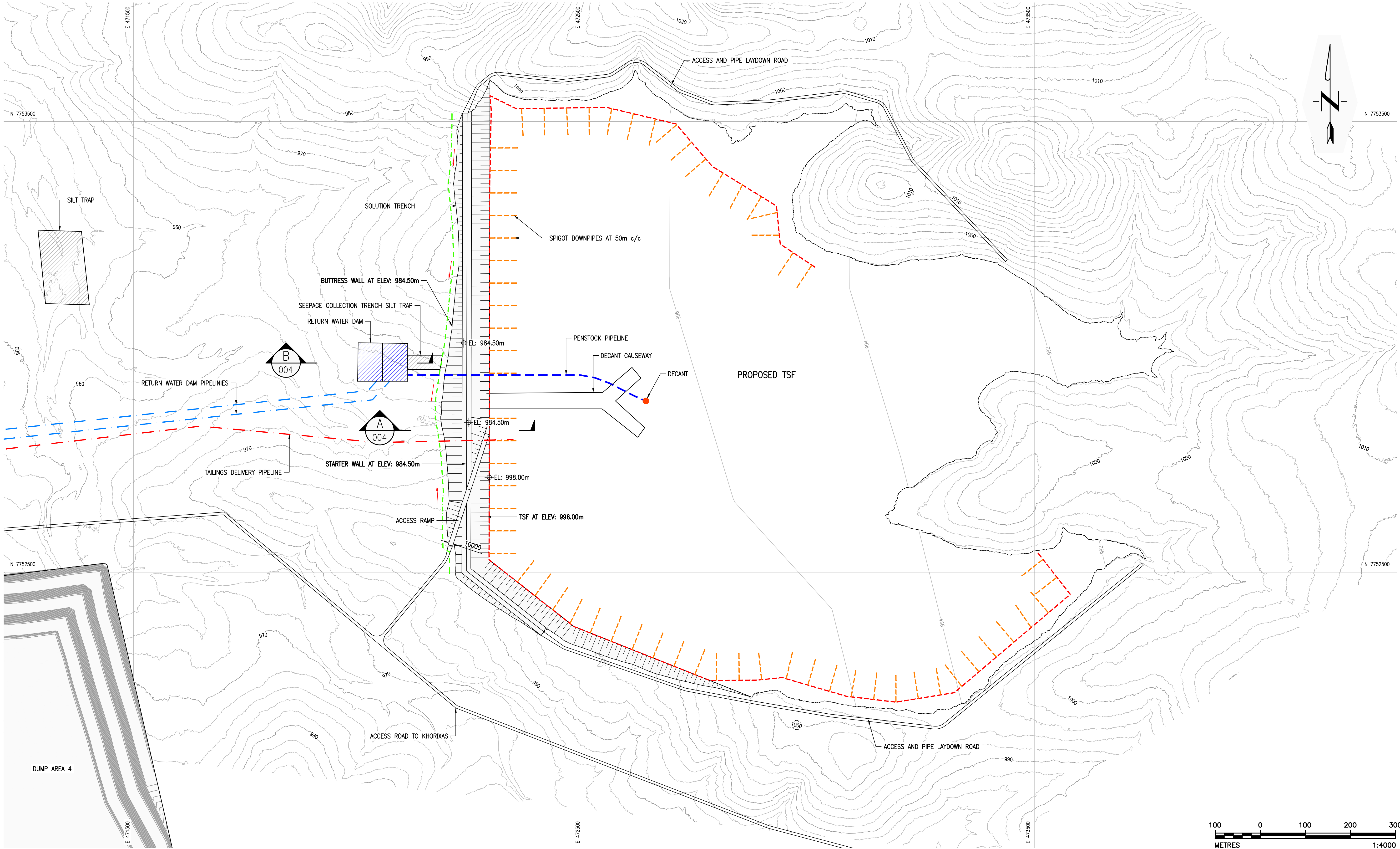
Concept Drawings



 <div>Knight Piesold CONSULTING</div>	PRIMARY DISCIPLINE	CIVIL		REVISION						<div>DISCLAIMER</div> <p>These drawings have been prepared by Knight Piesold Consulting (Pty) Ltd based on all information available at the time of preparation. Where site conditions necessitate an adaption or amendment to these drawings, no adaption or amendment may be made by any party without the prior written consent of Knight Piesold Consulting (Pty). Knight Piesold Consulting (Pty) Ltd will not be liable for any loss, damages or consequential loss suffered by any party resulting from these drawings where and adaption or amendment to the drawings has taken place without obtaining the prior written consent of Knight Piesold Consulting (Pty) Ltd.</p>	REFERENCE DRAWINGS				RARE EARTHS INC. LOFDAL PROJECT FINAL STAGE TYPICAL SECTIONS AND DETAILS
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	DESIGN	LM	19/08/2021	B	19/09/22	BUTTRISS AND ST ELEVATION RECEIVED	RC	LM	JV		—	—	—		
	DESIGN CHECK	VD	19/08/2021	—	—	—	—	—	—		—	—	—		
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DESIGN	LM 19/08/2021
DESIGN CHECK	VD 19/08/2021
PROJECT ENGINEER	VD -

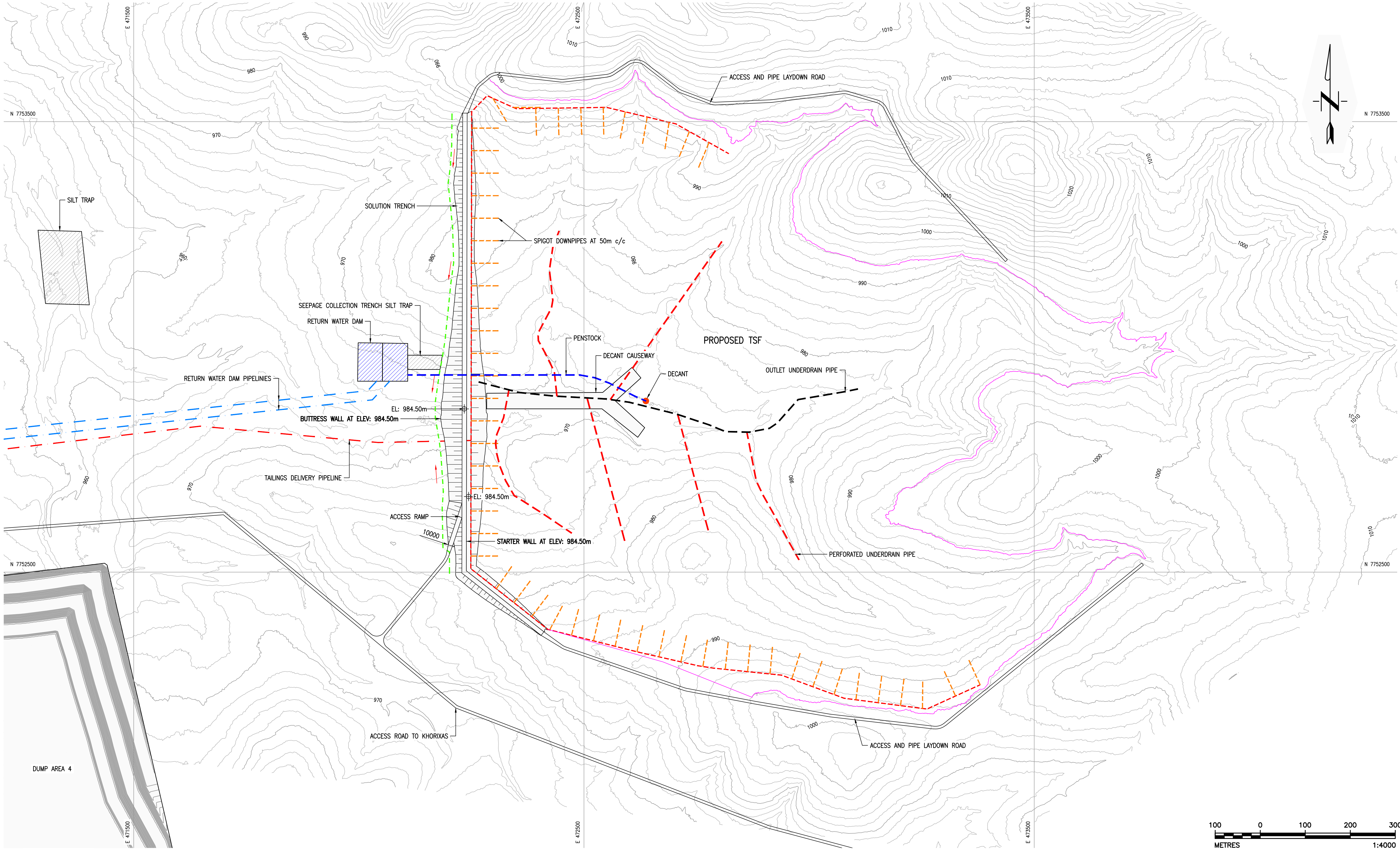
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RARE EARTHS INC. LOFDAL PROJECT FINAL LAYOUT PLAN		
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DESIGN CHECK	VD	19/08/2021
PROJECT ENGINEER	VD	-

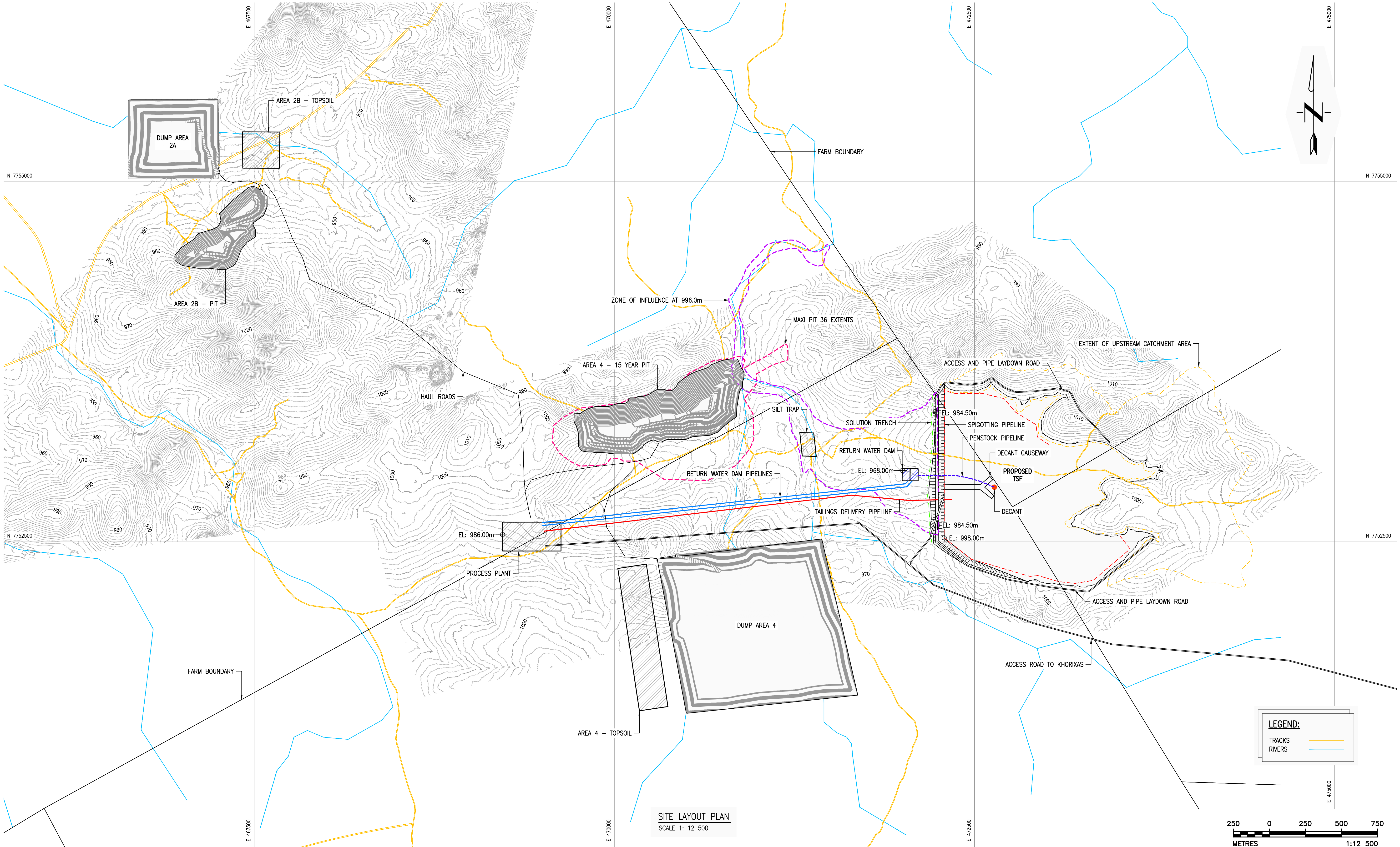
REVISION			
REV.No.	DATE	DESCRIPTION	
A	19/08/21	ISSUED FOR INFORMATION	
B	19/09/22	BUTTRESS AND ST ELEVATION RECEIVED	
-	-	-	
-	-	-	
-	-	-	
-	-	-	
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-	-	-	

DISCLAIMER		
DRAWN	CHKD.	APPD.
RW	TK	VD
RC	LM	JV
-	-	-
-	-	-
-	-	-
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REFERENCE DRAWINGS		
DRAWING No.	MAKERS No.	TITLE
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RARE EARTHS INC. LOFDAL PROJECT STAGE 1 LAYOUT PLAN		
DRAWING NUMBER 301-00928/01-002	SCALE 1: 4000	REV. B



PRIMARY DISCIPLINE	CIVIL
DRAWN	RC
DRAWING CHECK	LM
DESIGN	LM
DESIGN CHECK	VD
PROJECT ENGINEER	VD

REV.No.	DATE	DESCRIPTION	DRAWN	CHKD.	APPD.
A	19/08/21	ISSUED FOR INFORMATION	RW	TK	VD
B	19/08/22	BUTTRISS AND ST ELEVATION RECEIVED	RC	LM	JV
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DRAWING No.	MAKERS No.	TITLE
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RARE EARTHS INC.
LOFDAL PROJECT
SITE
LAYOUT PLAN

DRAWING NUMBER 301-00928/01-001	SCALE 1: 12 500	REV. B
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APPENDIX C

GISTM Consequence Classification

TABLE 1

**NAMIBIA RARE EARTHS INC.
LOFDAL RARE EARTHS PROJECT - NAMIBIA**

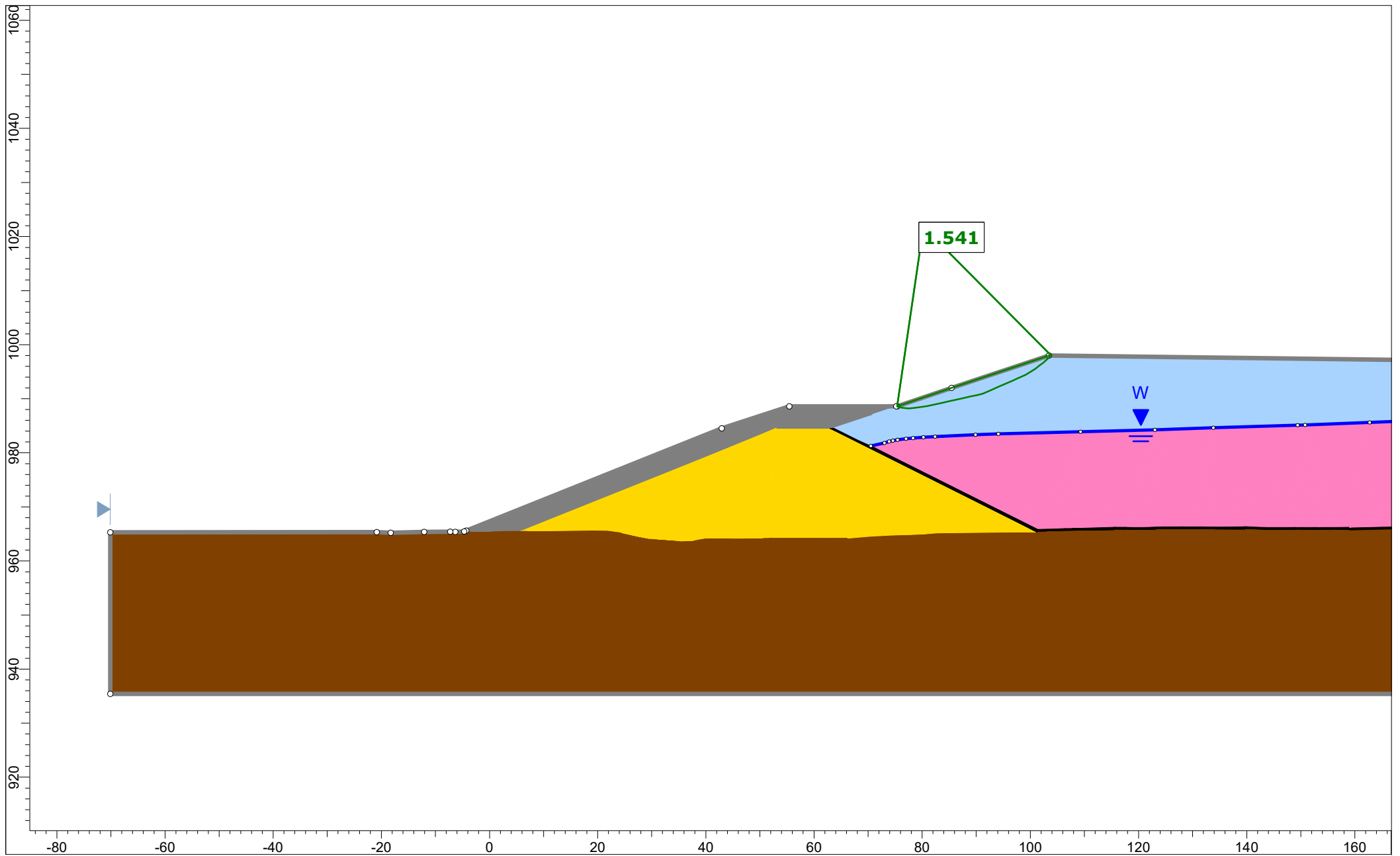
**CONCEPTUAL DESIGN OF TAILINGS STORAGE FACILITY AND WATER MANAGEMENT MANAGEMENT STRUCTURES
TSF GISTM CLASSIFICATION**

	Potential Population at Risk	Potential Loss of Life	Environment	Health, Social and Cultural (Number of people affected)	Infrastructure and Economics
Low	None	None expected	Minimal short-term impact	0	<US\$ 1m
Significant	10-Jan	Unspecified	Restoration possible in 1 – 5 years	< 500	<US\$ 10m
High	10-100	Possible (1–10)	Impact area 10 – 20 km ² Restoration > 5 years	500 – 1 000	<US\$ 100m
Very High	100 – 1 000	10 – 100	Impact area >20km ² Restoration 5 – 20 years	1,000	<US\$1b
Extreme	> 1 000	> 100	Impact area >20km ² Restoration > 20 years	5,000	>US&1bil

0	17OCT'22	ISSUED WITH REPORT W1301-00928/01	LM	VD
REV	DATE	DESCRIPTION	PREP'D	RVW'D

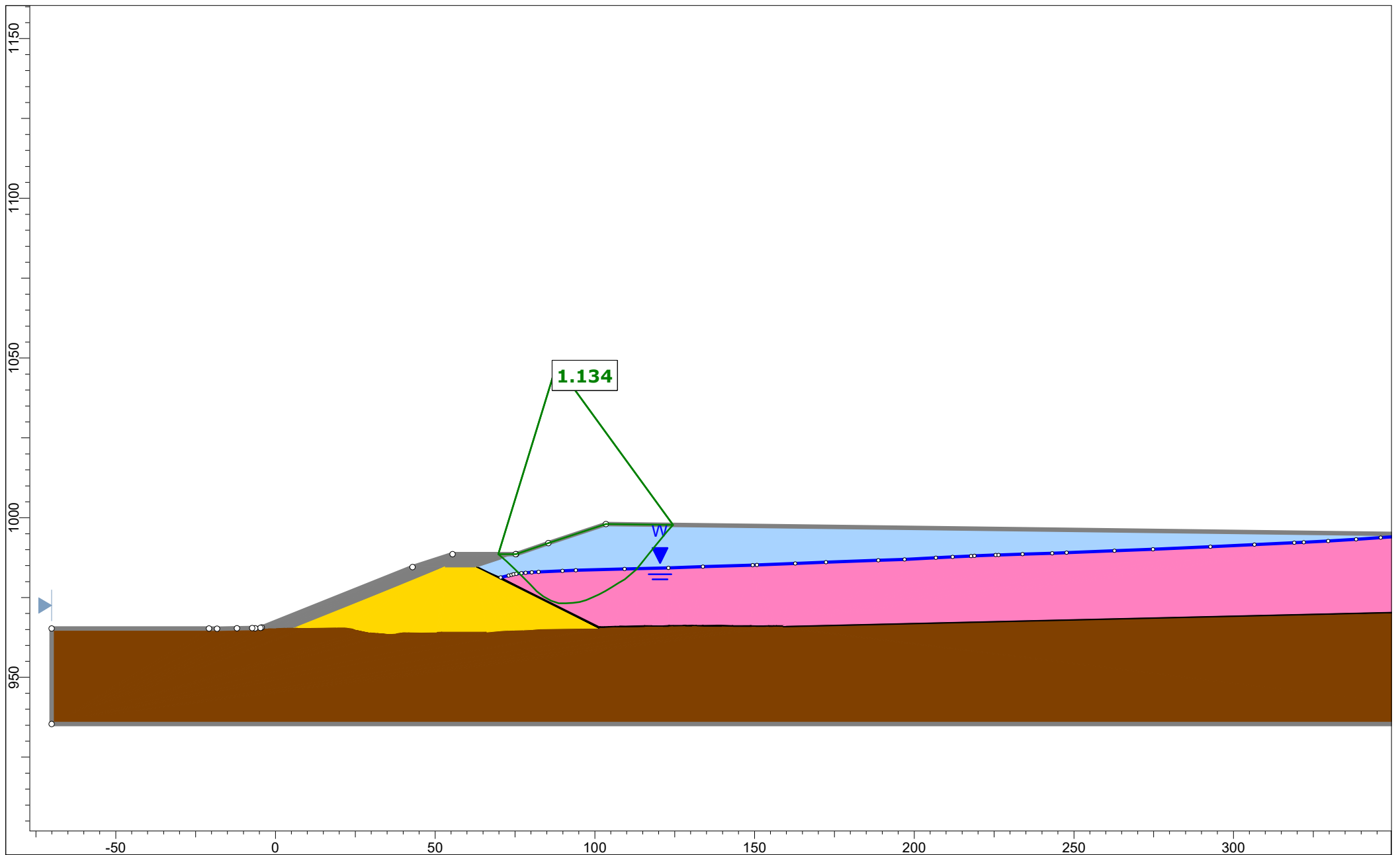
APPENDIX D

Stability Analysis Results - Peak Strength



APPENDIX E

Stability Analysis Results - Post-Peak



Appendix C:
Initial Site Visit To Lofdal REE Project report from Knight Piésold

INITIAL SITE VISIT TO LOFDAL REE PROJECT

SITE LAYOUT AND PHOTOGRAPHS

PROJECT SITE LAYOUT

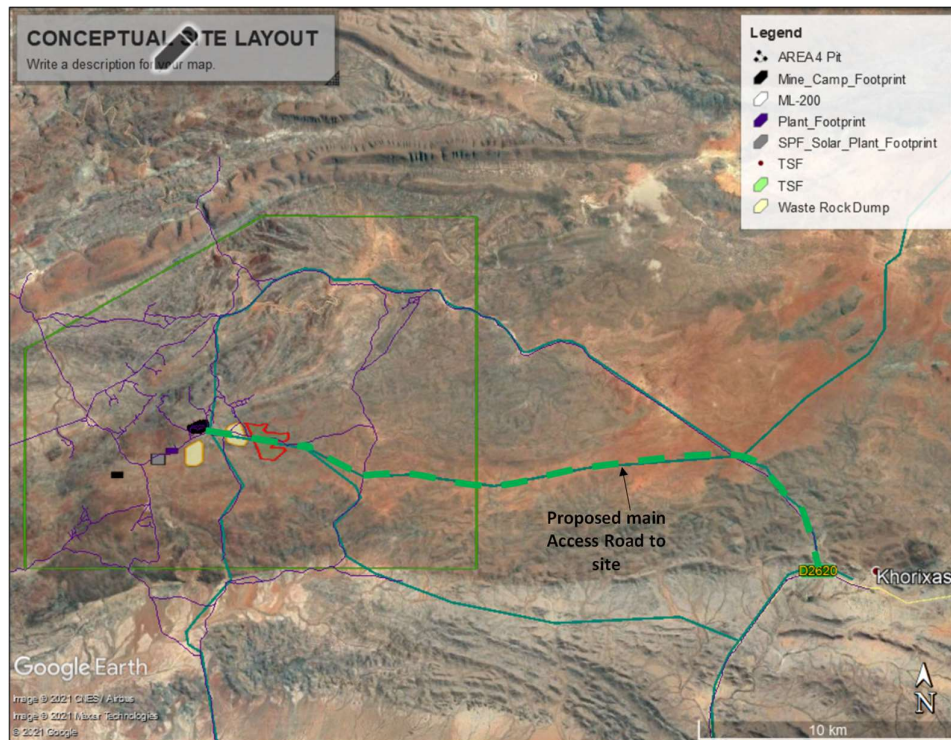


Photo 1. General layout and envisioned infrastructure corridors for the Lofdal Project.

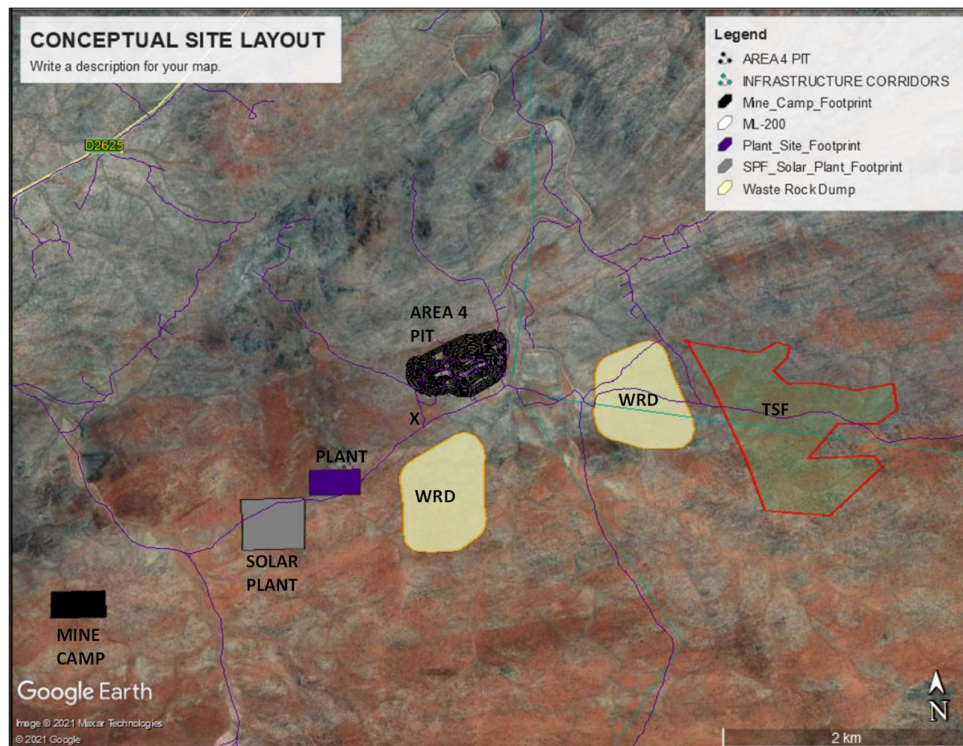


Photo 2. General layout of the Lofdal Project.

AREA 4 – ENVISIONED MAIN PIT



Photo 3. Rolling hill with gentle slope at Area 4 envisioned pit area. Note pit-transverse exploration trenches oriented south-north. Photograph taken looking north.

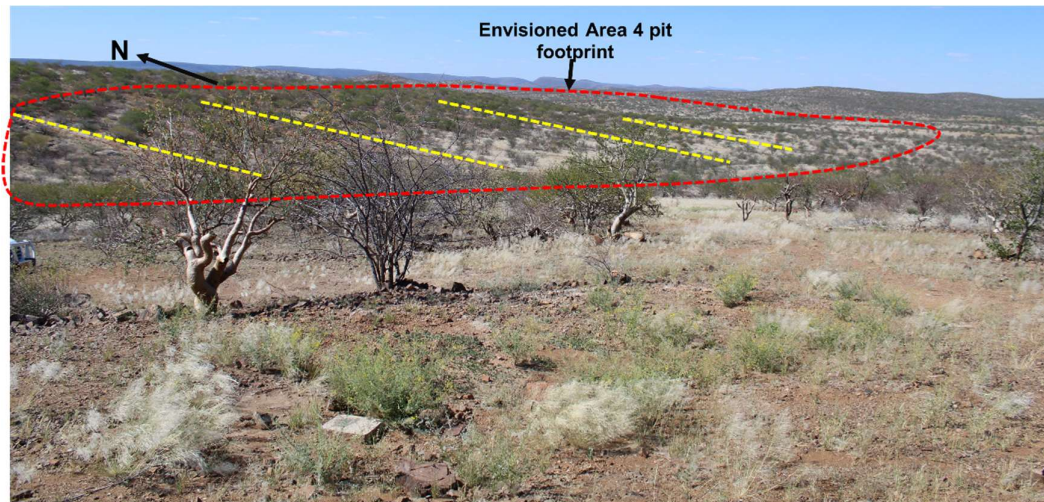


Photo 4. View of envisioned Area 4 pit from the south-western end.



Photo 5. Typical N-S transverse exploration trenches through starter pit, area 4.

ENVISIONED PROCESS PLANT SITE

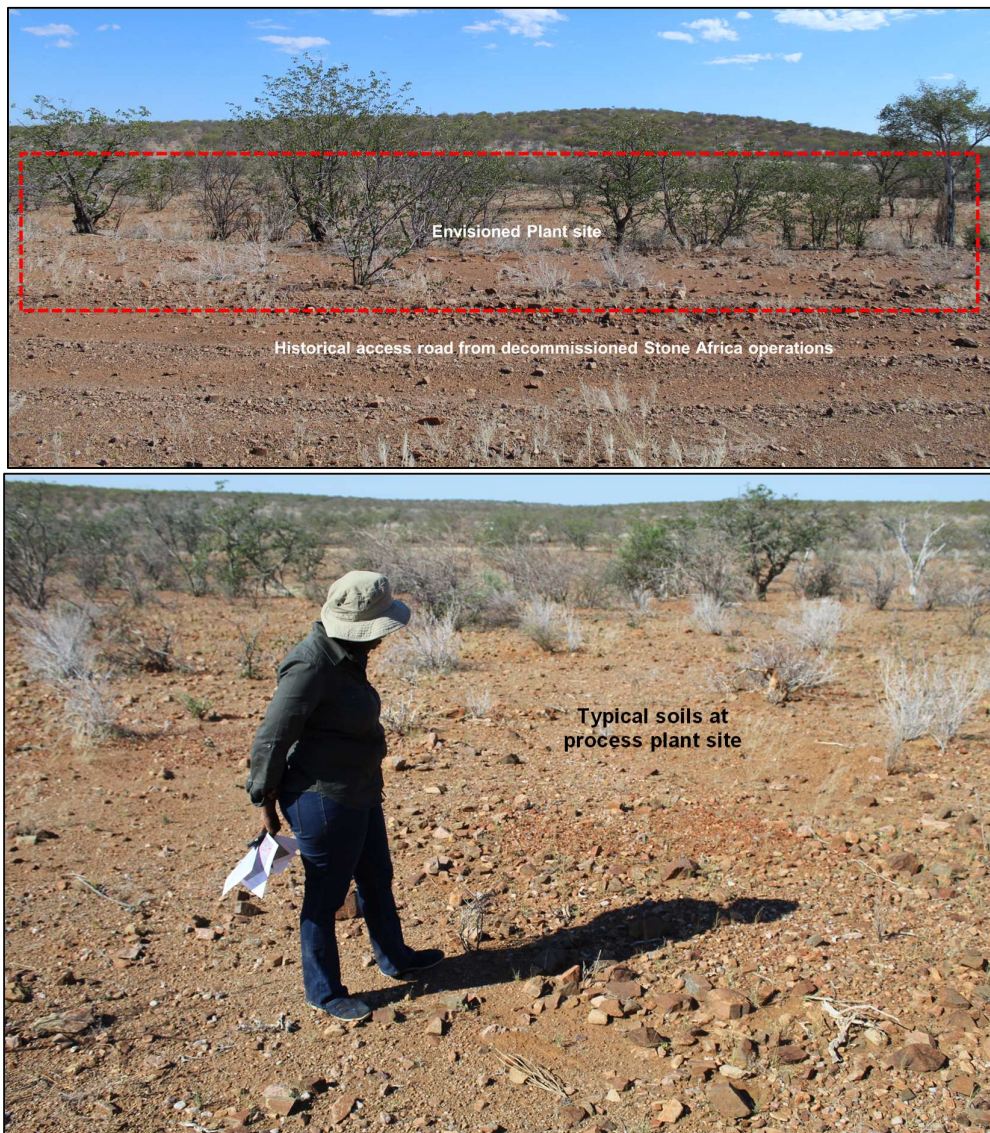


Photo 6. Relatively flat area and typical soils (sands and gravels) encountered at the envisioned process plant site.

WESTERN WRD AREA

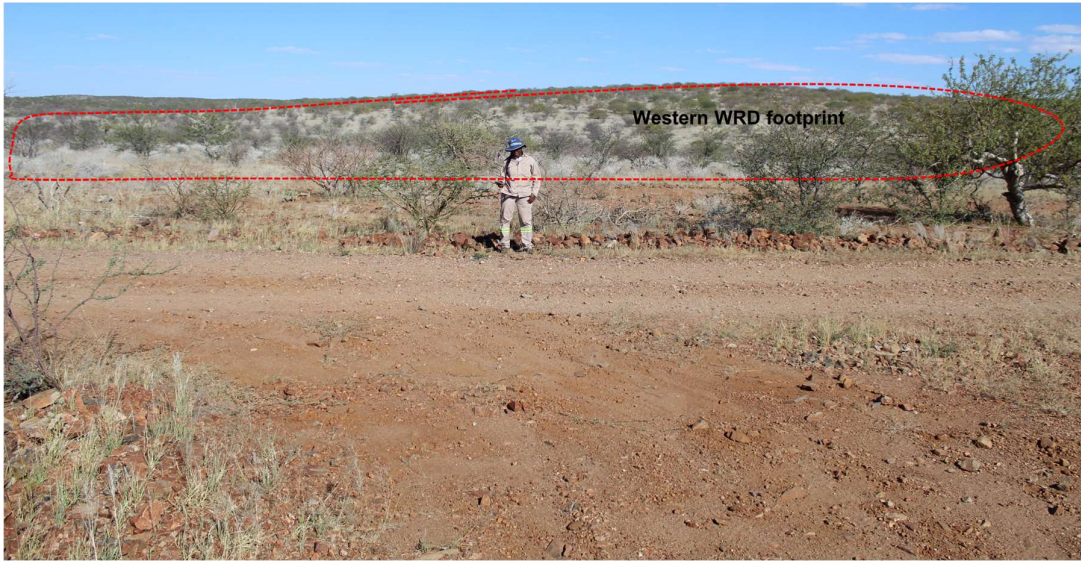


Photo 7. Gentle slopes over envisioned footprint of the western WRD. Photo taken facing south from point X on layout map. Thickly vegetated zone at toe of hill represents drainage channel in photo 8.



Photo 8. Drainage channel along northern toe of the western WRD.

TAILINGS STORAGE FACILITY AREA



Photo 9. Overview of valley (densely vegetated) where the TSF will be placed. 1st photo taken from hill on the northern of the proposed main access road to site (see layout map) looking towards the south east. 2nd photo shows current state of proposed main access road

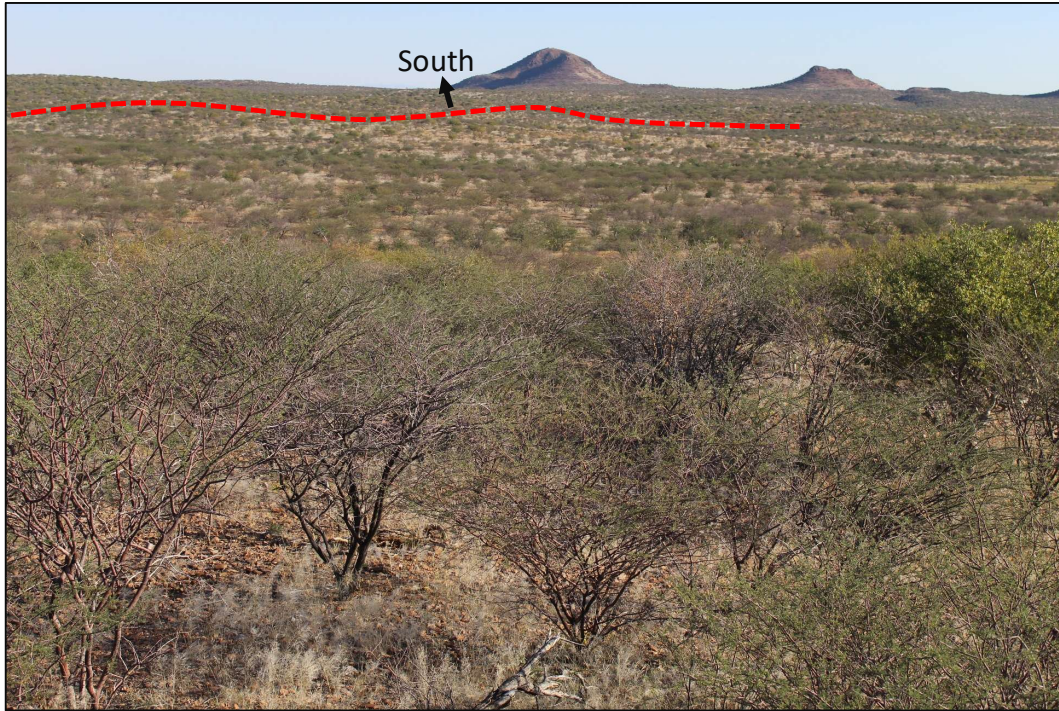
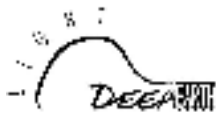


Photo 10. Looking south from proposed main access road over the main drainage channel and flood plain envisioned for TSF. The red line shows the approximate southern boundary of the TSF.



Photo 11. Typical vegetation cover and soils observed in various drainage channels truncating the envisioned TSF.

Appendix D:
Technical Memorandum from Light Deep Earth (LDE)



To:	Rainer Ellmies President, Namibia Critical Metals
From:	Jac Grobler Managing Director Technical, LightDeepEarth (Pty) Ltd
Date:	15 April 2021
Subject:	Lofdal gravity and magnetic concentration

1. DOCUMENT PURPOSE

To summarize the results of the gravity and magnetic separation testwork executed by LightDeepEarth on the Lofdal material.

2. BACKGROUND

Namibia Critical Metals have been investigating the separation of the Lofdal material to upgrade the TREO (total rare-earth oxides) from a typical 0.3% to a target grade of above 20%. The main mineral of interest in this deposit is xenotime containing significant concentrations of valuable heavy rare earths. Previous studies conducted by Mintek and Nagrom resulted in the following important conclusions:

- The valuable mineral (xenotime) liberation size to allow for effective separation is 38 µm.
- Gravity separation fractionations in general produced poor separation. Grinding sizes above 100 µm showed poor upgrades due to lack of liberation. Grinding sizes below 50 µm showed poor upgrades due to difficulty of fine gravity separation. Investigations into MGS (multi gravity separation) was recommended.
- Wet magnetic separation on the fine material (below 100 µm) showed the most potential but most of the work was conducted on efficient bench scale units. The magnetic separation needs to be demonstrated on pilot scale type units to confirm separation.
- The upgrade to above 20% TREO could not be achieved and the final TREO recovery was less than 60%.

Some alternative methods have been considered post the work conducted by Mintek and Nagrom in 2015:

- Utilizing XRF-sorting technology as a preconcentration step to upgrade the TREO prior more expensive fine grinding and processing.
- Re-considering gravity separation not for the purpose of upgrading xenotime but by selective rejection of liberated ankerite (problematic carbonate in leach circuit).
- Investigate the potential of the MGS for fine mineral gravity separation.

The purpose of the work executed at LDE can be summarized as follows:

- Upgrade fines in DMS by density differential on the fines which would by-pass sorters.
- Upgrade coarse sorted products (XRF sorted products selected over XRT sorted products due to slightly better performance).
- Ankerite (2.9-3.1 g/cm³) gravity rejection (shaking tables) on coarse particle size ranges (due to coarse liberation seen for gangue during previous mineralogy investigations).
- MGS investigations for fine mineral gravity separation.
- Investigate the potential of magnetic separation by evaluation of horizontal ring type units (Mineral Technology WHIMS and Gaustec WHIMS) and wet belt magnets (Longi and Malvern Engineering Magmizer)
- No flotation evaluation was done on this material.

This memo should be read together with the two supporting data files:

- QS results_NC20-03_a_Lofdal => feed mineralogy
- Testwork data_NC20-03_a_Lofdal => separation data

3. METHODOLOGY

The sample evaluation methodology can be summarized as follows:

1. Two samples were considered in the testwork:
 - a. 2.68 ton sample (0 x 10 mm) of natural fines that was screened as part of the crushing process applied on the as-received sample. No pre-concentration was done on this sample. This sample was screened on 8 mm and 1 mm to produce the correct sized feed for dense medium separation (DMS). 58% of the feed material fell in the 1 x 8 mm size fraction, 12% in 8 x 10mm and, 30% in 0 x 1mm size fraction. The TREO content in the targeted size fraction was 0.2%.
 - b. 1.27 ton sample that was pre-concentrated by means of XRF sorting (by Rados) and was crushed to 100% passing 1mm. This sample was screened into 5 size ranges (0 x 45, 45 x 106, 106 x 212, 212 x 500, 500 x 1000 µm) for a combination of gravity and magnetic separation. The TREO content in the crushed product prior size classification was 0.6%.
2. Size by assay on DMS head: The DMS feed material was screened into 6 size fractions and each size fraction was analysed by means of XRF (handheld, Olympus: Vanta C series on loose powder).
3. DMS at Pesco: The dense medium separation was done using a 50kg sample per test and performed at 7 different densities (relative density = 2.4, 2.5, 2.6, 2.7, 2.8, 2.9, 3.0). The aim was to isolate a low TREO concentration float fraction for rejection prior further size reduction and TREO upgrade. Sink and float fractions were analysed by XRF and XRD (quant).
4. Size by assay on the 0 x 1000 µm fraction: The gravity and magnetic evaluation feed material was split into 10 size fractions and analysed by means of XRF.
5. Qemscan on 5 sized fractions: Qemscan PMA (particle map analyses) was performed on the feed samples of each of the prepared size fractions (0 x 45, 45 x 106, 106 x 212, 212 x 500, 500 x 1000 µm).
6. Fraction 1: 500 x 1000 µm: Gravity fractionation by means of shaking table was conducted on this fraction to isolate liberated ankerite and not to upgrade xenotime since the xenotime is not liberated at these coarse size ranges. No magnetic separation was done on this fraction due to poor liberation.
7. Fraction 2: 212 x 500 µm: Gravity fractionation by means of shaking table was conducted on this fraction to isolate liberated ankerite and not to upgrade xenotime since the xenotime is not liberated at these coarse size ranges. No magnetic separation was done on this fraction due to poor liberation.
8. Fraction 3: 106 x 212 µm: Gravity fractionation by means of shaking table was conducted on this fraction to isolate liberated ankerite and not to upgrade xenotime since the xenotime is not liberated at these coarse size ranges. Carpc (dry, variable magnetic fractionation device) was used to evaluate the potential of dry separation. WHIMS (Mineral Technology unit) at 500 kg/hr and at maximum Gauss was used to evaluate wet magnetic separation.
9. Fraction 4: 45 x 106 µm: Gravity fractionation by means of shaking table and MGS (multi gravity separator) was conducted on this fraction to upgrade xenotime. Carpc (dry, variable magnetic fractionation device) was used to evaluate the potential of dry separation. WHIMS (Mineral Technology unit) and WRE (wet rare-earth), Longi Belt was used to evaluate wet magnetic separation. Further magnetic separation evaluation was done on the belt magnet from Malvern Engineering.
10. Fraction 5: 0 x 45 µm: Gravity fractionation by means of shaking table and MGS (multi gravity separator) was conducted on this fraction to upgrade xenotime. WRE (wet rare-earth), Longi Belt and Malvern Engineering Belt magnets were used to evaluate wet magnetic separation.
11. Operational settings: The testwork settings applied on the different units can be seen in the testwork data file (Testwork data_NC20-03_a_Lofdal)

4. DENSE MEDIUM SEPARATION (1 x 8 mm)

Table 1 provides the assay by size data by means of XRF. 58% of the mass reported 1 x 8 mm prepared fraction from the original supplied 0 x 10 mm material. The remaining 42% mass is represented by the 8 x 10 mm (12%) and 0 x 1 mm (30%). The 2.3% of the 0 x 1 mm indicated on Table 1 is due to screening efficiency on the bulk sample (2.68 t) as well as material degradation.

Table 2a and Table 2b illustrates the XRF and XRD of the density fractions from the DMS testwork. Ankerite concentrations were below 1% and therefore not detected by the XRD. Since this material is a sample from the fine fraction that by-passed the XRF sorting step it has a different mineral assemblage compared to the XRF sorted product. In this case the ankerite was absent indicating that the XRF sorting was upgrading the ankerite rich phases to the product. The TREO content was also three times lower compared to sorted product.

Calcite being the lighter mineral of the three majors (quartz, albite and calcite) could be rejected using DMS. The albite and quartz could not be effectively rejected. This could be considered as a pre-concentration method for a certain TREO loss to float due to poor liberation.

Table 1: XRF by size fraction on the DMS feed material, with TREO distribution

Fraction (mm)	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
7.1x8	0.00	0.69	0.04	0.48	15.36	2.3%	3.9%	0.0
6x7.1	0.24	3.23	0.27	7.08	6.77	4.1%	7.0%	7.8
5x6	0.25	3.24	0.28	7.27	6.75	17.4%	29.8%	35.1
3.35x5	0.21	3.14	0.26	6.87	5.08	6.4%	11.0%	10.8
2.36x3.35	0.20	3.15	0.26	6.57	5.10	9.8%	16.8%	15.7
1.8x2.36	0.20	3.27	0.26	6.87	6.20	10.5%	17.9%	17.3
1x1.8	0.20	3.43	0.26	7.86	6.44	5.6%	9.6%	9.3
0x1	0.21	4.03	0.31	9.75	4.78	2.3%	3.9%	3.9
DMS head	0.21	3.17	0.26	6.91	6.42	58.4%	100.0%	100.0

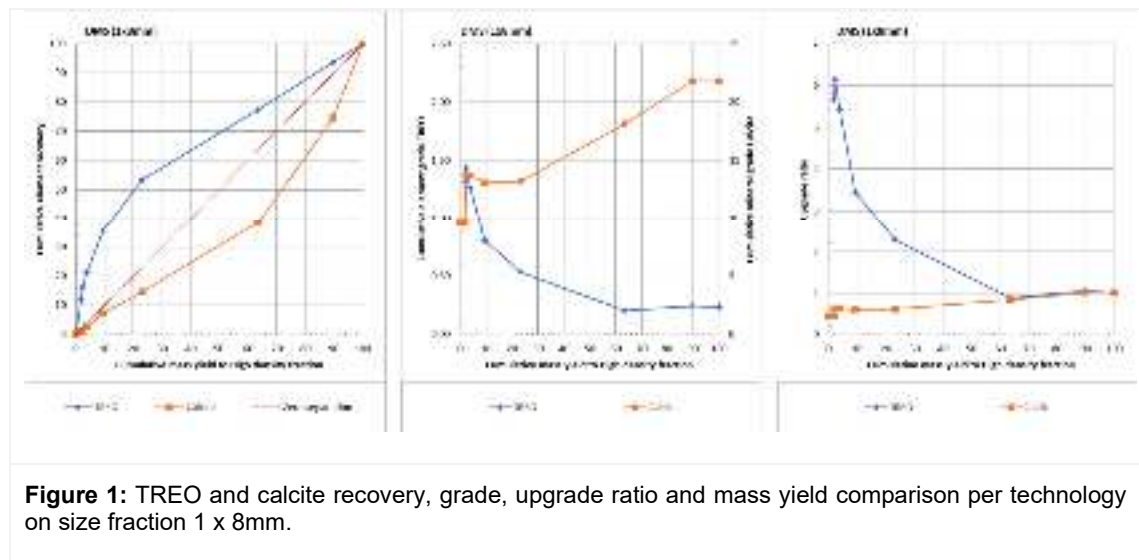
*%TREO is obtained from handheld XRF and is estimated % based on ppm Y. %Fe, %Ti, %Ca and %Si is obtained from handheld XRF. These are not precise analytical (ICP-MS) values. Applicable on all tables in the memorandum. T.Ms% = mass% expressed relative to the head of the total (including all processes). P.Ms% = the mass% relative to this specific process. TREO_D = TREO distribution or recovery into the respective fractions.

Table 2a: XRF by density fraction on the DMS process, with TREO distribution

Density	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
2.4 Float	0.14	2.54	0.20	15.69	2.38	5.6%	10.3%	6.3
2.4 Sink	0.24	3.05	0.27	5.82	5.35	49.2%	89.7%	93.7
2.5 Float	0.10	1.95	0.21	12.80	6.04	19.8%	36.4%	22.5
2.5 Sink	0.21	3.48	0.25	4.19	5.27	34.7%	63.6%	77.5
2.6 Float	0.14	2.22	0.23	8.31	7.75	41.8%	76.8%	46.6
2.6 Sink	0.53	5.98	0.36	3.90	5.21	12.6%	23.2%	53.4
2.7 Float	0.15	2.79	0.24	5.93	5.35	52.9%	90.6%	64.6
2.7 Sink	0.81	9.15	0.39	4.49	4.71	5.5%	9.4%	35.4
2.8 Float	0.19	2.57	0.28	7.61	5.78	56.2%	96.2%	78.8
2.8 Sink	1.27	15.82	0.50	4.51	4.40	2.2%	3.8%	21.2
2.9 Float	0.18	3.16	0.25	7.12	5.47	57.0%	97.7%	83.8
2.9 Sink	1.44	24.77	0.51	3.92	3.81	1.3%	2.3%	16.2
3.0 Float	0.19	2.80	0.27	7.11	6.39	57.2%	98.1%	88.1
3.0 Sink	1.33	29.77	0.41	3.15	3.44	1.1%	1.9%	11.9

Table 2b: XRD by density fraction on the DMS process, with calcite distribution

Density	Quartz	Albite	Calcite	Ankerite	T.Ms%	P.Ms%	Calcite_D
2.4 Float	11.3	32.0	53.6	0.0	5.6%	10.3%	25.2
2.4 Sink	15.3	62.4	18.2	0.0	49.2%	89.7%	74.8
2.5 Float	9.8	48.2	36.9	0.0	19.8%	36.4%	61.5
2.5 Sink	18.9	62.0	13.2	0.0	34.7%	63.6%	38.5
2.6 Float	12.8	58.5	23.3	0.0	41.8%	76.8%	85.4
2.6 Sink	20.0	54.0	13.1	0.0	12.6%	23.2%	14.6
2.7 Float	15.8	63.9	18.3	0.0	52.9%	90.6%	92.8
2.7 Sink	19.5	45.3	13.7	0.0	5.5%	9.4%	7.2
2.8 Float	16.0	61.5	20.0	0.0	56.2%	96.2%	97.4
2.8 Sink	18.7	31.1	13.8	0.0	2.2%	3.8%	2.6
2.9 Float	16.2	61.2	20.0	0.0	57.0%	97.7%	98.9
2.9 Sink	18.6	24.9	9.7	0.0	1.3%	2.3%	1.1
3.0 Float	16.8	61.0	19.8	0.0	57.2%	98.1%	99.1
3.0 Sink	17.7	22.4	9.7	0.0	1.1%	1.9%	0.9


Figure 1: TREO and calcite recovery, grade, upgrade ratio and mass yield comparison per technology on size fraction 1 x 8mm.

Technology evaluation on 1 x 8 mm

Only DMS was evaluated on this size fraction. Figure 1 illustrates a TREO and calcite response. If a 25% TREO loss can be accepted for this material, DMS can produce a 40% mass reduction. The cut-point density to achieve 40% mass rejection to floats is around 2.52 g/cm³. Table 2c shows the standard technology parameter table.

Table 2c: Technology review on size fraction 1x8mm

Size Fraction (mm)	Particle attribute	Technology	Main test parameters	Potential for improvement	Current performance rating
1x8	Gravity	DMS	Density cut point	Medium	Promising
1x8	Magnetic	Dry rare-earth magnet	Speed, feedrate, magnet strength	Medium	Not tested (recommended)

5. HEAD CHARACTERISATION (0 x 1 mm)

The 5 sized fractions prepared from the sorted products were individually characterised by means of Qemscan PMA, XRF and XRD. A separate head sample was screened into 10 size fractions to evaluate the TREO distribution. Table 3 illustrates the XRF assay by size data for the prepared 0 x 1 mm size fraction. Table 4a and Table 4b provides the basic XRF and the major phases by XRD of the individual bulk size fractions prepared for testwork. Table 4b indicates that this feed material (sorted product) contained noticeable higher levels of ankerite (6%) when compared to the 0% in the DMS feed material (Table 2b).

Table 3: XRF by size fraction on the 0 x 1 mm prepared material, with TREO distribution

Fraction (µm)	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	TREO_D
850 x 1000	0.67	4.93	0.32	12.48	3.69	8.2%	8.8
710 x 850	0.66	4.93	0.35	12.78	4.41	9.8%	10.4
600 x 710	0.65	4.92	0.32	12.54	3.52	7.2%	7.5
500 x 600	0.64	4.75	0.30	12.52	3.13	11.2%	11.4
300 x 500	0.62	4.81	0.33	12.89	3.69	11.4%	11.5
212 x 500	0.55	5.07	0.32	12.00	3.51	8.9%	7.9
150 x 212	0.57	5.81	0.32	12.74	2.74	7.7%	7.1
106 x 150	0.60	7.78	0.37	12.77	3.80	1.8%	1.8
45 x 106	0.47	6.70	0.34	13.45	3.08	9.7%	7.3
0 x 45	0.68	5.81	0.38	14.25	3.16	24.2%	26.3
Head	0.62	5.41	0.34	13.07	3.41	100.0%	100.0

Table 4a: XRF on prepared bulk size fractions, with TREO distribution

Fraction (µm)	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	TREO_D
500 x 1000	0.65	5.06	0.33	12.35	3.35	34.1%	35.2
212 x 500	0.62	5.12	0.34	11.87	2.90	24.3%	23.9
106 x 212	0.58	6.26	0.35	12.77	3.02	9.1%	8.3
45 x 106	0.51	6.67	0.32	14.45	3.07	10.2%	8.2
0 x 45	0.69	5.69	0.35	14.74	2.93	22.4%	24.4
Head	0.63	5.49	0.34	13.02	3.09	100.0%	100.0

Table 4b: XRD on prepared bulk size fractions, with ankerite distribution

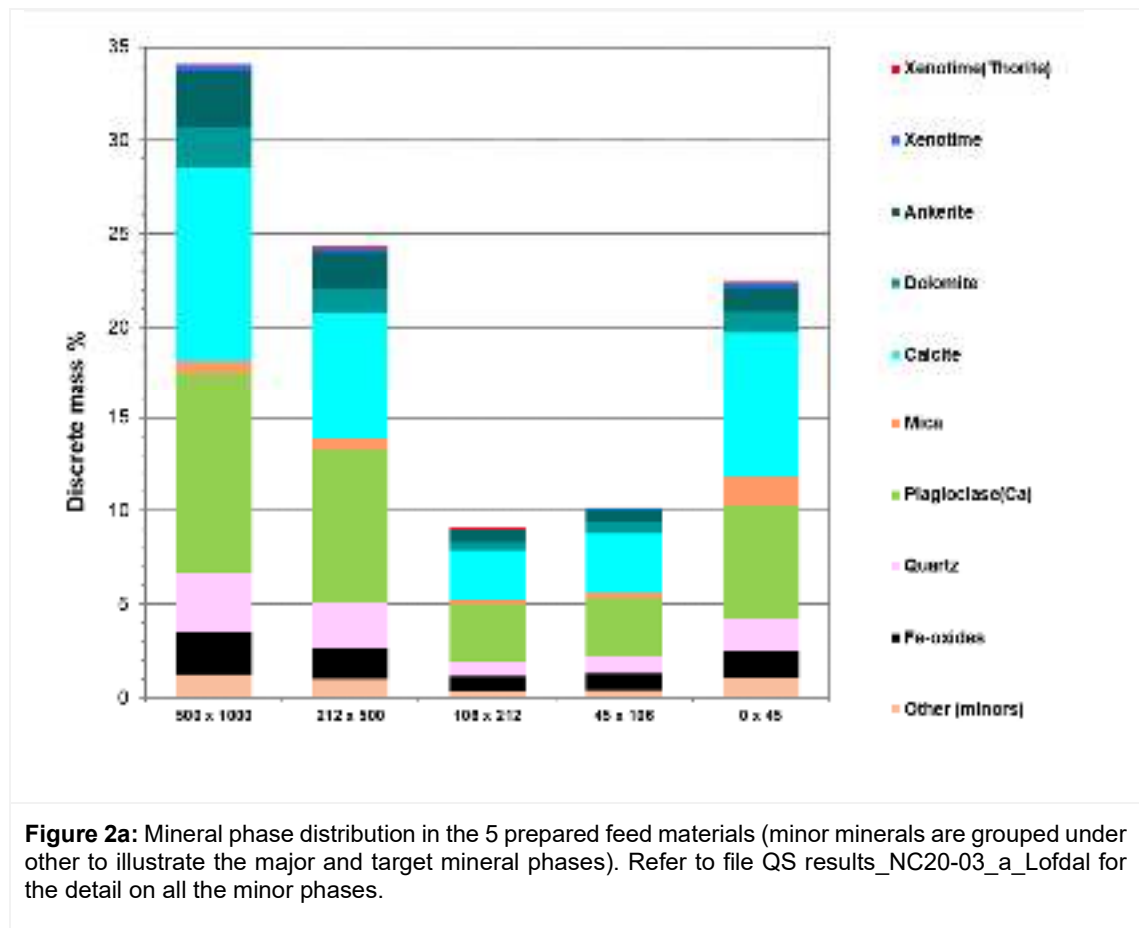
Fraction (µm)	Quartz	Albite	Calcite	Ankerite	Hematite	T.Ms%	Ank_D
500 x 1000	11.1	45.0	33.7	6.0	1.7	34.1%	39.2
212 x 500	10.9	43.3	35.9	4.7	2.2	24.3%	13.3
106 x 212	9.9	42.8	35.5	6.5	2.8	9.1%	10.1
45 x 106	8.5	39.0	38.0	7.7	3.4	10.2%	13.5
0 x 45	8.6	32.6	44.6	6.2	0.0	22.4%	24.0
Head	9.9	42.1	37.0	5.8	1.8	100.0%	100.0

The different mineral phases present in the 5 prepared head feeds is illustrated in Figure 2a. The three main REE phases (xenotime, xenotime(Thorite) and monazite) are illustrated in Figure 2b. Comparing Figure 2a and Figure 2b the xenotime distribution follows the mass distribution closely. The phase composition in each size fraction is similar, which indicates that there is no preferential phase breakage with resulting upgrades towards the finer fractions.

Figure 2c provides a summary of the liberation data. Each particle class represents a certain degree of liberation. The liberation variation is indicated in the list below:

- Xenotime-I = xenotime mass% in particle > 75%
- Xenotime-II = xenotime mass% in particle > 50-75%
- Xenotime-III = xenotime mass% in particle > 25-50%
- Xeno-Thor-I = xenotime(Thorite) mass% in particle > 75%
- Xeno-Thor-II = xenotime(Thorite) mass% in particle > 50-75%
- Xeno-Thor-III = xenotime(Thorite) mass% in particle > 25-50%
- REE Low + Fe = Combined zircon, xenotime, xenotime(thorite) > 10%

The ideal is to have high mass% of Xenotime-I (well liberated particle) to behave independently during separation ensuring a high grade and high recovery towards the product. Figure 2c illustrates that only the 0 x 45 μm feed contains these particles. This information aligns well with the Mintek liberation sizes (38 μm) previously measured. A high recovery is expected for Xenotime-I, the recovery likelihood decreases for Xenotime-II particles and even less for Xenotime-III. The same principal applies for the Xeno-Thor-I, II and III particles. From an ankerite liberation point of view, only the 0 x 45 μm contained liberated ankerite illustrated in Figure 2d. The dolomite liberation was better with clean dolomite particles (Dolomite-I) being present in all size fractions.



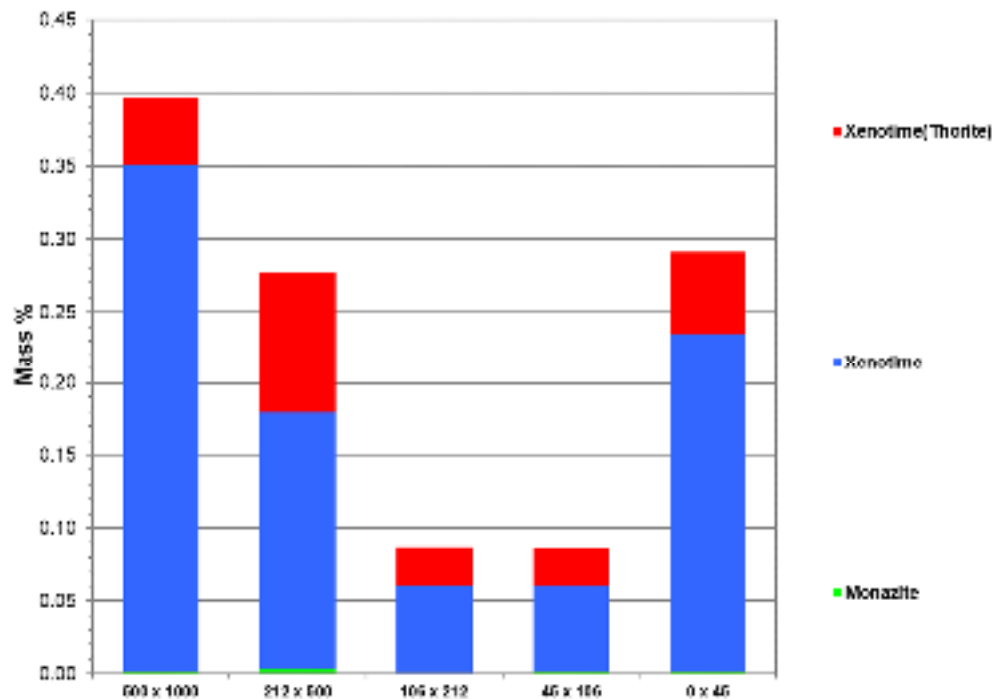


Figure 2b: REE Mineral phase distribution in the 5 prepared feed materials

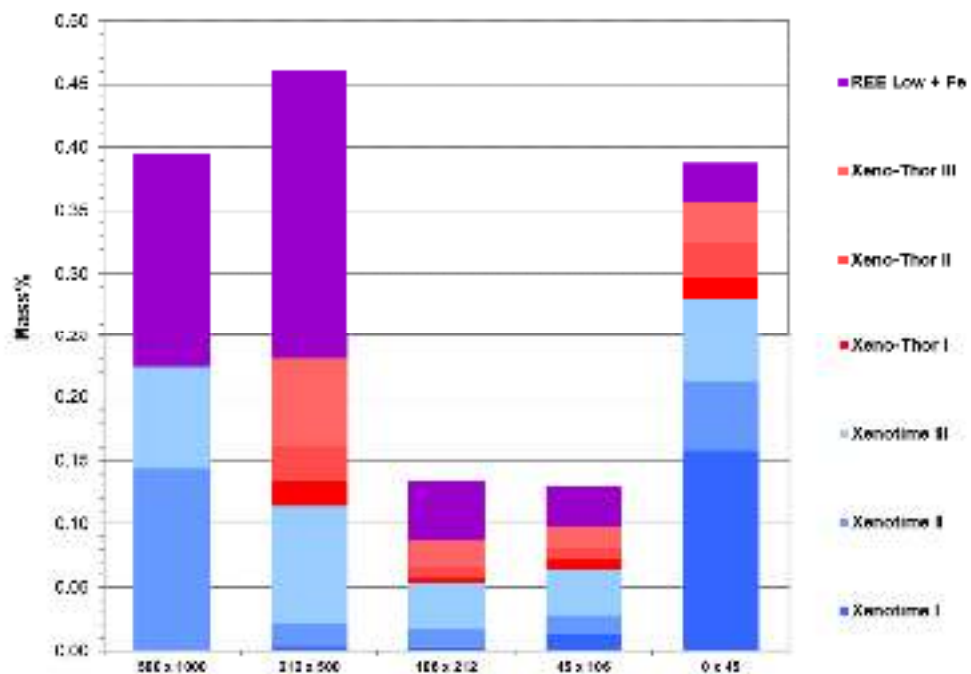


Figure 2c: Mineral particles (degrees of liberation) across prepared feeds

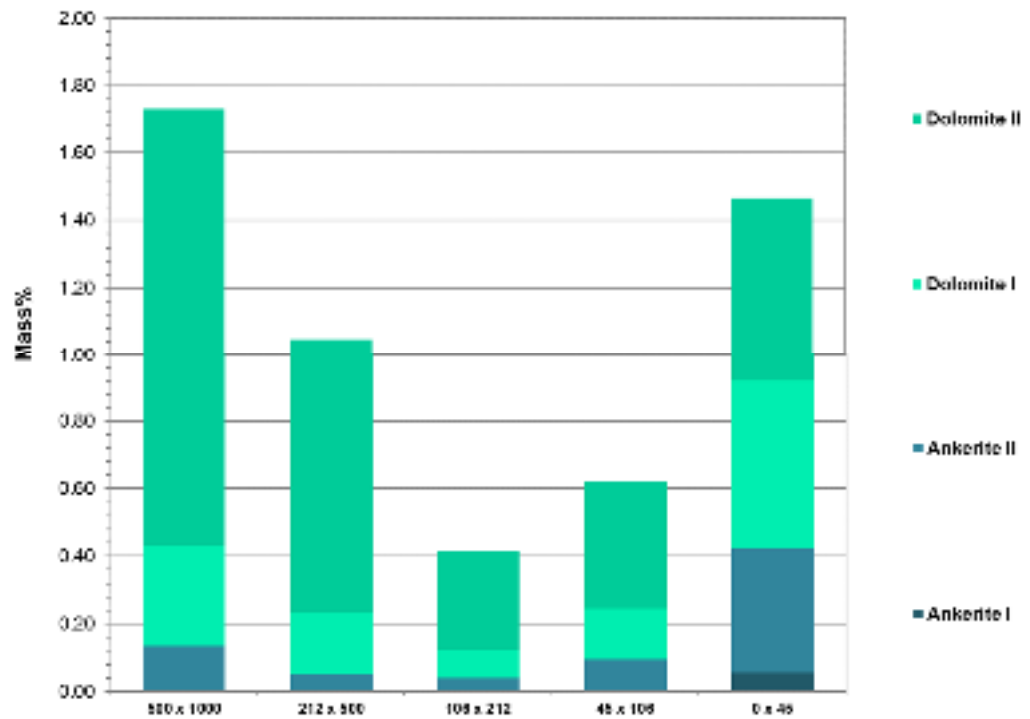


Figure 2d: Mineral particles (degrees of liberation) across prepared feeds

6. SEPARATION OF FRACTION 1: 500 x 1000 µm

The 500 x 1000 µm fraction represents 34% of the feed (Table 4a) with 35% of the TREO and contained only poorly liberated xenotime particles. This feed material was subjected to shaking table fractionation with the aim to isolate a mass fraction containing low levels of TREO (pre-concentration). Table 5a illustrates that the TREO was upgraded from 0.7 to 1.2 at 20% TREO recovery. The ankerite showed upgrade towards the midd1 and midd2 fractions (Table 5b) but showed strong overlap with the TREO distribution (Table 5a) which indicate that these two are most likely associated. Hematite showed high levels of concentration towards the concentrate fraction, this indicates that the hematite was well liberated at this particle size range.

Table 5a: XRF on shaking table fraction of the 500 x 1000 µm feed, with TREO distribution

Density	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
conc	1.16	11.52	0.39	11.98	3.69	3.9%	11.4%	20.2
midd1	0.74	5.62	0.30	12.58	3.13	9.7%	28.6%	32.4
midd2	0.60	3.82	0.32	12.11	3.50	12.8%	37.6%	34.4
midd3	0.46	3.33	0.31	12.24	4.49	3.6%	10.5%	7.4
tail	0.34	2.77	0.29	11.20	4.04	4.1%	11.9%	6.2
Hd 500 x 1000	0.65	5.06	0.33	12.35	3.35	34.1%	100.0%	100.0

Table 5b: XRD of shaking table fractions of the 500 x 1000 µm feed, with Ankerite distribution

Density	Quartz	Albite	Calcite	Ankerite	Hematite	T.Ms%	P.Ms%	Ank_D
conc	13.3	34.1	32.3	7.7	9.8	3.9%	11.4%	14.6
midd1	11.4	41.9	35.4	7.2	2.2	9.7%	28.6%	34.1
midd2	11.0	44.2	35.1	7.0	0.0*	12.8%	37.6%	43.5
midd3	10.7	55.8	29.3	2.5	0.0*	3.6%	10.5%	4.4
tail	9.4	56.3	30.7	1.8	0.0*	4.1%	11.9%	3.5
Hd 500 x 1000	11.1	45.0	33.7	6.0	1.7	34.1%	100.0%	100.0

*Zero numbers on this table are below the detection limit of the XRD and is most likely not zero but a low number below 0.5%

Technology evaluation on 500 x 1000 µm

Only shaking table was evaluated on this size fraction. Figure 3 illustrates a single line with the TREO recovery (left) and grade with upgrade ratio (middle and right) showing weak upgrade response. This fraction represents a large portion of feed mass (34%) and is to be crushed finer to improve on liberation. Table 5c shows the standard technology parameter table.

Table 5c: Technology review on size fraction 500x1000 µm

Size Fraction (micron)	Particle attribute	Technology	Main test parameters	Potential for improvement	Current performance rating
500x1000	Gravity	Shaking Table	Angle, waterflow, feedrate	Low	Poor
500x1000	Magnetic	Dry rare-earth magnet	Speed, feedrate, magnet strength	Medium	Not tested (recommended)

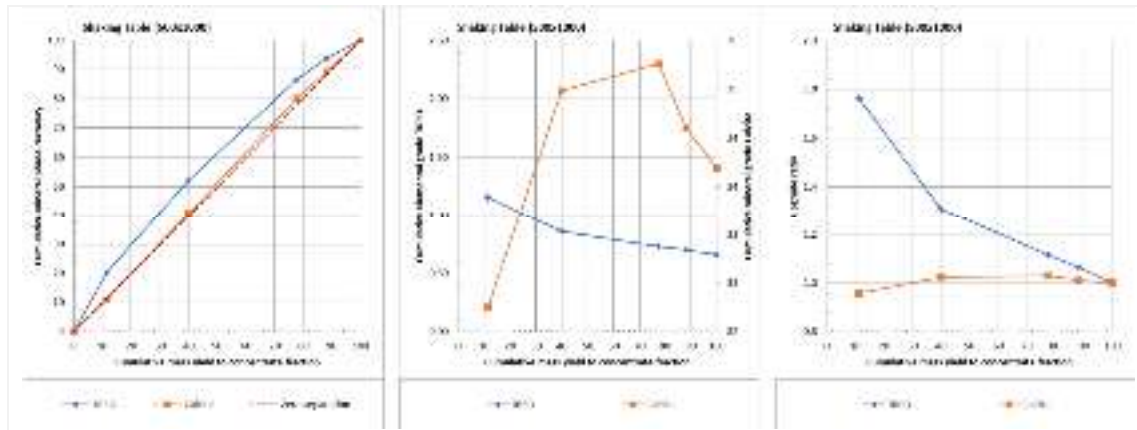


Figure 3: TREO and calcite recovery, grade, upgrade ratio and mass yield on size fraction 500x1000 µm.

7. SEPARATION OF FRACTION 2: 212 x 500 µm

The 212 x 500 µm fraction represents 24% of the feed (Table 4a) with 25% of the TREO and contained only poorly liberated xenotime particles. This feed material was subjected to shaking table fractionation with the aim to isolate a mass fraction containing low levels of TREO (pre-concentration). Table 6a illustrates that the TREO was upgraded from 0.6 to 1.3 at 27% TREO recovery. The ankerite had better upgrade (compared to Table 5b) towards the midd1 and midd2 fractions (Table 6b). The ankerite and TREO showed a strong overlap in the gravity concentration, which showed low potential to isolate an ankerite rich fraction without the sacrifice of significant TREO recovery units.

Table 6a: XRF on shaking table fraction of the 212 x 500 µm feed, with TREO distribution

Density	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
conc	1.25	13.81	0.46	11.68	2.00	3.3%	13.7%	27.6
midd1	0.65	4.32	0.32	12.89	2.31	7.4%	30.6%	31.8
midd2	0.53	3.51	0.31	13.42	3.63	9.5%	39.2%	33.7
midd3	0.38	2.90	0.29	11.81	3.00	2.8%	11.4%	6.9
tail	0.30	2.55	0.28	11.05	3.14	1.2%	5.1%	2.4
Hd 212 x 500	0.62	5.12	0.34	11.87	2.90	24.3%	100.0%	100.0

Table 6b: XRD of shaking table fractions of the 212 x 500 µm feed, with ankerite distribution

Density	Quartz	Albite	Calcite	Ankerite	Hematite	Ms%	P.Ms%	Ank_D
conc	10.0	33.2	32.9	7.6	12.4	3.3%	13.7%	22.2
midd1	11.9	40.1	36.9	6.6	1.7	7.4%	30.6%	42.8
midd2	10.6	46.4	36.8	3.1	0.0*	9.5%	39.2%	26.0
midd3	10.7	49.1	35.5	2.8	0.0*	2.8%	11.4%	6.9
tail	9.9	53.3	33.1	2.0	0.0*	1.2%	5.1%	2.2
Hd 212 x 500	10.9	43.3	35.9	4.7	2.2	24.3%	100.0%	100.0

*Zero numbers on this table are below the detection limit of the XRD and is most likely not zero but a low number below 0.5%

Technology evaluation on 212 x 500 μm

Similar to the coarser size fractions, only shaking table was evaluated on this size fraction. Figure 4 illustrates a single line with the TREO recovery (left), grade (middle) and upgrade ratio (right) showing weak response. The upgrade was slightly better compared to Figure 3 due to more liberation (Figure 2c). This fraction represents a large portion of feed mass (24%) and is to be crushed finer to improve on liberation. Table 6c shows the standard technology parameter table.

Table 6c: Technology review on size fraction 212x500 μm

Size Fraction (micron)	Particle attribute	Technology	Main test parameters	Potential for improvement	Current performance rating
212x500	Gravity	Shaking Table	Angle, waterflow, feedrate	Low	Poor
212x500	Magnetic	Dry rare-earth magnet	Speed, feedrate, magnet strength	Medium	Not tested

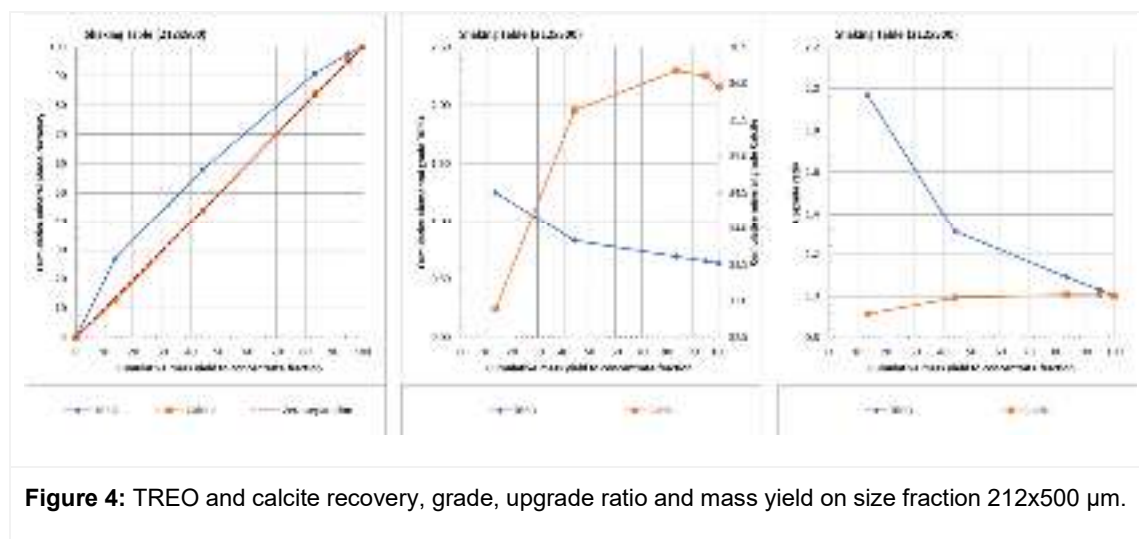


Figure 4: TREO and calcite recovery, grade, upgrade ratio and mass yield on size fraction 212x500 μm .

8. SEPARATION OF FRACTION 3: 106 x 212 μm

The 106 x 212 μm fraction represents 9% of the feed (Table 4a) with 8% of the TREO and contained mostly poorly liberated xenotime particles. This feed material was subjected to shaking table fractionation with the aim to isolate a mass fraction containing low levels of TREO (pre-concentration). Table 7a illustrates that the TREO was upgraded from 0.6 to 1.5 at 18% TREO recovery. The ankerite showed a higher upgrade the midd1 fraction (Table 7b). Although the tail showed a lower concentration of TREO (0.37%) the 36% TREO loss due to poor liberation is still a concern.

Table 7a: XRF on shaking table fractions of the 106 x 212 μm feed, with TREO distribution

Density	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
conc	1.45	49.49	0.62	4.38	1.65	0.6%	7.0%	17.9
midd1	1.13	9.62	0.60	13.72	2.08	0.6%	7.0%	13.9
midd2	0.71	4.79	0.42	14.26	2.39	1.0%	11.2%	14.0
midd3	0.52	3.56	0.29	13.48	2.70	1.8%	19.5%	17.8
tail	0.37	2.87	0.25	12.79	3.59	5.0%	55.4%	36.3
Hd 106 x 212	0.57	6.95	0.33	12.57	3.04	9.1%	100.0%	100.0

Table 7b: XRD of shaking table fractions of the 106 x 212 µm feed, with ankerite distribution

Density	Quartz	Albite	Calcite	Ankerite	Hematite	T.Ms%	P.Ms%	Ank_D
conc	6.9	8.5	10.2	1.4	64.4	0.6%	7.0%	2.0
midd1	7.7	33.1	37.6	9.4	9.0	0.6%	7.0%	13.3
midd2	8.7	39.3	40.1	9.0	0.0*	1.0%	11.2%	20.2
midd3	8.9	43.2	38.4	7.1	0.0*	1.8%	19.5%	28.0
tail	9.8	49.3	35.4	3.3	0.0*	5.0%	55.4%	36.4
Hd 106 x 212	9.2	43.0	34.9	5.0	5.1	9.1%	100.0%	100.0

*Zero numbers on this table are below the detection limit of the XRD and is most likely not zero but a low number below 0.5%

Dry magnetic separation by means of Carpcio is illustrated in Table 8a and Table 8b. The TREO (0.6 to 2.0%) upgrade occurred in the intermediate gauss ranges (12-19 kG) but 55% of the TREO remained in the non-magnetic (nmg) fraction due to poor xenotime liberation. The Fe grade distribution showed a strong correlation with the magnetic strength, which is expected, and similar trend was seen with the XRD data for hematite. Ankerite was concentrated towards the same magnetic fractions as the xenotime. This implies that most of the ankerite is mineralised (containing REE).

Table 8a: XRF on Carpcio fractions of the 106 x 212 µm feed, with TREO distribution

Magnetic	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
1.7kG mag	0.52	56.68	0.21	1.21	1.24	0.3%	3.0%	2.7
6.1kG mag	1.02	41.08	0.34	2.72	2.95	0.4%	4.1%	7.3
15.4kG mag	2.02	16.85	0.67	10.80	1.94	0.6%	6.2%	21.7
18.9kG mag	1.86	8.69	0.77	14.56	1.56	0.3%	3.6%	11.5
nmg	0.37	2.48	0.32	13.42	3.44	7.5%	83.1%	52.9
Magsep head	0.58	6.26	0.35	12.77	3.02	9.1%	100.0%	100.0

Table 8b: XRD of Carpcio fractions of the 106 x 212 µm feed, with ankerite distribution

Magnetic	Quartz	Albite	Calcite	Ankerite	Hematite	T.Ms%	P.Ms%	Ank_D
1.7kG mag	7.5	0.2	0.0*	0.0*	74.4	0.3%	3.0%	0.0*
6.1kG mag	19.1	4.7	1.4	0.0*	68.3	0.4%	4.1%	0.0*
15.4kG mag	12.2	13.9	28.1	15.4	15.9	0.6%	6.2%	14.8
18.9kG mag	8.7	14.3	37.2	23.0	7.3	0.3%	3.6%	12.7
nmg	8.6	51.9	34.9	2.5	0.0*	7.5%	83.1%	31.6
Magsep head	9.9	42.8	35.5	6.5	2.8	9.1%	100.0%	100.0

*Zero numbers on this table are below the detection limit of the XRD and is most likely not zero but a low number below 0.5%

Wet magnetic separation by means of Mineral Technology WHIMS at maximum gauss (16Amp, 17 kG) is illustrated in Table 9a and Table 9b. A TREO (0.6 to 1.2%) upgrade was achieved but 73% of the TREO remained in the non-magnetic (nmg) fraction due to poor xenotime liberation or possibly not high enough magnetic pull. Wet separation requires higher magnetic field strengths to counter the vertical drag forces that wash fine weakly magnetic materials from the matrix. It is suspected that higher magnetic field strengths (>18 kG) will achieve higher TREO recovery. The high Fe recovery (>58%), primarily hematite, to small mass magnetic fraction (12% mass) showed that the mineral separation is taking place.

Table 9a: XRF on WHIMS fractions of the 106 x 212 µm feed, with TREO distribution

Magnetic	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
mag	1.22	33.21	0.44	6.40	3.35	1.1%	12.3%	26.4
mid	1.10	33.11	1.03	5.26	2.76	0.0%	0.5%	1.0
nmg	0.47	3.00	0.34	13.45	3.08	7.9%	87.2%	72.6
WHIMS hd	0.57	6.86	0.35	12.54	3.11	9.1%	100.0%	100.0

Table 9b: XRD of WHIMS fractions of the 106 x 212 µm feed, with ankerite distribution

Magnetic	Quartz	Albite	Calcite	Ankerite	Hematite	T.Ms%	P.Ms%	Ank_D
mag	12.7	26.0	13.9	2.1	38.1	1.1%	12.3%	6.9
mid	15.7	18.0	13.7	1.5	44.0	0.0%	0.5%	0.2
nmg	9.4	47.8	36.6	4.0	0.0*	7.9%	87.2%	92.9
WHIMS hd	9.8	44.9	33.6	3.8	4.9	9.1%	100.0%	100.0

*Zero numbers on this table are below the detection limit of the XRD and is most likely not zero but a low number below 0.5%

Technology evaluation on 106 x 212 µm

One gravity and two magnetic technologies were evaluated. Figure 5 illustrates three lines with the TREO recovery (left), grade (middle) and upgrade ratio (right) showing medium to weak response. The dry magnetic separation (CRP) showed the best performance but could not be considered as a separation technology since it was only for characterisation. The upgrade was slightly better compared to Figure 4 due to more liberation (Figure 2c). This fraction represents a smaller portion of feed mass (9%). Table 9c shows the standard technology parameter table. Dry rare earth magnets can be considered to isolate a non-magnetic fraction for upfront rejection provided that it contains low levels of TREO (low recovery loss). The Carpc fractionation showed that a high magnetic (hematite as identified by XRD) could be isolated with relatively low TREO recovery loss to high mag fraction. Note that the Carpc fractions were plotted according to grade in mag fraction and not according to magnetic field strength. The standard WHIMS unit was operated at its highest gauss setting and could not extract further magnetic material from the non-magnetic fraction unless the matrix is changed to finer plate spacing.

Table 9c: Technology review on size fraction 106x212 µm

Size Fraction (micron)	Particle attribute	Technology	Main test parameters	Potential for improvement	Current performance rating
106x212	Gravity	Shaking Table	Angle, waterflow, feedrate	Low	Poor
106x212	Magnetic	Dry fractions (multiple gauss)	Magnet strength	Low	Poor
106x212	Magnetic	WHIMS	Feedrate, magnet strength	Low	Poor
106x212	Magnetic	Dry rare-earth magnet	Speed, feedrate, magnet strength	Medium	Not tested
106x212	Magnetic	WHIMS – fine matrix	Feedrate, magnet strength	Medium	Not tested

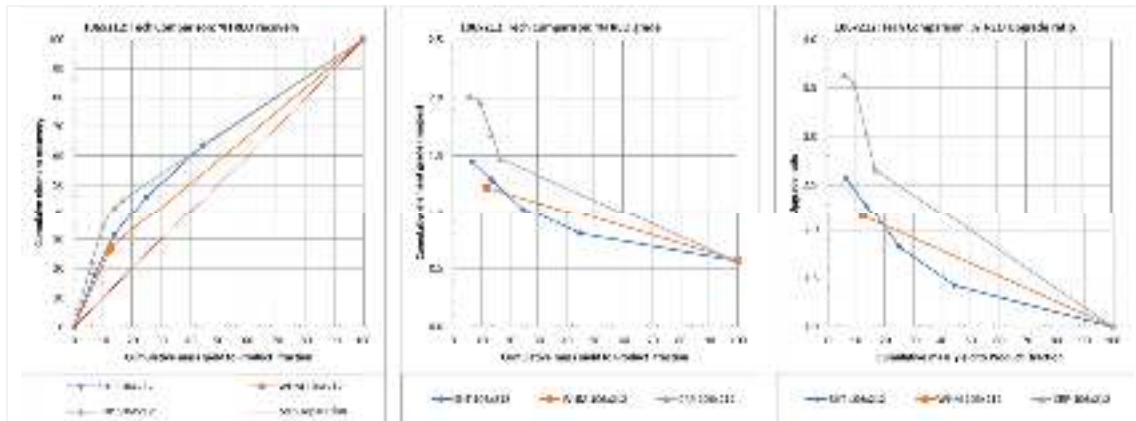


Figure 5: TREO recovery-yield, grade-recovery and upgrade-ratio-yield comparison per technology on size fraction 106 x 212 µm.

9. SEPARATION OF FRACTION 4: 45 x 106 µm

The 45 x 106 µm fraction represents 10% of the feed (Table 4a) with 8% of the TREO and contained mostly poorly liberated xenotime particles. This feed material was subjected to shaking table gravity fractionation with the aim to concentrate xenotime. Table 10a illustrates that the TREO was upgraded from 0.5 to 1.6 at 21% TREO recovery. The other remaining TREO was distributed across all other gravity ranges which indicates various degrees of xenotime liberation.

Table 10a: XRF on shaking table fractions of the 45 x 106 µm feed, with TREO distribution

Density	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
conc	1.55	51.22	0.69	4.56	1.42	0.7%	6.6%	21.0
midd1	1.21	13.52	0.66	14.88	1.80	0.4%	3.5%	8.6
midd2	0.64	5.43	0.42	14.87	1.93	1.7%	16.8%	21.9
midd3	0.41	3.50	0.29	14.67	2.57	1.9%	19.0%	16.1
tail	0.29	3.04	0.22	13.35	3.26	5.5%	54.2%	32.5
Hd 45 x 106	0.49	7.08	0.31	13.32	2.73	10.2%	100.0%	100.0

Table 10b: XRD of shaking table fractions of the 45 x 106 µm feed, with ankerite distribution

Density	Quartz	Albite	Calcite	Ankerite	Hematite	T.Ms%	P.Ms%	Ank_D
conc	3.1	4.9	8.9	2.1	68.3	0.7%	6.6%	2.2
midd1	6.0	25.4	37.1	13.5	14.3	0.4%	3.5%	7.6
midd2	8.0	32.8	43.4	11.0	1.8	1.7%	16.8%	29.7
midd3	10.0	38.0	39.5	9.4	0.0*	1.9%	19.0%	28.7
tail	10.5	43.4	37.2	3.7	0.0*	5.5%	54.2%	31.9
Hd 45 x 106	9.3	37.4	36.8	6.2	5.3	10.2%	100.0%	100.0

*Zero numbers on this table are below the detection limit of the XRD and is most likely not zero but a low number below 0.5%

The MGS is another gravity separation technique that was evaluated on this size fraction (more than 23 separate tests). The MGS results on this fraction showed a large degree of variation from a TREO grade-recovery perspective. The reasons and test condition details are documented in the Coremet technical memorandum (22 Feb 2021) (3rd party who conducted these tests). Table 11 illustrates the results of selected tests that showed the most potential (all the results can be viewed in the testwork file). The TREO could be upgraded from 0.5 to 1.5%. 72% of the TREO could be concentrated in 40% of the mass (Test 2 in Table 11).

Table 11: XRF on MGS gravity fractions of the 45 x 106 µm feed, with TREO distribution

Density	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
T1-conc	0.76	11.13	0.45	16.77	3.46	6.7%	66.3%	83.5
T1-tail	0.30	2.92	0.22	11.71	3.74	3.4%	33.7%	16.5
T1-head	0.61	8.37	0.37	15.07	3.55	10.2%	100.0%	100.0
T2-conc	1.14	15.31	0.66	13.26	1.73	4.1%	40.2%	71.9
T2-tail	0.30	3.06	0.22	12.77	3.41	6.1%	59.8%	28.1
T2-head	0.64	7.99	0.40	12.97	2.74	10.2%	100.0%	100.0
T3-conc	1.41	30.35	0.65	8.72	1.19	1.8%	17.9%	48.7
T3-tail	0.32	3.04	0.23	13.22	2.70	8.3%	82.1%	51.3
T3-head	0.52	7.93	0.31	12.41	2.43	10.2%	100.0%	100.0
T4-conc	1.52	37.94	0.71	8.28	1.62	1.2%	12.2%	30.4
T4-tail	0.36	3.45	0.28	14.50	4.31	8.9%	87.8%	52.5
T4-head	0.50	7.64	0.33	13.75	3.98	10.2%	100.0%	82.9

Dry magnetic separation by means of Carpcio is illustrated in Table 12a and Table 12b. The TREO (0.6 to 1.3%) upgrade took place in the intermediate gauss ranges (12-19 kG) and more TREO is recovered to the 15.4kG fraction. The TREO recovery to the mag fractions improved from 45 to 67% due to improved xenotime liberation.

Table 12a: XRF on Carpcio fractions of the 45 x 106 µm feed, with TREO distribution

Magnetic	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
1.7kG mag	0.34	56.24	0.13	1.18	0.91	0.2%	2.3%	1.5
6.1kG mag	0.93	33.46	0.30	5.93	1.70	0.7%	7.2%	13.2
15.4kG mag	1.34	8.66	0.56	15.01	1.82	1.3%	13.2%	34.7
18.9kG mag	0.89	6.05	0.50	15.84	1.38	0.8%	7.9%	13.8
nmG	0.23	1.97	0.24	14.45	3.92	7.1%	69.4%	31.5
Magsep head	0.51	6.67	0.32	14.45	3.07	10.2%	100.0%	100.0

Table 12b: XRD of Carpcio fractions of the 45 x 106 µm feed, with ankerite distribution

Magnetic	Quartz	Albite	Calcite	Ankerite	Hematite	T.Ms%	P.Ms%	Ank_D
1.7kG mag	3.9	0.5	0.0*	0.0*	69.4	0.2%	2.3%	0.0
6.1kG mag	10.1	16.8	16.3	2.8	42.9	0.7%	7.2%	2.6
15.4kG mag	9.4	15.4	36.0	21.6	1.6	1.3%	13.2%	37.0
18.9kG mag	7.5	18.5	44.1	19.0	0.0*	0.8%	7.9%	19.5
nmG	9.7	46.5	40.3	2.4	0.0*	7.1%	69.4%	21.5
Magsep head	8.5	39.0	38.0	7.7	3.4	10.2%	100.0%	100.0

*Zero numbers on this table are below the detection limit of the XRD and is most likely not zero but a low number below 0.5%

Three different horizontal ring WHIMS tests were performed on 45 x 106 µm. The first was the Mineral Technologies (MT) WHIMS using the normal 2mm matrix at maximum gauss (16Amp, 17 kG). This test is illustrated as the first test in Table 13a and Table 13b. Due to the low mass yield

to magnetic fraction as second batch of tests were performed using the fine 1mm matrix for the MT WHIMS. This matrix was able to achieve a higher mass yield to the magnetic fraction resulting in 62% TREO recovery. The third test was conducted using the Gaustec “Mini-mag” (GTC in Table 13a) unit that could achieve higher gauss ranges (above 18 kG). 44% of the TREO could be recovered in 16% percent of the mass (1.3% TREO grade) which indicated a higher level of selectivity.

Table 13a: XRF on WHIMS fractions of the 45 x 106 µm feed, with TREO distribution

Magnetic	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
mag (2mm)	1.09	40.24	0.31	5.05	1.83	0.8%	8.1%	18.7
mid (2mm)	1.06	15.76	0.37	10.69	1.89	0.5%	5.0%	11.2
nmg (2mm)	0.38	3.04	0.35	14.33	3.20	8.8%	86.9%	70.1
WHIMS hd	0.47	6.69	0.35	13.40	3.03	10.2%	100.0%	100.0
mag (1mm)	0.53	4.86	0.35	14.27	2.16	4.9%	48.2%	62.1
nmg (1mm)	0.30	2.80	0.26	13.14	2.77	5.3%	51.8%	37.9
WHIMS hd	0.41	3.79	0.31	13.68	2.47	10.2%	100.0%	100.0
GTC-mag	1.32	26.02	0.42	9.95	1.92	1.6%	15.9%	44.4
GTC-mid	0.40	3.11	0.32	15.59	2.82	3.4%	33.4%	28.4
GTC-nmg	0.25	2.19	0.27	14.46	3.77	5.2%	50.7%	26.7
GTC hd	0.47	6.29	0.31	14.12	3.16	10.2%	100.0%	99.4

Table 13b: XRD of WHIMS fractions of the 45 x 106 µm feed, with ankerite distribution

Magnetic	Quartz	Albite	Calcite	Ankerite	Hematite	T.Ms%	P.Ms%	Ank_D
mag (2mm)	8.50	12.92	10.76	2.82	52.37	0.8%	8.1%	2.7
mid (2mm)	7.75	30.26	26.84	12.42	14.21	0.5%	5.0%	7.3
nmg (2mm)	9.49	41.06	37.07	8.82	0.00	8.8%	86.9%	90.1
WHIMS hd	9.32	38.25	34.43	8.51	4.95	10.2%	100.0%	100.0
mag (1mm)	9.57	36.00	36.69	10.46	1.64	4.9%	48.2%	59.2
nmg (1mm)	9.24	52.09	34.56	2.66	0.00	5.3%	51.8%	16.2
WHIMS hd	9.40	44.34	35.59	6.42	0.79	10.2%	100.0%	75.4

*Zero numbers on this table are below the detection limit of the XRD and is most likely not zero but a low number below 0.5%. XRD was not perform on the GTC fractions.

Wet magnetic separation by means of Longi and Malvern Engineering (ME) BMG (belt magnet) is illustrated in Table 14a and Table 14b. These unit favours fine mineral magnetic recovery since the wash water is not vertical but at an angle to allow better retention of the fine magnetic materials. The Longi has a lower Gauss (12 kG) permanent magnet compared to the ME-BMG at 15kG. Due to the magnet difference more mass could be pulled to the magnetic fraction of the ME BMG resulting in higher TREO recovery. The lower gauss on these units had better results compared to the WHIMS at 17kG due to the configuration of the material flow to limit the drag force on the fine particles.

The ME BMG was able to recover 78% of the TREO in 39% of the mass resulting in the lowest tail grade (0.15% TREO) from all the technologies evaluated. The Longi BMG could only achieve a 22% mass yield to mag at higher TREO grade (1.1% TREO).

Table 14a: XRF on BMG fractions of the 45 x 106 µm feed, with TREO distribution

Magnetic	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
Longi-mag	1.10	20.12	0.48	10.98	2.44	2.2%	21.6%	52.7
Longi-nmg	0.27	2.41	0.26	13.85	2.55	8.0%	78.4%	47.3
Longi-head	0.45	6.24	0.31	13.23	2.52	10.2%	100.0%	100.0
ME-mag	0.85	8.26	0.52	16.17	1.36	4.0%	39.4%	78.1
ME-nmg	0.15	1.55	0.17	12.42	3.22	6.2%	60.6%	21.9
ME-head	0.43	4.19	0.31	13.89	2.49	10.2%	100.0%	100.0

Table 14b: XRD of BMG fractions of the 45 x 106 μ m feed, with ankerite distribution

Magnetic	Quartz	Albite	Calcite	Ankerite	Hematite	T.Ms%	P.Ms%	Ank_D
Longi-mag	7.6	25.5	26.0	14.0	17.3	2.2%	21.6%	34.8
Longi-nmg	9.3	43.8	36.9	7.3	0.0	8.0%	78.4%	65.2
Longi-head	8.9	39.8	34.5	8.7	3.7	10.2%	100.0%	100.0
ME-mag	7.7	23.4	39.2	16.3	3.6	4.0%	39.4%	89.1
ME-nmg	9.9	54.3	33.6	1.3	0.0	6.2%	60.6%	10.9
ME-head	9.0	42.1	35.8	7.2	1.4	10.2%	100.0%	100.0

Technology evaluation on 45 x 106 μ m

Two gravity and three magnetic technologies were evaluated. Figure 6 illustrates five lines with the TREO recovery-yield (left) and recovery-grade (middle) and upgrade ratio-yield (right) showing medium response. The upgrade was slightly better compared to Figure 5 due to more liberation (Figure 2c). This fraction represents a smaller portion of feed mass (10%). Table 14c shows the standard technology parameter table.

The shaking table performed reasonably well (70% TREO recovery at 50% mass yield) since the feed material was in the optimal size range for this technology and xenotime was reasonably liberated. The MGS performed better at 72% TREO recovery in 40% of the concentrate mass (4 of the most promising test of 23 tests were plotted).

The Carpc dry magnetic fractionation (CRP) showed that a high magnetic (hematite as identified by XRD) could be isolated with relatively low TREO recovery loss to high mag fraction. It should be noted that the Carpc fractions were plotted with decreasing TREO grade and not decreasing magnetic field strength since the TREO is concentrated in the middling fractions of the Carpc.

The standard Mineral Technologies (MT) WHIMS (WHM) unit was operated at its highest gauss setting and could not extract further magnetic material (15% mass yield) from the non-magnetic fraction. The fine matrix (1mm instead of 2mm) enabled higher mass pull (50%) to the magnetic fraction which resulted in the 2nd set of data points. The overall recovery was still low (63% TREO recovery in 48% of the mass). The Gaustec WHIMS (GTC) showed higher selectivity and could upgrade the TREO to above 1.3% at 45% mass yield to magnetic fraction.

The belt magnet (BMG) evaluation with the Longi belt representing the lower yield points (below 20%) and ME belt representing the higher yield points (39%) in Figure 6, showed the highest TREO recovery potential. This technology could be used as a pre-concentration step to the higher-grade technologies like MGS.

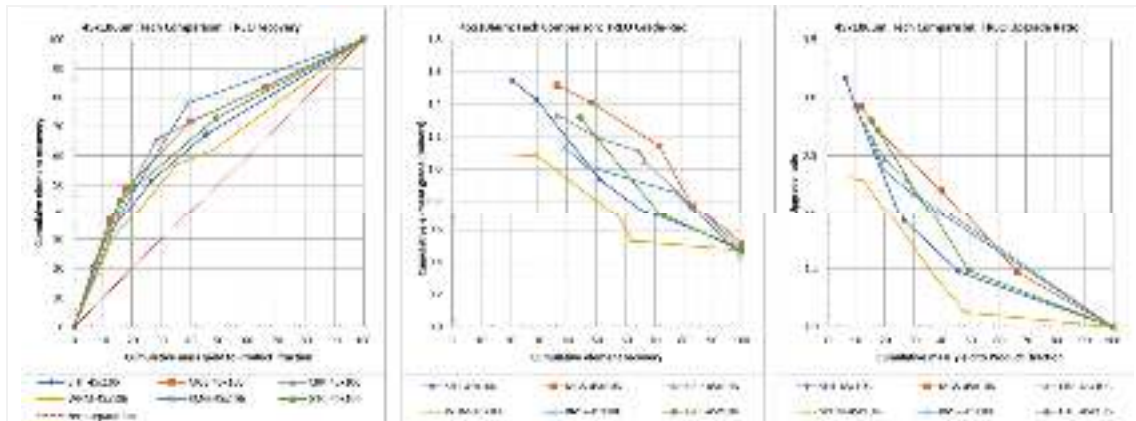


Figure 6: TREO recovery-yield, grade-recovery and upgrade-ratio-yield comparison per technology on size fraction 45x106 μm .

Table 14c: Technology review on size fraction 45x106 μm

Size Fraction (micron)	Particle attribute	Technology	Main test parameters	Potential for improvement	Current performance rating
45x106	Gravity	Shaking Table	Angle, waterflow, feedrate	Medium	Poor
45x106	Gravity	Multi gravity separator	RPM, waterflow, feedrate, angle	Medium	Promising
45x106	Magnetic	Dry fractions (multiple gauss)	Magnet strength	Low	Poor
45x106	Magnetic	WHIMS	Feedrate, magnet strength	Low	Poor
45x106	Magnetic	Dry rare-earth magnet	Speed, feedrate, magnet strength	Medium	Not tested
45x106	Magnetic	WHIMS – fine matrix	Feedrate, magnet strength	Medium	Poor
45x106	Magnetic	Minimag - fine matrix	Feedrate, magnet strength	Medium	Promising
45x106	Magnetic	Wet rare earth belt	Belt speed, feedrate, waterflow	Medium	Promising
45x106	Magnetic	Wet rare earth belt - high Gauss	Belt speed, feedrate, waterflow	Medium	Not tested

10. SEPARATION OF FRACTION 5: 0 x 45 µm

The 0 x 45 µm fraction represents 22% of the feed (Table 4a) with 24% of the TREO and contained mostly (>60%) well liberated xenotime particles. This feed material was subjected to shaking table gravity fractionation with the aim to concentrate xenotime. Table 15a illustrates that the TREO was upgraded from 0.7 to 1.6 at 2% TREO recovery. The bulk of the TREO was distributed to the lower gravity fractions (midd2, midd3 and tail). It is suspected that the shaking table is unable to concentrate these ultrafine particles although they are well liberated.

Table 15a: XRF on shaking table fractions of the 0 x 45 µm feed, with TREO distribution

Density	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
conc	1.64	24.37	0.51	10.16	1.98	0.2%	0.8%	1.9
midd1	0.91	9.15	0.39	13.50	2.30	0.7%	2.9%	3.9
midd2	0.80	6.72	0.38	14.04	2.08	3.4%	15.2%	17.6
midd3	0.68	5.25	0.36	13.69	1.89	8.2%	36.5%	35.8
tail	0.63	5.01	0.34	13.98	2.20	10.0%	44.5%	40.7
Hd 0 x 45	0.69	5.69	0.35	14.74	2.93	22.4%	100.0%	100.0

Table 15b: XRD of shaking table fractions of the 0 x 45 µm feed, with ankerite distribution

Density	Quartz	Albite	Calcite	Ankerite	Hematite	T.Ms%	P.Ms%	Ank_D
conc	6.7	26.2	26.1	6.2	25.1	0.2%	0.8%	1.0
midd1	8.3	31.7	39.3	5.8	8.1	0.7%	2.9%	3.3
midd2	9.5	34.0	42.4	7.1	2.7	3.4%	15.2%	20.6
midd3	8.4	34.1	41.6	6.2	2.3	8.2%	36.5%	42.9
tail	7.9	35.5	44.8	3.8	0.0	10.0%	44.5%	32.2
Hd 0 x 45	8.3	34.6	42.9	5.3	1.7	22.4%	100.0%	100.0

The MGS on the 0 x 45 µm produced improved results compared to the shaking table and TREO could be upgraded to above 4.9% from 0.7% in feed. Table 16 illustrates selected tests from the MGS. The TREO recovery of above 60% could only be achieved at higher mass yield to concentrate above 40%.

Table 16: XRF on MGS gravity fractions of the 0 x 45 µm feed, with TREO distribution

Density	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
T1-conc	4.89	32.03	1.61	6.14	1.01	1.3%	6.0%	42.0
T1-tail	0.43	4.32	0.31	14.91	3.20	21.1%	94.0%	58.0
T1-head	0.70	5.98	0.39	14.39	3.06	22.4%	100.0%	100.0
T2-conc	2.88	17.25	0.91	11.85	1.65	3.0%	13.4%	52.8
T2-tail	0.40	4.23	0.27	14.40	2.41	19.4%	86.6%	47.2
T2-head	0.73	5.97	0.36	14.06	2.31	22.4%	100.0%	100.0
T3-conc	0.86	5.45	0.40	14.81	2.79	9.7%	43.4%	60.9
T3-tail	0.42	5.02	0.28	13.88	2.57	12.7%	56.6%	39.1
T3-head	0.61	5.21	0.33	14.28	2.67	22.4%	100.0%	100.0

The fine matrix (1 mm) from the MT WHIMS and Gaustec Mini-mag unit was evaluated on this fine material fraction to determine the potential of the horizontal ring design. Table 17 illustrates selected tests from these two units. The separation response was similar with 50% TREO recovery in 39% of the mass. It is suspected that the drag force of the vertical wash water misplaces the fine magnetic particles causing the low TREO recovery to magnetic fraction.

Table 17: XRF on WHIMS fractions of the 0 x 45 µm feed, with TREO distribution

Magnetic	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
mag (1mm)	0.73	5.40	0.35	14.07	2.59	8.7%	38.9%	49.0
nmg (1mm)	0.48	3.96	0.35	14.27	2.74	13.7%	61.1%	51.0
WHIMS hd	0.58	4.52	0.35	14.20	2.68	22.4%	100.0%	100.0
GTC-mag	2.16	15.68	0.50	10.87	1.62	1.8%	8.2%	28.7
GTC-mid	0.48	3.96	0.32	14.47	2.25	7.0%	31.4%	24.3
GTC-nmg	0.48	3.93	0.33	14.62	2.49	13.5%	60.4%	47.1
GTC hd	0.62	4.91	0.34	14.26	2.35	22.4%	100.0%	100.0

Selected wet magnetic separation results by means of the two BMG units (Longi and ME) are illustrated in Table 18. This unit favours fine mineral magnetic recovery since the wash water is not vertical but at an angle to allow better retention of the fine magnetic materials. The 70% TREO recovery at 42% of the mass is a promising result with the Malvern Engineering (ME) BMG producing the lowest TREO tail grade (0.3%) from all units tested. This is a similar outcome on the 45 x 106 µm fraction.

Table 18: XRF on BMG fractions of the 0 x 45 µm feed, with TREO distribution

Magnetic	%TREO	%Fe	%Ti	%Ca	%Si	T.Ms%	P.Ms%	TREO_D
Longi-mag	2.94	23.93	0.57	8.51	1.75	1.8%	8.2%	36.7
Longi-nmg	0.45	3.93	0.36	14.44	2.83	20.6%	91.8%	63.3
Longi-head	0.66	5.57	0.38	13.95	2.74	22.4%	100.0%	100.0
Longi-mag	1.84	14.01	0.51	11.71	2.12	3.8%	17.1%	47.0
Longi-nmg	0.43	3.83	0.33	14.54	2.36	18.6%	82.9%	53.0
Longi-head	0.67	5.57	0.36	14.06	2.32	22.4%	100.0%	100.0
ME-mag	1.00	6.91	0.47	15.83	3.88	9.3%	41.5%	69.1
ME-nmg	0.32	3.62	0.28	15.41	4.20	13.1%	58.5%	30.9
ME-head	0.60	4.98	0.36	15.58	4.07	22.4%	100.0%	100.0
ME-mag	0.87	5.99	0.42	14.32	2.14	10.4%	46.3%	70.9
ME-nmg	0.31	3.51	0.25	13.90	2.64	12.0%	53.7%	29.1
ME-head	0.57	4.66	0.33	14.09	2.41	22.4%	100.0%	100.0

Technology evaluation on 0 x 45 µm

Figure 7 illustrates the comparison of the five technologies tested on the 0 x 45 µm fraction. The shaking table had a poor response with low TREO grade and recovery to concentrate. The MGS was able to produce high TREO grades but the recovery as below 55%. A second round of optimisation tests to obtain higher recovery produced some data points above 60% recovery but at significantly high mass yields resulting in lower grades.

The two horizontal ring WHIMS units (MT and Gaustec) evaluated showed limited promise and it is suspected that the vertical slurry flow is not effective for the TREO recovery of fine particles (below 30 µm). The BMG units (Longi and ME) that were evaluated showed promise with more than 70% TREO recovery in 45% of the mass. The Longi unit was not able to produce higher mass yields than 20% to magnetic fraction due to lower Gauss permanent magnets. The ME unit fitted with the higher Gauss magnets produced the most promising results. Table 19 summarises the technologies and what has been evaluated to date. It is suspected that a pulsating type of

magnetic separation unit, like the SLon could also be effective in recovering TREO from this fine mineral fraction. Due to the fineness of the particles in this fraction no dry separation can be considered.

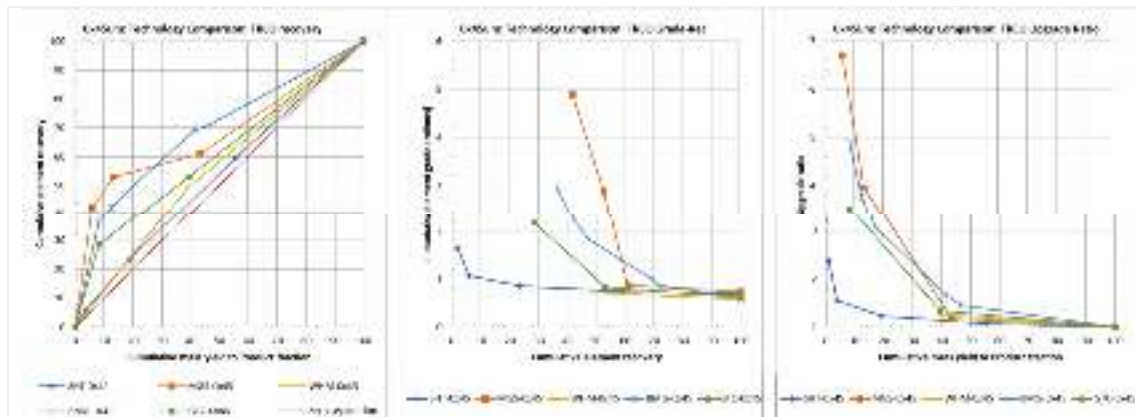


Figure 7: TREO recovery-yield, grade-recovery and upgrade-ratio-yield comparison per technology on size fraction 0 x 45 µm.

Table 19: Technology review on size fraction 0x45 µm

Size Fraction (micron)	Particle attribute	Technology	Main test parameters	Potential for improvement	Current performance rating
0x45	Gravity	Shaking Table	Angle, waterflow, feedrate	Low	Poor
0x45	Gravity	Multi gravity separator	RPM, waterflow, feedrate, angle	Medium	Promising
0x45	Magnetic	Wet rare earth belt	Belt speed, feedrate, waterflow	Medium	Promising
0x45	Magnetic	Wet rare earth belt - high Gauss	Belt speed, feedrate, waterflow	Medium	Promising
0x45	Magnetic	WHIMS – fine matrix	Feedrate, magnet strength	Medium	Poor
0x45	Magnetic	Minimag - fine matrix	Feedrate, magnet strength	Medium	Poor
0x45	Magnetic	Pulsating magsep	Feedrate, magnet strength	Medium	Not tested (recommended)

11. CONCLUSIONS

- DMS to isolate ankerite: The DMS separation was unable to isolate a high-density fraction rich in ankerite with low levels of xenotime. The high-density fraction contained high percentages of TREO.
- DMS to reject calcite: The DMS separation showed promise to reject mass (30% mass) at low TREO recovery loss (20%) through a density cut point of 2.5 g/cm³.
- Xenotime liberation: The Qemscan indicated that the xenotime is reasonably (>60%) liberated below 45 µm.
- Coarse gravity separation to isolate ankerite: The three coarser fractions (106 x 212 µm, 212 x 500 µm and 500 x 1000 µm) contained up to 6% ankerite. The ankerite could not be isolated to a high-density fraction without the significant sacrifice of TREO recovery. Ankerite concentrated predominantly to the middling fractions.
 - 500 x 1000 µm fraction: poor upgrade response in this size fraction.
 - 212 x 500 µm fraction: poor upgrade response in this size fraction.
 - 106 x 212 µm fraction: slightly improved upgrade response as compared to coarser fractions. Magnetic separation (WHIMS) and gravity separation (shaking table) achieved between 30 to 40 % TREO recovery at a ~20% mass pull and 2 x upgrade ratio.
- 45 x 106 µm fraction – gravity separation: The MGS showed the most promise from a gravity separation point of view with 70% TREO recovery at 40% concentrate yield.
- 45 x 106 µm fraction – magnetic separation: The Malvern Engineering Belt magnet (BMG) showed the most promise for TREO recovery (>75%) in 40% of the mass.
- 0 x 45 µm fraction – gravity separation: The MGS was able to produce high TREO grades (>5%) but recovery greater than 60% was challenging on the unit.
- 0 x 45 µm fraction – magnetic separation: The Malvern Engineering Belt magnet (BMG) showed the most promise for TREO recovery (>70%) in 45% of the mass.

12. RECOMMENDATIONS

- Dry magnetic separation on 1 x 8 mm: The dry magnetic separation of this fraction on 11 kG rare-earth roll (Debtech) unit should be considered as an alternative to DMS to reject calcite and pre-concentrate this size fraction before further size reduction.
- Magnetic separation: This technology seems to be the most effective to upgrade the xenotime, but high field strengths will be required to influence the ultrafine particle. Initial potential was demonstrated with the belt magnet (BMG). More work is required on other fine wet high intensity magnetic separation devices and alternative fine matrices (SLon, Gaustec alternative matrices, other models of BMG). Reasonable throughputs are important on a rougher step in which TREO recovery (>70%) can be achieved.
- 500 x 1000, 212 x 500 and 106 x 212 µm fractions: 1) Crush finer for improved liberation and subsequent treatment with the finer fraction. 2) Evaluate dry rare earth magnetic separation, although dry processing at these size ranges may not be feasible in an operating environment.
- Magnetic separation stages: The BMG testwork showed potential and rougher, scavenger and cleaner runs are to be considered in the next stage of testwork.
- Liberation size: From this work xenotime liberation only occurs below 45 µm and then only 60% of the xenotime particles are fully liberated. A large portion of the xenotime should be recoverable if high TREO grades are not pursued (less than 3% TREO).
- MGS gravity separation: This technology can be well applied on a rougher concentrate after 30-50% of the mass is rejected with magnetic separation.
- 0x45 and 45 x 106 µm fractions: Treat <106 µm material as a combined fraction using 1) rare earth belt magnet and 2) MGS gravity separation.
- Flotation: This technology was not considered in this study and comparative study on this same material fraction (0 x 45 µm) is required.



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Date

Appendix E:

HLS and Bond Testing on Lofdal Samples report from Geolabs Global Pty Ltd

REPORT: HLS and Bond Testing on Lofdal Samples

Report ID:	2021-03-10_NAM1_HLS and Bond Testing on Lofdal Samples
Date Issued:	2021-03-10
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1. INTRODUCTION

Namibian Critical Metals commissioned Geolabs Global to conduct the following tests:

- Bond Comminution testing: To evaluate the sample's resistance to crushing and milling.
- Heavy liquid separation (HLS): Amenability of rare earth minerals to upgrade, by means of density separation.

Approximately 238 kg of ROM material, 344 kg of -20 mm fines and 2318 kg ore sorting products were delivered to Geolabs for the tests. The material underwent preparation to form three samples for the following scope:

- 'ROM no sorting' - CWi, Ai, BBWi and BRWi
- 'Sorted Products and fines' - BBWi and BRWi
- '-20 mm ROM fines' - HLS

2. BRIEF METHODOLOGY

2.1 Sample Preparation

'ROM no sorting' sample :

Separate samples were submitted for Bond Low Energy Impact testing (CWi) and Bond Abrasion Index (Ai) testing. These sample did not require any additional preparation.

The 'ROM no sorting' sample material was screened to determine the particle size distribution, as received. The screened fractions were recombined, and crushed in a 8 x 6-inch jaw crusher and cone crusher to -12.50 mm and -3.35 mm for the Bond Rod Work Index and Bond Ball Work Index tests, respectively.

'Sorted Products and fines' sample:

The 'sorted products' sample was reconstituted for comminution testing from sorted products, under the client's instructions. The composition of the reconstituted sample is contained in Table 1:

Table 1: Composition of 'Sorted Products and fines' sample

Sample ID	Mass (kg)
Coarse Conc 4	20.0
Fines Conc 4	8.0
-20 mm Fines	30.5

'-20 mm ROM fines' sample:

The '-20 mm ROM fines' sample was crushed to -10 mm using a 8 x 6 jaw crusher. The material was subsequently screened at 1 mm to generate the -10+1 mm fraction for the HLS test.

2.2 Particle Size Distribution

A PSD analysis was conducted on the 'ROM no sorting' sample as received. The following screen sizes were used:

Size
mm
63
45
31.5
22.4
16.5
12.5
9.5
6.7
4.75
3.35
2.36

The material was screened on a gyratory sieve shaker down to 9.5 mm. The -9.5 mm material was screened using a Fritsch Analysette 3 Spartan 200 mm sieve shaker down to 2.36 mm. The masses retained as oversize on each screen was weighed and recorded.

2.3 Bond Ball Work Index

The Bond ball test was designed by Fred C. Bond as a standardised test to measure a sample's resistance to ball milling. It is expressed as a power draw in kWh/t, also known as the Bond Ball Work Index (BBWi).

The BBWi test provides a Work Index (kWh/t) that is widely used to estimate the energy required for grinding. The Work index is defined as the energy (kWh) required for grinding a short ton of material from a notional infinite size to a product size of 80% passing 100 µm. The expression relates the work input to the work index as follows (also known as Bond's Third Law of Comminution):

$$E = 10W_i \left(\frac{1}{\sqrt{P_{80}}} - \frac{1}{\sqrt{F_{80}}} \right)$$

Where E = specific energy (kWh/t)

W_i = work index, which is an ore-specific constant (kWh/t)

P_{80} = size at which 80% of the product passes (µm)

F_{80} = size at which 80% of the feed passes (µm)

The bulk density of the feed material was determined using a one litre measuring cylinder and placing a representative known mass of material (M , in kg) inside the cylinder and shaken for 5-10 minutes on a vibratory shaker. The settled volume (V , in litre) of the sample was recorded and the bulk density (ρ_b) was then calculated by:

$$\rho_b = \frac{M}{V}$$

The feed mass (M_f) for the test was then calculated for a 700 ml sample:

$$M_f = 0.7 \times \rho_b$$

The feed mass was then split out from a bulk sample using a rotary splitter and following a calculated splitting route. The remainder of the sample was split into 10 aliquots which were used for topping up the feed material after every cycle during the test.



Figure 1: Bond Ball Work Index tester at Geolabs Global

The 700 ml feed material was then placed inside the mill and milled for an arbitrary number of revolutions (usually 100 to 150). After milling, the entire mill was discharged onto a grate which allows the material to pass through with ease while not allowing the ball charge to pass. The milled material was then evenly distributed across four 106 μm screens and screened for 20 minutes; on a Fritsch Analysette 3 Spartan, shown in figure 3.



Figure 2: Fritsch Analysette 3 Spartan at Geolabs Global

After 20 minutes of vibration, the samples were removed, and the test sieves cleaned to remove all near size particles to prevent potentially blinding the screen. The oversize material was then replaced into the sieves for an additional 10 minutes of vibration.

The mass of the oversize material was recorded, and the net grams of product produced per revolution G_{bp} was calculated by:

$$G_{bp} = \frac{p_f - p_i}{70t}$$

Where

p_f : product mass after material has been milled, in grams

p_i : product mass initially present in material before being milled, in grams.

t : residence time of material being milled, in minutes

The residence time required for the next cycle is calculated based on the product present in the feed material (p), the target circulation load and the net grams of product produced per revolution for the previous cycle (G_{bp}). For the test, a circulation load of 250% is targeted, therefore:

$$t = \left(\frac{M_f}{3.5} - p \right) / (70G_{bp})$$

The oversize material was then topped up to the feed mass (M_f) using the before mentioned aliquots. The new feed was then placed inside the Bond Ball test mill and milled for a residence time of t . This process was then repeated for at least 7 cycles, until a $250 \pm 5\%$ circulation load was reached for at least three consecutive cycles, and the net grams of product produced per revolution did not vary by more than 3% over the last three cycles.

The average net grams product produced per revolution was calculated from the last three cycles:

$$G_{bp} = \frac{1}{3} \sum_{i=0}^2 G_{bp,i}$$

With G_{bp} , F_{80} and P_{80} known, the Bond Work Index was calculated using the empirical equation:

$$W_i = \frac{44.5}{P_{100}^{0.23} G_{bp}^{0.82} \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right)} kWh/st$$

With P_{100} : Closing screen size used in μm

G_{bp} : net grams of product produced per revolution inside the ball mill in g/rev

P_{80} : the size which 80% of the product (at P_{100}) passes

F_{80} : the size which 80% of the feed passes.

2.4 Bond Rod Work Index

The Bond rod test was designed by Fred C. Bond as a standardised test to measure a sample's resistance to rod milling. It is expressed as a power draw in kWh/t, also known as the Bond Rod Work Index (BRWi).

The bulk density of the feed material was determined using a one-litre measuring cylinder by placing a representative known mass of material (M , in kg) inside the cylinder and shaken for 5-10 minutes on a vibratory shaker. The settled volume (V , in litre) of the sample was recorded and the bulk density (ρ_b) was then calculated by:

$$\rho_b = \frac{M}{V}$$

The feed mass (M_f) for the test was then calculated for a 1 250 ml sample:

$$M_f = 1.25 \times \rho_b$$

The feed mass was split from a bulk sample using a rotary divider. The remainder of the sample was screened at 10 mm and 3.35 mm, creating three fractions: +10 mm, -10+3.35 mm and -3.35 mm. These fractions were used for topping up the feed material after every cycle during the test.



Figure 3: Bond Rod Work Index tester at Geolabs Global

The 1250 ml feed material was then placed inside the mill and milled for an arbitrary number of revolutions (usually 50). After milling, the ground material was distributed evenly across three 1180 μm screens and screened for 15 minutes on a Fritsch Analysette 3 Spartan.

After 15 minutes of vibration, the samples were removed, and the test sieves cleaned. The mass of the oversize material was recorded, and the net grams of product produced per revolution G_{bp} calculated by:

$$G_{bp} = \frac{p_f - p_i}{46t}$$

Where

p_f : product mass after material has been milled, in grams

p_i : product mass initially present in material before being milled, in grams.

t : residence time of material being milled, in minutes

The residence time required for the next cycle is calculated based on the product present in the feed material (p), the target circulation load and the net grams of product produced per revolution for the previous cycle (G_{bp}). For the test, a circulation load of 100% is targeted, therefore:

$$t = \left(\frac{M_f}{2} - p \right) / (46G_{bp})$$

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The oversize material was topped up to the feed mass (M_f) using the screened fractions mentioned before. The new feed was then placed inside the Bond rod test mill and milled for a residence time of t . This process was then repeated for at least 7 cycles, until a $100 \pm 5\%$ circulation load was reached for at least three consecutive cycles, and the net grams of product produced per revolution did not vary by more than 3% for the last three cycles.

The average net grams product produced per revolution was calculated from the last three cycles:

$$G_{bp} = \frac{1}{3} \sum_{i=0}^2 G_{bp,i}$$

With G_{bp} , F_{80} and P_{80} known, the Bond Work Index was calculated using the empirical equation:

$$W_i = \frac{62}{P_{100}^{0.23} G_{bp}^{0.625} \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right)} kWh/st$$

With P_{100} : Closing screen size used in μm

G_{bp} : net grams of product produced per revolution inside the ball mill in g/rev

P_{80} : the size which 80% of the product (at P_{100}) passes

F_{80} : the size which 80% of the feed passes.

2.5 Bond Abrasion Index

The Bond Abrasion test was designed to determine the steel media and liner wear in rod mills, ball mills and crushers. The test consists of a 7.62 cm x 2.54 cm x 0.64 cm alloy steel paddle, that has been hardened to 500 Brinell, which is subjected to wear from a given sample.

The paddle is weighed and placed 2.54 cm deep into a rotor with a diameter of 11.43 cm. A 400g -19+12.7 mm sample is placed in the steel drum, which has dimensions of 11.43 cm x 30.54 cm (WxD). The rotor is rotated at 632 rpm while the steel drum at 70 rpm, in the same direction, for 15 minutes. After 15 minutes, the 400 g sample is removed and replaced with another 400g sample. This process is repeated four times in total, such that the paddle has been subjected to wear from a total of 1600 g of sample over a period of one hour.



Figure 4: Bond Abrasion Index tester at Geolabs Global

After four 400g batches have been tested, the paddle is removed and the mass recorded. The mass loss that occurred from the paddle (in grams) is the Abrasion Index.

The balance used to weigh the Abrasion test paddles is SANAS accredited and weighs in grams up to the fourth decimal place at 0.0001g intervals.



Figure 5 Vibra 0.0001g increment balance

2.6 Bond Low Energy Impact test

Fred C. Bond designed the Bond Low Energy Impact test to enable the sizing of crushers with data generated from a simple laboratory test. The test machine consists of two 13.6 kg hammers, each mounted on a pendulum allowing the hammers to swing freely.



Figure 6: Bond Low Energy Impact Test at Geolabs Global

A rock specimen is placed in a position within the tester where the two hammers would meet simultaneously after being released by electromagnets. The rock specimen is subjected to a known energy, calculated from the angle at which the 13.60 kg hammers are released, and is inspected afterwards for any breakage. If the specimen has not broken, the angle at which the hammers are released is increased incrementally, until the specimen is broken. The energy at which the particle breaks is known as the Impact crushing strength (a) in N (newtons):

$$a = \frac{2 \times m \times g \times h}{c}$$

Where m = mass of each hammer (kg)

g = gravity constant 9.81 m/s^2

h = height, in metres, related to the angle at which the specimen fractured

c = smallest dimension of the specimen in metres.

The impact crushing strength is then related to crushability work index (CWi) through the empirical formula:

$$CWi = \frac{0.0485 \times a}{d}$$

Where a = impact strength in Newton (N)

d = particle density in t/m^3

2.7 Heavy Liquid Separation

Heavy liquid separation was conducted on the -10+1.0 mm fraction using lithium heteropolytungstate (LST) and tetrabromoethane (TBE). A cascaded format was used during the HLS test, with descending liquid densities. The floats from each density were carried to the next density, and the sinks were taken as products. The only exception to this was the 2.8 g/cm³ sinks which was carried over 3.0 g/cm³ because of the change in dense media used. The following liquid densities were used: 3.0, 2.90, 2.80, 2.75, 2.70, 2.65, 2.60, 2.55 and 2.50 g/cm³. Figure 7 shows the flow diagram for the cascaded format used in this HLS program:

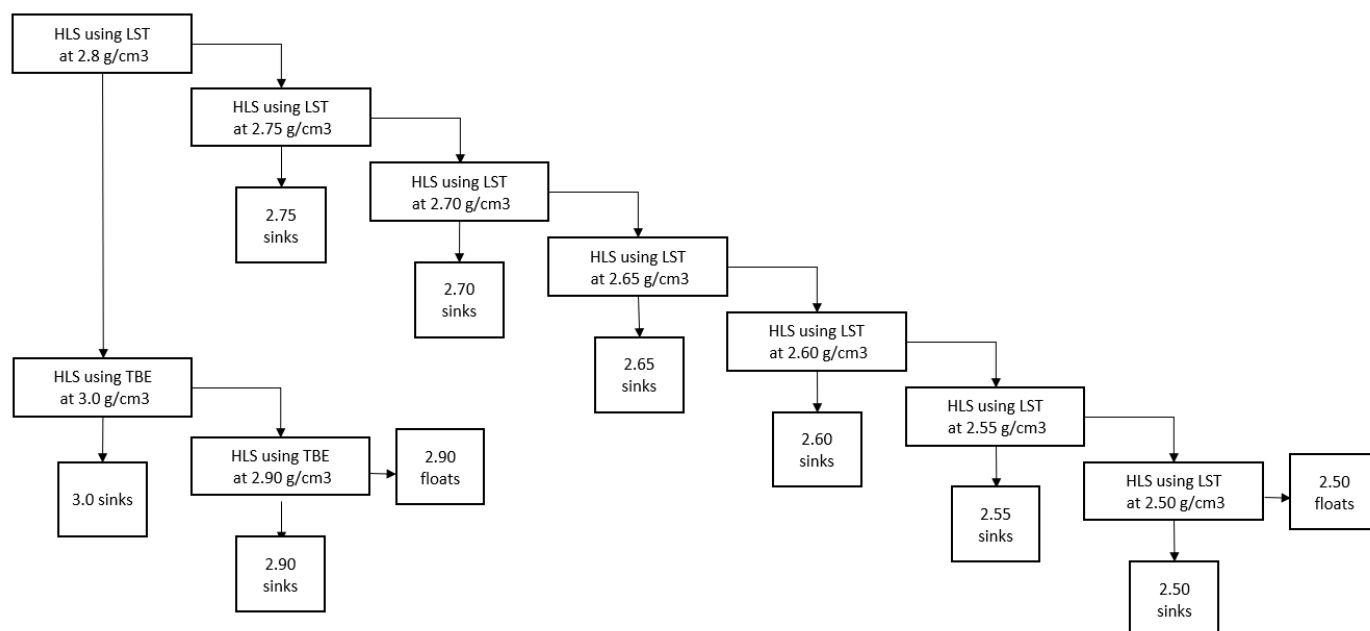


Figure 7: Diagram of the cascaded HLS format

All HLS products and -1.0 mm fraction were rotary split and pulverized. Chemical assays and x-ray diffraction analysis (XRD) were conducted on these pulps.

2.8 Chemical Assays

The chemical assays were carried out by ALS in Moderfontein, using scheme ME-MS61r. Results for the chemical assays are contained in the accompanying ALS results certificate (ALS ref: JB22036671). A total of 12 samples were analyzed, which included all HLS products and the -1 mm undersize fraction.

2.9 X-ray Diffraction Analysis

X-ray diffraction analyses were performed in PANalytical Aeris diffractometer with a PIXcel detector with fixed slits and Co-K α radiation. All XRD samples were prepared using the backloading preparation method. Minerals were identified and quantified using SiroQuant 5.0. The phase name might not give the actual composition of the minerals but instead the mineral group. Due to factors such as crystallite size and preferred orientation effects, results may be affected. Traces of additional phases may also be present. Amorphous phases, which could be present, were not considered during the quantification. A total of 12 samples were analysed by XRD, which included all HLS products and the -1 mm undersize fractions..

3. RESULTS

This section summarises the results obtained in the scope requested. Detailed results can be viewed in the accompanying sheet '2022-03-04_NAM1_Data.xlsx'

3.1 Particle Size Distribution

Table 2 contains the particle size distribution of the 'ROM no sorting' sample as received:

Table 2: PSD of 'ROM no sorting' Sample

Size	Mass Retained	Mass Retained	Cumulative Passing
mm	kg	%	%
63	145.5	61.10	38.90
45	21.6	9.07	29.83
31.5	13.7	5.75	24.08
22.4	9.55	4.01	20.07
16.5	9.55	4.01	16.06
12.5	5.9	2.48	13.58
9.5	6.15	2.58	11.00
6.7	6.55	2.75	8.25
4.75	4.75	1.99	6.26
3.35	1.65	0.69	5.56
2.36	3.3	1.39	4.18
PAN	9.95	4.18	0.00
Total	238.15	100.00	-

A graph of the PSD for the 'ROM no sorting' sample is shown in Figure 8:

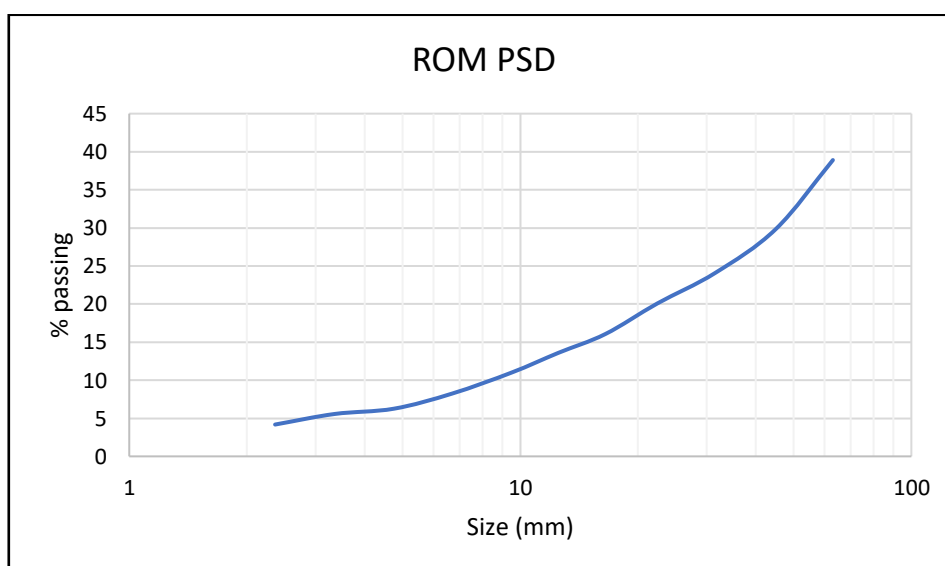


Figure 8: PSD of 'ROM no sorting' sample as received

3.2 Bond Ball Work Index

The Bond Ball Work Index ranged between 14.2 kWh/t to 16.7 kWh/t across various limiting screen sizes tested. The material exhibited an average to moderately high resistance to milling at these target sizes.

	Sample ID	BWi (kWh/t)	P ₁₀₀ (µm)	F ₈₀ (µm)	P ₈₀ (µm)	Gbp (g/rev)
1	ROM no sorting	16.7	53	2 134	39	0.82
2		16.2	75	2 134	60	1.05
3		15.7	106	2 133	82	1.25
4	Sorted Product & Fines	15.7	53	2 230	39	0.89
5		14.6	75	2 230	55	1.12
6		14.2	106	2 230	77	1.35

3.3 Bond Rod Work Index

The Bond Rod Work Index for both samples tested was similar at 14.5 kWh/t for 'ROM no sorting' and 14.7 kWh/t for 'Sorted Product & Fines'.

	Sample ID	BWi (kWh/t)	P ₁₀₀ (µm)	F ₈₀ (µm)	P ₈₀ (µm)	Gbp (g/rev)
1	ROM no sorting	14.5	1 180	10 223	938	9.48
2	Sorted Product & Fines	14.7	1 180	7 536	893	9.78

3.4 Bond Abrasion Index

The Bond Abrasion Index was measured at 0.1408 g for the 'ROM no sorting' sample. This sample has a low abrasion index relative to other ores.

Bond Abrasion Index test results							
Sample ID	Sample 1 (g)	Sample 2 (g)	Sample 3 (g)	Sample 4 (g)	Paddle Before test (g)	Paddle After test (g)	Ai (g)
ROM no sorting	400.89	400.82	400.62	400.94	96.1596	96.0188	0.1408

3.5 Bond Low Energy Impact Test

The Bond Low Energy Impact test results summary is shown in Table 3. The test yielded an average Crushability Work Index (CWi) of 7.9 kWh/t with a maximum of 11.5 kWh/t across 17 specimens tested. This is considered to be low CWi to crushing when compared to other ore.

Table 3: BLEI test results summary

Sample ID	Average CWi kWh/t	Maximum CWi kWh/t	SG
ROM no sorting	7.9 ± 1.9	11.5	2.66

3.6 Heavy Liquid Separation

Results for the HLS products and the -1 mm fraction are summarised in this report focusing on TREO (Total Rare Earth Oxides).

Mass yields, grades and recovery to the sinks from the HLS test are shown in Table 4.

Table 4: HLS results of the 'ROM -20mm fines'

Fraction	Density g/cm ³	Mass Sinks g	Mass Yield %	Discrete		Cumulative	
				TREO %	Recovery %	TREO %	Recovery %
-10+1mm	3.00	161.50	2.83	0.362	12.22	0.362	12.22
	2.90	53.62	0.94	0.238	2.67	0.332	14.89
	2.80	91.48	1.60	0.148	2.84	0.277	17.73
	2.75	220.06	3.85	0.185	8.51	0.239	26.24
	2.70	207.93	3.64	0.145	6.29	0.212	32.53
	2.65	864.60	15.14	0.068	12.37	0.134	44.90
	2.60	2483.73	43.50	0.040	20.52	0.077	65.42
	2.55	391.32	6.85	0.038	3.12	0.073	68.54
	2.50	84.08	1.47	0.040	0.70	0.073	69.24
	<2.50	41.14	0.72	0.051	0.44	0.073	69.67
-1 mm	-	1110.21	19.44	0.131	30.33	0.084	100.00
Total		5709.67	100.00	0.084	-	-	-

Figure 9 shows the grade-recovery curve of the TREO to the HLS sinks products vs. mass cumulative mass pull to sinks. At a density of 3.0 g/cm³ a mass pull of 2.83% yielded a TREO grade of 0.362% at a recovery of 12.22%.

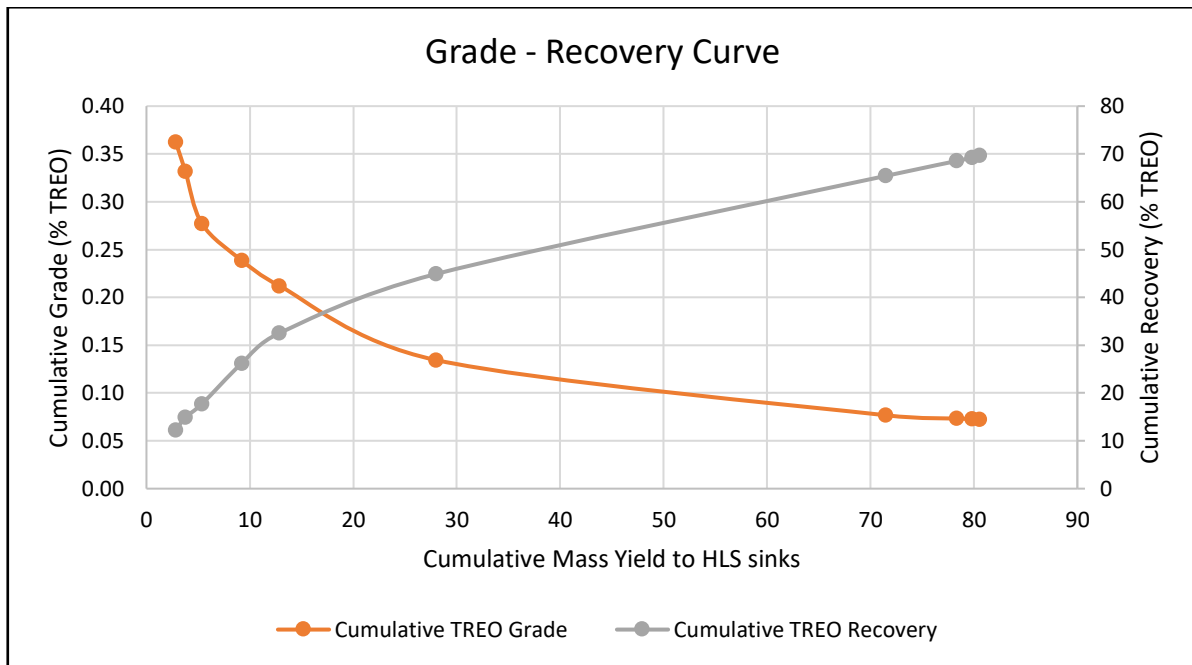


Figure 9: Grade - Recovery curve of TREO to HLS sinks of 'ROM -20mm fines'

In Table 5 , the data of the graph in Figure 9 is shown:

Table 5: TREO grade - Recovery versus Mass Pull

Cumulative Mass Pull %	Upgrade Ratio	Cumulative TREO GradeCumulative %	Cumulative TREO Recovery %
2.83	4.32	0.362	12.22
3.77	3.95	0.332	14.89
5.37	3.30	0.277	17.73
9.22	2.84	0.239	26.24
12.87	2.53	0.212	32.53
28.01	1.60	0.134	44.90
71.51	0.91	0.077	65.42
78.36	0.87	0.073	68.54
79.84	0.87	0.073	69.24
80.56	0.86	0.073	69.67

3.7 Chemical Assays

Table 6 shows the measured chemical composition for the HLS products and the -1 mm fraction. In the 3.0 g/cm³ and 2.9 g/cm³ sinks products, the Ce values were above the detection limit for ME-MS61r method of 500 ppm. Therefore, Ce (highlighted in red) was calculated using a Ce/La ratio of 1.8766 under the client's recommendation.

Table 6: Chemical Analyses of HLS products (ME-MS61r)

Fraction	Density	Ag	Al	As	Ba	Be	Bi	Ca	Cd	Ce	Co	Cr	Cs	Cu	Fe	Ga
	g/cm ³	ppm	%	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm
-10+1mm	3.00	0.02	2.54	80.8	210	8.87	7.46	3.3	0.21	1109	307	1065	0.69	40.2	32.1	20.6
	2.90	0.01	2.99	26.8	380	4.97	3.09	11.95	0.19	614	87.3	58	1.54	27.9	10.05	17.25
	2.80	<0.01	5.67	20.1	630	5.59	1.2	10.35	0.14	374	48	85	1.54	21.2	6.99	25.2
	2.75	<0.01	6.23	23.4	560	6	2.33	7.92	0.24	500	77.6	63	1.44	25.8	7.69	27.1
	2.70	<0.01	7.19	15	510	5.37	1.38	7.02	0.28	399	49.6	72	1.22	17.5	5.03	28.5
	2.65	<0.01	7.14	9	360	4.17	0.72	5.57	0.21	179.5	32.4	80	0.63	12.1	3.02	25.3
	2.60	<0.01	7.02	6.1	180	3.08	0.41	3.88	0.18	101.5	22.6	77	0.37	9.1	1.55	20.6
	2.55	<0.01	7.27	7.4	140	3	0.62	3.58	0.26	97.3	35.1	79	0.36	12.4	1.73	20.2
	2.50	<0.01	7.82	9.2	170	3.11	0.91	1.64	0.32	102	31.7	84	0.38	15	1.9	20.9
	<2.50	<0.01	7.5	11	190	3.11	1.03	2.07	0.47	133.5	50.8	131	0.5	15.4	2.16	22.2
-1 mm	-	<0.01	7.54	17	540	6	1.79	4.97	0.08	392	113	114	1.09	24.8	6.41	29.5
Head		<0.01	7.33	12.1	320	3.95	0.98	4.96	0.04	261	56.3	100	0.61	14.7	4.18	23.6

Fraction	Density	Ge	Hf	In	K	La	Mg	Mn	Mo	Na	Nb	Ni	P	Pb	Rb	Re
	g/cm ³	ppm	ppm	ppm	%	ppm	%	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm
-10+1mm	3.00	1.38	10.6	0.134	0.74	590	0.92	1415	83.9	0.62	176.5	59.8	1580	73.4	42.1	0.099
	2.90	0.72	8.1	0.152	1.44	327	5.12	2640	26.5	0.62	136	35.3	1320	37.6	86.5	0.066
	2.80	0.44	8.2	0.171	2.57	201	3.99	2110	18.3	1.16	145	29.3	970	26.9	130.5	0.051
	2.75	0.59	9	0.157	2.25	286	1.97	1750	19.45	1.83	146	28.5	1010	54.3	116.5	0.089
	2.70	0.47	7.1	0.116	2.07	216	1.36	1415	14.65	2.81	127	21.6	890	53.5	102	0.059
	2.65	0.25	5.8	0.08	1.5	97.4	0.61	954	10.05	3.96	88.1	15.7	510	41.1	63.5	0.037
	2.60	0.22	4.5	0.053	0.72	56.2	0.21	494	8.08	5.1	54.3	10.2	310	32	32.3	0.033
	2.55	0.21	4.2	0.046	0.52	56	0.21	501	7.66	5.49	49.4	12.3	310	49.2	21.2	0.044
	2.50	0.24	5.2	0.042	0.49	48.4	0.24	679	7.84	5.99	53.6	17.7	300	78.4	20.5	0.061
	<2.50	0.31	5.1	0.057	0.36	67.4	0.25	879	12.2	5.84	100.5	20.4	340	101.5	14.6	0.03
-1 mm	-	0.56	7.5	0.103	1.78	198.5	0.81	1305	15.9	3.09	159	29.8	750	17.2	85.7	0.011
Head		0.37	5.8	0.067	1.22	138.5	0.67	921	13.25	4.22	113.5	18.3	610	8.4	55.7	0.005

Fraction	Density	S	Sb	Sc	Se	Sn	Sr	Ta	Te	Th	Ti	Tl	U	V	Y	Zn
	g/cm ³	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm
-10+1mm	3.00	0.06	1.92	25.6	10	8.9	446	0.49	0.43	620	0.231	1.77	25.6	404	334	298
	2.90	0.02	0.49	31.6	2	5.9	423	0.66	0.18	323	0.247	1.32	13.3	179	344	202
	2.80	0.02	0.3	34.4	2	8.4	301	1.17	0.1	227	0.381	1.26	9.9	192	235	148
	2.75	0.03	0.41	30.3	2	7.9	239	1.23	0.15	271	0.347	1.82	13.5	177	229	320
	2.70	0.02	0.36	25.6	<1	8.3	181.5	1.78	0.1	184.5	0.371	1.26	9.3	137	190	239
	2.65	0.01	0.18	15.4	<1	5.8	136	1.74	0.06	89.6	0.278	0.82	5.7	88	92.8	195
	2.60	0.01	0.14	8.3	<1	3.6	108.5	1.2	<0.05	53.8	0.207	0.6	4	49	54.5	150
	2.55	0.01	0.15	8.7	1	3.7	107	0.96	<0.05	54.4	0.23	0.8	3.8	48	54.2	274
	2.50	0.02	0.41	10.3	<1	4.1	104	0.9	<0.05	89.6	0.284	1.16	5.3	54	69.8	399
	<2.50	0.01	0.21	11.5	1	5	102.5	0.94	0.05	87.1	0.397	0.6	6	86	81	155
-1 mm	-	0.02	0.35	23.1	1	7.4	227	2.45	0.12	231	0.359	0.28	9.2	163	140.5	53
Head		0.01	0.22	15.2	2	5	166	1.54	0.1	137	0.277	0.15	6.5	107	103	15

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Fraction	Density	Zr	Dy	Er	Eu	Gd	Ho	Lu	Nd	Pr	Sm	Tb	Tm	Yb
	g/cm ³	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
-10+1mm	3.00	>500	82.7	39.9	30.7	100	14.5	5.6	466	141	103.5	14.55	5.7	38.4
	2.90	>500	85.4	43	23.9	88.2	15.05	5.76	259	75.5	72.6	14.05	6.07	41.1
	2.80	>500	57.5	29.7	14.95	55.3	10.6	3.93	144.5	42.5	41.5	9.96	4.08	28
	2.75	>500	57.9	29.8	17.75	63.5	10.4	4.01	209	62.3	54.9	10.05	4.27	28.6
	2.70	464	44.6	23.2	13.45	48.5	7.93	2.89	160	46.5	41.9	7.77	3.08	20.7
	2.65	324	23.7	12	6.77	25	4.32	1.56	76.7	21.8	20.7	4.08	1.66	11.2
	2.60	240	13.6	7.57	3.76	13.7	2.6	0.99	45.3	12.85	11.65	2.22	1.05	6.93
	2.55	220	13.15	6.93	3.79	13.6	2.44	0.91	41.5	11.75	11.15	2.23	0.97	6.6
	2.50	289	16.85	9.66	3.44	13.75	3.23	1.32	35.7	10.5	9.61	2.56	1.37	9.39
	<2.50	282	18.6	11.1	4.29	16.3	3.64	1.56	48.1	14.35	12.7	2.9	1.62	10.9
-1 mm	-	409	36.5	17.65	12.3	42.7	6.35	2.18	152	43.9	38.6	6.66	2.41	16.25
Head		331	25.6	13.25	8.33	29.1	4.72	1.59	106	30.5	26.6	4.88	1.77	11.6

Note: Ce reported was above detection limit of 500 ppm. Ce is estimated as 1.8766 x La (as per client's request)

3.8 X-ray Diffraction Analysis

The HLS products and -1.0 mm fraction were analysed by XRD, which were validated against the assays. In these products, REE-bearing minerals were below detection limit for XRD and therefore excluded. At a density of 3.0 g/cm³, the sinks contained 43.8% hematite, while the fraction between 3.0 g/cm³ and 2.90 g/cm³ mostly contained dolomite.

Table 7: Assay-validated quantitative XRD mineralogy for the Head sample

Fraction	Density g/cm3	Quartz %	Albite %	Muscovite %	Calcite %	Chlorite %	Hematite %	Dolomite %	Goethite %	Actinolite %	Smectite %
-10+1mm	3.00	26.7	7.5	8.1	3.4	0.0	43.8	8.1	2.3	0.0	0.0
	2.90	17.6	7.3	12.3	7.7	0.0	8.9	40.1	6.1	0.0	0.0
	2.80	12.8	13.7	22.8	9.5	0.0	6.4	28.6	3.5	2.6	0.0
	2.75	18.8	21.7	20.2	13.5	2.4	9.0	9.0	1.1	4.2	0.0
	2.70	17.4	33.2	18.3	14.9	5.2	5.2	2.8	0.8	2.2	0.0
	2.65	19.8	46.6	12.7	12.7	3.6	2.9	0.4	0.7	0.5	0.0
	2.60	21.5	60.3	6.2	8.5	0.9	1.9	0.2	0.1	0.5	0.0
	2.55	19.5	64.8	4.5	7.5	0.8	2.2	0.3	0.1	0.5	0.0
	2.50	18.5	70.7	4.2	2.4	1.0	2.3	0.2	0.1	0.6	0.0
	<2.50	19.7	69.0	3.1	3.6	1.2	2.6	0.1	0.2	0.6	0.0
-1 mm	-	17.1	36.3	16.6	9.7	1.6	8.1	2.9	0.6	1.8	5.2
Head		19.2	49.2	10.5	9.6	1.6	5.0	2.3	0.5	1.0	1.0

4. CONCLUSION

The Lofdal samples tested exhibited a medium to moderately high resistance to grinding. The 'ROM no sorting' sample's BBWi ranged from 15.7 kWh/t to 16.7 kWh/t using limiting screen sizes from 106 µm down to 53 µm. For 'Sorted products and fines' sample, the BBWi ranged from 14.2 kWh/t to 15.7 kWh/t using limiting screen sizes from 106 µm down to 53 µm. The BRWi for both samples measured in a similar range to the BBWi, at 14.5 kWh/t and 14.7 kWh/t for 'ROM no sorting' and 'Sorted products and fines', respectively.

The 'ROM no sorting' showed lower resistance to crushing and lower abrasive characteristics than other ores.

Through HLS, upgrade ratios up to 4.32 were achieved. The highest TREO grade was attained in the 3.0 g/cm³ sinks, yielding a TREO grade of 0.362% at a recovery of 12.22%.

****END OF REPORT****

Appendix F:

An Investigation into Hydrometallurgical Recovery of Rare Earth Elements from the Lofdal Deposit
prepared for Namibia Critical Metals and dated January 20, 2022 from SGS Canada Inc.



An Investigation into
THE HYDROMETALLURGICAL RECOVERY OF RARE EARTH ELEMENTS FROM THE
LOFDAL DEPOSIT

prepared for

NAMIBIA CRITICAL METALS INC.

Project 18299-02 – Final Report
January 20, 2022

NOTES

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ACCREDITATION: SGS Minerals Lakefield is accredited to the requirements of ISO/IEC 17025 for specific tests as listed on our scope of accreditation, including geochemical, mineralogical, and trade mineral tests. To view a list of the accredited methods, please visit the following website and search SGS Lakefield: <http://palcan.scc.ca/SpecsSearch/GLSearchForm.do>.

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Executive Summary

Two flotation samples produced from the Lofdal Deposit were tested separately to determine the potential for producing a marketable rare earth product with minimal impurities.

The assay results of the two received flotation concentrates were dissimilar as shown in **Error! Reference source not found.** Assayed rare earth elements were 4-5 times more prevalent in the second concentrate while aluminum and calcium were reduced by approximately half.

Table I: Flotation Concentrate Assays

Sample ID	Date Rec'd	La, g/t	Ce, g/t	Pr, g/t	Nd, g/t	Sm, g/t	Eu, g/t	Gd, g/t	Tb, g/t	Dy, g/t
Conc 1	24-Mar	426	683	<256	257	<431	--	--	145	1160
Conc 2	14-May	1950	3980	338	1330	725	435	2380	729	4860
		Ho, g/t	Y, g/t	Er, g/t	Tm, g/t	Yb, g/t	Lu, g/t	Sc, g/t	Th, g/t	U, g/t
Conc 1	24-Mar	--	9056	--	--	--	--	--	1494	<85
Conc 2	14-May	1150	38900	3560	494	3100	455	38	8240	154
		Si, %	Al, %	Fe, %	Mg, %	Ca, %	Na, %	K, %	Ti, %	P, %
Conc 1	24-Mar	12.0	2.99	18.7	1.00	12.4	1.12	0.81	0.935	0.450
Conc 2	14-May	10.0	1.66	22.8	0.25	6.58	0.979	0.15	3.84	2.03
		Mn, %	Cr, %	V, %	S, %	Cl, g/t				
Conc 1	24-Mar	0.24	0.01	0.04	0.03	--				
Conc 2	14-May	0.39	0.03	0.10	0.11	62				

This test program included:

- Consideration of gangue acid leaching with HCl to minimize calcium-bearing minerals in the feed to any subsequent process step (either two cycles of caustic cracking and sulphuric acid leaching, or sulphuric acid baking and water washing). Gangue acid leaching was rejected after reviewing initial results.
- Two cycles of caustic cracking and sulphuric acid leaching of gangue acid leach residue with the expectation that the rare earth elements would primarily report to the second cycle acid leach liquor. This was not observed in the test program and this process was similarly rejected.
- Sulphuric acid baking followed by a water leaching, impurity removal with magnesium carbonate, rare earth precipitation with sodium carbonate, sulphuric acid re-leaching, uranium ion exchange with Purolite A660, thorium solvent extraction, and final rare earth precipitation and calcination.

The acid baking flowsheet produced a final purified heavy rare earth oxide product containing 98% TREO with low levels of impurities (381 g/t Mo, 105 g/t Bi, 100 g/t Cr, 100 g/t Ni, 1.1 g/t Th and <0.3% Na, Mg, Si, and Ca). A detailed analysis of the final calcined product is presented in Table II. Terbium and dysprosium were highlighted as elements of primary interest and were tracked through the test program after the receipt of the second flotation concentrate (Conc 2, May 14, 2021).

Table II: Final Calcined Product (C-RP3) Assay Summary

Sample & Quant.	RP-3 Precip	C-RP3 Calcine
(mL or g)	22	10
La	11002	23800
Ce	19924	43100
Pr	2094	4530
Nd	7489	16200
Sm	3467	7500
Eu	2140	4630
Gd	13961	30200
Tb	3587	7760
Dy	28985	62700
Ho	6703	14500
Y	226056	489000
Er	21404	46300
Tm	2922	6320
Yb	17243	37300
Lu	2325	5030
Sc	<12	<25
Th	0.51	1.1
U	<0.2	<0.5
Si	878	1900
Al	<23	<50
Fe	32	70
Mg	1387	3000
Ca	740	1600
Na	1433	3100
K	<37	<80
Ti	<28	<60
P	18	40
Mn	<37	<80
Zn	20	44
S (%)		0.01
TREE (%)	36.9	79.9

Add'n Elements (g/t)	
Ag	<50
As	<200
Ba	15
Be	0.90
Bi	105
Cd	<3
Co	<200
Cr	100
Li	<30
Mo	381
Nb	
Ni	99
Pb	<200
Sb	35
Se	<50
Sn	40
Sr	16.7
Ta	
Tl	<50
V	<60
Zr	
LOI (%)	0.49

The conceptual flowsheet is shown in Figure I below. High leach extractions of 97% Tb/Dy were accomplished and though some losses of REE were encountered in the subsequent batch liquor treatment steps, these are expected to be minimum during continuous operations.

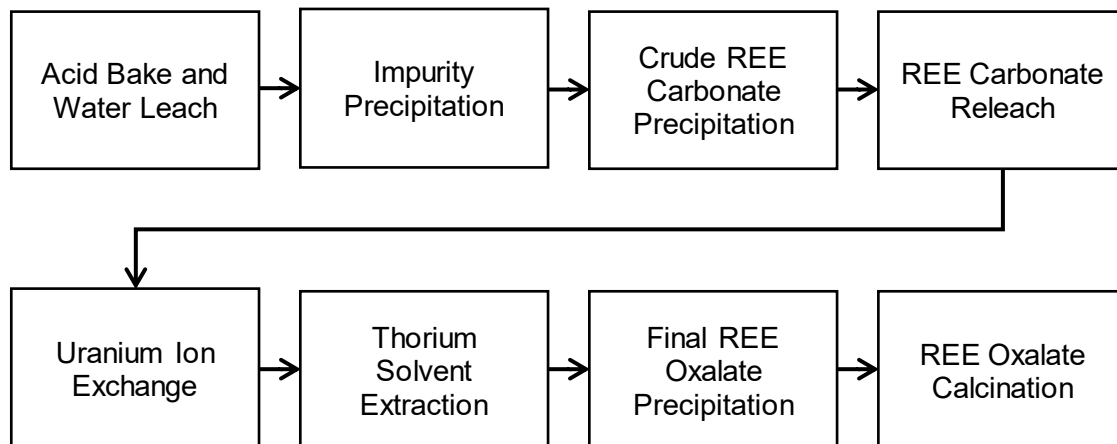


Figure I: Conceptual Flowsheet for the Recovery of REE from Flotation Concentrate

While gangue acid leaching (GAL) was successful in removing calcium from the flotation concentrate with hydrochloric acid, rare earth losses were high, with 22-56% of the neodymium dissolved at acidities needed to remove 91-96% of the calcium. This residue would move on to sulphuric acid baking. Ultimately, the addition of a second acid reagent (at 207 kg/t HCl) and associated handling system, an additional solid-liquid separation step, and the reactors required for this process outweighed the benefit of gangue removal in advance of acid baking and this process step was rejected. This was further evidenced when comparing REE dissolution and reagent requirements in the sulphuric acid bake testwork where REE recoveries were fairly similar whether the GAL step was included or not.

Two-cycle caustic cracking, followed by water leaching and hydrochloric acid leaching was not successful in liberating the rare earth elements, allowing them to be solubilized in the acid leach step. Overall, 26% of the yttrium was dissolved in the acid leach steps, approximately evenly split between each. This process was therefore eliminated from further consideration.

Twelve sulphuric acid bake (AB) tests were completed: three using the initial flotation concentrate as feed, two using a blend of gangue acid leach residues as feed, and seven using the second flotation concentrate as feed (including the bulk acid bake test, AB12). Acid baking of Conc 1 resulted in the recovery of 91% of the yttrium (used as a proxy element for all REE in initial testing). Acid baking of the gangue acid leach residue resulted in 96% yttrium recovery, though this was understood to only include 45-80% of the REE originally in the flotation concentrate due to Nd losses in the gangue acid leaching (GAL) step. Partial optimization of the acid bake and water leach process provided conditions for the bulk acid bake test (1250 kg/t sulphuric acid addition, 300°C, three hours, with 20% solids (w/w) followed by water leaching at ambient temperature for two hours). Total rare earth recovery in the bulk acid bake test was 97% with a leach liquor containing 17.2 g/L TREE and 39.1 g/L iron.

A combined acid bake with water leach liquor from initial tests (Conc 1 and GAL residue as feed) was used to investigate reagents and pH targets for impurity removal (IR) testing. These results were confirmed with a similar leach solution generated from Conc 2 flotation concentrate before a bulk IR test was completed. The bulk IR test precipitated 94% of the iron and thorium and 40% of the aluminum with the addition of 381 kg/t magnesium carbonate, which was needed to maintain pH 2.9 for one hour at 50°C. In the bulk test, 11% of the neodymium and 9% of the yttrium co-precipitated, but it is expected that these co-precipitation values can be reduced through additional optimization.

The filtrate from bulk impurity removal (IR-5) was delivered for crude rare earth precipitation testing with bulk test (RP-2) target of pH 6.50, maintained for one hour at ambient temperature, using sodium carbonate. This process resulted in the precipitation of 100% TREE, along with most remaining impurities (Sc, Th, U, Al, Fe, Ti), and required 90.5 kg/t sodium carbonate.

The crude rare earth precipitate was re-leached using conditions selected based on previous experience on similar materials. The re-leach test consisted of repulping most of the precipitate and acidifying to pH 1. This was maintained for three hours before using more of the precipitate to increase to pH 3.5. The leach required 48 kg/t sulphuric acid and successfully redissolved 99% of the REE while partially rejecting thorium, aluminum, and iron. This liquor contained 28 g/L TREE, 23.3 mg/L U, and 1.98 mg/L Th.

To remove the residual uranium and thorium, the re-leach liquor was contacted sequentially with Purolite A660 for two contacts and then an organic mixture of 0.5% Primene JMT, 2.5% isodecanol in Aromatic 150ND. These reagents and contact ratios were not optimized, but were selected based on previous test programs removing uranium and thorium from a concentrated rare earth liquor. As a result, 99.9% of the uranium was removed in the first IX contact while 15% of the REE was captured across the two contacts. Ninety-four percent of the thorium was also collected in the ion exchange contacts while a further 75% was removed in the solvent extraction contact. Rare earth losses at this stage should be minimized with proper optimization and under continuous operation, employing crowding or scrubbing of co-extracted REE. After these two process steps, 21.4 g/L TREE remained with few remaining impurities.

Two final rare earth precipitation tests were completed using this liquor, with each precipitate being subjected to a high-temperature calcination step. Oxalic acid precipitation and calcination produced a final solid containing 98.1% total rare earth oxides (TREO, direct assay) representing 94% of the rare earth elements present in the feed liquor for this test. Sodium carbonate precipitation and calcination produced a final solid containing 95.0% TREO (direct assay) representing 94% of the feed liquor rare earth elements.

These observations and measurements are valid for a hydrometallurgical feed sample that is largely similar to that tested but would not necessarily apply for flotation concentrates with different quantities of TREE and impurities. Any changes to the feed will require confirmatory and/or optimization testwork to produce reliable results.

Introduction

SGS Canada Inc. in Lakefield, Ontario was requested to conduct an investigation into the recovery of rare earth elements (especially terbium and dysprosium) from flotation concentrates generated from samples originating from the Lofdal Deposit in Namibia by Ms. Bernadine Ballington of SGS South Africa, on behalf of Namibia Critical Metals Inc. (NCM, formerly Namibia Rare Earths Inc.).

The program outlined in this report included concentrate receipt and characterization, acid bake with water leaching, gangue acid pre-leaching, caustic cracking with water wash and acid leaching, acidic leach liquor impurity removal, crude rare earth precipitation and re-leaching, uranium removal with ion exchange, thorium removal with solvent extraction, and comparative final rare earth precipitation and calcination testing.

The testwork was initially based on results provided by NCM followed by efforts to optimize the rare earth element (REE) dissolution, impurity removal, and crude REE precipitation steps. Once a crude rare earth precipitate was formed, there was insufficient material for optimization and test conditions were selected based on SGS Canada experience with similar flowsheets.

Feed preparation was reported in *Scoping Metallurgical Testwork on Samples from the Lofdal Heavy Rare Earth Project* (July 23, 2021, SGS Project Number 18299-01).

Throughout the test program, test proposals and results were shared with Ms. Barbara Mulcahy and Mr. Rainer Ellmies for review and comment.



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Testwork Summary

Two unique process paths were identified prior to the receipt of flotation concentrate – an elevated temperature acid bake process (with or without pre-leaching to remove gangue minerals) and a two-stage mixable caustic crack with acid leach process wherein the rare earth elements (REE) would remain primarily in the solid phase in the first stage and would be adequately accessible to acid solubilization after the second caustic crack. In an effort to streamline testwork, many decisions were made based on yttrium behaviour before all assay results became available.

Two flotation concentrates generated over the course of the hydrometallurgical test program, were provided on March 24 (“Longi Combined Mags” or “Conc 1”) and May 14, 2021 (“Conc 2”). Available assays for these two concentrates are shown in Table 1. The second concentrate contained around five times higher concentration of REE, and improved rejection of most impurities. Test data sheets can be found in Appendix A.

Table 1: Flotation Concentrate Assay Summary

Sample ID	Date Rec'd	La, g/t	Ce, g/t	Pr, g/t	Nd, g/t	Sm, g/t	Eu, g/t	Gd, g/t	Tb, g/t	Dy, g/t
Conc 1	24-Mar	426	683	< 256	257	<431	--	--	145	1160
Conc 2	14-May	1950	3980	338	1330	725	435	2380	729	4860
		Ho, g/t	Y, g/t	Er, g/t	Tm, g/t	Yb, g/t	Lu, g/t	Sc, g/t	Th, g/t	U, g/t
Conc 1	24-Mar	--	9056	--	--	--	--	--	1494	< 85
Conc 2	14-May	1150	38900	3560	494	3100	455	38	8240	154
		Si, %	Al, %	Fe, %	Mg, %	Ca, %	Na, %	K, %	Ti, %	P, %
Conc 1	24-Mar	12.0	2.99	18.7	1.00	12.4	1.12	0.81	0.935	0.450
Conc 2	14-May	10.0	1.66	22.8	0.25	6.58	0.979	0.15	3.84	2.03
		Mn, %	Cr, %	V, %	Ag, g/t	As, g/t	Ba, g/t	Be, g/t	Bi, g/t	Cd, g/t
Conc 1	24-Mar	0.24	0.01	0.04	< 2	< 40	405	7.35	< 20	< 2
Conc 2	14-May	0.39	0.03	0.10	--	--	--	--	--	--
		Co, g/t	Cu, g/t	Li, g/t	Mo, g/t	Ni, g/t	Pb, g/t	Sb, g/t	Se, g/t	Sn, g/t
Conc 1	24-Mar	118	42	27	< 40	< 70	< 30	< 50	< 30	< 20
Conc 2	14-May	--	--	--	--	--	--	--	--	--
		Sr, g/t	Tl, g/t	Zn, g/t	S, %	Cl, g/t				
Conc 1	24-Mar	179	< 30	38	0.03	--				
Conc 2	14-May	--	--	--	0.11	62				

1. Gangue Acid Pre-leach (GAL) Testwork

1.1. GAL Test Procedure

Three gangue acid pre-leach (GAL) tests were completed early in the test program to investigate the removal of gangue minerals through the addition of hydrochloric acid. The initial pulp density (prior to acid addition) for each test was 50% (w/w) solids and the slurry was heated to 50°C before adding reagent. Intermediate solution samples and final wash liquors were analyzed for yttrium, neodymium, calcium, and iron. Final solution samples were also analyzed by ICP-AES for a full scan and by ICP-MS for a REE scan. Final residues were analyzed for a whole rock analysis (WRA) and REE by X-Ray Fluorescence (XRF). The REE by XRF scan only provided data on a subset of REE (La, Ce, Pr, Nd, Sm, Y, Th, and U).

Test GAL1 was a titration-style test in which acid was added to achieve a series of five pH or free acidity targets sequentially with solution samples taken after 30 min of stability at each target. This test indicated that calcium dissolution was immediate and complete (approximately 99%) at the highest pH tested (pH 3.0) while iron dissolution remained near zero regardless of acid addition (pH 3.0 to 25 g/L HCl). Meanwhile, the tracked REE (Nd and Y) dissolution increased with increasing acidity to 56% and 1% at the conclusion of the test, respectively. The GAL1 final washed residue was supplied to caustic crack testwork.

Test GAL2 applied this result and targeted pH 3.0 for a duration of two hours, including kinetic liquor sampling at 30 min and one hour. These samples indicated that there was no benefit to extending the residence time beyond one hour, though final calcium dissolution was slightly lower than GAL1 at 91%.

Test GAL3 investigated two pH points – pH 5.0 (at the recommendation of the Namibia Critical Metals team) and pH 1.5 in search of lower REE loss. Calcium dissolution at pH 5.0 was quite low (estimated 15%), improving to 96% at pH 1.5. Neodymium loss was 33%. The final residues from GAL2 and GAL3 were blended before being advanced to acid bake testing.

1.2. GAL Test Results

The series of three GAL tests indicated that calcium-bearing gangue minerals could be removed through the addition of hydrochloric acid to achieve less than pH 3.0 (ideally pH 1.5) when starting at 50% (w/w) solids and maintaining 50°C for at least 30 min. Increasing the acidity of the slurry had no effect on iron dissolution; however, REE (exemplified by yttrium and neodymium) dissolution increased with increasing acidity, resulting in up to 56% solubilized (Nd at 25 g/L HCl). These REE losses can be minimized through targeting a higher pH, but still represented a 22% Nd loss at the lowest acidity target of pH 3.0. Acid addition ranged from 207 kg/t to 270 kg/t (100% reagent and dry flotation concentrate basis).

Partial metallurgical balances for the gangue acid pre-leach tests can be found in Table 2 through Table 4, and solution tenors of select elements from all three tests have been charted in Figure 1.

Table 2: GAL1 Summary Metallurgical Balance

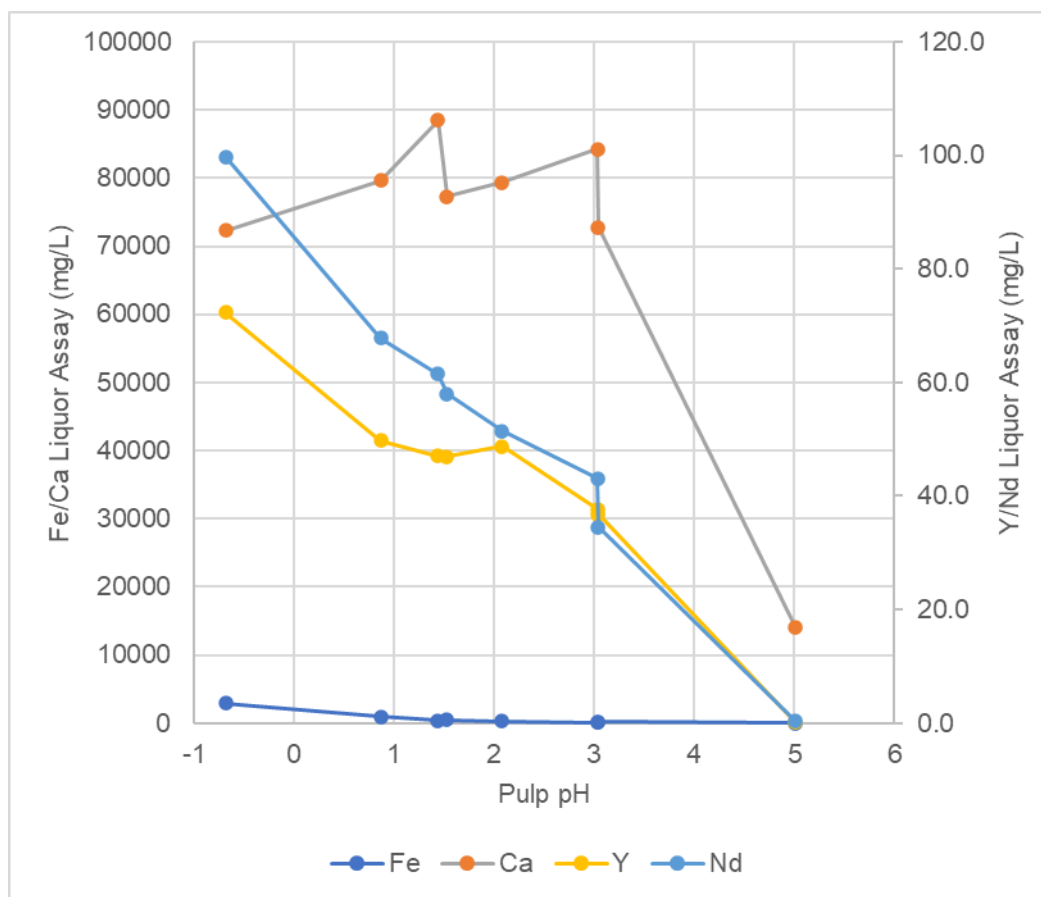
Sample & Quant.	Assay Units	Longi Combined Mags	pH 3 Filtrate	pH 2 Filtrate	pH 1.5 Filtrate	pH 1 Filtrate	25 g/L Filtrate	Wash	25 g/L Residue	Leaching	Account. out/in	Calc Head
(mL or g)		500	23	35	30	32	579	1215	335	%	%	
Nd	mg/L, g/t	257	34.5	51.4	58	67.8	99.7	8.1	<171	56	102	263
Y	mg/L, g/t	9056	36.8	48.7	46.9	49.8	72.3	5.94	13386	1	100.3	9080
Fe	mg/L, %	18.7	164	241	442	903	2920	312	28.7	2	105	19.6
Ca	mg/L, %	12.4	72800	79400	77300	79600	72300	6340	0.2	99	96	11.9

Table 3: GAL2 Summary Metallurgical Balance

Sample & Quant.	Assay Units	Longi Combined Mags	30 min Filtrate	60 min Filtrate	Final Filtrate	Wash	Final Residue	Leaching	Account. out/in	Calc Head
(mL or g)		500	40	33	555	1315	353	%	%	
Nd	mg/L, g/t	257	42.7	45.2	43.1	2.14	300	22	106	272
Y	mg/L, g/t	9056	38.1	40.6	37.7	1.98	12363	1	97	8790
Fe	mg/L, %	18.7	89.3	99.6	129	4.7	26.4	0	100	18.7
Ca	mg/L, %	12.4	85500	85300	84200	5980	1.60	91	108	13.3

Table 4: GAL3 Summary Metallurgical Balance

Sample & Quant.	Assay Units	Longi Combined Mags	pH 5 Filtrate	pH 1.5 Filtrate	Wash	pH 1.5 Residue	Leaching	Account. out/in	Calc Head
(mL or g)		500	33	637	1584	346	%	%	
Nd	mg/L, g/t	257	0.29	61.5	3.23	257	33	104	267
Y	mg/L, g/t	9056	0.1	47.1	3.29	12442	1	96	8678
Fe	mg/L, %	18.7	0.4	421	19.8	27.1	0	100	18.8
Ca	mg/L, %	12.4	14000	88500	5180	0.69	96	109	13.5


Figure 1: GAL Solution Tenors at Various pH

While these results support the use of hydrochloric acid for the removal of calcium-bearing gangue minerals in advance of other process steps, the added complexity of the overall process and the potential loss of REE units likely outweighs the apparent benefits. This will depend on the overall process selected: the calcium would be expected to be leached in the hydrochloric acid leaching stage of a caustic crack with water wash and acid leach process; calcium would remain in the solids as calcium sulphate in a sulphuric acid bake and water leach process. Testing of these two process routes (described in the next sections) demonstrated that the benefits of gangue acid leaching did not justify its inclusion in the overall flowsheet.

2. Caustic Crack with Water Wash and Acid Leach (CC and AL) Testwork

2.1. CC and AL Test Procedure

One full sequence of caustic crack with water wash and acid leaching was completed consisting of a caustic crack with water wash (CC) using GAL1 residue as feed, an acid leach (AL) on the resultant residue, a second CC using the AL residue as feed, and finally a second AL test. This sequence was selected to determine the potential of a two-stage caustic process for the recovery of REE. The first caustic crack test (CC-1) was known to dissolve some impurities and crack some of the xenotime mineralization allowing for the first acid leach test to dissolve further impurities and some REE. It was expected that the residue would contain most of the REE, which would then have to be further processed. The intention of the second CC and AL tests was to further crack the mineralization and solubilize the REE in the final acidic liquor. In practice, any REE dissolved in the first acid leach step would be precipitated and blended with the feed to the second AL step.

The feed for each caustic crack test was mixed with 50% sodium hydroxide at a ratio of two tonnes NaOH per tonne of test feed (100% reagent and dry feed basis). Once combined, the slurry was heated to 140°C and mixed for the target time before deionized water (DI) was added to reduce the sodium hydroxide concentration to 15%. This diluted slurry was mixed for a further two hours at 90°C before filtration on a glass fibre filter. These conditions are summarized in Table 5. Final solution samples (filtrate and wash) were analyzed for a full ICP-AES scan, REE scan by ICP-MS, and chloride content. The final residues were analyzed for WRA and REE by XRF plus chlorine content.

Table 5: Caustic Crack and Water Wash Conditions Summary

Test ID	Caustic Crack					Water Wash		
	Feed	Lixiviant	Dosage, kg/t	Temperature, °C	Reaction Time, h	NaOH Target	Temperature, °C	Reaction Time, h
CC-1	GAL-1 Res	50% NaOH	2000	140	3	15% NaOH	90	2
CC-2	CC 1 AL-1 Res	50% NaOH	2000	140	3	15% NaOH	90	2

Both acid leaches were titration-style tests in which acid was added to sequentially achieve a series of three pH or free acidity targets, with solution samples taken after one hour of stability at each target. The residue from the preceding caustic crack test was mixed with DI water to achieve a target slurry solids content of

15% (w/w) before heating to 50°C. Concentrated hydrochloric acid was then added to achieve each target before sampling. These conditions are summarized in Table 6.

Table 6: Acid Leach Conditions Summary

Test ID	Feed	Initial % solids	Acid Addition Targets			Temperature, °C
CC 1 AL-1	CC-1 Res	15	pH 3	25 g/L	50 g/L	50
CC 2 AL-1	CC-2 Res	15	pH 1	25 g/L	50 g/L	50

2.2. CC and AL Test Results

The first caustic crack with water leach test (CC-1) resulted in the dissolution of less than half of any tracked impurity in the caustic stage. The caustic crack portion of the test only ran for two hours (instead of the target three hours) before dilution for the water wash. The acid leach test (CC 1 AL-1 at 50 g/L HCl) mostly dissolved calcium (68%) and thorium (77%) that had carried over in the gangue acid leach (GAL1) residue used as feed. In addition, 14% of the yttrium reported to the filtrates.

The second caustic crack/water leach test (CC-2) resulted in the dissolution of less than half of any tracked impurity in the caustic stage, while the acid leach stage (CC 2 AL-1 at 50 g/L HCl) mostly dissolved thorium (64%). Only 18% of the yttrium reported to the filtrates in this test.

Taken as a whole, this process route did not achieve suitable cracking of the mineralization to allow for the acidic dissolution of REE. This process was therefore not investigated any further.

Partial metallurgical balances for the caustic crack/water leach tests can be found in Table 7 and Table 9 while the acid leach test balances are in Table 8 and Table 10.

Table 7: CC-1 Summary Metallurgical Balance

Element	Units	Feed GAL1	NaOH	WW PLS	Wash	Final res	Extraction calc head	Accountability out/in	Calc Head	Extraction based on Res/Fd
Quant (mL/g)		277	1108	3011	3018	234.1	%	%		%
		Assay (mg/L, %, g/t)								
La	mg/L, g/t	171		0.29	0.69	341	3.6	175	1000	
Ce	mg/L, g/t	<171		0.41	1.29	512	4.1	264	1500	
Pr	mg/L, g/t	<256		0.06	0.14	<256	1.0	86	710	
Nd	mg/L, g/t	<171		0.19	0.54	171	5.2	89	500	
Sm	mg/L, g/t	<431		<0.04	0.06	<431	0.3	85	1190	
Y	mg/L, g/t	13386		0.05	0.13	14961	0.0	95	41215	
Th	mg/L, g/t	2109		<0.03	0.03	2285	0.0	92	6295	
U	mg/L, g/t	<85		2.13	0.06	<85	24.9	112	310	
Si	mg/L, %	16.5				13.3	0.0	68	36.56	32
Al	mg/L, %	4.13		741	24.6	3.90	20.2	100	13.44	20
Fe	mg/L, %	28.7		70.0	0.7	32.2	0.3	95	89.06	5
Mg	mg/L, %	1.18		0.64	<0.07	1.31	0.1	94	3.62	6
Ca	mg/L, %	0.2		3.3	<0.9	0.20	2.6	87	0.57	15
Na	mg/L, %	1.51		93900	3060	1.99				
K	mg/L, %	1.21		127	4	1.22	12.1	97	3.83	15
Ti	mg/L, %	1.41		2.85	0.02	1.61	0.2	97	4.45	3
P	mg/L, %	0.585		36	<5	0.607	8.0	95	1.82	12
Mn	mg/L, %	0.06		0.47	<0.04	0.07	0.9	99	0.19	2
Cr	mg/L, %	0.01		0.2	<0.1	0.01	2.7	119	0.04	-16
V	mg/L, %	0.062		6.3	0.2	0.056	13.0	88	0.18	24

Table 8: CC 1 AL-1 Summary Metallurgical Balance

Element	Units	CC-1 WL Res	pH 3 PLS	25 g/L PLS	50 g/L PLS	Wash	pH 3 res	25 g/L res	50 g/L res	Extraction	Accountability out/in	Calc Head
Quant (mL/g)		213	52.6	52.9	1329.6	2062.3	8.6	7.5	190.2	%	%	g/t, %
		Assay (mg/L, %, g/t)										
La	mg/L, g/t	341					85	<85	<85	8	24	100
Ce	mg/L, g/t	512					256	<171	<171	10	33	200
Pr	mg/L, g/t	<256					<256	<256	<256	8	97	250
Nd	mg/L, g/t	171	14.7	22.6	22.7	0.89	<171	<171	<171	53	190	300
Sm	mg/L, g/t	<431					<431	<431	<431	8	97	420
Y	mg/L, g/t	14961	115	138	135	3.36	14646	14883	14095	14	98	14640
Th	mg/L, g/t	2285	0.06	242	258	6.50	2373	703	615	77	105	2400
U	mg/L, g/t	<85	<1	<1	<1	<1	<85	<85	<85	23	116	99
Si	mg/L, %	13.3					13.7	13.9	13.6	8	99	13.15
Al	mg/L, %	3.90	106	380	414	12.3	3.95	3.93	3.69	15	100	3.88
Fe	mg/L, %	32.2	1.3	889	1280	46.1	32.1	32.5	33.5	10	103	33.23
Mg	mg/L, %	1.31	120	246	273	8.35	1.24	1.21	1.04	23	92	1.21
Ca	mg/L, %	0.20	183	193	183	6.3	0.086	0.079	0.071	68	100	0.20
Na	mg/L, %	1.99	843	814	753	21	1.52	1.57	1.57	32	103	2.05
K	mg/L, %	1.22	11	33	55	2	1.21	1.27	1.11	12		1.12
Ti	mg/L, %	1.61	0.08	153	179	4.89	1.65	1.61	1.47	16	97	1.56
P	mg/L, %	0.607	<5	<5	<5	<5	0.620	0.642	0.602	9	98	0.59
Mn	mg/L, %	0.07	5.06	11.6	12.8	0.39	0.06	0.06	0.06	20	99	0.07
Cr	mg/L, %	0.01	<0.1	1.8	2.0	<0.1	0.02	0.01	0.01	18	109	0.01
V	mg/L, %	0.056	<2	<2	<2	<2	0.05	0.062	0.056	13	103	0.06

Table 9: CC-2 Summary Metallurgical Balance

Element	Units	Feed	NaOH	WW PLS	Wash	Final res	Extraction calc head	Accountability out/in	Calc Head	Extraction based on Res/Fd
Quant (mL/g)		171	697	1823	1680	149.2	%	%		%
Assay (mg/L, %, g/t)										
La	mg/L, g/t	<85		0.08	0.06	<85		89	200	
Ce	mg/L, g/t	<171		0.13	0.04	<171		89	300	
Pr	mg/L, g/t	<256		<0.03	<0.03	<256		88	450	
Nd	mg/L, g/t	<171		<0.06	<0.06	<171		88	300	
Sm	mg/L, g/t	<431		<0.04	<0.04	<431		88	760	
Y	mg/L, g/t	14095		0.06	<0.01	17560		109	30898	
Th	mg/L, g/t	615		<0.03	<0.03	615		88	1084	
U	mg/L, g/t	<85		0.43	<0.02	<85		93	160	
Si	mg/L, %	13.6				10.3	0.0	67	18.18	33
Al	mg/L, %	3.69		806	42.1	3.49	22.8	107	7.95	17
Fe	mg/L, %	33.5		59.8	0.6	37.3	0.2	98	65.72	3
Mg	mg/L, %	1.04		1.01	<0.07	1.39	0.1	118	2.45	-17
Ca	mg/L, %	0.071		4.9	<0.9	0.093	7.0	122	0.18	-14
Na	mg/L, %	1.57		105000	5210	1.58				
K	mg/L, %	1.11		113	5	1.35	9.6	118	2.63	-6
Ti	mg/L, %	1.47		4.34	0.02	1.85	0.3	110	3.26	-10
P	mg/L, %	0.602		23	<5	0.742	4.3	113	1.36	-8
Mn	mg/L, %	0.062		1.02	<0.04	0.077	1.6	111	0.14	-9
Cr	mg/L, %	<0.01		<0.1	<0.1	0.02	1.1	133	0.04	-31
V	mg/L, %	0.056		3.8	<0.2	0.062	7.3	104	0.12	4

Table 10: CC 2 AL-1 Summary Metallurgical Balance

Element	Units	CC-2 WL	pH 1	25 g/L	50 g/L	Wash	pH 1	25 g/L	50 g/L	Extraction	Accountability	Calc
Quant (mL/g)		Res	PLS	PLS	PLS	1515.5	res	res	res		out/in	Head
		134	52.0	54.4	826.9		8.5	10.7	111.2	%	%	g/t, %
Assay (mg/L, %, g/t)												
La	mg/L, g/t	<85					<85	<85	<85	15	97	100
Ce	mg/L, g/t	<171					<171	<171	<171	15	97	200
Pr	mg/L, g/t	<256					<256	<256	<256	15	97	250
Nd	mg/L, g/t	<171	1.58	5.58	5.35	0.21	<171	<171	<171	31	120	200
Sm	mg/L, g/t	<431					<431	<431	<431	15	97	420
Y	mg/L, g/t	17560	29.6	73.2	68.3	1.64	17087	17875	16457	18	95	16640
Th	mg/L, g/t	615	<0.03	46.6	48.7	1.28	615	264	264	64	100	610
U	mg/L, g/t	<85	<1	<1	<1	<1	<85	<85	<85	30	119	101
Si	mg/L, %	10.3					9.82	10.2	10.0	15	94	9.73
Al	mg/L, %	3.49	14.1	115	150	5.9	3.43	3.58	3.41	18	99	3.44
Fe	mg/L, %	37.3	84.2	955	1330	52.8	38.6	37.9	39.0	17	104	38.76
Mg	mg/L, %	1.39	144	202	228	7.83	1.25	1.29	1.19	26	96	1.34
Ca	mg/L, %	0.093	27.8	27.6	26.6	1.4	0.093	0.10	0.086	32	113	0.11
Na	mg/L, %	1.58	586	571	524	13	1.19	1.24	1.23	35	100	1.58
K	mg/L, %	1.35	12	42	67	3	1.31	1.39	1.29	18		1.31
Ti	mg/L, %	1.85	0.13	671	707	17.8	1.83	1.43	1.32	39	98	1.81
P	mg/L, %	0.742	<5	<5	<5	<5	0.720	0.759	0.716	16	96	0.71
Mn	mg/L, %	0.077	9.65	12.4	12.4	0.36	0.06	0.07	0.07	24	98	0.08
Cr	mg/L, %	0.02	<0.1	2.6	2.6	<0.1	0.03	0.01	0.01	29	78	0.02
V	mg/L, %	0.062	<1	<1	<1	<1	0.062	0.062	0.062	17	100	0.06

3. Acid Bake and Water Leach (AB) Testwork

3.1. AB Test Procedure

A total of 12 acid bake and water leach (AB) tests were completed throughout the test program to investigate the dissolution of rare earth elements (REE) and the behaviour of gangue minerals. Acid baking was conducted with sulphuric acid at elevated temperature (200-300°C). Acid was added to the feed solids (one of two flotation concentrates or GAL residue) and manually homogenized. Once blended, the crucible containing the mixture was placed in a furnace at ambient temperature and the furnace was gradually heated to the target temperature. After the furnace reached the operating temperature, the contents of the crucible were mixed (“rabbled”) once per hour during the three hours at temperature. After three hours, the furnace was allowed to cool slightly before the baked calcine was removed to come to ambient temperature in preparation for water leaching. A summary of acid bake conditions is presented in Table 11.

Table 11: Acid Bake Test Conditions Summary

Test ID	AB1	AB2	AB3	AB4	AB5	AB6	AB7	AB8	AB9	AB10	AB11	AB12 (Bulk)
Feed	Composite REE Concentrate (March 24, 2021)	Blended GAL2 and GAL3 Residue	Blended GAL2 and GAL3 Residue	Composite REE Concentrate (March 24, 2021)	Composite REE Concentrate (March 24, 2021)	Composite REE Concentrate (May 14, 2021)	Composite REE Concentrate (May 14, 2021)	Composite REE Concentrate (May 14, 2021)	Composite REE Concentrate (May 14, 2021)	Composite REE Concentrate (May 14, 2021)	Composite REE Concentrate (May 14, 2021)	Composite REE Concentrate (May 14, 2021)
Reagent	H ₂ SO ₄	H ₂ SO ₄	H ₂ SO ₄	H ₂ SO ₄	H ₂ SO ₄	H ₂ SO ₄	H ₂ SO ₄	H ₂ SO ₄	H ₂ SO ₄	H ₂ SO ₄	H ₂ SO ₄	H ₂ SO ₄
Reagent Strength	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%
Target Reagent Dose (kg/t)	1500	1500	1000	1250	1000	1500	1500	1500	1250	1000	750	1250
Effective Reagent Dose (kg/t)	1380	1502	1003	1242	890	1440	1446	1175	1052	948	681	1194
Target Temperature (°C)	280	200	280	200	250	280	300	300	300	300	300	300
Test Time (h)	3	3	3	3	3	3	3	3	3	3	3	3

Most of the calcines were subjected to standard water leaching conditions: a target of 10% solids (on bake feed basis) and mixed for four hours at ambient temperature. Liquor samples were taken after one and two hours before final filtration. Intermediate solution samples and final wash liquors were analyzed for yttrium, neodymium, and iron. Final solution samples were analyzed by ICP-AES for a full scan, ICP-MS for a REE scan, and analyzed for chloride and sulphate content. Final residues were analyzed for a whole rock analysis (WRA) by X-Ray Fluorescence (XRF), chlorine and sulphur content, and REE scan by XRF (first five tests) or by ICP-MS. The REE by XRF scan only provided data on a subset of REE (La, Ce, Pr, Nd, Sm, Y, Th, and U). The calcine from test AB9 was split to investigate alternative water leach conditions (WL1-AB9 was the baseline, WL2-AB9 increased leach temperature to 80°C, and WL3-AB9 ran at 20% solids vs. acid bake feed). Test WL-AB12 followed the conditions of WL3-AB9 after positive results from that test.

3.2. AB Test Results

3.2.1. Test Results from Conc 1 (March 24, 2021)

The three AB tests using the original flotation concentrate (Conc 1 in AB1, AB4, and AB5) compared the recovery of REE under three acid dosage and temperature conditions: high acid with high temperature (AB1), mid acid with low temperature (AB4), and low acid with mid temperature (AB5). The solids samples from this series of tests were analyzed for a subset of rare earth elements by X-Ray Fluorescence (REE by XRF: La, C, Pr, Nd, Sm, Y, Th, U). While this method can provide results rapidly, the results are reported to two decimals as percent as oxides, a granularity that converts to approximately 100 g/t (compared to a granularity of 1-10 g/t achievable when analyzing by ICP-MS). The calculated recoveries from these tests based on the XRF data were reported at less than 60%, though yttrium recovery for the first test (AB1) was 91% owing to its greater concentration in the feed (i.e. the granularity effect was minimized). Later, the feed and AB1 residue were analyzed for terbium and dysprosium by ICP-MS, with calculated recoveries of 84% and 86%, respectively.

In addition to the REE, 50-100% of the sulphuric acid-soluble impurities also dissolved during the water leach (Th, Al, Fe, Mg, K, Ti, P, Mn, V) while silicon, calcium, and sodium remained in the solids phase. Most of these dissolved impurities can be removed through pH adjustment with magnesium carbonate, as reported in Section 4 (Impurity Removal (IR) Testwork). Metal extractions for these three tests are summarized in Table 12 and liquor assays are summarized in Table 13.

Table 12: Acid Bake and Water Leach Extraction Summary with Conc 1 Feed

Element	WL-AB1	WL-AB4	WL-AB5
La	60	61	59
Ce	58	51	56
Pr	16	15	14
Nd	56	53	52
Sm	21	15	17
Tb	84	-	-
Dy	86	-	-
Y	91	45	62
Th	75	80	76
U	30	28	28
Si	0	0	0
Al	48	40	40
Fe	70	83	78
Mg	97	93	92
Ca	8	8	7
Na	1	0	0
K	85	65	61
Ti	69	49	42
P	83	53	59
Mn	84	84	79
Cr	0	0	0
V	66	66	57

Table 13: Acid Bake and Water Leach Filtrate Assay Summary with Conc 1 Feed

Element	WL-AB1	WL-AB4	WL-AB5
La	25.2	26.5	24.4
Ce	46.7	45.5	42.7
Pr	4.68	4.52	4.25
Nd	17.4	15.9	14.7
Sm	11.1	7.58	8.93
Eu	7.8	4.58	5.84
Gd	51.4	30.4	38.9
Tb	13	7.71	10
Dy	109	60.4	81.7
Ho	23.6	13.4	18.2
Y	828	450	613
Er	77.1	42.4	58.6
Tm	10.6	5.92	8
Yb	66.7	35.4	50
Lu	8.96	4.92	6.8
Sc	2.45	2.21	2.24
Th	132	141	137
U	3.49	3.41	3.24
Si	0	0	0
Al	1710	1550	1460
Fe	15600	20300	18300
Mg	1050	1130	1080
Ca	1150	1210	1020
Na	13	5	--
K	800	655	603
Ti	789	591	489
P	462	309	323
Mn	233	252	232
Cr	10.1	8.7	8
V	32.1	32.3	29.9
S	76000	96000	59000
Cl	1	4	2

Test AB1 (1500 kg/t H₂SO₄, 280°C) resulted in the highest recovery of yttrium from the first flotation concentrate (Conc 1). This result was used for planning of tests AB6 through AB12 (Conc 2 feed). These solutions were combined with liquors produced in tests AB2 and AB3 for initial impurity removal (IR) tests.

3.2.2. Test Results from Combined GAL Residue

The two AB tests using combined GAL residue (AB2 and AB3) compared the recovery of REE under two acid dosage and temperature conditions: high acid with low temperature (AB2) and low acid with high temperature (AB3). The solid samples from this pair of tests were analyzed for a subset of rare earth elements by X-Ray Fluorescence (REE by XRF: La, Ce, Pr, Nd, Sm, Y, Th, U) with the same benefits and drawbacks described in the previous section. The calculated recoveries in these tests, based on the XRF data, were up to 80%, though yttrium recovery for test AB3 was 96%. Later, the GAL Res and AB3 residue were analyzed for terbium and dysprosium by ICP-MS, with calculated recoveries of 93% and 94%, respectively.

In addition to the REE, sulphuric acid-soluble impurities also dissolved during the water leach (Th, Al, Fe, Mg, K, Ti, P, Mn, V) while silicon and sodium remained in the solids phase. Calcium was largely removed from the solids in the gangue acid leach tests (90-100%); the calcium remaining in the solids was solubilized in the AB tests. Calcium tenors were similar between these tests and those using Conc 1 as feed (approximately 1000 mg/L). As noted in the previous section, these dissolved impurities can be removed through pH adjustment. Metal extractions for these tests are summarized in Table 14 and liquor assays are summarized in Table 15.

Table 14: Acid Bake and Water Leach Extraction Summary with GAL Res Feed

Element	WL-AB2	WL-AB3
La	83	83
Ce	84	83
Pr	25	25
Nd	68	68
Sm	27	30
Tb	-	93
Dy	-	94
Y	66	96
Th	61	57
U	36	36
Si	0	0
Al	38	48
Fe	39	40
Mg	91	94
Ca	92	88
Na	1	2
K	51	82
Ti	32	44
P	65	79
Mn	77	77
Cr	0	0
V	37	50

Table 15: Acid Bake and Water Leach Filtrate Assay Summary with GAL Res Feed

Element	WL-AB2	WL-AB3
La	41.9	36.3
Ce	86.1	72.7
Pr	8.42	7.52
Nd	31.2	28.6
Sm	16	16.7
Eu	9.89	11.4
Gd	65.7	77.8
Tb	17.1	20.4
Dy	136	168
Ho	30.6	37.9
Y	972	1230
Er	96	118
Tm	13.3	16.5
Yb	77.7	96.9
Lu	10.6	13.5
Sc	3.24	3.24
Th	172	146
U	4.63	4.23
Si	0	0
Al	2080	2430
Fe	12900	12500
Mg	1270	1260
Ca	1010	929
Na	21.7	50
K	744	1070
Ti	555	720
P	535	639
Mn	75.2	71.7
Cr	11.9	13.7
V	29.4	35
S	160000	65000
Cl	1	1

Test AB3 (1000 kg/t H₂SO₄, 280°C) resulted in the highest recovery of yttrium from the GAL residue. These solutions were combined with liquors produced in tests AB1, AB4, and AB5 for initial impurity removal (IR) tests.

Based on the added complexity of the gangue acid leach process (requirement of hydrochloric acid, leach reactors, and a solid-liquid separation step) and the similarity in impurities in leach liquors after acid bake testing, the GAL was abandoned and the remainder of the acid bake test program focussed on evaluating the new flotation concentrate with a higher concentration of REE and lower concentration of calcium.

3.2.3. Test Results from Conc 2 (May 14, 2021)

The final six AB tests plus one bulk AB test (AB12) investigated the limits of the acid bake and water leach process using a new flotation concentrate (Conc 2). These tests considered the effect of elevated bake temperature (AB6 vs. AB7), high intensity mixing compared to infrequent rabbling (AB8 vs. AB7), and decreasing acid dosage (AB7, AB9 through AB11). The calcine from AB9 was also split to allow a comparison of water leach conditions, summarized in Table 16, with WL1-AB9 representing the baseline water leach conditions.

Table 16: AB9 Water Leach Test Condition Summary

Test ID	WL1-AB9	WL2-AB9	WL3-AB9
Feed	AB9 Calcine	AB9 Calcine	AB9 Calcine
Initial Pulp Density (% Solids)	10%	10%	20%
Test Time (h)	4	4	4
Temperature (°C)	25	80	25

The solids samples from these tests were analyzed for rare earth content by ICP-MS. Light rare earth dissolution remained high under all acid bake and water leach conditions tested, but heavy rare earth recovery was significantly reduced when decreasing the acid dosage (tests AB6, AB9, AB10, and AB11; 1500 kg/t down to 750 kg/t H₂SO₄).

Test AB12 was performed as a set of three acid bake tests using 240 g Conc 2 as feed before combining all three calcines for the water leach step. Bulk acid bake and water leach conditions were selected based on the optimum results from previous tests (acid bake: 1250 kg/t H₂SO₄, 300°C; water leach: 20% solids by weight, 25°C for two hours, as in test WL2-AB9). Test WL2-AB9 and AB12 both had very good REE recovery (97-98% Y, 95% Dy, and 94-95% Tb) under these conditions.

In addition to the REE, sulphuric acid-soluble impurities also dissolved during the water leach (Th, U, Al, Fe, Mg, K, Ti, P, Mn, V) while silicon, calcium, and sodium remained in the solids phase. As noted in the previous section, these dissolved impurities can be removed through pH adjustment. Metal extractions for these three tests are summarized in Table 17 and liquor assays are summarized in Table 18.

Table 17: Acid Bake and Water Leach Extraction Summary with Conc 2 Feed

Element	WL-AB6	WL-AB7	WL-AB8	WL1-AB9	WL2-AB9	WL3-AB9	WL-AB10	WL-AB11	WL-AB12
La	95	95	96	96	96	95	94	92	95
Ce	96	96	96	95	95	95	93	91	96
Pr	95	96	96	95	95	94	91	88	96
Nd	96	96	96	64	95	95	90	86	95
Sm	91	92	94	90	91	87	86	63	92
Eu	93	93	95	91	92	90	85	58	93
Gd	95	95	96	93	93	92	87	57	94
Tb	95	95	97	94	94	92	87	54	95
Dy	95	96	97	95	95	93	89	55	95
Ho	96	96	97	95	95	94	89	52	96
Y	98	98	98	98	98	97	92	59	97
Er	96	96	97	95	96	94	89	54	96
Tm	95	96	97	95	96	94	89	54	96
Yb	95	94	97	94	96	93	87	54	96
Lu	94	94	97	94	96	92	88	53	96
Sc	58	59	-	-	-	-	-	-	57
Th	17	74	81	68	65	63	57	45	75
U	84	86	92	85	86	82	78	64	88
Si	0	0	0	0	0	0	0	0	2
Al	20	22	30	24	29	19	24	23	28
Fe	58	60	82	63	64	63	55	54	59
Mg	71	71	87	75	88	71	77	77	83
Ca	13	14	15	17	15	5	17	16	8
Na	2	4	8	4	5	3	3	0	6
K	61	55	54	57	75	54	50	44	62
Ti	52	50	68	50	40	46	48	35	64
P	52	10	33	30	48	36	19	23	60
Mn	69	68	87	74	88	68	78	84	79
Cr	100	0	0	0	0	0	0	0	0
V	62	60	70	61	71	56	52	41	65

Table 18: Acid Bake and Water Leach Filtrate Assay Summary with Conc 2 Feed

Element	WL-AB6	WL-AB7	WL-AB8	WL1-AB9	WL2-AB9	WL3-AB9	WL-AB10	WL-AB11	WL-AB12
La	172	196	189	169	174	467	176	174	487
Ce	336	384	370	285	290	809	296	321	922
Pr	34	39	37.4	34.9	36.6	104	39.6	31.9	107
Nd	127	145	141	121	124	352	131	112	381
Sm	57.4	64.9	66.9	61.4	62.6	153	60.9	47.7	188
Eu	36.6	40.5	41.5	36.2	37	110	36.8	27.5	121
Gd	235	263	254	216	220	560	217	173	656
Tb	61	69	66.2	57.2	59.2	143	57.1	41.3	183
Dy	459	540	511	508	501	1170	508	343	1340
Ho	106	123	122	108	109	306	110	70.3	326
Y	3560	4350	4000	3720	3780	8200	3660	2450	10400
Er	328	381	380	323	329	877	331	225	961
Tm	47.3	54.3	52	46	46.8	126	45.6	32.1	147
Yb	281	308	309	249	257	694	251	194	833
Lu	36.4	41.5	42.6	38.2	39.5	101	37.7	24.9	119
Sc	3.15	3.46	4.11	4.17	4.87	8.03	4.15	3.5	7.36
Th	47.3	645	669	545	527	1290	484	362	1550
U	12	13.8	15.1	13.9	14.5	33.1	13.1	9.94	39.4
Si	0	0	0	0	0	0	0	0	0
Al	415	435	560	494	589	1160	499	473	1390
Fe	14300	15500	20200	15900	16100	37100	14500	14300	39100
Mg	178	187	211	187	219	436	208	215	517
Ca	957	988	978	1170	1000	1170	1200	1140	1370
Na	28	52	86	55	59	129	39	<20	130
K	95	98	91	98	211	235	90	78	269
Ti	2380	2280	2840	2380	1880	5750	2260	1640	7350
P	1300	223	734	684	1190	2300	439	531	3670
Mn	294	298	352	308	371	714	352	381	839
Cr	17.8	18.3	18.1	17.7	22.3	36.7	17.3	13.9	40.1
V	67.9	65.7	70.8	68.6	80	173	61.3	48.7	162
S	92000	66000	77000	86000	85000	180000	62000	53000	0
Cl	3	2	2	3	34	8	2	2	1

Solutions from tests with adequate recovery were combined for additional impurity removal (IR) tests. This includes AB6 through AB10, and AB12.

4. Impurity Removal (IR) Testwork

4.1. IR Test Procedure

Two types of impurity removal testing were completed as part of this test program. In the first, called a “titration-style” test, the feed solution was heated to the target temperature of 50°C before reagent was added sequentially to a series of pH targets between pH 2.0 and pH 4.0. After achieving a pH target, the slurry was maintained for one hour and the solution sampled before moving to the next target. This process was repeated until the final pH target, whereupon the reaction slurry was fully filtered. In this program, the solutions were analyzed for key impurities (Al, Fe, Th) and rare earth elements (Y, Nd) to gauge the extent of precipitation, with a goal of maximizing impurity removal while minimizing rare earth losses.

Tests IR-1 and IR-2 compared the difference in performance between magnesium carbonate and calcium carbonate. Once the results became available, the next test (IR-3) further explored magnesium carbonate with the addition of phosphoric acid to achieve a molar ratio of 1:1 (Fe:P) to improve iron precipitation. Each of these tests used combined leach liquor from the first flotation concentrate and GAL Res acid bake tests.

Test IR-4 duplicated the conditions of IR-1 to confirm performance using a different feed liquor (generated from acid bake testing of the second flotation concentrate).

The final impurity removal test in this program (IR-5) was a bulk “endpoint” test that included a stoichiometric addition of hydrogen peroxide to oxidize iron in solution in order for it to precipitate. In this test, the magnesium carbonate reagent was added to achieve a single pH target (pH 2.9) and then maintained for one hour before filtration. This pH target was selected based on titration test results (IR-1 and IR-4).

4.2. IR Test Results

Titration style tests to compare magnesium carbonate (IR-1) and calcium carbonate (IR-2) resulted in a similar precipitation profile for the tracked impurities, but calcium carbonate addition resulted in elevated co-precipitation of neodymium and yttrium at all pH targets. The extent of precipitation of the tracked elements for these tests can be seen in Figure 2 and Figure 3. The chart for magnesium carbonate (Figure 2) shows that neodymium and yttrium precipitation are near-zero when the slurry is below pH 3.3.

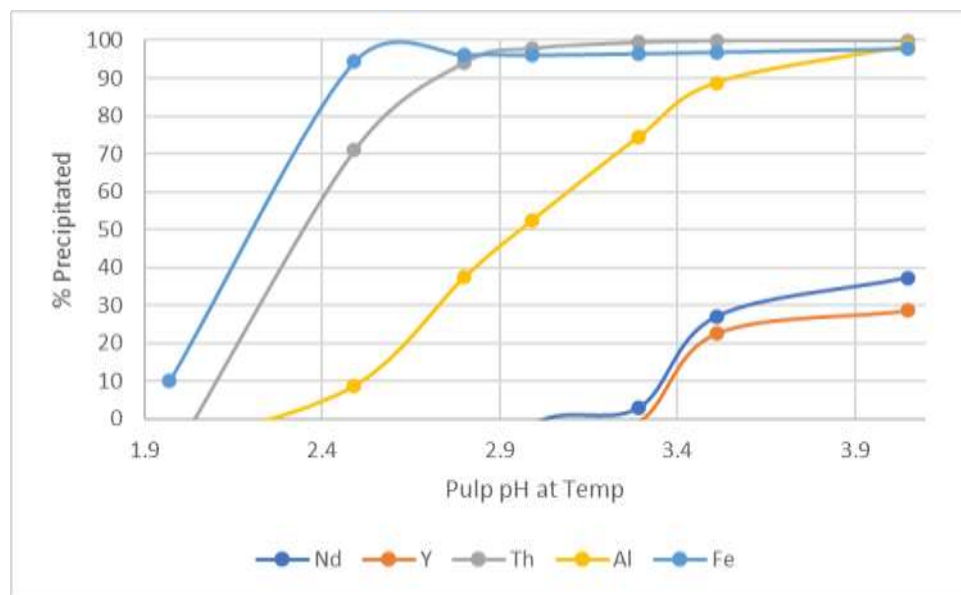


Figure 2: Extent of Precipitation of Select Elements in IR-1 with Magnesium Carbonate

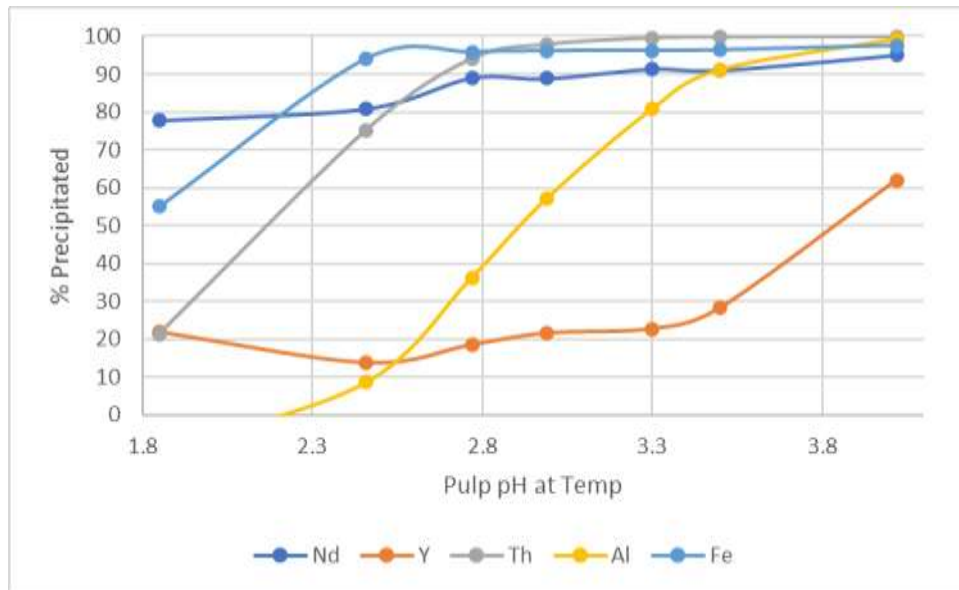


Figure 3: Extent of Precipitation of Select Elements in IR-2 with Calcium Carbonate

The addition of phosphoric acid to a magnesium carbonate titration test (IR-3) resulted in a significant increase in iron and thorium precipitation at the lowest pH values (pH 2.0 and pH 2.5) when compared to IR-1. However, precipitation without phosphoric acid addition was similar from pH 2.8 and above. This can be seen in Figure 4. These benefits do not outweigh the need for a phosphoric acid delivery system and additional reagent required to neutralize; this reagent was not investigated further.

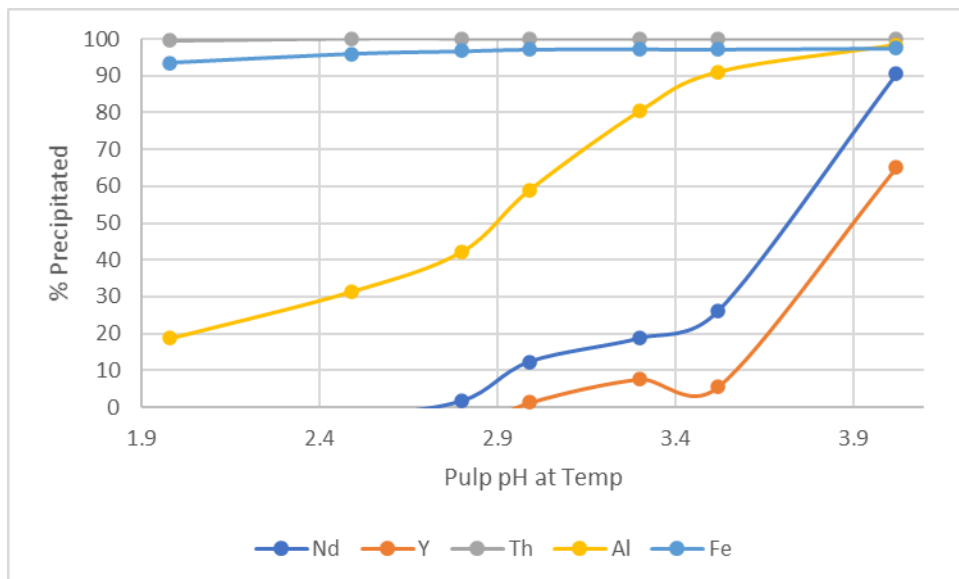


Figure 4: Extent of Precipitation of Select Elements in IR-3 with MgCO₃ and Phosphoric Acid

Test IR-4 was a duplicate of test IR-1 – a titration test using magnesium carbonate – using acid bake water leach liquor produced from the second flotation concentrate (containing higher levels of REE and lower

calcium content). This test confirmed the observations of test IR-1, showing that the maximum precipitation of iron and thorium while minimizing neodymium and yttrium co-precipitation occurred around pH 2.9. These results are displayed in Figure 5.

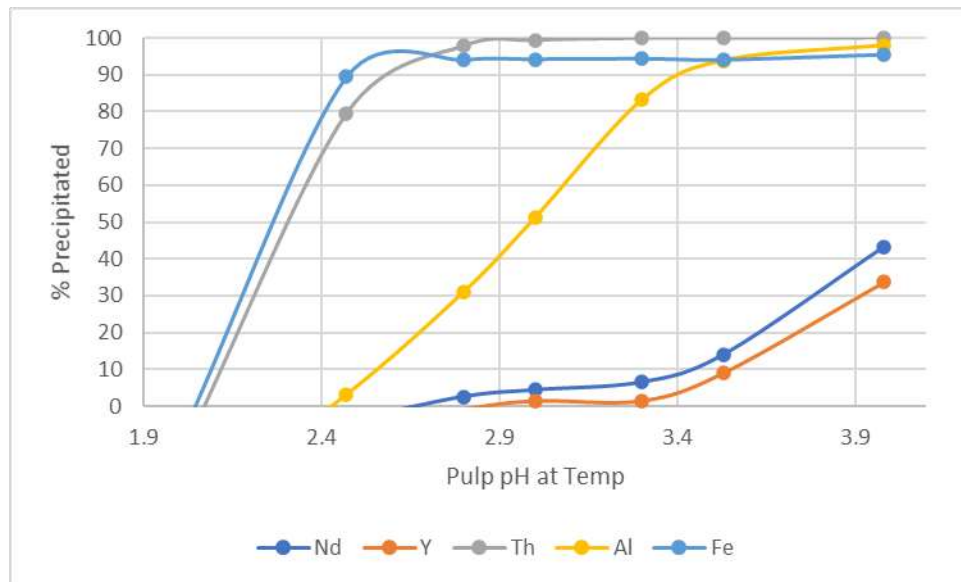


Figure 5: Extent of Precipitation of Select Elements in IR-4 with Magnesium Carbonate

Test IR-5 was designed as a confirmatory bulk endpoint test to produce liquor for further downstream testwork. This test aimed to achieve a final target of pH 2.9-3.0 with the addition of hydrogen peroxide to further precipitate iron. In this test, 94% of the iron and thorium precipitated along with 40% of the aluminum. Approximately 11% of the neodymium and 9% of the yttrium co-precipitated, which is greater than what might be expected in an optimized process, as evidenced by low co-precipitation values in titration style tests (IR-1 and IR-4), with each showing less than 3% co-precipitation based on solution assays only. The metallurgical balance for IR-5 can be seen in Table 19. Magnesium carbonate dosage to achieve the target pH was 381 kg/t (based on flotation concentrate), a value that is slightly higher than the expected value of 372 kg/t estimated from IR-4 testing.

Table 19: IR-5 Summary Metallurgical Balance

Sample & Quant.	Assay Units	Feed	Final Filt	Final Wash	Final PPT	PPT % Res/Feed	Calc Head	Acc %
(mL or g)		7838	6880.7	10224.5	561.0			
La	mg/L, %	288	208		0.04	--	213	74
Ce	mg/L, %	513	382		0.08	10.7	390	76
Pr	mg/L, %	60.4	40.3		<0.03	--	54	89
Nd	mg/L, %	215	145	39.3	0.03	11.4	203	94
Sm	mg/L, %	103	68.7		<0.04	--	91	89
Eu	mg/L, %	65.6	41.7			--	37	56
Gd	mg/L, %	367	278			--	244	66
Tb	mg/L, %	99.6	71.4			--	63	63
Dy	mg/L, %	724	553			--	485	67
Ho	mg/L, %	181	126			--	111	61
Y	mg/L, %	5760	4220	1120	0.71	8.8	5673	98
Er	mg/L, %	522	399			--	350	67
Tm	mg/L, %	79.7	54.8			--	48	60
Yb	mg/L, %	480	333			--	292	61
Lu	mg/L, %	64.8	44.2			--	39	60
Sc	mg/L, %	4.33	0.83			--	1	17
Th	mg/L, %	840	7.46	<1.87	1.11	94.3	801	95
U	mg/L, %	22.1	7.27		0.02	54.9	19	84
Si	mg/L, %				0.18	--	130	--
Al	mg/L, %	733	326	111	0.413	40.3	726	99
Fe	mg/L, %	21900	33.3	12.3	28.6	93.5	20518	94
Mg	mg/L, %	290	13700		1.57	--	--	--
Ca	mg/L, %	1170	1120		0.50	30.6	1341	115
Na	mg/L, %	75	290		0.04	35.4	281	375
K	mg/L, %	144	108		0.02	8.3	107	74
Ti	mg/L, %	3640	2.74		4.86	95.5	3477	96
P	mg/L, %	1550	<5		1.99	91.7	1425	92
Mn	mg/L, %	466	347		0.04	5.9	332	71
Cr	mg/L, %	24.8	<0.3		0.04	118.5	30	120
V	mg/L, %	94.0	<0.2		0.14	106.6	100	107
Cu	mg/L, %	<40	11.0			--	10	24
Pb	mg/L, %	<8	<4			--	4	44
Zn	mg/L, %	<5	6			--	5	105
TREE	mg/L, %	10390	6965	1159	0.9	--	8293	80
LREE	mg/L, %	1179	844	39	0.2	--	951	81
HREE	mg/L, %	8344	6121	1120	0.7	--	7341	88

5. Crude Rare Earth Precipitation Testwork

5.1. Rare Earth Precipitation (RP) Test Procedure

Rare earth precipitation testwork also included two types of testing – titration-style and endpoint testing. In the titration-style test (RP-1), sodium carbonate was added sequentially to a series of pH targets between pH 6.00 and pH 7.25. After achieving a pH target, the slurry was maintained for 30 min and the solution sampled before moving to the next target. This process was repeated until the final pH target, whereupon the reaction slurry was fully filtered. In this program, the solutions were analyzed for yttrium to gauge the extent of rare earth precipitation.

The second crude rare earth precipitation test (RP-2) was a bulk “endpoint” test with a single pH target selected based on the titration style test results (pH 6.50) and then maintained for one hour before filtration.

5.2. Crude RP Test Results

Test RP-1 showed that yttrium precipitation was approximately 91% at the lowest pH tested (pH 6.10) and increased to greater than 99% at pH 6.50 as shown in Figure 6. The bulk endpoint test (RP-2) was planned on this basis and successfully precipitated the rare earth elements along with impurity elements (Th, U, Al, Fe). The final residue assayed at 43% total REE and 1.86% Al. Thorium, uranium, and iron all assayed <0.5%. The metallurgical balance for the endpoint test (RP-2) is included in Table 20. Sodium carbonate dosage to achieve the target pH was 90.5 kg/t (based on flotation concentrate), a value that is slightly higher than the expected value of 78 kg/t estimated from RP-1 testing.

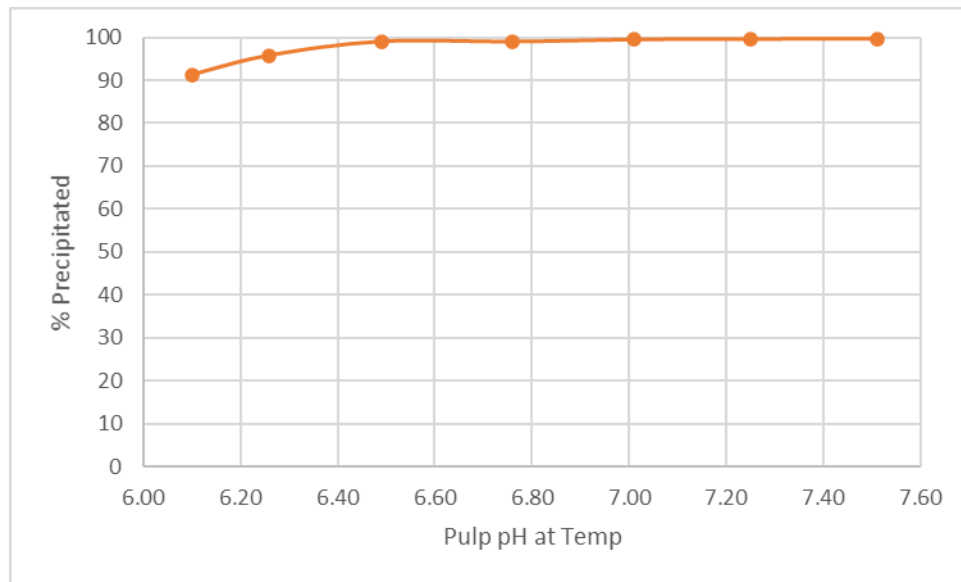


Figure 6: Extent of Yttrium Precipitation in RP-1

Table 20: RP-2 Summary Metallurgical Balance

Sample & Quant.	Assay Units	Feed	Final Filt	Final Wash	Final PPT	PPT % Res/Feed	Calc Head	Acc %
(mL or g)		5283	5599.9	5969.6	86.3			
La	mg/L, %	208	0.09		1.26	99.0	206	99
Ce	mg/L, %	382	0.09		2.3	98.4	376	98
Pr	mg/L, %	40.3	<0.03		0.244	98.9	40	99
Nd	mg/L, %	145	<0.06		0.876	98.7	143	99
Sm	mg/L, %	68.7	<0.04		0.409	97.3	67	97
Eu	mg/L, %	41.7	<0.03		0.248	97.2	41	97
Gd	mg/L, %	278	0.11		1.65	97.0	270	97
Tb	mg/L, %	71.4	0.05		0.413	94.5	68	95
Dy	mg/L, %	553	0.66		3.43	101.3	561	101
Ho	mg/L, %	126	0.26		0.746	96.7	122	97
Y	mg/L, %	4220	24.1	3.63	26.7	103.4	4392	104
Er	mg/L, %	399	1.30		2.43	99.5	398	100
Tm	mg/L, %	54.8	0.25		0.317	94.5	52	95
Yb	mg/L, %	333	1.87		1.86	91.3	306	92
Lu	mg/L, %	44.2	0.31		0.256	94.6	42	95
Sc	mg/L, %	0.83	<0.07		<0.004	78.7	1	88
Th	mg/L, %	7.46	<0.03		0.0438	95.9	7	96
U	mg/L, %	7.27	1.19		0.0334	75.1	7	92
Si	mg/L, %				0.15	--	24	--
Al	mg/L, %	326	<2		1.86	93.1	306	94
Fe	mg/L, %	33.3	0.9		0.21	103.0	35	106
Mg	mg/L, %	13700	11900		0.55	0.7	12703	93
Ca	mg/L, %	1120	683		0.69	10.0	836	75
Na	mg/L, %	290	5420		0.16	--	--	--
K	mg/L, %	108	87		<0.008	1.3	94	87
Ti	mg/L, %	2.74	<0.02		0.01	71.5	2	72
P	mg/L, %	<5	<5		<0.004	--	--	--
Mn	mg/L, %	347	259		0.27	12.8	319	92
Cr	mg/L, %	<0.3	<0.1		<0.007	--	--	--
V	mg/L, %	<0.2	<0.2		<0.006	--	--	--
Cu	mg/L, %	11.0	<0.2			--	--	2
Pb	mg/L, %	<4	<2			--	--	53
Zn	mg/L, %	6	1.9			--	--	34
TREE	mg/L, %	6981	29	4	43.1	101.0	7084	101.5
LREE	mg/L, %	844	0	0	5.1	98.5	832	98.6
HREE	mg/L, %	6121	29	4	38.1	101.6	6252	102.1

6. Rare Earth Purification Testwork

Once a crude rare earth precipitate had been generated, there was insufficient material to perform any kind of optimization testing. From this point in the test program, all test conditions were based on SGS experience with similar processes and typically only a single test was carried out for each process step to maximize the amount of final product that could be produced. In some cases, this resulted in an excess of reagent or losses of rare earth elements. It is understood that these values can be used as guidelines for design, but with an opportunity for significant optimization. This applies to the rare earth precipitate re-leach (RL), uranium ion exchange (UIX), thorium solvent extraction (ThSX), rare earth precipitation of treated RL liquor (RP), and calcination (C-RP).

6.1. Rare Earth Precipitate Re-leach (RL)

Rare earth precipitate re-leach (RL) testwork consisted of a two-stage sulphuric acid process wherein crude RP solids were slurried to 20% (w/w) solids in deionized water and heated to 50°C. Sulphuric acid was

then added to achieve pH 1.0 and maintained for three hours. A solution sample was then collected and assayed for yttrium content, before additional RP solids were added to raise the slurry to pH 3.5 and allowed to mix for another hour.

In this test, 91% of the feed mass was dissolved, resulting in a concentrated rare earth liquor containing 28 g/L total rare earth elements (TREE), which represented 99% of the available TREE. This leach procedure also rejected 94% of the thorium, 85% of the aluminum, and 99% of the iron. The acid required in this redissolution step was 48 kg/t (flotation concentrate basis), but is expected to be approximately 70 kg/t when correcting for any rare earth units that reported to various wash phases and/or precipitates and thus were not advanced to the re-leach step. The metallurgical balance for the re-leach test (RL-1) is included as Table 21.

Table 21: RL-1 Summary Metallurgical Balance

Sample & Quant.	Assay Units	RP-2 Final Residue	pH 1 Filtrate	pH 3.5 Filtrate	Wash	pH 3.5 Residue	Leached	Account.	Calc
(mL or g)		80	46	1018	312	7	%	out/in	Head
La	mg/L, %	1.26		842		0.0971	99	85	1.07
Ce	mg/L, %	2.3		1540		0.278	99	86	1.97
Pr	mg/L, %	0.244		165		0.0307	99	87	0.21
Nd	mg/L, %	0.876		577		0.119	99	85	0.74
Sm	mg/L, %	0.409		270		0.0683	98	85	0.35
Eu	mg/L, %	0.248		181		0.037	99	94	0.23
Gd	mg/L, %	1.65		1090		0.202	99	85	1.40
Tb	mg/L, %	0.413		281		0.0571	99	87	0.36
Dy	mg/L, %	3.43		2210		0.547	98	83	2.84
Ho	mg/L, %	0.746		501		0.117	98	86	0.64
Y	mg/L, %	26.7	13500	17100	1930	2.47	99	88	23.4
Er	mg/L, %	2.43		1580		0.48	98	84	2.04
Tm	mg/L, %	0.317		216		0.0827	97	88	0.28
Yb	mg/L, %	1.86		1270		0.681	97	90	1.67
Lu	mg/L, %	0.256		177		0.0967	96	91	0.23
Sc	mg/L, %	<0.004		1.66		0.0278	47	112	0.004
Th	mg/L, %	0.0438		1.98		0.489	6	101	0.04
U	mg/L, %	0.0334		23.3		0.0302	92	96	0.03
Si	mg/L, %	0.15		22.2		1.49	18	104	0.16
Al	mg/L, %	1.86		210		17.4	15	94	1.76
Fe	mg/L, %	0.21		1.2		2.57	1	105	0.22
Mg	mg/L, %	0.55		324		0.03	99	75	
Ca	mg/L, %	0.69		519		0.02	100	96	0.66
Na	mg/L, %	0.16		32		0.03	94	26	0.04
K	mg/L, %	<0.008		29		<0.008	98	--	0.04
Ti	mg/L, %	0.01		<5		0.12	38	138	0.02
P	mg/L, %	<0.004		<5		0.057	57	--	0.01
Mn	mg/L, %	0.27		219		0.008	100	103	0.28
Cr	mg/L, %	<0.007		<0.3		0.01	24	23	0.00
V	mg/L, %	<0.006		<0.2		<0.006	35	13	0.00
Cu	mg/L, %			45.0			-	-	0.06
Pb	mg/L, %			<20			-	-	0.03
Zn	mg/L, %			20.3			-	-	0.03
S	mg/L, g/t						-	-	-
TREE	mg/L, %	43.1		28000		5.36	99	83	-

6.2. Uranium Ion Exchange (UIX) and Thorium SX (ThSX)

Uranium ion exchange (UIX) testwork was designed with two stages of IX contact with Purolite A660 resin (strong base anion) in order to ensure all uranium was removed from the liquor. Each UIX contact used 133 mL wet settled resin (wsr), targeting a feed-to-resin volume ratio of 7.5. Once the resin had been added to the feed liquor, the contents were gently mixed with a paddle-type impeller for 24 h at ambient temperature, before the resin was filtered from the solution. A sample of the liquor was collected after each contact and analyzed for uranium and thorium after both contacts had been completed. Other elements were added to the UIX assay request after thorium SX (ThSX) testwork was completed.

In the UIX test, 99.9% of the uranium was removed in the first contact along with 75% of the thorium. The second contact resulted in uranium concentration falling below the detection limit of 0.02 mg/L while 75% of the remaining thorium was removed (an overall removal of thorium equivalent to 94%). After pH adjustment, this liquor was sampled and analyzed for a full ICP scan and REE scan – values that were used retroactively to determine the behaviour of the rare earth elements during the UIX test. It was determined that 3% of the total rare earth content of the re-leach liquor was collected by the IX resin in the first contact and 12% of the remaining REE in the second contact (approximately 15% REE collected on the resin in total). This REE extraction was a known risk of IX testing with the A660 resin, but this process can be optimized to bring REE losses to near-zero when operated in a continuous manner and when high uranium loading crowd off co-extracted REE. The metallurgical balance for this test is presented in Table 22.

Table 22: Uranium Ion Exchange Metallurgical Balance Summary

		UIX1-1 (Sol/Resin=952/133)					UIX1-2 (Sol/Resin=939/133)				
		Feed Sol'n	24 h Solution	Loaded Resin	Extraction %	Account. out/in	Feed Sol'n	24 h Solution	Loaded Resin	Extraction %	Account. out/in
Quant (mL; g)		951.6	1007.8	72.8			938.8	1000.6	76.4		
Element	Units	Assay (mg/L, %, g/t)			%	%	Assay (mg/L, %, g/t)			%	%
Tb	mg/L, g/t	281	244		8	92	244	197		14	86
Dy	mg/L, g/t	2210	1980		5	95	1980	1650		11	89
Y	mg/L, g/t	17100	15800		2	98	15800	13100		12	88
Th	mg/L, g/t	1.98	0.46		75	25	0.46	0.11		75	25
U	mg/L, g/t	23.3	0.03		99.9	0	0.03	<0.02		29	71
La	mg/L, %	842	757		5	95	757	631		11	89
Ce	mg/L, %	1540	1420		2	98	1420	1160		13	87
Pr	mg/L, %	165	149		4	96	149	123		12	88
Nd	mg/L, %	577	524		4	96	524	433		12	88
Sm	mg/L, %	270	238		7	93	238	201		10	90
Eu	mg/L, %	181	150		12	88	150	120		15	85
Gd	mg/L, %	1090	997		3	97	997	785		16	84
Ho	mg/L, %	501	449		5	95	449	366		13	87
Er	mg/L, %	1580	1480		1	99	1480	1230		11	89
Tm	mg/L, %	216	195		4	96	195	163		11	89
Yb	mg/L, %	1270	1170		2	98	1170	1010		8	92
Lu	mg/L, %	177	158		5	95	158	134		10	90
Sc	mg/L, %	1.66	0.70		55	45	0.70	0.27		59	41
Th	mg/L, %	1.98	2		0	100		0.16		91	9
U	mg/L, %	23.3	22		0	100		<0.02		100	0
Si	mg/L, %	22.2	21		0	100				100	0
Al	mg/L, %	210	198		0	100		174		6	94
Fe	mg/L, %	1.2	1		0	100		<0.6		44	56
Mg	mg/L, %	324	306		0	100		274		5	95
Ca	mg/L, %	519	490		0	100		447		3	97
Na	mg/L, %	32	30		0	100		<30		-6	106
K	mg/L, %	29	27		0	100		<30		-17	117
Ti	mg/L, %	5	5		0	100		<4		10	90
P	mg/L, %	5	5		0	100		<5		-13	113
Mn	mg/L, %	219	207		0	100		181		7	93
Cr	mg/L, %	0.3	0		0	100		<0.2		25	75
V	mg/L, %	0.2	0		0	100		<0.2		-13	113
Cu	mg/L, %	45	42		0	100		<40		0	100
Pb	mg/L, %	20	19		0	100		<9		49	51
Zn	mg/L, %	20.3	19		0	100		15		17	83
TREE	mg/L, %	28000	25711		3	97	25711	21303		12	88

assumes zero extraction in UIX-1

after pH adjust for ThSX

The uranium-depleted liquor after UIX was adjusted to pH 1.5 using sulphuric acid prior to the ThSX test. An organic mixture of 0.5% Primene JMT and 2.5% tridecanol in Aromatic 150ND was contacted twice with deionized water adjusted to pH 1.0 with sulphuric acid at a ratio of 1:1 (aqueous-to-organic, A/O) at ambient temperature. These two solutions were contacted at a phase ratio of 10:1 (A/O) for ten minutes at ambient temperature and then allowed to separate. This test was designed prior to receipt of the UIX results and executed without regard for the high level of thorium removed in the UIX test. While very little additional thorium was removed from the liquor, this test still demonstrated the selectivity of these conditions.

The thorium SX contact successfully loaded 75% of the residual thorium (combined with UIX, 98.4% total Th removal) to the organic mixture while tracking of yttrium, terbium, and dysprosium showed 0% (Y and Tb) and 2% (Dy) loading. A multi-stage counter current SX process should lead to higher levels of thorium

loading and allow for scrubbing of the loaded organic to recover any co-extracted rare earth elements. Data collected in this test is summarized in Table 23.

Table 23: Thorium Solvent Extraction Summary

Contact			Feed	ThSX-1-1
Aqueous Feed				UIX1-2 Filtrate
Aqueous Feed pH				1.5
Phase Ratio	A/O			10
Temperature	°C			ambient
Volumes In/Out	Org. In	mL		89
	Aq. In	mL		885
	Org. Out	mL		88
	Aq. Out	mL		880
Disengagement Time		sec*		8
First Phase to Separate				Aq
Emulsion (quantity)		mL		-
Emulsion Location				-
Organic Density		g/mL	0.881	0.884
Aqueous Density		g/mL	1.057	1.056
Phases	Aqueous	colour	lt pink	lt pink
		clarity	clear	clear
		pH	1.46	1.48
		ORP (mV)	327	350
	Organic	colour	lt yellow	lt yellow
		clarity	clear	clear
Free Acid Data	Sample		Feed	ThSX-1-1R
	Aliquot	mL	2.0	2.0
	NaOH	N	0.2	0.2
	Titrant	mL	1.30	1.34
	Acid Type		H2SO4	H2SO4
	MW	g/mole	98.1	98.1
			g/L	acid
			6.4	6.6
Contact			Feed	ThSX-1-1
Aqueous Feed pH				1.5
Thorium	Org. Assay	mg/L		1.06
	Aq. Assay	mg/L	0.16	0.04
	Loaded	%		73
	Distrib.	O/A		27
	Calc. Org In	mg/L		-0.15
	Separation	Th/Y		6188
	Factors	Th/Dy		147
Yttrium	Org. Assay	mg/L		57
	Aq. Assay	mg/L	13100	13200
	Loaded	%		0.0
	Distrib.	O/A		0.0
	Calc. Org In	mg/L		309
Terbium	Calc. Org.	mg/L		-19
	Aq. Assay	mg/L	197	200
	Loaded	%		-0.9
	Distrib.	O/A		-0.1
	Calc. Org In	mg/L		0.0
Dysprosium	Calc. Org.	mg/L		294
	Aq. Assay	mg/L	1650	1630
	Loaded	%		2
	Distrib.	O/A		0
	Calc. Org In	mg/L		0

The thorium-free raffinate from the solvent extraction step was advanced to final rare earth precipitation and calcination. No stripping testwork was conducted, but it is expected that thorium strip liquor be recycled to the acid bake water leach stages, where any co-extracted REE can be recovered. The majority of the thorium contained in the strip liquor will exit the circuit via the Impurity Removal (IR) stages, which are able to remove > 90% of the contained thorium at low REE losses.

6.3. Final Rare Earth Precipitation (RP) and Calcination (C-RP)

The final step of this test program was the precipitation of rare earth elements with minimal impurities. Originally, this was envisioned as an oxalic acid precipitation exclusively; however, assay results of the ThSX raffinate showed low levels of impurities, so precipitation with sodium carbonate was tested as well.

The oxalic acid precipitation test (RP-3) targeted the addition of 110% of the stoichiometric requirement of oxalate based on the quantity of rare earth elements in the volume of feed liquor. This oxalic acid was added as a 10% (w/w) solution after the feed liquor had been heated to 50°C and maintained for two hours before filtration. Due to the expected low solids mass, no solids samples were taken for analysis, though the final filtrate (barren solution) was analyzed for a full suite ICP scan and REE scan while the wash liquor was analyzed for yttrium. The precipitate was dried and then subjected to a calcination step to decompose the REE oxalates to rare earth oxides (C-RP3). Calcination was a three-hour process at a temperature of 1200°C with the cool calcine was submitted for Whole Rock Analysis, ICP scan, and REE scan.

The sodium carbonate precipitation test (RP-4) followed the same procedure as the crude rare earth precipitation test (RP-2): the feed liquor was heated to 50°C before adding sodium carbonate solution (100 g/L concentration) to achieve a target of pH 6.50, which was maintained for one hour. Due to the expected low solids mass, no solids samples were taken for analysis, though the final filtrate (barren solution) was analyzed for a full suite ICP scan and REE scan while the wash liquor was analyzed for yttrium. The precipitate was dried and then subjected to a calcination step to decompose the REE carbonates to rare earth oxides (C-RP4). Calcination was a three-hour process at a temperature of 1200°C, and the cool calcine was submitted for Whole Rock Analysis, ICP scan, and REE scan. It is noted that this calcination step is generally not required as a rare earth carbonate product is desirable to potential customers.

The solids assay values measured after calcination were used to calculate elemental concentrations in the solids after precipitation (assuming no loss). These calculated values were used to complete metallurgical balances for the precipitation tests.

Oxalic acid precipitation and calcination (RP-3 and C-RP3) produced a final rare earth product containing 79.9% REE (equivalent to 98.1% rare earth oxides). This represented 94% of the total rare earth elements present in the ThSX raffinate. The main impurities in the calcine include Na, Mg, Si, and Ca (0.2%-0.3%). Other minor impurities included Mo (381 g/t), Bi (105 g/t), Cr and Ni (100 g/t). The metallurgical balance

for the precipitation test and the assays of the final calcine are available in Table 24 and Table 25, respectively.

Table 24: Oxalic Acid Precipitation (RP-3) Summary Metallurgical Balance

Sample & Quant.	Assay Units	Feed	Final Filt	Final Wash	Final PPT	PPT % Res/Feed	Calc Head	Acc %
(mL or g)		400	465.5	401.5	21.7			
La	mg/L, %	637	14.1		11002	93.9	614	96
Ce	mg/L, %	1160	8.52		19924	93.4	1093	94
Pr	mg/L, %	124	0.42		2094	91.8	114	92
Nd	mg/L, %	434	0.92		7489	93.8	408	94
Sm	mg/L, %	198	0.23		3467	95.2	189	95
Eu	mg/L, %	120	0.14		2140	96.9	116	97
Gd	mg/L, %	800	1.18		13961	94.8	760	95
Tb	mg/L, %	200	0.49		3587	97.5	196	98
Dy	mg/L, %	1630	5.94		28985	96.6	1582	97
Ho	mg/L, %	369	2.11		6703	98.7	367	99
Y	mg/L, %	13200	212	8.51	226056	93.1	12541	95
Er	mg/L, %	1210	10.7		21404	96.1	1176	97
Tm	mg/L, %	164	2.22		2922	96.8	161	98
Yb	mg/L, %	993	15.8		17243	94.4	956	96
Lu	mg/L, %	135	2.40		2325	93.6	129	96
Sc	mg/L, %	0.14	<0.07		<12	448.7	1	507
Th	mg/L, %	0.04	<0.03		0.51	69.1	0	156
U	mg/L, %	<0.02	<0.02		<0.2	62.8	0	179
Si	mg/L, %				0.088	--	48	--
Al	mg/L, %	171	147		<0.002	0.7	172	101
Fe	mg/L, %	<0.6	1.9		0.003	--	--	--
Mg	mg/L, %	268	217		0.14	28.1	328	122
Ca	mg/L, %	439	328		0.074	9.2	422	96
Na	mg/L, %	<30	14		0.14	--	--	--
K	mg/L, %	<30	34		<0.004	--	--	--
Ti	mg/L, %	<4	0.42		<0.003	--	--	--
P	mg/L, %	<5	<5		0.002	--	--	--
Mn	mg/L, %	178	145		<0.004	1.1	171	96
Cr	mg/L, %	<0.2	0.2			--	--	--
V	mg/L, %	<0.2	<0.2			--	--	--
Cu	mg/L, %	<40	18.7			--	--	--
Pb	mg/L, %	<9	<2			--	--	--
Zn	mg/L, %	15	14.4		0.00	0.0	17	112
TREE	mg/L, %	21374	277	9	369303	93.9	20403	95.5
LREE	mg/L, %	2553	24	0	43977	93.6	2418	94.7
HREE	mg/L, %	18821	253	9	325326	93.9	17984	95.6

Table 25: Oxalic Acid Precipitate Calcination (C-RP3) Assay Summary

Sample & Quant.	RP-3 Precip	C-RP3 Calcine
(mL or g)	22	10
La	11002	23800
Ce	19924	43100
Pr	2094	4530
Nd	7489	16200
Sm	3467	7500
Eu	2140	4630
Gd	13961	30200
Tb	3587	7760
Dy	28985	62700
Ho	6703	14500
Y	226056	489000
Er	21404	46300
Tm	2922	6320
Yb	17243	37300
Lu	2325	5030
Sc	<12	<25
Th	0.51	1.1
U	<0.2	<0.5
Si	878	1900
Al	<23	<50
Fe	32	70
Mg	1387	3000
Ca	740	1600
Na	1433	3100
K	<37	<80
Ti	<28	<60
P	18	40
Mn	<37	<80
Zn	20	44
S (%)		0.01
TREE (%)	36.9	79.9

Add'n Elements (g/t)	
Ag	<50
As	<200
Ba	15
Be	0.90
Bi	105
Cd	<3
Co	<200
Cr	100
Li	<30
Mo	381
Nb	
Ni	99
Pb	<200
Sb	35
Se	<50
Sn	40
Sr	16.7
Ta	
Tl	<50
V	<60
Zr	
LOI (%)	0.49

Sodium carbonate precipitation and calcination (RP-4 and C-RP4) produced a final rare earth product containing 77.3% REE (equivalent to 95.0% rare earth oxides with direct assay, or 92.4% rare earth oxides based on 7.6% total oxide impurities). This represented 94% of the total rare earth elements present in the ThSX raffinate. The main impurities in the calcine included 1.7% Si, along with Ca, Na, Mg, and Mn (0.2%-0.7%). The increase in impurities when using sodium carbonate as the precipitant was expected. Other minor impurities included Mo, Ni, and Sr (~225 g/t) and Ba and Bi (~150 g/t). Some of these impurities may be removed with additional solids washing. The metallurgical balance for the precipitation test and the assays of the final calcine are available in Table 24 and Table 25, respectively.

Table 26: Sodium Carbonate Precipitation (RP-4) Summary Metallurgical Balance

Sample & Quant.	Assay Units	Feed	Final Filt	Final Wash	Final PPT	PPT % Res/Feed	Calc Head	Acc %
(mL or g)		400	432.4	371.7	17.8			
La	mg/L, %	637	0.24		13129	91.9	586	92
Ce	mg/L, %	1160	0.12		23691	91.1	1057	91
Pr	mg/L, %	124	<0.03		2457	88.4	110	88
Nd	mg/L, %	434	<0.05		8928	91.7	398	92
Sm	mg/L, %	198	<0.04		4038	91.0	180	91
Eu	mg/L, %	120	<0.03		2521	93.7	112	94
Gd	mg/L, %	800	0.16		16689	93.0	744	93
Tb	mg/L, %	200	0.05		4266	95.1	190	95
Dy	mg/L, %	1630	0.55		34428	94.2	1536	94
Ho	mg/L, %	369	0.17		7994	96.6	357	97
Y	mg/L, %	13200	16.1	3.68	280090	94.6	12513	95
Er	mg/L, %	1210	0.72		25383	93.6	1133	94
Tm	mg/L, %	164	0.12		3542	96.3	158	96
Yb	mg/L, %	993	0.73		20948	94.1	935	94
Lu	mg/L, %	135	0.11		2824	93.3	126	93
Sc	mg/L, %	0.14	<0.07		<15	464.7	1	519
Th	mg/L, %	0.04	<0.03		1.1	123.6	0	205
U	mg/L, %	<0.02	<0.02		<0.3	65.1	0	173
Si	mg/L, %				0.992	--	442	--
Al	mg/L, %	171	<0.2		0.337	87.8	150	88
Fe	mg/L, %	<0.6	<0.2		0.01	--	--	--
Mg	mg/L, %	268	167		0.21	35.0	274	102
Ca	mg/L, %	439	153		0.429	43.6	357	81
Na	mg/L, %	<30	12500		0.34	--	--	--
K	mg/L, %	<30	14		<0.005	--	--	--
Ti	mg/L, %	<4	<0.02		0.006	--	--	--
P	mg/L, %	<5	<5		0.01	--	--	--
Mn	mg/L, %	178	51.3		0.14	35.1	118	66
Cr	mg/L, %	<0.2	<0.1			--	--	--
V	mg/L, %	<0.2	<0.2			--	--	--
Cu	mg/L, %	<40	<0.1			--	--	--
Pb	mg/L, %	<9	<2			--	--	--
Zn	mg/L, %	15	0.8		0.00	0.0	1	6
TREE	mg/L, %	21374	19	4	450927	94.1	20136	94.2
LREE	mg/L, %	2553	0	0	52243	91.3	2331	91.3
HREE	mg/L, %	18821	19	4	398684	94.5	17805	94.6

Table 27: Sodium Carbonate Precipitate Calcination (C-RP4) Assay Summary

Sample & Quant.	RP-4 Precip	C-RP4 Calcine
(mL or g)	18	10
La	13129	22500
Ce	23691	40600
Pr	2457	4210
Nd	8928	15300
Sm	4038	6920
Eu	2521	4320
Gd	16689	28600
Tb	4266	7310
Dy	34428	59000
Ho	7994	13700
Y	280090	480000
Er	25383	43500
Tm	3542	6070
Yb	20948	35900
Lu	2824	4840
Sc	14.588	<25
Th	1.1	1.9
U	<0.3	<0.5
Si	9920	17000
Al	3367	5770
Fe	117	200
Mg	2101	3600
Ca	4295	7360
Na	3384	5800
K	<47	<80
Ti	58	100
P	117	200
Mn	1400	2400
Zn	0	
S (%)		0.03
TREE (%)	45.1	77.3

Add'n Elements (g/t)	
Ag	<50
As	<200
Ba	158
Be	18.9
Bi	151
Cd	<3
Co	<200
Cr	<70
Li	<30
Mo	225
Nb	
Ni	232
Pb	<200
Sb	41
Se	<50
Sn	51
Sr	214
Ta	
Tl	<50
V	<60
Zr	
LOI (%)	0.69

Conclusions and Recommendations

Based on the testwork results reported herein, the following conclusions can be made:

- A successful flowsheet was developed capable of extraction 97% REE (95% Dy and 95% Tb) into a rare earth oxide product stream containing 98% TREO (49% Y, 0.8% Tb, and 6.3% Dy). Key residual impurities included 1.1 g/t Th, < 0.3% Mg, Si, and Ca, 381 g/t Mo, 105 g/t Bi, 100 g/t Cr and 100 g/t Ni. The flowsheet consisted of acid baking of a flotation concentrate, followed by water leaching and impurity removal (to remove majority of Fe and Th) as well as some of the Al. A crude and impure rare earth precipitate was produced via carbonate precipitation, which was subsequently re-leached in sulphuric acid and the liquor further treated with IX and SX to remove U and Th. Rare earth elements were recovered from this liquor with oxalic acid addition and the final precipitate was calcined to form an oxide.
- Other flowsheet options including gangue leaching and caustic cracking were considered but rejected. Calcium-bearing gangue minerals can be readily leached (>99%) from the flotation concentrate with hydrochloric acid in advance of acid baking. However, the rare earth elements also partially leached in this step and would be lost in this flowsheet. Therefore, this approach was not adopted into the recommended flowsheet. The caustic cracking approach was also tested and ultimately not adopted. The main reason for this is that REE solubilization in the subsequent acid leaching step was poor, even after aggressive two stage caustic cracking.
- Sulphuric acid baking of the flotation concentrate at 300°C for three hours followed by water washing at 20% solids resulted in ~97% REE extraction and left most of the calcium in the leach residue as insoluble gypsum. The leach liquor contained >17 g/L TREE and ~40 g/L iron, along with lower levels of other impurities. Acid addition was about 1.2 tonne acid per tonne of concentrate.
- Neutralization of the acid bake/water leach liquor to pH 2.9 with ~400 kg/t magnesium carbonate (1 hour at 50°C) along with minimal peroxide to fully oxidize the dissolved iron to the ferric state, precipitated most of the iron and thorium (>90%), along with 40% of the aluminum. Approximately 11% of the neodymium and 9% of the yttrium co-precipitated in this test; however, these values may be reduced with further optimization of this step in the process. In addition, a two-stage counter current precipitation process is worth testing, since this may allow for more efficient impurity precipitation (particularly aluminium) with lower REE losses.
- Raising the pH of the liquor after impurity removal to 6.5 with sodium carbonate (one hour at ambient temperature) successfully precipitated all of the REE from solution. Residual impurities (Sc, Th, U, Al, Fe, Ti) also precipitated under these conditions. Soda ash consumption was ~90 kg/t.

The following stages of the proposed flowsheet were not optimized in any way due to the limited quantity of available material from preceding steps. Targets were selected based on SGS experience with similar materials. These results are presented to allow for a preliminary assessment of process potential and can be refined through further testing, which would require a much larger quantity of flotation concentrate. An

optimized bulk crude rare earth precipitation test would produce adequate feed for optimization of the following process steps.

- Re-leaching of solids from soda ash precipitation with sulphuric acid at 50°C, performed in a two-stage process, successfully redissolved the precipitated rare earth elements while leaving most of the thorium, aluminum, and iron in the leach residue. The acid required in this redissolution step is expected to be approximately 70 kg/t. This process produced a liquor containing 28 g/L total rare earth elements, 23 mg/L uranium and 2 mg/L thorium.
- Treatment of the re-leach liquor by ion exchange with Purolite A660 resin under fairly aggressive conditions (high resin:solution ratio) extracted ~100% of the uranium and 94% of the thorium, along with 15% of the REE. Optimization of this process will significantly reduce REE losses and may eliminate the co-extraction of thorium.
- Thorium solvent extraction (SX) was completed on the assumption that the thorium would not be extracted in the prior ion exchange step to remove uranium. However, in this test program, 94% of the thorium was extracted by the IX resin making the SX step redundant. Further testing of the UIX and ThSX process steps is strongly recommended to clarify the behaviour of thorium and REE. A single SX contact with the organic solvent (0.5% Primene JMT, 2.5% isodecanol, balance Aromatic 150ND) at an aqueous-to-organic phase ratio of 10:1 extracted a further 75% of the thorium without loading any REE.
- The purified solution after SX, which contained 21.4 g/L TREE, was treated to precipitate the REE, either oxalic acid or sodium carbonate. Oxalic acid is known to be more selective, but expensive, while soda ash precipitation will co-precipitate any impurities remaining in solution. The precipitates from both tests were calcined to produce rare earth oxides, although this step may not be required in practice.
 - Oxalic acid precipitation and calcination resulted in a solid sample assaying at 98.1% total rare earth oxides with very low levels of impurities, primarily Na, Mg, Si, and Ca. This process required 1145 kg/t oxalic acid (100% reagent, final REO product basis).
 - Sodium carbonate precipitation and calcination resulted in a solid sample assaying at 95.0% total rare earth oxides with moderate levels of impurities, primarily Si, Ca, Na, Al, Mg, and Mn. This process required 1010 kg/t sodium carbonate (100% reagent, rare earth carbonate product basis) or 1730 kg/t on a REO product basis.

Near the conclusion of the testwork it was revealed that the feed for the flotation process was expected to be different from that which was tested for this program. It is strongly recommended that this updated feed material is provided for flotation testing and the produced concentrate used in confirmatory hydrometallurgical testing. A much larger sample of flotation concentrate would allow for optimization of every process step and would provide better estimates of reagent demands at each stage. This change in feed may require the reintroduction of process steps that were rejected or eliminated through the testwork reported herein.

Appendix A – Test Sheets

Project: 18299-02
Client: Lofdal (NCM)

Date: Mar 26, 2021
Technologist: M Rosborough

Test: GAL1

Purpose: To leach gangue from a REE concentrate using HCl, effect of pH

Sample: Longi Combined Mags LIMS CA03074-MAR21

H&S: Review MSDS for HCl

Procedure:

1. Add the target amount of water into a suitably sized reactor. Prepare target charge weight of feed and add to water while mixing. If slurry is not mixable, add additional DI water as needed. Record additions.
2. Once pulp is mixing, begin heating to the target temperature. Begin slowly adding acid once at temperature to reach the target pH. Record all acid additions.
3. Maintain the pulp at the target pH for the target retention time at each pH point. Sample the liquor as instructed and advance to the next pH point.
4. Once the test is complete, weigh and filter the pulp.
5. Record the total weight, density, free acid, pH, and ORP of the filtrate. Collect a sample for assay.
6. Repulp and displacement wash the final residue (roughly equal to final pulp volume for repulp). Collect a sample of the wash water for assay, recording the total weight, density, free acid, etc.
7. Record the wet weight of any solids left, and dry to constant weight at 80-100°C. Record the dry weight and submit for assay.

Assays:	#	Elements	Streams
	5	liquor samples - Y, Nd, Ca, Fe	partial liquor samples, wash
	1	liquor samples - ICP, REE	final filtrate
	1	residue sample - WRA, REE by XRF	final residue

Conditions:

Feed to Add:	607.5	g
Feed Moisture:	18%	
Dry Equivalent:	500	g
Initial Pulp Density:	50%	solids
Water to Add:	392.5	g
Reagent	HCl	
Reagent Strength:	36	%
Retention Time:	0.5	h at each pH target
Temperature:	50	°C
Target Acidity:	variable	pH

Acidity Targets:

Target #	pH Target
1	3.0
2	2.0
3	1.5
4	1.0
5 (final)	25 g/L HCl

Project: 18299-02
 Client: Lofdal (NCM)

Date: Mar 26, 2021
 Technologist: M Rosborough

Test: GAL1

Test Data:

Time		Reactor			Reagents / Feed			Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	HCl 36% g	H2O g	
9:05		24.3	8.21	-295	608.23		393.16	feed addition. Heat on
9:20		52.0	7.63	-241				begin pH adjustment
9:40		59.0	4.46	-281		110.53		reacts aggressively, steam fills headspace and disperse
9:50		61.0	3.85	-150		121.84		
10:15	0.00	54.5	3.01	92		38.82		steam disipates, surface visibly bubbling
10:45	0.50	52.8	3.04	87		12.61		sample 1
11:00	0.00	51.4	2.10	570		8.75		
11:30	0.50	48.6	2.08	545		5.35		sample 2
11:40	0.00	47.8	1.45	575		3.44		
12:20	0.67	54.1	1.53	580		5.39		Sample 3 FAT= 0g/L
12:30	0.00	52.5	0.80	605		7.55		
13:00	0.50	50.2	0.87	598		1.91		Sample 4 FAT= 1.3g/L
13:10	0.00	49.9	-0.69	594		56.53		
13:40	0.50	49.7	-0.68	598		2.65	7	End test, filter FAT= 23.9g/L
Totals/Avg.		50.6	2.69		608	375	400	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
pH 3		48.39	26.39	1.162	23	313	3.17			fast	
pH 2		69.92	40.47	1.168	35	498	1.5			moderate	
pH 1.5		60.68	34.58	1.170	30	526	0.75			moderate	
pH 1		67.34	37.40	1.174	32	550	0.12			moderate	
25 g/L		1151.92	682.45	1.178	579	535	-1.33			moderate	29%
Wash			1233.65	1.015	1215	537	0.83	418.5	335.06	moderate	

Final Filtration/Washing:

Diameter of filtration paper:	150	mm
Type of Paper (Whatman #):	3	
Filtration Time:	18	minutes
Washing Time:	25	minutes
Cake Moisture:	20%	
Weight Loss:	33%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	orange
Clarity of Wash:	clear
Colour of Wash:	light yellow
Colour of Residue:	Orange/brown

Residue:

Tare:	10.9 g
Tare + Wet:	429.4 g
Tare + Dry:	346.0 g

Acid Addition: 270 kg/t

Project: 18299-02
 Client: Lofdal (NCM)

Date: Mar 26, 2021
 Technologist: M Rosborough

Test: GAL1

Metallurgical Balance		3.04	2.08	1.53	0.87	-0.7						
Sample & Quant.	Assay Units	Longi Combined Mags	pH 3 Filtrate	pH 2 Filtrate	pH 1.5 Filtrate	pH 1 Filtrate	25 g/L Filtrate	Wash	25 g/L Residue	Leaching	Account. out/in	Calc Head
(mL or g)		500	23	35	30	32	579	1215	335	%	%	
La	mg/L, g/t	426					144		171	59	66	281
Ce	mg/L, g/t	683					284		<171	74	65	444
Pr	mg/L, g/t	<256					27.6		<256	16	79	204
Nd	mg/L, g/t	257	34.5	51.4	58	67.8	99.7	8.1	<171	56	102	263
Sm	mg/L, g/t	<431					27.7		<431	10	74	321
Eu	mg/L, g/t						9.47			-	-	11
Gd	mg/L, g/t						32.0			-	-	37
Tb	mg/L, g/t	145					4.19			-	3	5
Dy	mg/L, g/t	1160					18.2			-	2	21
Ho	mg/L, g/t						2.92			-	-	3
Y	mg/L, g/t	9056	36.8	48.7	46.9	49.8	72.3	5.94	13386	1	100.3	9080
Er	mg/L, g/t						6.59			-	-	8
Tm	mg/L, g/t						0.80			-	-	1
Yb	mg/L, g/t						4.54			-	-	5
Lu	mg/L, g/t						0.60			-	-	1
Sc	mg/L, g/t						2.68			-	-	3
Th	mg/L, g/t	1494					133		2109	10	105	1567
U	mg/L, g/t	<85					1.65		<85	3	69	59
Si	mg/L, %	12.0							16.5	0	93	11.1
Al	mg/L, %	2.99					529		4.13	2	95	2.83
Fe	mg/L, %	18.7	164	241	442	903	2920	312	28.7	2	105	19.6
Mg	mg/L, %	1.00					1320		1.18	16	94	0.94
Ca	mg/L, %	12.4	72800	79400	77300	79600	72300	6340	0.2	99	96	11.9
Na	mg/L, %	1.12					39		1.51	0	90	1.01
K	mg/L, %	0.81					97		1.21	1	101	0.82
Ti	mg/L, %	0.935					5.18		1.41	0	101	0.95
P	mg/L, %	0.450					471		0.585	12	99	0.45
Mn	mg/L, %	0.24					1190		0.06	77	75	0.18
Cr	mg/L, %	0.01					10.8		0.01	12	76	0.01
V	mg/L, %	0.04					23.0		0.062	6	112	0.04
TREE	mg/L, %	1.2					735		1.5	8.0	86	1.1
LREE	mg/L, %	0.2					583		0.1	45.6	72	0.1
HREE	mg/L, %	1.0					152		1.3	2	88	0.9

Project: 18299-02
Client: Lofdal (NCM)

Date:	Mar 31, 2021
Technologist:	M Rosborough

Test: GAL2

Purpose: To leach gangue from a REE concentrate using HCl, effect of pH

Sample: Longi Combined Mags LIMS CA03074-MAR21

H&S: Review MSDS for HCl

Procedure:

1. Add the target amount of water into a suitably sized reactor. Prepare target charge weight of feed and add to water while mixing. If slurry is not mixable, add additional DI water as needed. Record additions.
2. Once pulp is mixing, begin heating to the target temperature. Begin slowly adding acid once at temperature to reach the target pH. Record all acid additions.
3. Maintain the pulp at the target pH for the target retention time. Sample the liquor at requested time intervals.
4. Once the test is complete, weigh and filter the pulp.
5. Record the total weight, density, free acid, pH, and ORP of the filtrate. Collect a sample for assay.
6. Repulp and displacement wash the final residue (roughly equal to final pulp volume for repulp). Collect a sample of the wash water for assay, recording the total weight, density, free acid, etc.
7. Record the wet weight of any solids left, and dry to constant weight at 80-100°C. Record the dry weight and submit for assay.

Assays:	#	Elements	Streams
	3	liquor samples - Y, Nd, Ca, Fe	partial liquor samples, wash
	1	liquor samples - ICP, REE	final filtrate
	1	residue sample - WRA, REE by XRF	final residue

Conditions:

Feed to Add:	607.5	g
Feed Moisture:	18%	
Dry Equivalent:	500	g
Initial Pulp Density:	50%	solids
Water to Add:	392.5	g
Reagent	HCl	
Reagent Strength:	36	%
Retention Time:	2	h
Temperature:	50	°C
Target Acidity:	3.0	pH

Acidity Targets:

Sample after 30 min, 60 min

Project: 18299-02
 Client: Lofdal (NCM)

Date: Mar 31, 2021
 Technologist: M Rosborough

Test: GAL2

Metallurgical Balance		3.10	3.01	3.03						
Sample & Quant.	Assay Units	Longi Combined Mags	30 min Filtrate	60 min Filtrate	Final Filtrate	Wash	Final Residue	Leaching	Account. out/in	Calc Head
(mL or g)		500	40	33	555	1315	353	%	%	
La	mg/L, g/t	426			60.1		400	19	82	349
Ce	mg/L, g/t	683			101		600	21	79	536
Pr	mg/L, g/t	<256			11.5		<300	6	88	225
Nd	mg/L, g/t	257	42.7	45.2	43.1	2.14	300	22	106	272
Sm	mg/L, g/t	<431			13.3		<500	4	85	368
Eu	mg/L, g/t				4.73			-	-	5
Gd	mg/L, g/t				16.6			-	-	18
Tb	mg/L, g/t	145			2.06			-	2	2
Dy	mg/L, g/t	1160			9.92			-	1	11
Ho	mg/L, g/t				1.51			-	-	2
Y	mg/L, g/t	9056	38.1	40.6	37.7	1.98	12363	1	97	8790
Er	mg/L, g/t				3.52			-	-	4
Tm	mg/L, g/t				0.43			-	-	0
Yb	mg/L, g/t				2.53			-	-	3
Lu	mg/L, g/t				0.36			-	-	0
Sc	mg/L, g/t				0.08			-	-	0
Th	mg/L, g/t	1494			0.16		2109	0	100	1491
U	mg/L, g/t	<85			0.13		<85	0	71	60
Si	mg/L, %	12.0					16.6	0	98	11.7
Al	mg/L, %	2.99			66.3		4.16	0	99	2.95
Fe	mg/L, %	18.7	89.3	99.6	129	4.7	26.4	0	100	18.7
Mg	mg/L, %	1.00			1130		1.13	14	92	0.92
Ca	mg/L, %	12.4	85500	85300	84200	5980	1.60	91	108	13.3
Na	mg/L, %	1.12			41		1.61	0	102	1.14
K	mg/L, %	0.81			145		1.15	2	102	0.83
Ti	mg/L, %	0.935			<0.02		1.31	0	99	0.93
P	mg/L, %	0.450			<5		0.62	0	98	0.44
Mn	mg/L, %	0.24			1160		0.10	65	83	0.20
Cr	mg/L, %	0.01			0.3		0.02	0	104	0.01
V	mg/L, %	0.04			<0.3		0.06	0	108	0.04
TREE	mg/L, %	1.2			308		1.4	3.2	85	1.1
LREE	mg/L, %	0.2			229		0.2	14.6	85	0.2
HREE	mg/L, %	1.0			79		1.2	1	85	0.9

Project: 18299-02
Client: Lofdal (NCM)

Date:	Apr 01, 2021
Technologist:	M Rosborough

Test: GAL3

Purpose: To leach gangue from a REE concentrate using HCl, effect of pH

Sample: Longi Combined Mags LIMS CA03074-MAR21

H&S: Review MSDS for HCl

Procedure:

1. Add the target amount of water into a suitably sized reactor. Prepare target charge weight of feed and add to water while mixing. If slurry is not mixable, add additional DI water as needed. Record additions.
2. Once pulp is mixing, begin heating to the target temperature. Begin slowly adding acid once at temperature to reach the target pH. Record all acid additions.
3. Maintain the pulp at the target pH for the target retention time at each pH point. Sample the liquor as instructed and advance to the next pH point.
4. Once the test is complete, weigh and filter the pulp.
5. Record the total weight, density, free acid, pH, and ORP of the filtrate. Collect a sample for assay.
6. Repulp and displacement wash the final residue (roughly equal to final pulp volume for repulp). Collect a sample of the wash water for assay, recording the total weight, density, free acid, etc.
7. Record the wet weight of any solids left, and dry to constant weight at 80-100°C. Record the dry weight and submit for assay.

Assays:	#	Elements	Streams
	2	liquor samples - Y, Nd, Ca, Fe	partial liquor samples, wash
	1	liquor samples - ICP, REE	final filtrate
	1	residue sample - WRA, REE by XRF	final residue

Conditions:

Feed to Add:	607.5	g
Feed Moisture:	18%	
Dry Equivalent:	500	g
Initial Pulp Density:	50%	solids
Water to Add:	392.5	g
Reagent	HCl	
Reagent Strength:	36	%
Retention Time:	1	h at each pH target
Temperature:	50	°C
Target Acidity:	variable	pH

Acidity Targets:

Target #	pH Target
1	5.0
(final)	1.5

Project: 18299-02
 Client: Lofdal (NCM)

Date: Apr 01, 2021
 Technologist: M Rosborough

Test: GAL3

Test Data:

Time		Reactor			Reagents / Feed			Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	HCl 36% g	H2O g	
7:10	-0.67	23.3	8.68	193	608		394.25	all feed added, heat on
7:30	-0.33	49.9	7.88	167				begin pH adjustment
7:50	0.00	55.1	5.00	0		23.84		at target 1
8:05	0.25	53.7	4.95	5		8.76		
8:20	0.50	51.6	4.97	-8		1.83		
8:35	0.75	50.1	5.01	20		1.34		
8:50	1.00	49.8	5.01	80		1.80		Sample 1
9:00	-1.00	54.0	4.28	48		84.29		begin pH adjustment
9:20	-0.67	56.2	3.60	90		142.72		
10:00	0.00	52.6	1.40	557		37.43		at target 2
10:15	0.25	50.6	1.36	542		1.83		
10:30	0.50	49.4	1.47	530		0.58		
11:00	1.00	51.2	1.44	530		3.34		end test, Filter
Totals/Avg.		49.8	4.23		608	308	394	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
pH 5		77.39	33.98	1.030	33	318	6.65			fast	
pH 1.5		1216.64	747.5	1.173	637	521	0.91			fast	28%
Wash			1599.25	1.010	1584	450	3.18	425.71	345.92	slow	

Final Filtration/Washing:

Diameter of filtration paper:	150	mm
Type of Paper (Whatman #):	3	
Filtration Time:	15	minutes
Washing Time:	43	minutes
Cake Moisture:	19%	
Weight Loss:	31%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	yellow
Clarity of Wash:	clear
Colour of Wash:	none
Colour of Residue:	brown

Residue:

Tare:	10.8 g
Tare + Wet:	436.5 g
Tare + Dry:	356.8 g

Acid Addition: 222 kg/t

Project: 18299-02
 Client: Lofdal (NCM)

Date: Apr 01, 2021
 Technologist: M Rosborough

Test: GAL3

Metallurgical Balance		5.01	1.4						
Sample & Quant.	Assay Units	Longi Combined Mags	pH 5 Filtrate	pH 1.5 Filtrate	Wash	pH 1.5 Residue	Leaching	Account. out/in	Calc Head
(mL or g)		500	33	637	1584	346	%	%	
La	mg/L, g/t	426		85.1		341	31	81	344
Ce	mg/L, g/t	683		143		768	26	105	714
Pr	mg/L, g/t	<256		16.6		<256	11	77	199
Nd	mg/L, g/t	257	0.29	61.5	3.23	257	33	104	267
Sm	mg/L, g/t	<431		18.3		<431	7	75	322
Eu	mg/L, g/t			6.34			-	-	8
Gd	mg/L, g/t			22.4			-	-	29
Tb	mg/L, g/t	145		2.78			-	2	4
Dy	mg/L, g/t	1160		13.0			-	1	17
Ho	mg/L, g/t			1.88			-	-	2
Y	mg/L, g/t	9056	0.1	47.1	3.29	12442	1	96	8678
Er	mg/L, g/t			4.57			-	-	6
Tm	mg/L, g/t			0.53			-	-	1
Yb	mg/L, g/t			3.12			-	-	4
Lu	mg/L, g/t			0.44			-	-	1
Sc	mg/L, g/t			0.31			-	-	0
Th	mg/L, g/t	1494		1.49		2197	0	102	1522
U	mg/L, g/t	<85		0.55		<85	1	70	59
Si	mg/L, %	12.0				17.0	0	98	11.8
Al	mg/L, %	2.99		143		4.26	1	99	2.96
Fe	mg/L, %	18.7	0.4	421	19.8	27.1	0	100	18.8
Mg	mg/L, %	1.00		1260		1.12	17	94	0.94
Ca	mg/L, %	12.4	14000	88500	5180	0.69	96	109	13.5
Na	mg/L, %	1.12		39		1.62	0	100	1.12
K	mg/L, %	0.81		80		1.19	1	102	0.83
Ti	mg/L, %	0.935		<0.02		1.35	0	100	0.93
P	mg/L, %	0.450		<5		0.642	0	99	0.44
Mn	mg/L, %	0.24		1290		0.07	77	89	0.21
Cr	mg/L, %	0.01		4.7		0.02	4	108	0.01
V	mg/L, %	0.04		1.0		0.062	0	109	0.04
TREE	mg/L, %	1.2		427		1.4	5.1	85	1.1
LREE	mg/L, %	0.2		325		0.2	22.5	89	0.2
HREE	mg/L, %	1.0		102		1.2	1	84	0.9

Project: 18299-02
Client: Lofdal (NCM)

Date: 7-Apr-21
Technologist(s): M Rosborough

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Test: CC-1

Purpose: To Conduct a Caustic Crack on a GAL Residue from Flotation Concentrate

Sample: GAL1 Residue

Grind: as received
K80= μm

Procedure:

1. Mix wet residue and required amount of 50% NaOH in Monel reactor
2. The reactor is closed without condenser and heatup to 140°C is commenced.
3. The reaction time begins when the reaction slurry is at temperature. Maintain temperature for three hours.
4. Monitor and record conditions during test. Record observations in log sheet.
6. At end of reaction time, allow slurry to cool slightly before dilution with DI for WW portion of test.
8. Heat up to 90°C and maintain for two hours.
9. At end of test, the pulp weight was determined and the solids filtered. The primary filtrate was collected and submitted as PLS.
10. The residue was re-slurried to approximately original volume and re-filtered, followed by a displacement wash. The combined washes (repulp + displacement) were submitted as WASH. Washed solids were submitted as washed residue.

CC Conditions:

	Target	Actual		
Weight of feed solids:	277	277	g	2.0 t/t NaOH
Moisture content:	0	0	%	
Wet solids weight:	277	277		
Total pulp wt:	1385	1385	g	
50% NaOH Solution	1108	1108	g	
Pulp Density:	20.0	20.0	% solids	
Leach time:	3	2	h	
Leach Temperature:	140	140	°C	
Equipment	2 L Monel reactor			

WW Conditions:

CC Pulp weight:	1385	1364	g
Target NaOH concentration	15	15.0	% NaOH (w/w)
Pulp Weight calc	3970	3949.7	g
Water (DI) to add:	2585	2586	g
Test temperature:	90	90	°C
Leach time:	2	2	h
Equipment:	2 L Monel reactor		

Assays:

#	Elements	Streams
2	liquor samples - ICP, REE, Cl	Final + Wash
2	residue sample - WRA, REE by XRF, Cl	combined feed, washed residue

Caustic Leach Data:

45

Crack Vessel Tare:	10632.5
Crack Vessel + Feed Mass:	12015.5
Crack Vessel + Final Slurry Mass:	11996.5

Time		Crack Conditions				Reagents / Feed			Comments
(24 h)	(h) elapsed	a Temp °C			RPM	a Feed g	a NaOH g	a DI g	
9:04		65.9			400	277	554	554	heat on
9:45	0	140.3			400				time=0
10:45	1	138.8			400				
11:50	2	139.0			400				heat off
									** test was mistakenly ended an hour ealry

Water Leach Data: Gross end wt: 14590 g
Reactor Tare a: 10633 g

Time		Reactor				Reagents / Feed			Comments
(24 h)	(h) elapsed	a Temp °C			RPM	Crack Res g	H ₂ O g		
11:55		139.0			400				begin dilution addition
12:00		86.3			550		2586		all water added
12:05	0	89.7			600				
13:05	1	91.8			600				
14:05	2	89.8			600				
Totals/Avg:		99.3				0	2586	0	0.0

Sampling INFO

Sample #	Weight		Volume PLS, ml	emf at room T	pH at room T	SG g/mL	Calc PLS Vol, mL	Wet res, g	Dry res, g	%H ₂ O	Colours		Filtration fst /slw	Pulp % solids
	pulp, g	PLS, g									PLS	Residue		
Final WW Pulp	3958	3524	3011	-50	12.48	1.170	3011.0				lt brown	brown	fast	
Combined Wash		3033	3018	-20	12.82	1.005	-233.0	302.4	234.1	23%	none	brown	fast	5.9

Final Filtration:

Diameter of filtration paper:	185	Washing time:	20	min
type of paper (Whatman ##):	GF	Clarity of wash:	clear	
Filtration time:	25	Volume of wash:	3000	mL
Clarity of filtrate:	clear	Colour of wash:	none	
Colour of filtrate:	light brown	Colour of solids:	brown	
Cake thickness:	~3	mm	Vt. of Assay Cut	16.74 g
			Dry Wt. Of Assay Cut	13.0 g
				CCr PLS
% Moisture	22.6			g/L NaOH
% Weightloss:	15.5			S NaOH: 162 g/L NaOH
				Wash NaOH: 5.3 g/L NaOH
				16.7 kg/t NaOH Cons

Other Notes / Observations

Metallurgical Balance

Element	Units	Feed GAL1	NaOH	WW PLS	Wash	Final res	Extraction calc head	countabil out/in	Calc Head	Extraction based on Res/Fd
Quant (mL/g)		277	1108	3011	3018	234.1				
Assay (mg/L, %, g/t)							%	%		%
La	mg/L, g/t	171		0.29	0.69	341	3.6	175	1000	
Ce	mg/L, g/t	<171		0.41	1.29	512	4.1	264	1500	
Pr	mg/L, g/t	<256		0.06	0.14	<256	1.0	86	710	
Nd	mg/L, g/t	<171		0.19	0.54	171	5.2	89	500	
Sm	mg/L, g/t	<431		<0.04	0.06	<431	0.3	85	1190	
Eu	mg/L, g/t			<0.03	<0.03				2	
Gd	mg/L, g/t			<0.03	0.04				0	
Tb	mg/L, g/t			<0.03	<0.03				2	
Dy	mg/L, g/t			<0.05	<0.05				4	
Ho	mg/L, g/t			<0.02	<0.02				1	
Y	mg/L, g/t	13386		0.05	0.13	14961	0.0	95	41215	
Er	mg/L, g/t			<0.04	<0.04				0	
Tm	mg/L, g/t			<0.04	<0.04				3	
Yb	mg/L, g/t			<0.02	<0.02				1	
Lu	mg/L, g/t			<0.03	<0.03				2	
Sc	mg/L, g/t			<0.07	<0.07				5	
Th	mg/L, g/t	2109		<0.03	0.03	2285	0.0	92	6295	
U	mg/L, g/t	<85		2.13	0.06	<85	24.9	112	310	
Si	mg/L, %	16.5				13.3	0.0	68	36.56	32
Al	mg/L, %	4.13		741	24.6	3.90	20.2	100	13.44	20
Fe	mg/L, %	28.7		70.0	0.7	32.2	0.3	95	89.06	5
Mg	mg/L, %	1.18		0.64	<0.07	1.31	0.1	94	3.62	6
Ca	mg/L, %	0.2		3.3	<0.9	0.20	2.6	87	0.57	15
Na	mg/L, %	1.51		93900	3060	1.99				
K	mg/L, %	1.21		127	4	1.22	12.1	97	3.83	15
Ti	mg/L, %	1.41		2.85	0.02	1.61	0.2	97	4.45	3
P	mg/L, %	0.585		36	<5	0.607	8.0	95	1.82	12
Mn	mg/L, %	0.06		0.47	<0.04	0.07	0.9	99	0.19	2
Cr	mg/L, %	0.01		0.2	<0.1	0.01	2.7	119	0.04	-16
V	mg/L, %	0.062		6.3	0.2	0.056	13.0	88	0.18	24
Ba	mg/L, %			0.23	0.009					
Sr	mg/L, %			0.022	0.002					
F	mg/L, %									
Cl	mg/L, g/t			9	<1	46		#####		
S	mg/L, %									

Project: 18299-02
Client: Lofdal (NCM)

Date: April 8, 2021
Technologist: M Rosborough

47

Test: CC1 AL-1

Purpose: To evaluate the CC-HCl process

Sample: CC-1 Residue (Wet)
H&S: Review MSDS for HCl

Geiger count: <1 µSv/h (feed)
<1 µSv/h (residue)

Target Grind: as is
Actual K80: µm

Procedure:

1. Prepare target weight of feed and DI water into 2 L reactor. Commence heatup. Secure concentrated HCl.
2. Add acid to pH 3 and hold for 60 minutes.
3. Add acid to 25 g/L, hold for 60 min, then add acid to 50 g/L HCl and hold for 60 min.
4. Monitor and record pH, ORP and T during test but do not leave probes hanging in tanks. Record observations, weights of sample taken, reagents/water added to/from test in log sheet.
5. At end of test, the pulp + reactor was weighed and filtered. Filter and wash times are monitored by technologist or video camera and recorded
6. The contents of the reactor was washed out onto the filter. A PLS sample was obtained and submitted for analysis. The remainder was kept wet for further testing
7. The residue was repulped using a known amount of DI water (approx the original volume).
8. The contents of the reactor was washed out onto the filter followed by 2 x 500 mL additional displacement washes.
9. Displacement wash times are monitored by technologist or video camera and recorded
10. All wash solutions were combined, weighed, SG measured and a sample submitted as final WASH.
11. A subsample of the washed filtercake was dried and weighed and submitted for analysis.

Assays:	#	Elements	Streams
	2	intermediate liquor samples - Nd, Y, Th, ICP	pH 3, 25 g/L
	2	liquor samples - Nd, Y, Th, ICP	final PLS (50 g/L) and wash
	3	residue sample - WRA (reported as elements), REE by XRF	washed residues

Conditions:	target	actual	
CC WL Wet Cake Feed Wt.:	285.7	275.0	g use available sample
CCr WL Cake %H2O:	22.6%	22.6%	
CCr WL Cake Feed Wt. Dry Eq:	221	213	g
Target % solids (before reagent):	15.0	13.60	% solids
Calculated pulp weight:	1474	1565	
Water to add:	1189	1290	g DI water
Lixiviant type:	HCl	HCl	
target final Lixiviant Concentration:	50	50.0	g/L HCl
initial Acid to add:			g HCl, 37%
% solids (after reagent):			% solids
Total Time (h):	3	3	h
Temperature (°C):	50	50.0	(temperature controlled with heating mantle)

Test Data

Time		Reactor			Reagents / Feed / Sample				Comments
(24 h)	(min) elapsed	Temp °C	pH	ORP	Feed g	H ₂ O g	HCl 37% g	Sample out g	
10:07	-23	23.3	11.09	27	275	1290			heat on
10:22	-8	47.7	10.63	59					begin pH adjustment
10:30	0	58.4	2.98	561			10.59		target 1 T=0
11:00	30	58.4	3.01	532			0.3		
11:30	60	54.0	3.00	526			0.41	65.47	sample target 1
11:38	0	53.8	0.20	683			91.16	2	acid addition, FA Check = 26.1g/L
12:05	27	51.0	0.12	663				2	FA check = 20.6g/L
12:10	32	50.7	0.06	652			21.12		acid addition, FA Check = 26.1g/L
12:38	60	50.8	0.03	645				64.76	sample target 2
12:45	0	52.1	-0.15	618			92.64	2	acid addition, FA check = 48.2g/L
12:55	10	51.5	-0.24	614			8.5		acid addition
13:15	30	49.5	-0.27	606				2	FA check = 49.9g/L
Totals/Avg.		52.5			275	1290	225	138	

Test: CC1 AL-1

Sampling INFO

Sample #	Weight		Volume PLS, mL	emf at room T	pH at room T	SG g/mL	Calc PLS Vol, mL	Wet res, g	Dry res, g	%H ₂ O	Colours		Filtration fst /slw	Pulp % solids
	pulp, g	PLS, g									PLS	Residue		
pH 3 Sample	65.5	52.7	53	557	2.74	1.0019	56.7	11.4	8.6	24%	none	brown	fast	13.2
25 g/L Sample	64.8	53.9	53	579	0.75	1.0196	56.2	9.4	7.5	21%	yellow	brown	fast	11.5
50 g/L Final	1617.6	1371.4	1330	538	0.59	1.0314	1384.0	236.0	190.2	19%	yellow	brown	fast	11.8
Wash		2059.8	2062			0.9988	2062.3				lt yellow	brown	slow	

Free Acid Data *Fill out SG data. Enter aliquot data in weight or volume basis. Enter vol of titrant. Enter type of acid (HCl, H₂SO₄ or HNO₃)*

Sample #	SG g/mL	Sample Aliquot (wght or vol based)			Aliquot mL	Titrant		Which Acid	Stoich mol/mol	MW g/mole	g/L acid	g acid
		g	mL calc	mL pipet		N	mL					
pH 3 Sample				2	2	0.2	0.00	HCl	1	36.4606	0	0.0
25 g/L Sample				2	2	0.2	7.05	HCl	1	36.4606	26	1.4
50 g/L Final				2	2	0.2	13.14	HCl	1	36.4606	48	63.7
Wash				5	5	0.2	0.88	HCl	1	36.4606	1	2.6
sum:											66.3 g	HCl (100%)

Filtration:

Diameter of filtration paper:	185	Clarity of filtrate:	clear
type of paper (Whatman #):	3	Clarity of wash:	clear
Filtration time:	9 mins	Colour of filtrate:	yellow
Washing time:	33 mins	Colour of wash:	light yellow
Volume of wash:	2000	Colour of solids:	brown
Cake thickness:	4.0 mm		

% Moisture	19%
% Weightloss:	14

Overall % Weightloss: 37 CCr-AL Residue

Other Notes / Observations

wet cut =	19.8	g
dry cut =	15.94	
total cake wet weight before assay cut =	236.0	g

Acid Consumption

Acid Added:	83.6	g HCl, 100%	Acid Addition:	393	kg/t	278	kg/t CCr Feed
Acid Left:	66.3	g HCl, 100%					
Consumed:	17.2	g HCl, 100%	Acid consumption:	81	kg/t	57	kg/t CCr Feed

Test: CC1 AL-1

Metallurgical Balance

Element	Units	CC-1 WL Res	pH 3 PLS	25 g/L PLS	50 g/L PLS	Wash	pH 3 res	25 g/L res	50 g/L res
Quant (mL/g)		213	52.6	52.9	1329.6	2062.3	8.6	7.5	190.2
Assay (mg/L, %, g/t)									
La	mg/L, g/t	341					85	<85	<85
Ce	mg/L, g/t	512					256	<171	<171
Pr	mg/L, g/t	<256					<256	<256	<256
Nd	mg/L, g/t	171	14.7	22.6	22.7	0.89	<171	<171	<171
Sm	mg/L, g/t	<431					<431	<431	<431
Y	mg/L, g/t	14961	115	138	135	3.36	14646	14883	14095
Th	mg/L, g/t	2285	0.06	242	258	6.50	2373	703	615
U	mg/L, g/t	<85	<1	<1	<1	<1	<85	<85	<85
Si	mg/L, %	13.3					13.7	13.9	13.6
Al	mg/L, %	3.90	106	380	414	12.3	3.95	3.93	3.69
Fe	mg/L, %	32.2	1.3	889	1280	46.1	32.1	32.5	33.5
Mg	mg/L, %	1.31	120	246	273	8.35	1.24	1.21	1.04
Ca	mg/L, %	0.20	183	193	183	6.3	0.086	0.079	0.071
Na	mg/L, %	1.99	843	814	753	21	1.52	1.57	1.57
K	mg/L, %	1.22	11	33	55	2	1.21	1.27	1.11
Ti	mg/L, %	1.61	0.08	153	179	4.89	1.65	1.61	1.47
P	mg/L, %	0.607	<5	<5	<5	<5	0.620	0.642	0.602
Mn	mg/L, %	0.07	5.06	11.6	12.8	0.39	0.06	0.06	0.06
Cr	mg/L, %	0.01	<0.1	1.8	2.0	<0.1	0.02	0.01	0.01
V	mg/L, %	0.056	<2	<2	<2	<2	0.05	0.062	0.056
Ba	mg/L, %		13.1	17.9	17.8	0.50			
Sr	mg/L, %		11.2	12.1	11.5	0.31			
F	mg/L, %								
Cl	mg/L, g/t	46							
S	mg/L, %								
TREE	mg/L, %	1.67	130	161	158	4	1.58	1.60	1.52
LREE	mg/L, %	0.17	15	23	23	1	0.12	0.11	0.11
HREE	mg/L, %	1.50	115	138	135	3	1.46	1.49	1.41
La+Pr+Nd+Sm	mg/L, %	0.1	15	23	23	1	0.1	0.1	0.1
TREE (ex Ce)	mg/L, %	1.6	130	161	158	4	1.6	1.6	1.5

Extraction	Accountability out/in	Calc Head
%	%	g/t, %
8	24	100
10	33	200
8	97	250
53	190	300
8	97	420
14	98	14640
77	105	2400
23	116	99
8	99	13.15
15	100	3.88
10	103	33.23
23	92	1.21
68	100	0.20
32	103	2.05
12		1.12
16	97	1.56
9	98	0.59
20	99	0.07
18	109	0.01
13	103	0.06
		0.01
		0.01

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20
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La
TF

Project: 18299-02
Client: Lofdal (NCM)

Date: 9-Apr-21
Technologist(s): M Rosborough

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Test: CC-2

Purpose: To Conduct a Caustic Crack on a CC-WW-AL Residue

Sample: CC 1 AL-1 Res

Grind: as received
K80= μm

Procedure:

1. Mix wet residue and required amount of 50% NaOH in Monel reactor
2. The reactor is closed without condenser and heatup to 140°C is commenced.
3. The reaction time begins when the reaction slurry is at temperature. Maintain temperature for three hours.
4. Monitor and record conditions during test. Record observations in log sheet.
6. At end of reaction time, allow slurry to cool slightly before dilution with DI for WW portion of test.
8. Heat up to 90°C and maintain for two hours.
9. At end of test, the pulp weight was determined and the solids filtered. The primary filtrate was collected and submitted as PLS.
10. The residue was re-slurried to approximately original volume and re-filtered, followed by a displacement wash. The combined washes (repulp + displacement) were submitted as WASH. Washed solids were submitted as washed residue.

CC Conditions:

	Target	Actual		
Weight of feed solids:	174	171	g	2.0 t/t NaOH
Moisture content:	19%	19%	%	
Wet solids weight:	216	212		
Total pulp wt:	913	909	g	
50% NaOH Solution	697	697	g	
Pulp Density:	19.1	19.7	% solids	
Leach time:	3	3	h	
Leach Temperature:	140	140	°C	
Equipment	2 L Monel reactor			

WW Conditions:

CC Pulp weight:	913	876.5	g	% NaOH (w/w)
Target NaOH concentration	15	15.3		
Pulp Weight calc	2497	2459.8	g	
Water (DI) to add:	1584	1583	g	
Test temperature:	90	90	°C	
Leach time:	2		h	
Equipment:	2 L Monel reactor			

Assays:

#	Elements	Streams
2	liquor samples - ICP, REE, Cl	Final + Wash
1	residue sample - WRA, REE by XRF, Cl	washed residue

Caustic Leach Data:

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Crack Vessel Tare:	10533
Crack Vessel + Feed Mass:	11441.5
Crack Vessel + Final Slurry Mass:	11409.5

Time (24 h)	(h) elapsed	Crack Conditions				Reagents / Feed			Comments
		a Temp °C			RPM	a Feed g	a NaOH g	a DI g	
8:35	-0.9	75.3			200		349	349	prepare NaOH solution
8:50	-0.6	76.4			290	212		12.4	feed addition, Heat on
9:26	0.0	140.5			290				time 0
11:30	2.1	139.7			290				
12:26	3.0	139.8			290				heat off begin WW

Water Leach Data: Gross end wt: 12972.5 g
Reactor Tare a: 10533 g

Time (24 h)	(h) elapsed	a Temp °C	Reactor			Reagents / Feed			Comments
					RPM	Crack Res g	H ₂ O g		
12:33	-0.2	82.1			360		1583		dilution water added, heat on
12:43	0.0	90.1			360.0				Time 0
13:43	1.0	90.0			360.0				
14:44	2.0	90.4			360				end test
Totals/Avg:		88.2				0	1583	0	0.0

Sampling INFO

Sample #	Weight		Volume PLS, mL	emf at room T	pH at room T	SG g/mL	Calc PLS Vol, mL	Wet res, g	Dry res, g	%H ₂ O	Colours		Filtration fst /slw	Pulp % solids
	pulp, g	PLS, g									PLS	Residue		
Final WW Pulp	2440	2141	1823	-37	12.47	1.174	1823.2				lt yl/brn	brown	fast	
Combined Wash		1689	1680	-30	12.70	1.005	-148.4	208.9	149.2	29%	none	brown	fast	6.1

Final Filtration:

Diameter of filtration paper:	150	Washing time:	20	min
type of paper (Whatman ##):	GF	Clarity of wash:	clear	
Filtration time:	11	Volume of wash:	2000	mL
Clarity of filtrate:	clear	Colour of wash:	none	
Colour of filtrate:	light yellow/brown	Colour of solids:	brown	
Cake thickness:	0.4	Wet Wt. of Assay Cut	13.16	g
		Dry Wt. Of Assay Cut	9.4	g
% Moisture	28.6	CCr PLS		g/L NaOH
% Weightloss:	12.5	S NaOH:	181	g/L NaOH
		Wash NaOH:	9.0	g/L NaOH
			1.6	kg/t NaOH Cons

Other Notes / Observations

Metallurgical Balance

Element	Units	Feed	NaOH	WW PLS	Wash	Final res
Quant (mL/g)		171	697	1823	1680	149.2
Assay (mg/L, %, g/t)						
La	mg/L, g/t	<85		0.08	0.06	<85
Ce	mg/L, g/t	<171		0.13	0.04	<171
Pr	mg/L, g/t	<256		<0.03	<0.03	<256
Nd	mg/L, g/t	<171		<0.06	<0.06	<171
Sm	mg/L, g/t	<431		<0.04	<0.04	<431
Eu	mg/L, g/t			<0.03	<0.03	
Gd	mg/L, g/t			<0.03	<0.03	
Tb	mg/L, g/t			<0.03	<0.03	
Dy	mg/L, g/t			<0.05	<0.05	
Ho	mg/L, g/t			<0.02	<0.02	
Y	mg/L, g/t	14095		0.06	<0.01	17560
Er	mg/L, g/t			<0.04	<0.04	
Tm	mg/L, g/t			<0.04	<0.04	
Yb	mg/L, g/t			0.02	<0.02	
Lu	mg/L, g/t			<0.03	<0.03	
Sc	mg/L, g/t			<0.07	<0.07	
Th	mg/L, g/t	615		<0.03	<0.03	615
U	mg/L, g/t	<85		0.43	<0.02	<85
Si	mg/L, %	13.6				10.3
Al	mg/L, %	3.69		806	42.1	3.49
Fe	mg/L, %	33.5		59.8	0.6	37.3
Mg	mg/L, %	1.04		1.01	<0.07	1.39
Ca	mg/L, %	0.071		4.9	<0.9	0.093
Na	mg/L, %	1.57		105000	5210	1.58
K	mg/L, %	1.11		113	5	1.35
Ti	mg/L, %	1.47		4.34	0.02	1.85
P	mg/L, %	0.602		23	<5	0.742
Mn	mg/L, %	0.062		1.02	<0.04	0.077
Cr	mg/L, %	<0.01		<0.1	<0.1	0.02
V	mg/L, %	0.056		3.8	<0.2	0.062
Ba	mg/L, %			0.46	0.008	
Sr	mg/L, %			0.018	<0.002	
F	mg/L, %					
Cl	mg/L, g/t			9	<1	100
S	mg/L, %					

Extraction calc head	Accountabil out/in	Calc Head	Extraction based on Res/Fd
%	%		%
	89	200	
	89	300	
	88	450	
	88	300	
	88	760	
		1	
		0	
		1	
		2	
		1	
	109	30898	
		0	
		2	
		1	
		1	
		3	
	88	1084	
	93	160	
0.0	67	18.18	33
22.8	107	7.95	17
0.2	98	65.72	3
0.1	118	2.45	-17
7.0	122	0.18	-14
9.6	118	2.63	-6
0.3	110	3.26	-10
4.3	113	1.36	-8
1.6	111	0.14	-9
1.1	133	0.04	-31
7.3	104	0.12	4

Project: 18299-02
Client: Lofdal (NCM)

Date: April 12, 2021
Technologist: M Rosborough

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Test: CC 2 AL-1

Purpose: To evaluate the CC-HCl process

Sample: CC-2 Residue (Wet)
H&S: Review MSDS for HCl

Geiger count: <1 µSv/h (feed)
<1 µSv/h (residue)

Target Grind: as is
Actual K80: µm

Procedure:

1. Prepare target weight of feed and DI water into 2 L reactor. Commence heatup. Secure concentrated HCl.
2. Add acid to pH 3 and hold for 60 minutes.
3. Add acid to 25 g/L, hold for 60 min, then add acid to 50 g/L HCl and hold for 60 min.
4. Monitor and record pH, ORP and T during test but do not leave probes hanging in tanks. Record observations, weights of sample taken, reagents/water added to/from test in log sheet.
5. At end of test, the pulp + reactor was weighed and filtered. Filter and wash times are monitored by technologist or video camera and recorded
6. The contents of the reactor was washed out onto the filter. A PLS sample was obtained and submitted for analysis. The remainder was kept wet for further testing
7. The residue was repulped using a known amount of DI water (approx the original volume).
8. The contents of the reactor was washed out onto the filter followed by 2 x 500 mL additional displacement washes.
9. Displacement wash times are monitored by technologist or video camera and recorded
10. All wash solutions were combined, weighed, SG measured and a sample submitted as final WASH.
11. The washed filtercake was dried and weighed and submitted for analysis.

Assays:	#	Elements	Streams
	2	intermediate liquor samples - Nd, Y, Th, ICP	pH 3, 25 g/L
	2	liquor samples - Nd, Y, Th, ICP	final PLS (50 g/L) and wash
	3	residue sample - WRA (reported as elements), REE by XRF	washed residues

Conditions:	target	actual	
CC WL Wet Cake Feed Wt.:	195.7	187.7	g use available sample
CCr WL Cake %H ₂ O:	29%	28.6%	
CCr WL Cake Feed Wt. Dry Eq:	140	134	g
Target % solids (before reagent):	15.0	13.426	% solids
Calculated pulp weight:	932	999	
Water to add:	736	811	g DI water
Lixiviant type:	HCl	HCl	
target final Lixiviant Concentration:	50	50.0	g/L HCl
initial Acid to add:			g HCl, 37%
% solids (after reagent):			% solids
Total Time (h):	3	3	h
Temperature (°C):	50	50.0	(temperature controlled with heating mantle)

Test Data

Time		Reactor			Reagents / Feed / Sample				Comments
(24 h)	(min) elapsed	Temp °C	pH	ORP	Feed g	H ₂ O g	HCl 37% g	Sample out g	
8:10	-33	23.0	10.12	133	188	811			heat on
8:40	-3	50.4	9.63	116					
8:43	0	54.1	3.01	375			3.69		target 1
9:13	30	51.9	3.00	292			0.48		
9:45	62	49.3	3.01	290			0.39	65	sample 1
9:55	0	51.4	0.31	596			58.66		acid addition target 2
10:30	35	50.1	0.08	616				2	FA check = 21.7g/L
10:35	40	50.1	0.04	610			10.49		acid addition
10:55	60	50.8	0.00	613				67	sample 2
11:06	0	51.7	-0.20	585			70.14		acid addition target 3
11:38	32	49.6	-0.33	592				2	FA check = 53.0g/L
12:06	60	51.7	-0.33	595					end test
Totals/Avg.		51.0			188	811	144	136	

Test: CC 2 AL-1

Sampling INFO

Sample #	Weight		Volume PLS, mL	emf at room T	pH at room T	SG g/mL	Calc PLS Vol, mL	Wet res, g	Dry res, g	%H ₂ O	Colours		Filtration fst /slw	Pulp % solids
	pulp, g	PLS, g									PLS	Residue		
pH 1 Sample	65.0	52.0	52	337	3.58	1.0003	56.4	11.2	8.5	24%	none	brown	fast	13.1
25 g/L Sample	67.1	55.3	54	538	0.45	1.0172	55.5	14.7	10.7	27%	yellow	brown	fast	15.9
50 g/L Final	1002.7	852.5	827	517	0.11	1.0309	864.8	147.0	111.2	24%	yellow	brn/blk	fast	11.1
Wash		1513.9	1516			0.9989	1515.5				slight yl	brn/blk	moderate	

Free Acid Data *Fill out SG data. Enter aliquot data in weight or volume basis. Enter vol of titrant. Enter type of acid (HCl, H₂SO₄ or HNO₃)*

Sample #	SG g/mL	Sample Aliquot (wght or vol based)			Aliquot		Titrant		Which Acid	Stoich mol/mol	MW g/mole	g/L acid	g acid
		g	mL calc	mL pipet	mL	N	mL						
pH 1 Sample				2	2	0.2	0.00	HCl	1	36.4606		0	0.0
25 g/L Sample				2	2	0.2	6.86	HCl	1	36.4606		25	1.4
50 g/L Final				2	2	0.2	14.25	HCl	1	36.4606		52	43.0
Wash				5	5	0.2	0.89	HCl	1	36.4606		1	2.0
sum:												44.9 g	HCl (100%)

Filtration:

Diameter of filtration paper:	150
type of paper (Whatman #):	3
Filtration time:	4 min
Washing time:	39 min
Volume of wash:	1600
Cake thickness:	0.4

Clarity of filtrate:	clear
Clarity of wash:	clear
Colour of filtrate:	yellow
Colour of wash:	slight yellow
Colour of solids:	brown+black

% Moisture	24%
% Weightloss:	20

Overall % Weightloss: 41 CCr-AL Residue

Other Notes / Observations

Acid Consumption

Acid Added:	53.5	g HCl, 100%
Acid Left:	44.9	g HCl, 100%
Consumed:	8.6	g HCl, 100%

Acid Addition:	399	kg/t	282	kg/t CCr Feed
Acid consumption:	64	kg/t	45	kg/t CCr Feed

Test: CC 2 AL-1

Metallurgical Balance

Element	Units	CC-2 WL Res	pH 1 PLS	25 g/L PLS	50 g/L PLS	Wash	pH 1 res	25 g/L res	50 g/L res
Quant (mL/g)		134	52.0	54.4	826.9	1515.5	8.5	10.7	111.2
Assay (mg/L, %, g/t)									
La	mg/L, g/t	<85					<85	<85	<85
Ce	mg/L, g/t	<171					<171	<171	<171
Pr	mg/L, g/t	<256					<256	<256	<256
Nd	mg/L, g/t	<171	1.58	5.58	5.35	0.21	<171	<171	<171
Sm	mg/L, g/t	<431					<431	<431	<431
Y	mg/L, g/t	17560	29.6	73.2	68.3	1.64	17087	17875	16457
Th	mg/L, g/t	615	<0.03	46.6	48.7	1.28	615	264	264
U	mg/L, g/t	<85	<1	<1	<1	<1	<85	<85	<85
Si	mg/L, %	10.3					9.82	10.2	10.0
Al	mg/L, %	3.49	14.1	115	150	5.9	3.43	3.58	3.41
Fe	mg/L, %	37.3	84.2	955	1330	52.8	38.6	37.9	39.0
Mg	mg/L, %	1.39	144	202	228	7.83	1.25	1.29	1.19
Ca	mg/L, %	0.093	27.8	27.6	26.6	1.4	0.093	0.10	0.086
Na	mg/L, %	1.58	586	571	524	13	1.19	1.24	1.23
K	mg/L, %	1.35	12	42	67	3	1.31	1.39	1.29
Ti	mg/L, %	1.85	0.13	671	707	17.8	1.83	1.43	1.32
P	mg/L, %	0.742	<5	<5	<5	<5	0.720	0.759	0.716
Mn	mg/L, %	0.077	9.65	12.4	12.4	0.36	0.06	0.07	0.07
Cr	mg/L, %	0.02	<0.1	2.6	2.6	<0.1	0.03	0.01	0.01
V	mg/L, %	0.062	<1	<1	<1	<1	0.062	0.062	0.062
Ba	mg/L, %		5.99	9.58	9.75	0.29			
Sr	mg/L, %		3.18	3.44	3.27	0.092			
F	mg/L, %								
Cl	mg/L, g/t	100							
S	mg/L, %								
TREE	mg/L, %	1.87	31	79	74	2	1.82	1.90	1.76
LREE	mg/L, %	0.11	2	6	5	0	0.11	0.11	0.11
HREE	mg/L, %	1.76	30	73	68	2	1.71	1.79	1.65
La+Pr+Nd+Sm	mg/L, %	0.1	2	6	5	0	0.1	0.1	0.1
TREE (ex Ce)	mg/L, %	1.9	31	79	74	2	1.8	1.9	1.7

Extraction	Accountability out/in	Calc Head
%	%	g/t, %
15	97	100
15	97	200
15	97	250
31	120	200
15	97	420
18	95	16640
64	100	610
30	119	101
15	94	9.73
18	99	3.44
17	104	38.76
26	96	1.34
32	113	0.11
35	100	1.58
18		1.31
39	98	1.81
16	96	0.71
24	98	0.08
29	78	0.02
17	100	0.06
		0.01
		0.00

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Project: 18299-02
Client: Lofdal (NCM)

Date: 26-Mar-21
Technologist: M Rosborough

Test: AB1

Purpose: To static acid bake REE concentrate prior to water leaching - heated ore and acid contact

Sample: Composite REE Concentrate (March 24, 2021)

H&S: Review MSDS for H₂SO₄, hot surfaces, use heat protective PPE

Procedure:

1. Mix feed sample with the required amount of reagent in a tared crucible. Once blended, allow to cure.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature (if required).
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 h, rabbling every 60 min, before removing from the furnace while hot.
6. Allow the sample to cool to room temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.

Conditions:	Target	Actual		Results:	
Feed Weight:	200	243.67	g @ 17.92% moisture	Crucible Tare:	652.03 g
Reagent:	H ₂ SO ₄	H ₂ SO ₄		Starting Gross Weight:	1183.3 g
Reagent Strength:	96%	96%		Starting Net Weight:	531.27 g
Reagent Dosage:	1500	1500	kg/t	Calcine Gross Weight:	985.14 g
Reagent to Add:	313	312.5	g	Calcine Net Weight:	333.11 g
Total Weight:	513	556.17	g	Weight Loss:	37%
Cure Time:	0	0.08	h	Colour:	grey
Preheat Temperature:	-	-	°C	Consistency:	soft crumbly solids
Preheat Time:	-	-	h	Pulverized (yes/no):	no
Target Temperature:	280	280	°C		
Test Time:	3	3	h		
Sample Removal Temperature:	<100	127	°C		

287.6 g acid effective dose
1380.45 kg/t effective dose
40% overall weight loss

Observations:

Stage	Time	Comments:
mix/cure	7:40	immediate large brown bubbles+heat >130°C during addition, small grey bubbles after mixing
heat	7:45	Placed in furnace at ~20°C
roast	8:28	Furnace @280°C Time=0
roast	9:30	rabble, green fuming solids fairly soft
roast	10:25	rabble, green fuming solids fairly soft
cool	11:30	set point to 0°C begin cooling
out	13:00	sample out @ 127°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 30-Mar-21
Technologist: M Rosborough

Test: WL-AB1

Purpose: To water leach an acid bake calcine

Sample: AB1 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements	Streams
3	liquor samples - Y, Nd, Fe	partial liquor samples, final wash
1	liquor samples - ICP, REE, Cl	Final PLS
1	residue sample - WRA, REE by XRF, S, Cl	washed residue

Conditions:

AB1 Feed:	200	g
H2SO4 added:	300	g
Net Calcine Weight:	333.11	g
Calcine Feed to Test:	333.11	g
Calculated Equivalent Feed:	200	g
Calculated Equivalent H2SO4:	300	g
Target % Solids (vs. Feed):	10%	
Calc. Pulp Weight:	2000	g
DI Water to add:	1667	g
Resultant Pulp Density:	17%	(relative to calcine)
Test Time:	4	h
Temperature:	25	°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 30-Mar-21
Technologist: M Rosborough

Test: WL-AB1

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
6:45	0.00	25.7	1.27	466	329	1668	550 RPM
7:00	0.25	27.1	1.05	524			all feed added while mixing, Time=0
7:15	0.50	28.6	1.00	550			
7:45	1.00	30.4	0.95	570			Sample 1, solids returned
8:15	1.50	31.4	0.91	580			
8:45	2.00	31.6	0.88	585			Sample 2, solids returned
9:45	3.00	31.1	0.84	589			
10:45	4.00	30.4	0.81	589			End test, Filter
Totals/Avg.		29.5	0.96	556.63	329	1668	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h		66.79	54.25	1.0585	51	550	1.15			moderate	
2 h		70.06	58.87	1.0784	55	562	1.07			moderate	
Final		1876.73	1633.2	1.0857	1504	566	1.03			slow	7.8%
Wash			2232.97	1.0041	2224	585	2.05	205.78	146.81	slow	

Free Acid Data:

Sample #	Aliquot mL	Titrant		Which Acid	Stoich mol/mol	MW g/mole	g/L acid	g acid
		N	mL					
1 h	5	0.2	9.00	H2SO4	2	98.1	18	0.9
2 h	5	0.2	9.46	H2SO4	2	98.1	19	1.0
Final	5	0.2	9.57	H2SO4	2	98.1	19	28.2
Wash	10	0.2	0.86	H2SO4	2	98.1	1	1.9

Final Filtration/Washing:

Diameter of filtration paper:	185	mm
Type of Paper (Whatman #):	3	
Filtration Time:	52	minutes
Washing Time:	111	minutes
Cake Moisture:	29%	
Weight Loss:	16%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	brown
Clarity of Wash:	clear
Colour of Wash:	light brown/yellow
Colour of Residue:	beige/grey**

Residue: Total

Tare:	9.9 g
Tare + Wet:	215.6 g
Tare + Dry:	156.7 g
Acid Addition	1500 kg/t
Acid Remaining	160 kg/t
Acid Consumed	1340 kg/t

Comments:

** fines beige, coarser material grey/black

Project: 18299-02
Client: Lofdal (NCM)

Date: 30-Mar-21
Technologist: M Rosborough

Test: WL-AB1

Metallurgical Balance

Sample & Quant.	Assay Units	AB1 Feed	1 h Filtrate	2 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		200	51	55	1504	2224	147
La	mg/L, g/t	426			25.2		171
Ce	mg/L, g/t	683			46.7		342
Pr	mg/L, g/t	<256			4.68		<256
Nd	mg/L, g/t	257	16.8	18.1	17.4	1.61	<171
Sm	mg/L, g/t	<431			11.1		<431
Eu	mg/L, g/t				7.80		
Gd	mg/L, g/t				51.4		
Tb	mg/L, g/t	145			13.0		25.6
Dy	mg/L, g/t	1160			109		184
Ho	mg/L, g/t				23.6		
Y	mg/L, g/t	9056	832	857	828	35	945
Er	mg/L, g/t				77.1		
Tm	mg/L, g/t				10.6		
Yb	mg/L, g/t				66.7		
Lu	mg/L, g/t				8.96		
Sc	mg/L, g/t				2.45		
Th	mg/L, g/t	1494			132		439
U	mg/L, g/t	<85			3.49		<85
Si	mg/L, %	12.0					15.3
Al	mg/L, %	2.99			1710		1.87
Fe	mg/L, %	18.7	9270	14100	15600	592	7.48
Mg	mg/L, %	1.00			1050		0.04
Ca	mg/L, %	12.4			1150		14.0
Na	mg/L, %	1.12			13		1.42
K	mg/L, %	0.81			800		0.15
Ti	mg/L, %	0.935			789		0.366
P	mg/L, %	0.450			462		0.10
Mn	mg/L, %	0.24			233		0.05
Cr	mg/L, %	0.01			10.1		<0.007
V	mg/L, %	0.04			32.1		0.02
S / SO4	mg/L, %	0.03			76000		11.8
Cl	mg/L, g/t				<1		16

Extract.	Account.	Calc
%	out/in	Head
60	74	315
58	88	602
16	87	223
56	110	284
21	93	400
-	-	59
-	-	387
84	80	117
86	82	955
-	-	178
91	86	7754
-	-	580
-	-	80
-	-	502
-	-	67
-	-	18
75	88	1315
30	104	88
0	94	11
48	89	3
70	99	19
97	82	1
8	90	11
1	94	1
85	87	1
69	92	1
83	94	0
84	87	0
66	93	0
69	57	28
39	-	19

1.30 g/L TREE in final filtrate

Project: 18299-02
Client: Lofdal (NCM)

Date: 13-Apr-21
Technologist: M Rosborough

Test: AB2

Purpose: To static acid bake Gangue Acid Leach Residue prior to water leaching - heated ore and acid contact

Sample: Blended GAL2 and GAL3 Residue

H&S: Review MSDS for H₂SO₄, hot surfaces, use heat protective PPE

Procedure:

0. Combine and homogenize GAL2 and GAL3 residue. Submit a sample for assay (see WL tab for details).
1. Mix feed sample with the required amount of reagent in a tared crucible. Once blended, allow to cure.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature (if required).
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 h, rabbling every 60 min, before removing from the furnace while hot.
6. Allow the sample to cool to room temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.

Conditions:	Target	Actual	
Feed Weight:	200	200.03	
Reagent:	H ₂ SO ₄	H ₂ SO ₄	
Reagent Strength:	96%	96%	
Reagent Dosage:	1500	1503	kg/t
Reagent to Add:	313	313.09	g
Total Weight:	513	513.12	g
Cure Time:	0		h
Preheat Temperature:	-	-	°C
Preheat Time:	-	-	h
Target Temperature:	200	200	°C
Test Time:	3	3	h
Sample Removal Temperature:	<100	121	°C

Results:

Crucible Tare:	651.73	g
Starting Gross Weight:	1164.76	g
Starting Net Weight:	513.03	g
Calcine Gross Weight:	1156.31	g
Calcine Net Weight:	504.58	g
Weight Loss:	2%	
Colour:	grey	
Consistency:	wet sticky	
Pulverized (yes/no):	no	

313 g acid effective dose
 1502.175 kg/t effective dose
 2% overall weight loss

Observations:

Stage	Time	Comments:
mix/cure	8:25	minimal reaction during mixing
heat	8:30	heat on, some foaming visible on surface of sample
roast	8:56	Furnace @ 200°C time 0
roast	10:00	rabble, some fuming very wet sticky
roast	11:00	rabble, sticky
cool	11:56	heat off
out	15:00	sample out 121°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 14-Apr-21
Technologist: M Rosborough

Test: WL-AB2

Purpose: To water leach an acid bake calcine

Sample: AB2 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements	Streams
3	liquor samples - Y, Nd, Fe	partial liquor samples, final wash
1	liquor samples - ICP, REE, Cl, SO4 (Enviro)	Final PLS, separate cut for SO4
2	residue sample - WRA, REE by XRF, S, Cl	AB Feed, washed residue

Conditions:

AB2 Feed:	200	g
H2SO4 added:	300.6	g
Net Calcine Weight:	504.6	g
Calcine Feed to Test:	504.6	g
Calculated Equivalent Feed:	200	g
Calculated Equivalent H2SO4:	301	g
Target % Solids (vs. Feed):	10%	
Calc. Pulp Weight:	2000	g
DI Water to add:	1495	g
Resultant Pulp Density:	25%	(relative to calcine)
Test Time:	4	h
Temperature:	25	°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 14-Apr-21
Technologist: M Rosborough

Test: WL-AB2

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
7:40	0.00	35.5	0.60	524	504.6	1495.7	550 RPM sample added, agitation on
7:50	0.17	36.5	0.18	559			
8:10	0.50	36.4	-0.04	577			
8:40	1.00	35.5	-0.11	581			sample 1
9:20	1.67	34.3	-0.13	580			
9:40	2.00	33.6	-0.13	579			sample 2
10:10	2.50	32.9	-0.13	578			
10:40	3.00	32.4	-0.12	577			
11:40	4.00	31.3	-0.12	575			end test
Totals/Avg.		34.3	0.00	570.00	505	1496	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h		72.39	62.79	1.1385	55	484	0.88			slow	
2 h		70.28	62.62	1.1402	55	485	0.8			slow	
Final		1869.14	1646.76	1.1389	1446	488	0.78			moderate	7.6%
Wash		1662.64	2240.54	1.0034	2233	547	1.75	196.24	142.07	moderate	

Free Acid Data:

Sample #	Aliquot	Titrant		Which Acid	Stoich mol/mol	MW g/mole	g/L acid	g acid
	mL	N	mL					
1 h	2	0.2	24.43	H2SO4	2	98.1	120	6.6
2 h	1	0.2	12.51	H2SO4	2	98.1	123	6.7
Final	1	0.2	12.35	H2SO4	2	98.1	121	175.1
Wash	5	0.2	1.97	H2SO4	2	98.1	4	8.6

Final Filtration/Washing:

Diameter of filtration paper:	185	mm
Type of Paper (Whatman #):	3	
Filtration Time:	27	minutes
Washing Time:	25	minutes
Cake Moisture:	28%	
Weight Loss:	11%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	gold
Clarity of Wash:	clear
Colour of Wash:	light yellow
Colour of Residue:	brown

Residue: Total

Tare:	12.1 g
Tare + Wet:	208.3 g
Tare + Dry:	154.2 g

Acid Addition	1503	kg/t
Acid Remaining	986	kg/t
Acid Consumed	517	kg/t

Comments:

Project: 18299-02
 Client: Lofdal (NCM)

Date: 14-Apr-21
 Technologist: M Rosborough

Test: WL-AB2

Metallurgical Balance

Sample & Quant.	Assay Units	AB2 Feed	1 h Filtrate	2 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		200	55	55	1446	2233	142
La	mg/L, g/t	341			41.9		<85
Ce	mg/L, g/t	768			86.1		<171
Pr	mg/L, g/t	<256			8.42		<256
Nd	mg/L, g/t	257	30.5	30.3	31.2	1.19	<171
Sm	mg/L, g/t	<431			16.0		<431
Eu	mg/L, g/t				9.89		
Gd	mg/L, g/t				65.7		
Tb	mg/L, g/t				17.1		
Dy	mg/L, g/t				136		
Ho	mg/L, g/t				30.6		
Y	mg/L, g/t	12363	937	960	972	30.5	5670
Er	mg/L, g/t				96.0		
Tm	mg/L, g/t				13.3		
Yb	mg/L, g/t				77.7		
Lu	mg/L, g/t				10.6		
Sc	mg/L, g/t				3.24		
Th	mg/L, g/t	2197			172		1142
U	mg/L, g/t	<85			4.63		<85
Si	mg/L, %	16.6					22.8
Al	mg/L, %	4.14			2080		3.42
Fe	mg/L, %	26.8	11300	11400	12900	352	23.0
Mg	mg/L, %	1.13			1270		0.13
Ca	mg/L, %	1.14			1010		0.086
Na	mg/L, %	1.58			21.7		2.14
K	mg/L, %	1.14			744		0.71
Ti	mg/L, %	1.31			555		1.17
P	mg/L, %	0.628			535		0.29
Mn	mg/L, %	0.085			75.2		0.02
Cr	mg/L, %	0.01			11.9		0.01
V	mg/L, %	0.06			29.4		0.05
S / SO4	mg/L, %	0.03			160000		0.34
Cl	mg/L, g/t	70			1		20

Extract.	Account.	Calc
%	out/in	Head
83	107	363
84	97	744
25	95	243
68	147	377
27	98	422
-	-	72
-	-	475
-	-	124
-	-	983
-	-	221
66	96	11917
-	-	694
-	-	96
-	-	562
-	-	77
-	-	23
61	94	2055
36	111	94
0	97	16
38	95	4
39	100	27
91	89	1
92	70	1
1	97	2
51	92	1
32	95	1
65	94	1
77	83	0
37	102	0
99	79	39
34	0	21

1.61 g/L TREE in final filtrate

Project: 18299-02
Client: Lofdal (NCM)

Date: 13-Apr-21
Technologist: M Rosborough

Test: AB3

Purpose: To static acid bake Gangue Acid Leach Residue prior to water leaching - heated ore and acid contact

Sample: Blended GAL2 and GAL3 Residue

H&S: Review MSDS for H₂SO₄, hot surfaces, use heat protective PPE

Procedure:

1. Mix feed sample with the required amount of reagent in a tared crucible. Once blended, allow to cure.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature (if required).
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 h, rabbling every 60 min, before removing from the furnace while hot.
6. Allow the sample to cool to room temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.

Conditions:	Target	Actual	
Feed Weight:	200	200.01	
Reagent:	H ₂ SO ₄	H ₂ SO ₄	
Reagent Strength:	96%	96%	
Reagent Dosage:	1000	1004	kg/t
Reagent to Add:	208	209.22	g
Total Weight:	408	409.23	g
Cure Time:	0		h
Preheat Temperature:	-	-	°C
Preheat Time:	-	-	h
Target Temperature:	280	280	°C
Test Time:	3	3	h
Sample Removal Temperature:	<100	210	°C

Results:

Crucible Tare:	669.6	g
Starting Gross Weight:	1078.53	g
Starting Net Weight:	408.93	g
Calcine Gross Weight:	980.27	g
Calcine Net Weight:	310.67	g
Weight Loss:	24%	
Colour:	grey	
Consistency:	dry crumbly	
Pulverized (yes/no):	no	

208.92 g acid effective dose

1002.766 kg/t effective dose

24% overall weight loss

Observations:

Stage	Time	Comments:
mix/cure	9:08	minimal reaction on mixing some bubbles forming on surface after
heat	9:12	heat on
roast	9:40	furnace at 280°C time 0
roast	10:40	rabble significant fuming
roast	11:40	rabble significant fuming
cool	12:40	heat off
out	15:00	sample out hot 210°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 15-Apr-21
Technologist: M Rosborough

Test: WL-AB3

Purpose: To water leach an acid bake calcine

Sample: AB3 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements	Streams
3	liquor samples - Y, Nd, Fe	partial liquor samples, final wash
1	liquor samples - ICP, REE, Cl, SO4 (Enviro)	Final PLS, separate cut for SO4
1	residue sample - WRA, REE by XRF, S, Cl	washed residue

Conditions:

AB3 Feed:	200	g
H2SO4 added:	200.8512	g
Net Calcine Weight:	310.67	g
Calcine Feed to Test:	310.67	g
Calculated Equivalent Feed:	200	g
Calculated Equivalent H2SO4:	201	g
Target % Solids (vs. Feed):	10%	
Calc. Pulp Weight:	2000	g
DI Water to add:	1689	g
Resultant Pulp Density:	16%	(relative to calcine)
Test Time:	4	h
Temperature:	25	°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 15-Apr-21
Technologist: M Rosborough

Test: WL-AB3

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
7:20	0.00	26.9	1.38	455	311	1690	550 RPM
7:30	0.17	27.7	1.04	467			sample added, agitation on
8:20	1.00	29.6	0.86	489			sample 1
8:50	1.50	32.5	0.84	492			
9:20	2.00	32.0	0.83	493			sample 2
9:50	2.50	31.6	0.82	493			
11:20	4.00	30.5	0.81	491			end test
Totals/Avg.		30.1	0.94	482.86	311	1690	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h		66.9	58.36	1.0719	54	459	1.24			fast	
2 h		70.22	60.88	1.0779	56	464	1.13			slow	
Final		1875.02	1639.74	1.0776	1522	470	1			slow	7.3%
Wash		1507.45	2488.37	1.0016	2484	522	1.69	188.91	136.54	moderate	

Free Acid Data:

Sample #	Aliquot mL	Titrant		Which Acid	Stoich mol/mol	MW g/mole	g/L acid	g acid
		N	mL					
1 h	2	0.2	2.96	H2SO4	2	98.1	15	0.8
2 h	5	0.2	7.66	H2SO4	2	98.1	15	0.8
Final	5	0.2	7.44	H2SO4	2	98.1	15	22.2
Wash	10	0.2	0.66	H2SO4	2	98.1	1	1.6

Final Filtration/Washing:

Diameter of filtration paper:	185	mm
Type of Paper (Whatman #):	3	
Filtration Time:	67	minutes
Washing Time:	21	minutes
Cake Moisture:	28%	
Weight Loss:	20%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	gold
Clarity of Wash:	clear
Colour of Wash:	light yellow
Colour of Residue:	brown/ slt purp tint

Residue: Total

Tare:	12.6 g
Tare + Wet:	201.5 g
Tare + Dry:	149.1 g

Acid Addition	1004	kg/t
Acid Remaining	127	kg/t
Acid Consumed	877	kg/t

Comments:

Project: 18299-02
Client: Lofdal (NCM)

Date: 15-Apr-21
Technologist: M Rosborough

Test: WL-AB3

Metallurgical Balance

Sample & Quant.	Assay Units	AB3 Feed	1 h Filtrate	2 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		200	54	56	1522	2484	137
La	mg/L, g/t	341			36.3		<85
Ce	mg/L, g/t	768			72.7		<171
Pr	mg/L, g/t	<256			7.52		<256
Nd	mg/L, g/t	257	28.5	29.0	28.6	1.48	<171
Sm	mg/L, g/t	<431			16.7		<431
Eu	mg/L, g/t				11.4		
Gd	mg/L, g/t				77.8		
Tb	mg/L, g/t				20.4		16.3
Dy	mg/L, g/t				168		117
Ho	mg/L, g/t				37.9		
Y	mg/L, g/t	12363	1250	1250	1230	50.2	709
Er	mg/L, g/t				118		
Tm	mg/L, g/t				16.5		
Yb	mg/L, g/t				96.9		
Lu	mg/L, g/t				13.5		
Sc	mg/L, g/t				3.24		
Th	mg/L, g/t	2197			146		1230
U	mg/L, g/t	<85			4.23		<85
Si	mg/L, %	16.6					23.6
Al	mg/L, %	4.14			2430		2.96
Fe	mg/L, %	26.8	10200	11000	12500	326	23.5
Mg	mg/L, %	1.13			1260		0.084
Ca	mg/L, %	1.14			929		0.14
Na	mg/L, %	1.58			50		2.19
K	mg/L, %	1.14			1070		0.26
Ti	mg/L, %	1.307			720		1.00
P	mg/L, %	0.628			639		0.19
Mn	mg/L, %	0.09			71.7		0.02
Cr	mg/L, %	0.01			13.7		<0.007
V	mg/L, %	0.06			35		0.04
S / SO4	mg/L, %	0.03			65000		0.47
Cl	mg/L, g/t	70			1		17

Extract.	Account.	Calc
%	out/in	Head
83	98	334
83	87	670
25	91	232
68	143	369
30	98	421
-	-	87
-	-	592
93	-	166
94	-	1358
-	-	288
96	90	11159
-	-	898
-	-	126
-	-	737
-	-	103
-	-	25
57	89	1951
36	106	90
0	97	16
48	94	4
40	99	27
94	90	1
88	70	1
2	97	2
82	87	1
44	94	1
79	98	1
77	83	0
50	95	0
98	51	17
40	0	19

1.95 g/L TREE in final filtrate

Project: 18299-02
Client: Lofdal (NCM)

Date: 14-Apr-21
Technologist: M Rosborough

Test: AB4

Purpose: To static acid bake REE concentrate prior to water leaching - heated ore and acid contact

Sample: Composite REE Concentrate (March 24, 2021)

H&S: Review MSDS for H₂SO₄, hot surfaces, use heat protective PPE

Procedure:

1. Mix feed sample with the required amount of reagent in a tared crucible. Once blended, allow to cure.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature (if required).
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 h, rabbling every 60 min, before removing from the furnace while hot.
6. Allow the sample to cool to room temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.

Conditions:	Target	Actual		Results:	
Feed Weight:	200	243.79	g @ 17.92% moisture	Crucible Tare:	651.7 g
Reagent:	H ₂ SO ₄	H ₂ SO ₄		Starting Gross Weight:	1154.36 g
Reagent Strength:	96%	96%		Starting Net Weight:	502.66 g
Reagent Dosage:	1250	1247	kg/t	Calcine Gross Weight:	1047.33 g
Reagent to Add:	260	259.97	g	Calcine Net Weight:	395.63 g
Total Weight:	460	503.76	g	Weight Loss:	21%
Cure Time:	0		h	Colour:	grey
Preheat Temperature:	-	-	°C	Consistency:	crumbly
Preheat Time:	-	-	h	Pulverized (yes/no):	no
Target Temperature:	200	200	°C		
Test Time:	3	3	h		
Sample Removal Temperature:	<100	116	°C		

258.87 g acid effective dose
1241.937 kg/t effective dose
21% overall weight loss

Observations:

Stage	Time	Comments:
mix	7:40	significant off gassing/bubbling heat generation. Brown large bubbles to grey paste
heat	7:45	furnace on
roast	8:07	furnace at 200°C time 0
roast	9:25	rabble, grey+green soft solid some fuming
roast	10:10	rabble, grey+green soft solid some fuming
cool	11:10	heat off
out	13:58	sample out hot 116°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 16-Apr-21
Technologist: M Rosborough

Test: WL-AB4

Purpose: To water leach an acid bake calcine

Sample: AB4 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements	Streams
3	liquor samples - Y, Nd, Fe	partial liquor samples, final wash
1	liquor samples - ICP, REE, Cl, SO4 (Enviro)	Final PLS, separate cut for SO4
1	residue sample - WRA, REE by XRF, S, Cl	washed residue

Conditions:

AB4 Feed:	200	g
H2SO4 added:	249.5712	g
Net Calcine Weight:	395.63	g
Calcine Feed to Test:	395.63	g
Calculated Equivalent Feed:	200	g
Calculated Equivalent H2SO4:	250	g
Target % Solids (vs. Feed):	10%	
Calc. Pulp Weight:	2000	g
DI Water to add:	1604	g
Resultant Pulp Density:	20%	(relative to calcine)
Test Time:	4	h
Temperature:	25	°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 16-Apr-21
Technologist: M Rosborough

Test: WL-AB4

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
7:43	0.00	32.8	1.20	480	395	1605	550 RPM agitation on
7:58	0.25	35.0	0.53	534			
8:13	0.50	35.6	0.45	545			
8:43	1.00	35.8	0.37	555			sample 1
9:13	1.50	32.9	0.34	558			
9:43	2.00	32.5	0.33	558			sample 2
10:43	3.00	31.7	0.31	557			
11:45	4.03	30.2	0.31	554			end test
Totals/Avg.		33.3	0.48	542.63	395	1605	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h		69.3	58.01	1.1087	52	519	0.68			slow	
2 h		71.56	62.32	1.1107	56	521	0.64			slow	
Final		1871.25	1611.01	1.1097	1452	524	0.6			slow	7.8%
Wash		1481.28	2439.04	1.0053	2426	562	1.63	213.81	146.55	slow	

Free Acid Data:

Sample #	Aliquot	Titrant		Which Acid	Stoich mol/mol	MW g/mole	g/L acid	g acid
	mL	N	mL					
1 h	2	0.2	9.51	H2SO4	2	98.1	47	2.4
2 h	2	0.2	9.39	H2SO4	2	98.1	46	2.6
Final	2	0.2	9.15	H2SO4	2	98.1	45	65.1
Wash	10	0.2	1.96	H2SO4	2	98.1	2	4.7

Final Filtration/Washing:

Diameter of filtration paper:	185	mm
Type of Paper (Whatman #):	3	
Filtration Time:	45	minutes
Washing Time:	90	minutes
Cake Moisture:	31%	
Weight Loss:	14%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	amber
Clarity of Wash:	clear
Colour of Wash:	light yellow
Colour of Residue:	grey

Residue: Total

Tare:	12.3 g
Tare + Wet:	226.1 g
Tare + Dry:	158.8 g

Acid Addition	1248	kg/t
Acid Remaining	374	kg/t
Acid Consumed	874	kg/t

Comments:

Project: 18299-02
 Client: Lofdal (NCM)

Date: 16-Apr-21
 Technologist: M Rosborough

Test: WL-AB4

Metallurgical Balance

Sample & Quant.	Assay Units	AB4 Feed	1 h Filtrate	2 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		200	52	56	1452	2426	147
La	mg/L, g/t	426			26.5		171
Ce	mg/L, g/t	683			45.5		427
Pr	mg/L, g/t	<256			4.52		<256
Nd	mg/L, g/t	257	17.2	16.4	15.9	1.54	171
Sm	mg/L, g/t	<431			7.58		<431
Eu	mg/L, g/t				4.58		
Gd	mg/L, g/t				30.4		
Tb	mg/L, g/t				7.7		
Dy	mg/L, g/t				60.4		
Ho	mg/L, g/t				13.4		
Y	mg/L, g/t	9056	470	470	450	20.8	6221
Er	mg/L, g/t				42.4		
Tm	mg/L, g/t				5.92		
Yb	mg/L, g/t				35.4		
Lu	mg/L, g/t				4.92		
Sc	mg/L, g/t				2.21		
Th	mg/L, g/t	1494			141		352
U	mg/L, g/t	<85			3.41		<85
Si	mg/L, %	12.0					15.8
Al	mg/L, %	2.99			1550		2.28
Fe	mg/L, %	18.7	16800	18200	20300	681	4.77
Mg	mg/L, %	1.00			1130		0.078
Ca	mg/L, %	12.4			1210		13.5
Na	mg/L, %	1.12			<5		1.49
K	mg/L, %	0.81			655		0.36
Ti	mg/L, %	0.935			591		0.599
P	mg/L, %	0.450			309		0.27
Mn	mg/L, %	0.24			252		0.05
Cr	mg/L, %	0.01			8.7		<0.007
V	mg/L, %	0.04			32.3		0.02
S / SO4	mg/L, %	0.03			96000		11.1
Cl	mg/L, g/t				4		14

Extract.	Account. out/in	Calc Head
%	%	
61	74	317
51	94	643
15	86	221
53	105	269
15	86	371
-	-	33
-	-	221
-	-	56
-	-	438
-	-	97
45	92	8332
-	-	308
-	-	43
-	-	257
-	-	36
-	-	16
80	86	1281
28	102	87
0	97	12
40	94	3
83	107	20
93	88	1
8	87	11
0	98	1
65	91	1
49	93	1
53	94	0
84	90	0
66	91	0
74	77	31
74	-	39

0.76 g/L TREE in final filtrate

Project: 18299-02
Client: Lofdal (NCM)

Date: 22-Apr-21
Technologist: M Rosborough

Test: AB5

Purpose: To static acid bake REE concentrate prior to water leaching - heated ore and acid contact

Sample: Composite REE Concentrate (March 24, 2021)

H&S: Review MSDS for H₂SO₄, hot surfaces, use heat protective PPE

Procedure:

1. Mix feed sample with the required amount of reagent in a tared crucible. Once blended, allow to cure.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature (if required).
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 h, rabbling every 60 min, before removing from the furnace while hot.
6. Allow the sample to cool to room temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.

Conditions:

	Target	Actual	
Feed Weight:	200	243.70	
Reagent:	H ₂ SO ₄	H ₂ SO ₄	
Reagent Strength:	96%	96%	
Reagent Dosage:	1000	1001	kg/t
Reagent to Add:	208	208.64	g
Total Weight:	408	452.34	g
Cure Time:	0	0.08	h
Preheat Temperature:	-	-	°C
Preheat Time:	-	-	h
Target Temperature:	250	250	°C
Test Time:	3		h
Sample Removal Temperature:	<100	139	°C

Results:

Crucible Tare:	651.62	g
Starting Gross Weight:	1080.86	g
Starting Net Weight:	429.24	g
Calcine Gross Weight:	974.95	g
Calcine Net Weight:	323.33	g
Weight Loss:	25%	
Colour:	beige	
Consistency:	crumbly	
Pulverized (yes/no):	no	

185.54 g acid effective dose

890.4631 kg/t effective dose

29% overall weight loss

Observations:

Stage	Time	Comments:
mix	7:29	significant off gassing/bubbling heat generation. Brown large bubbles to grey paste
heat	7:34	into furnace 21°C heat on
roast	8:05	Furnace at 250°C time 0
roast	9:00	rabble, fuming yellow/green/grey solids
roast	10:07	rabble, fuming
cool	11:05	Furnace off
out	12:35	sample out 139°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 23-Apr-21
Technologist: M Rosborough

Test: WL-AB5

Purpose: To water leach an acid bake calcine

Sample: AB5 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements	Streams
3	liquor samples - Y, Nd, Fe	partial liquor samples, final wash
1	liquor samples - ICP, REE, Cl, SO4 (Enviro)	Final PLS, separate cut for SO4
1	residue sample - WRA, REE by XRF, S, Cl	washed residue

Conditions:

AB5 Feed:	200	g
H2SO4 added:	200.2944	g
Net Calcine Weight:	323.33	g
Calcine Feed to Test:	323.33	g
Calculated Equivalent Feed:	200	g
Calculated Equivalent H2SO4:	200	g
Target % Solids (vs. Feed):	10%	
Calc. Pulp Weight:	2000	g
DI Water to add:	1677	g
Resultant Pulp Density:	16%	(relative to calcine)
Test Time:	4	h
Temperature:	25	°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 23-Apr-21
Technologist: M Rosborough

Test: WL-AB5

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
7:50	0.00	25.7	1.36	559	223	1677	550 RPM agitation on
8:05	0.25	28.3	1.04	575			
8:20	0.50	29.5	0.96	583			
8:50	1.00	30.4	0.88	589			sample 1
9:20	1.50	30.4	0.85	591			
9:50	2.00	30.0	0.82	591			sample 2
10:20	2.50	29.6	0.80	590			
11:50	4.00	28.7	0.77	588			end test
Totals/Avg.		29.1	0.94	583.25	223	1677	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h		63.53	54.39	1.0767	51	569	1.44			slow	
2 h		69.21	60.2	1.0818	56	573	1.35			slow	
Final		1882.48	1632.93	1.082	1509	571	1.34			moderate	7.9%
Wash		1557.8	2304.54	1.004	2295	593	2.35	206.22	149.50	slow	

Free Acid Data:

Sample #	Aliquot	Titrant		Which Acid	Stoich mol/mol	MW g/mole	g/L	g acid
	mL	N	mL				acid	
1 h	2	0.2	1.76	H2SO4	2	98.1	9	0.4
2 h	2	0.2	1.92	H2SO4	2	98.1	9	0.5
Final	2	0.2	1.87	H2SO4	2	98.1	9	13.8
Wash	10	0.2	0.34	H2SO4	2	98.1	0	0.8

Final Filtration/Washing:

Diameter of filtration paper:	185	mm
Type of Paper (Whatman #):	3	
Filtration Time:	25	minutes
Washing Time:	91	minutes
Cake Moisture:	28%	
Weight Loss:	23%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	brown
Clarity of Wash:	clear
Colour of Wash:	light orange
Colour of Residue:	grey

Residue: Total

Tare:	12.0 g
Tare + Wet:	218.3 g
Tare + Dry:	161.5 g

Acid Addition	1001	kg/t
Acid Remaining	78	kg/t
Acid Consumed	924	kg/t

Comments:

Project: 18299-02
 Client: Lofdal (NCM)

Date: 23-Apr-21
 Technologist: M Rosborough

Test: WL-AB5

Metallurgical Balance

Sample & Quant.	Assay Units	AB5 Feed	1 h Filtrate	2 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		200	51	56	1509	2295	150
La	mg/L, g/t	426			24.4		171
Ce	mg/L, g/t	683			42.7		342
Pr	mg/L, g/t	<256			4.25		<256
Nd	mg/L, g/t	257	14.3	14.9	14.7	1.74	171
Sm	mg/L, g/t	<431			8.93		<431
Eu	mg/L, g/t				5.84		
Gd	mg/L, g/t				38.9		
Tb	mg/L, g/t				10.0		
Dy	mg/L, g/t				81.7		
Ho	mg/L, g/t				18.2		
Y	mg/L, g/t	9056	617	625	613	27.6	4331
Er	mg/L, g/t				58.6		
Tm	mg/L, g/t				8.00		
Yb	mg/L, g/t				50.0		
Lu	mg/L, g/t				6.80		
Sc	mg/L, g/t				2.24		
Th	mg/L, g/t	1494			137		439
U	mg/L, g/t	<85			3.24		<85
Si	mg/L, %	12.0					15.5
Al	mg/L, %	2.99			1460		2.25
Fe	mg/L, %	18.7	17100	17900	18300	668	5.71
Mg	mg/L, %	1.00			1080		0.096
Ca	mg/L, %	12.4			1020		13.4
Na	mg/L, %	1.12			--		1.47
K	mg/L, %	0.81			603		0.38
Ti	mg/L, %	0.935			489		0.677
P	mg/L, %	0.450			323		0.23
Mn	mg/L, %	0.24			232		0.06
Cr	mg/L, %	0.01			8.0		<0.007
V	mg/L, %	0.04			29.9		0.02
S / SO4	mg/L, %	0.03			59000		11.0
Cl	mg/L, g/t				2		20

Extract.	Account.	Calc
%	out/in	Head
59	73	312
56	85	577
14	87	224
52	104	267
17	90	390
-	-	44
-	-	294
-	-	75
-	-	616
-	-	137
62	94	8509
-	-	442
-	-	60
-	-	377
-	-	51
-	-	17
76	91	1362
28	104	88
0	97	12
40	93	3
78	105	20
92	89	1
7	87	11
0	98	1
61	91	1
42	94	1
59	93	0
79	92	0
57	100	0
64	70	23
50	-	30

0.99 g/L TREE in final filtrate

Project: 18299-02
Client: Lofdal (NCM)

Date: 17-May-21
Technologist: M Rosborough

Test: AB6

Purpose: To static acid bake REE concentrate prior to water leaching - heated ore and acid contact

Sample: Composite REE Concentrate (May 14, 2021) (Conc 2)

H&S: Review MSDS for H₂SO₄, hot surfaces, use heat protective PPE

Procedure:

1. Mix feed sample with the required amount of reagent in a tared crucible. Once blended, allow to cure.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature (if required).
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 h, rabbling every 60 min, before removing from the furnace while hot.
6. Allow the sample to cool to room temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.

Conditions:	Target	Actual	
Feed Weight:	200	200.44	
Reagent:	H ₂ SO ₄	H ₂ SO ₄	
Reagent Strength:	96%	96%	
Reagent Dosage:	1500	1499	kg/t
Reagent to Add:	313	313.04	g
Total Weight:	513	513.48	g
Cure Time:	0		h
Preheat Temperature:	-	-	°C
Preheat Time:	-	-	h
Target Temperature:	280	280	°C
Test Time:	3	3	h
Sample Removal Temperature:	<100	108	°C

Results:

Crucible Tare:	669.48	g
Starting Gross Weight:	1170.51	g
Starting Net Weight:	501.03	g
Calcine Gross Weight:	1012.04	g
Calcine Net Weight:	342.56	g
Weight Loss:	32%	
Colour:	grey	
Consistency:	crumbly	
Pulverized (yes/no):	no	

300.59 g acid effective dose

1439.665 kg/t effective dose

33% overall weight loss

Observations:

Stage	Time	Comments:
mix	8:40	significant foaming and heat generation during acid addition/mixing
heat	8:47	furnace SP to 280°C sample in
roast	9:15	furnace @ 280°C
roast	10:15	rabble, significant fuming
roast	11:20	rabble, significant fuming
cool	12:15	heat off
out	14:40	sample out 108°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 18-May-21
Technologist: M Rosborough

Test: WL-AB6

Purpose: To water leach an acid bake calcine

Sample: AB6 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements	Streams
3	liquor samples - Y, Nd, Fe	partial liquor samples, final wash
1	liquor samples - ICP, REE, Cl, SO4 (Enviro)	Final PLS, separate cut for SO4
1	residue sample - WRA, REE by ICP, S, Cl	washed residue

Conditions:

AB6 Feed:	200	g
H2SO4 added:	300.5184	g
Net Calcine Weight:	342.56	g
Calcine Feed to Test:	342.56	g
Calculated Equivalent Feed:	200	g
Calculated Equivalent H2SO4:	301	g
Target % Solids (vs. Feed):	10%	
Calc. Pulp Weight:	2000	g
DI Water to add:	1657	g
Resultant Pulp Density:	17%	(relative to calcine)
Test Time:	4	h
Temperature:	25	°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 18-May-21
Technologist: M Rosborough

Test: WL-AB6

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
8:08	0.00	32.3	1.22	550	343	1657	550 RPM agitation on
8:38	0.50	33.0	0.94	571			
9:08	1.00	32.8	0.91	573			sample 1
9:38	1.50	32.3	0.89	573			
10:08	2.00	31.9	0.87	573			sample 2
12:08	4.00	30.2	0.84	571			end test, filter
Totals/Avg.		32.1	0.95	568.50	343	1657	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h	1	72.89	61.34	1.0944	56	548	1.08			slow	
2 h	2	72.86	62.47	1.096	57	550	1.04			slow	
Final	4	1869.68	1598.37	1.0974	1457	550	0.97			slow	7.2%
Wash			1794.75	1.0069	1782	583	1.44	221.92	135.53	slow	

Free Acid Data:

Sample #	Aliquot	Titrant		Which Acid	Stoich mol/mol	MW g/mole	g/L acid	g acid
	mL	N	mL					
1 h	2	0.2	6.02	H2SO4	2	98.1	30	1.7
2 h	2	0.2	6.10	H2SO4	2	98.1	30	1.7
Final	2	0.2	6.24	H2SO4	2	98.1	31	44.6
Wash	10	0.2	2.44	H2SO4	2	98.1	2	4.3

Final Filtration/Washing:

Diameter of filtration paper:	150	mm
Type of Paper (Whatman #):	3	
Filtration Time:	52	minutes
Washing Time:	66	minutes
Cake Moisture:	39%	
Weight Loss:	19%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	brown
Clarity of Wash:	clear
Colour of Wash:	gold
Colour of Residue:	grey

Residue: Total

Tare:	10.7 g
Tare + Wet:	232.6 g
Tare + Dry:	146.3 g

Acid Addition	1503	kg/t
Acid Remaining	261	kg/t
Acid Consumed	1242	kg/t

Comments:

Project: 18299-02
Client: Lofdal (NCM)

Date: 18-May-21
Technologist: M Rosborough

Test: WL-AB6

Metallurgical Balance

Sample & Quant.	Assay Units	AB6 Feed	1 h Filtrate	2 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		200	56	57	1457	1782	136
La	mg/L, g/t	1950			172		90
Ce	mg/L, g/t	3980			336		148
Pr	mg/L, g/t	338			34.0		17.6
Nd	mg/L, g/t	1330	144	147	127	11.1	75
Sm	mg/L, g/t	725			57.4		58.7
Eu	mg/L, g/t	435			36.60		30.3
Gd	mg/L, g/t	2380			235		138
Tb	mg/L, g/t	729			61.0		33.8
Dy	mg/L, g/t	4860			459		243
Ho	mg/L, g/t	1150			106		53.3
Y	mg/L, g/t	38900	4130	4200	3560	259	945
Er	mg/L, g/t	3560			328		161
Tm	mg/L, g/t	494			47.3		24.5
Yb	mg/L, g/t	3100			281		173
Lu	mg/L, g/t	455			36.4		27.1
Sc	mg/L, g/t	38			3.15		<25
Th	mg/L, g/t	8240			47.3		2540
U	mg/L, g/t	154			12.0		25.2
Si	mg/L, %	10.0					14.1
Al	mg/L, %	1.66			415		1.77
Fe	mg/L, %	22.8	15000	15400	14300	959	13.0
Mg	mg/L, %	0.25			178		0.08
Ca	mg/L, %	6.58			957		6.93
Na	mg/L, %	0.979			28		1.33
K	mg/L, %	0.15			95		0.07
Ti	mg/L, %	3.84			2380		2.33
P	mg/L, %	2.03			1300		1.30
Mn	mg/L, %	0.39			294		0.14
Cr	mg/L, %	0.03			17.8		0.007
V	mg/L, %	0.10			67.9		0.04
S / SO4	mg/L, %	0.11			92000		8.16
Cl	mg/L, g/t	62			3		12

5.88 g/L TREE in final filtrate

Extract.	Account.	Calc	Si-tie
%	out/in	Head	%
95	67	1314	97%
96	64	2547	97%
95	77	260	96%
96	87	1157	96%
91	63	458	94%
93	66	287	95%
95	76	1805	96%
95	64	467	97%
95	72	3507	96%
96	70	808	97%
98	80	31229	98%
96	70	2498	97%
95	73	361	96%
95	70	2164	96%
94	62	283	96%
58	105	40	53%
17	25	2066	78%
84	68	104	88%
0	96	10	
20	90	2	25%
58	92	21	60%
71	74	0	78%
13	82	5	25%
2	94	1	4%
61	76	0	69%
52	86	3	57%
52	90	2	55%
69	80	0	74%
100	64	0	82%
62	79	0	69%
80	57	28	
73	48	30	

Project: 18299-02
Client: Lofdal (NCM)

Date: 17-May-21
Technologist: M Rosborough

Test: AB7

Purpose: To static acid bake REE concentrate prior to water leaching - heated ore and acid contact

Sample: Composite REE Concentrate (May 14, 2021) **(Conc 2)**

H&S: Review MSDS for H₂SO₄, hot surfaces, use heat protective PPE

Procedure:

1. Mix feed sample with the required amount of reagent in a tared crucible. Once blended, allow to cure.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature (if required).
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 h, rabbling every 60 min, before removing from the furnace while hot.
6. Allow the sample to cool to room temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.

Conditions:	Target	Actual	
Feed Weight:	200	200.08	
Reagent:	H ₂ SO ₄	H ₂ SO ₄	
Reagent Strength:	96%	96%	
Reagent Dosage:	1500	1502	kg/t
Reagent to Add:	313	313.13	g
Total Weight:	513	513.21	g
Cure Time:	0		h
Preheat Temperature:	-	-	°C
Preheat Time:	-	-	h
Target Temperature:	300		°C
Test Time:	3		h
Sample Removal Temperature:	<100		°C

Results:

Crucible Tare:	651.53	g
Starting Gross Weight:	1152.9	g
Starting Net Weight:	501.37	g
Calcine Gross Weight:	961.37	g
Calcine Net Weight:	309.84	g
Weight Loss:	38%	
Colour:	grey	
Consistency:	crumbly	
Pulverized (yes/no):	no	

301.29 g acid effective dose

1445.614 kg/t effective dose

40% overall weight loss

Observations:

Stage	Time	Comments:
mix	7:19	significant foaming and heat generation during acid addition/mixing
heat	7:23	furnace SP to 300°C sample in
roast	7:50	furnace @ 300°C
roast	8:50	rabble, significant fuming
roast	9:50	rabble, significant fuming
cool	10:51	heat off
out	14:50	sample out 88°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 19-May-21
Technologist: M Rosborough

Test: WL-AB7

Purpose: To water leach an acid bake calcine

Sample: AB7 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements	Streams
3	liquor samples - Y, Nd, Fe	partial liquor samples, final wash
1	liquor samples - ICP, REE, Cl, SO4 (Enviro)	Final PLS, separate cut for SO4
1	residue sample - WRA, REE by ICP, S, Cl	washed residue

Conditions:

AB7 Feed:	200	g
H2SO4 added:	300.6048	g
Net Calcine Weight:	309.84	g
Calcine Feed to Test:	309.84	g
Calculated Equivalent Feed:	200	g
Calculated Equivalent H2SO4:	301	g
Target % Solids (vs. Feed):	10%	
Calc. Pulp Weight:	2000	g
DI Water to add:	1690	g
Resultant Pulp Density:	15%	(relative to calcine)
Test Time:	4	h
Temperature:	25	°C

Project: 18299-02
 Client: Lofdal (NCM)

Date: 19-May-21
 Technologist: M Rosborough

Test: WL-AB7

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
9:15	0.00	23.9	1.56	553	310	1693	550 RPM agitation on
9:30	0.25	26.0	1.27	567			
10:15	1.00	28.0	1.12	592			sample 1
11:15	2.00	28.3	1.04	597			sample 2
12:15	3.00	27.7	1.00	597			
13:15	4.00	27.2	1.00	596			end test, filter
Totals/Avg.		26.9	1.17	583.67	310	1693	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h	1	65.55	55.22	1.0667	52	550	1.43			fast	
2 h	2	63.44	53.38	1.0765	50	553	1.37			fast	
Final	4	1889.51	1607.85	1.0822	1486	552	1.30			moderate	7.5%
Wash			1733.73	1.0068	1722	565	2.01	240.45	142.65	slow	

Free Acid Data:

Sample #	Aliquot	Titrant		Which Acid	Stoich mol/mol	MW g/mole	g/L acid	g acid
	mL	N	mL					
1 h	2	0.2	1.76	H2SO4	2	98.1	9	0.4
2 h	2	0.2	1.99	H2SO4	2	98.1	10	0.5
Final	2	0.2	1.98	H2SO4	2	98.1	10	14.4
Wash	10	0.2	0.99	H2SO4	2	98.1	1	1.7

Final Filtration/Washing:

Diameter of filtration paper:	150	mm
Type of Paper (Whatman #):	3	
Filtration Time:	27	minutes
Washing Time:	45	minutes
Cake Moisture:	41%	
Weight Loss:	19%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	brown
Clarity of Wash:	clear
Colour of Wash:	gold
Colour of Residue:	grey

Residue: Total

Tare:	10.6 g
Tare + Wet:	251.0 g
Tare + Dry:	153.2 g

Acid Addition	1503	kg/t
Acid Remaining	85	kg/t
Acid Consumed	1418	kg/t

Comments:

Project: 18299-02
 Client: Lofdal (NCM)

Date: 19-May-21
 Technologist: M Rosborough

Test: WL-AB7

Metallurgical Balance

Sample & Quant.	Assay Units	AB7 Feed	1 h Filtrate	2 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		200	52	50	1486	1722	143
La	mg/L, g/t	1950			196		100
Ce	mg/L, g/t	3980			384		165
Pr	mg/L, g/t	338			39.0		18.9
Nd	mg/L, g/t	1330	145	144	145	12.1	80
Sm	mg/L, g/t	725			64.9		58.4
Eu	mg/L, g/t	435			40.5		30.5
Gd	mg/L, g/t	2380			263		141
Tb	mg/L, g/t	729			69.0		34.8
Dy	mg/L, g/t	4860			540		249
Ho	mg/L, g/t	1150			123		55.0
Y	mg/L, g/t	38900	4320	4340	4350	315.0	1102
Er	mg/L, g/t	3560			381		169
Tm	mg/L, g/t	494			54.3		26.5
Yb	mg/L, g/t	3100			308		188
Lu	mg/L, g/t	455			41.5		28.8
Sc	mg/L, g/t	38			3.46		<25
Th	mg/L, g/t	8240			645		2420
U	mg/L, g/t	154			13.8		24.3
Si	mg/L, %	10.0					12.9
Al	mg/L, %	1.66			435		1.65
Fe	mg/L, %	22.8	11500	14200	15500	1100	12.2
Mg	mg/L, %	0.25			187		0.08
Ca	mg/L, %	6.58			988		6.35
Na	mg/L, %	0.979			52		1.19
K	mg/L, %	0.15			98		0.083
Ti	mg/L, %	3.84			2280		2.35
P	mg/L, %	2.03			223		2.06
Mn	mg/L, %	0.39			298		0.15
Cr	mg/L, %	0.03			18.3		0.007
V	mg/L, %	0.10			65.7		0.04
S / SO4	mg/L, %	0.11			66000		7.98
Cl	mg/L, g/t	62			2		12

Extract.	Account. out/in	Calc Head	Si-tie
%	%		%
95	78	1527	96%
96	75	2970	97%
96	90	303	96%
96	99	1312	95%
92	72	524	94%
93	74	323	95%
95	86	2054	95%
95	74	537	96%
96	86	4189	96%
96	83	953	96%
98	98	38007	98%
96	83	2951	96%
96	85	422	96%
94	78	2422	95%
94	72	329	95%
59	115	44	49%
74	79	6518	77%
86	78	120	88%
0	92	9	
22	90	2	23%
60	96	22	59%
71	79	0	75%
14	80	5	25%
4	91	1	5%
55	88	0	57%
50	88	3	53%
10	80	2	21%
68	84	0	71%
			81%
60	80	0	66%
74	45	22	
63	38	23	

7.00 g/L TREE in final filtrate

Project: 18299-02
Client: Lofdal (NCM)

Date:
Technologist:

Test: AB8

Purpose: To static acid bake REE concentrate prior to water leaching - heated ore and acid contact with mixing

Sample: Composite REE Concentrate (May 14, 2021) (Conc 2)

H&S: Review MSDS for H₂SO₄, hot surfaces, use heat protective PPE

Procedure:

1. Mix feed sample with the required amount of reagent in a glass reactor and overhead mixing.
2. Continue mixing with a overhead mixer and Teflon impellor while heating to ~150°C until a semi-solid consistency was produced. Take pictures at various stages of the process. Insert at bottom of tab and cut/paste as JPEG.
3. The acid/feed product was transferred to a tared crucible and bake in a muffle furnace for required time at temperature.
4. Hold the furnace at temperature for 3 h, rabbling every 60 min, before removing from the furnace while hot.
5. Allow the sample to cool to room temperature before handling (place in a bag if left overnight).
6. Record the crucible gross weight, remove the sample, and hold for further work.

Conditions:	Target	Actual	
Feed Weight:	200	200.14	
Reagent:	H ₂ SO ₄	H ₂ SO ₄	
Reagent Strength:	96%	96%	
Reagent Dosage:	1500	1502	kg/t
Reagent to Add:	313	313.18	g
Total Weight:	513	513.32	g
Cure Time:	0		h
Preheat Temperature:	-	-	°C
Preheat Time:	-	-	h
Target Temperature:	300		°C
Test Time:	3		h
Sample Removal Temperature:	<100		°C

Results:		
Crucible Tare:	669.43	g
Starting Gross Weight:	1114.56	g
Starting Net Weight:	445.13	g
Calcine Gross Weight:	976.83	g
Calcine Net Weight:	307.40	g
Weight Loss:	31%	
Colour:	grey	
Consistency:	solid	
Pulverized (yes/no):	yes	

244.99 g acid effective dose
 1175.129 kg/t effective dose
 40% overall weight loss

Observations:

Stage	Time	Temp°C	Comments:
mix	7:19	95	significant foaming on acid addition, brown pulp
mix/bake	7:30	155	grey pulp beginning to fume/smoke
transfer	7:50	173	thick grey paste fuming
heat	8:20	45	into furnace, setpoint to 300
roast	8:50	300	furnace at temp time 0
roast	9:50	300	solid mass unable to rabble effectively, significant fuming
cool	11:50	300	furnace off

Project: 18299-02
Client: Lofdal (NCM)

Date: 25-May-21
Technologist: M Rosborough

Test: WL-AB8

Purpose: To water leach an acid bake calcine

Sample: AB8 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements	Streams
3	liquor samples - Y, Nd, Fe	partial liquor samples, final wash
1	liquor samples - ICP, REE, Cl, SO4 (Enviro)	Final PLS, separate cut for SO4
1	residue sample - WRA, REE by ICP, S, Cl	washed residue

Conditions:

AB8 Feed:	200	g
H2SO4 added:	300.6528	g
Net Calcine Weight:	307.4	g
Calcine Feed to Test:	307.4	g
Calculated Equivalent Feed:	200	g
Calculated Equivalent H2SO4:	301	g
Target % Solids (vs. Feed):	10%	
Calc. Pulp Weight:	2000	g
DI Water to add:	1693	g
Resultant Pulp Density:	15%	(relative to calcine)
Test Time:	4	h
Temperature:	25	°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 25-May-21
Technologist: M Rosborough

Test: WL-AB8

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
8:23	0.00	26.1	1.95	581	307	1693	550 RPM agitation on
8:53	0.50	31.3	1.41	577			
9:25	1.03	32.8	1.29	574			sample 1
9:53	1.50	32.7	1.24	569			
10:23	2.00	32.4	1.21	567			sample 2
11:53	3.50	31.3	1.17	564			
12:23	4.00	31.0	1.16	563			end test
Totals/Avg.		31.1	1.35	570.71	307	1693	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h	1.03	64.6	55.0	1.0896	51	558	1.39			slow	
2 h	2	63.2	54.3	1.0955	50	552	1.32			slow	
Final	4	1884.2	1687.0	1.0986	1536	551	1.25			slow	5.4%
Wash			1540.0	1.0056	1531	553	2.18	174.95	102.14	slow	

Free Acid Data:

Sample #	Aliquot	Titrant		Which Acid	Stoich mol/mol	MW g/mole	g/L acid	g acid
	mL	N	mL					
1 h	2	0.2	1.79	H2SO4	2	98.1	9	0.4
2 h	2	0.2	2.12	H2SO4	2	98.1	10	0.5
Final	2	0.2	2.39	H2SO4	2	98.1	12	18.0
Wash	10	0.2	1.14	H2SO4	2	98.1	1	1.7

Final Filtration/Washing:

Diameter of filtration paper:	150	mm
Type of Paper (Whatman #):	3	
Filtration Time:	78	minutes
Washing Time:	overnight	minutes
Cake Moisture:	42%	
Weight Loss:	32%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	brown
Clarity of Wash:	clear
Colour of Wash:	yellow
Colour of Residue:	grey

Residue: Total

Tare:	11.1 g
Tare + Wet:	186.1 g
Tare + Dry:	113.3 g

Acid Addition	1503	kg/t
Acid Remaining	103	kg/t
Acid Consumed	1400	kg/t

Comments:

Project: 18299-02
 Client: Lofdal (NCM)

Date: 25-May-21
 Technologist: M Rosborough

Test: WL-AB8

Metallurgical Balance

Sample & Quant.	Assay Units	AB8 Feed	1 h Filtrate	2 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		200	51	50	1536	1531	102
La	mg/L, g/t	1950			189		131
Ce	mg/L, g/t	3980			370		207
Pr	mg/L, g/t	338			37.4		23.1
Nd	mg/L, g/t	1330	138	138	141	9.11	95
Sm	mg/L, g/t	725			66.9		62.2
Eu	mg/L, g/t	435			41.50		32.0
Gd	mg/L, g/t	2380			254		148
Tb	mg/L, g/t	729			66.2		35.6
Dy	mg/L, g/t	4860			511		255
Ho	mg/L, g/t	1150			122		54.9
Y	mg/L, g/t	38900	3980	4240	4000	207	1260
Er	mg/L, g/t	3560			380		160
Tm	mg/L, g/t	494			52.0		22.0
Yb	mg/L, g/t	3100			309		138
Lu	mg/L, g/t	455			42.6		18.8
Sc	mg/L, g/t	38			4.11		
Th	mg/L, g/t	8240			669		2340
U	mg/L, g/t	154			15.1		20.5
Si	mg/L, %	10.0					16.8
Al	mg/L, %	1.66			560		1.94
Fe	mg/L, %	22.8	18200	19400	20200	955	7.41
Mg	mg/L, %	0.25			211		0.05
Ca	mg/L, %	6.58			978		8.36
Na	mg/L, %	0.979			86		1.50
K	mg/L, %	0.15			91		0.12
Ti	mg/L, %	3.84			2840		1.99
P	mg/L, %	2.03			734		2.21
Mn	mg/L, %	0.39			352		0.077
Cr	mg/L, %	0.03			18.1		<0.007
V	mg/L, %	0.10			70.8		0.04
S / SO4	mg/L, %	0.11			77000		7.99
Cl	mg/L, g/t	62			2		30

6.58 g/L TREE in final filtrate

Extract.	Account. out/in	Calc Head	Si-tie
%	%		%
96	78	1518	96%
96	74	2947	97%
96	88	299	96%
96	95	1270	96%
94	75	545	95%
95	77	335	96%
96	85	2026	96%
97	72	526	97%
97	83	4054	97%
97	84	965	97%
98	90	34995	98%
97	84	2999	97%
97	83	410	97%
97	79	2443	97%
97	74	337	98%
-	83	32	
81	77	6332	83%
92	82	126	92%
0	86	9	
30	86	1	31%
82	92	21	81%
87	75	0	88%
15	76	5	24%
8	85	1	9%
54	86	0	54%
68	83	3	69%
33	83	2	35%
87	80	0	88%
			85%
70	77	0	74%
83	48	24	
50	49	31	

Project: 18299-02
Client: Lofdal (NCM)

Date: 02-Jun-21
Technologist: A. Rashleigh

Test: AB9

Purpose: To static acid bake REE concentrate prior to water leaching - heated ore and acid contact

Sample: Composite REE Concentrate (May 14, 2021) **(Conc 2)**

H&S: Review MSDS for H₂SO₄, hot surfaces, use heat protective PPE

Procedure:

1. Mix feed sample with the required amount of reagent in a tared crucible. Once blended, allow to cure.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature (if required).
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 h, rabbling every 60 min, before removing from the furnace while hot.
6. Allow the sample to cool to room temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.

Conditions:	Target	Actual	
Feed Weight:	300	300.30	
Reagent:	H ₂ SO ₄	H ₂ SO ₄	
Reagent Strength:	96%	96%	
Reagent Dosage:	1250	1251	kg/t
Reagent to Add:	391	391.3	g
Total Weight:	691	691.6	g
Cure Time:	0		h
Preheat Temperature:	-	-	°C
Preheat Time:	-	-	h
Target Temperature:	300		°C
Test Time:	3		h
Sample Removal Temperature:	<100		°C

Results:

Crucible Tare:	720.2	g
Starting Gross Weight:	1349.5	g
Starting Net Weight:	629.3	g
Calcine Gross Weight:	1188.3	g
Calcine Net Weight:	468.10	g
Weight Loss:	26%	
Colour:	dark grey	
Consistency:	very hard, chunky	
Pulverized (yes/no):	no	

329.00 g acid effective dose

1051.748 kg/t effective dose

32% overall weight loss

Observations:

Stage	Time	Comments:
mix		Very foamy, very hot >50C, brown to grey color after foam dissipated
heat	16:46	In room temperature oven
roast	17:16	At temperature - oven 300 C
roast	18:16	Rabble - difficult to rabble
roast	19:16	Rabble - difficult to rabble
cool	20:16	Oven off - sample removed to fumehood to cool
out	20:17	A lot of fuming when removed

Project: 18299-02
Client: Lofdal (NCM)

Date:	03-Jun-21
Technologist:	A. Rashleigh

Test: WL1-AB9

Purpose: To water leach an acid bake calcine

Sample: AB9 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements	Streams
3	liquor samples - Y, Nd, Fe	partial liquor samples, final wash
1	liquor samples - ICP, REE, Cl, SO4 (Enviro)	Final PLS, separate cut for SO4
1	residue sample - WRA+Y2O3, REE by ICP, S, Cl	washed residue

Conditions:

AB9 Feed:	100	g
H2SO4 added:	125.216	g
Net Calcine Weight:	156.0333	g
Calcine Feed to Test:	154.7333	g
Calculated Equivalent Feed:	99	g
Calculated Equivalent H2SO4:	124	g
Target % Solids (vs. Feed):	10%	
Calc. Pulp Weight:	992	g
DI Water to add:	837	g
Resultant Pulp Density:	16%	(relative to calcine)
Test Time:	4	h
Temperature:	25	°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 03-Jun-21
Technologist: A. Rashleigh

Test: WL1-AB9

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
15:24		22.2	1.11	488	155	837	550 RPM
15:35		25.1	0.90	537			
16:05	0.50	26.5	0.77	557			
16:35	1.00	27.2	0.71	563			Sample
17:05	1.50	27.3	0.67	564			
17:35	2.00	26.8	0.64	564			Sample
18:35	3.00	25.9	0.61	562			
19:35	4.00	25.3	0.59	561			End test - filter
Totals/Avg.		25.8	0.75	549.50	155	837	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h	1	57	47.9	1.0739	45	457	0.75			fast	
2 h	2	63	54.4	1.0844	50	456	0.70			fast	
Final	4	884.8	744.1	1.0862	685	454	0.65				6.8%
Wash			1279.7	1.0068	1271	482	2.08	95.70	60.30		

Free Acid Data:

Sample #	Aliquot	Titrant		Which Acid	Stoich mol/mol	MW g/mole	g/L	g acid
	mL	N	mL				acid	
1 h	2	0.2	2.95	H2SO4	2	98.1	14	0.6
2 h	2	0.2	3.14	H2SO4	2	98.1	15	0.8
Final	2	0.2	3.18	H2SO4	2	98.1	16	10.7
Wash	10	0.2	1.53	H2SO4	2	98.1	2	1.9

Final Filtration/Washing:

Diameter of filtration paper:	150	mm
Type of Paper (Whatman #):	3	
Filtration Time:	10	minutes
Washing Time:		minutes
Cake Moisture:	37%	
Weight Loss:	25%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	dark orange
Clarity of Wash:	slightly cloudy
Colour of Wash:	orange
Colour of Residue:	grey

Residue: Total

Tare:	19.0 g
Tare + Wet:	114.7 g
Tare + Dry:	79.3 g

Acid Addition	1252	kg/t
Acid Remaining	140	kg/t
Acid Consumed	1112	kg/t

Comments:

Project: 18299-02
Client: Lofdal (NCM)

Date: 03-Jun-21
Technologist: A. Rashleigh

Test: WL1-AB9

Metallurgical Balance

Sample & Quant.	Assay Units	AB9 Feed	1 h Filtrate	2 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		99	45	50	685	1271	60
La	mg/L, g/t	1950			169		88
Ce	mg/L, g/t	3980			285		160
Pr	mg/L, g/t	338			34.9		21.5
Nd	mg/L, g/t	1330	126	122	121	8.03	987
Sm	mg/L, g/t	725			61.4		73.9
Eu	mg/L, g/t	435			36.2		41.1
Gd	mg/L, g/t	2380			216		180
Tb	mg/L, g/t	729			57.2		44.1
Dy	mg/L, g/t	4860			508		315
Ho	mg/L, g/t	1150			108		66.2
Y	mg/L, g/t	38900	3730	3850	3720	182.0	1260
Er	mg/L, g/t	3560			323		198
Tm	mg/L, g/t	494			46.0		28.9
Yb	mg/L, g/t	3100			249		188
Lu	mg/L, g/t	455			38.2		29.3
Sc	mg/L, g/t	38			4.17		
Th	mg/L, g/t	8240			545		2930
U	mg/L, g/t	154			13.9		28.8
Si	mg/L, %	10.0					14.8
Al	mg/L, %	1.66			494		1.79
Fe	mg/L, %	22.8	11800	14200	15900	875	12.7
Mg	mg/L, %	0.25			187		0.072
Ca	mg/L, %	6.58			1170		6.62
Na	mg/L, %	0.979			55		1.34
K	mg/L, %	0.15			98		0.083
Ti	mg/L, %	3.84			2380		2.66
P	mg/L, %	2.03			684		1.80
Mn	mg/L, %	0.39			308		0.12
Cr	mg/L, %	0.03			17.7		0.01
V	mg/L, %	0.10			68.6		0.05
S / SO4	mg/L, %	0.11			86000		7.36
Cl	mg/L, g/t	62			3		<10

5.97 g/L TREE in final filtrate

Extract.	Account. out/in	Calc Head	Si-tie
%	%		%
96	63	1221	97%
95	52	2066	97%
95	75	254	96%
64	125	1657	50%
90	65	469	93%
91	63	275	94%
93	67	1602	95%
94	58	422	96%
95	76	3701	96%
95	68	786	96%
98	83	32422	98%
95	66	2352	96%
95	68	335	96%
94	59	1834	96%
94	62	282	96%
-	76	29	
68	67	5547	76%
85	74	114	87%
0	90	9	
24	86	1	27%
63	93	21	62%
75	70	0	80%
17	73	5	32%
4	87	1	7%
57	79	0	62%
50	85	3	53%
30	77	2	40%
74	74	0	78%
			66%
61	77	0	66%
82	59	24	
77	43	27	

Project: 18299-02
Client: Lofdal (NCM)

Date: 03-Jun-21
Technologist: A. Rashleigh

Test: WL2-AB9

Purpose: To water leach an acid bake calcine

Sample: AB9 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements	Streams
3	liquor samples - Y, Nd, Fe	partial liquor samples, final wash
1	liquor samples - ICP, REE, Cl, SO4 (Enviro)	Final PLS, separate cut for SO4
1	residue sample - WRA+Y2O3, REE by ICP, S, Cl	washed residue

Conditions:

AB9 Feed:	100	g
H2SO4 added:	125.216	g
Net Calcine Weight:	156.0333	g
Calcine Feed to Test:	154.7333	g
Calculated Equivalent Feed:	99	g
Calculated Equivalent H2SO4:	124	g
Target % Solids (vs. Feed):	10%	
Calc. Pulp Weight:	992	g
DI Water to add:	837	g
Resultant Pulp Density:	16%	(relative to calcine)
Test Time:	4	h
Temperature:	80	°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 03-Jun-21
Technologist: A. Rashleigh

Test: WL2-AB9

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
15:13		21.3	0.87	508	155	837	550 RPM
15:37	0.00	74.4	-0.38	654			Begin heating
16:07	0.50	83.3	-0.75	666			At temperature
16:37	1.00	81.0	-0.77	659			Sample
17:07	1.50	80.5	-0.80	655			
17:37	2.00	80.4	-0.83	651			Sample
18:37	3.00	80.8	-0.89	647			
19:37	4.00	80.8	-0.92	644			End test - filter
Totals/Avg.		72.8	-0.56	635.50	155	837	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h	1	54.5	44.6	1.0942	41	449	0.62			fast	
2 h	2	54.5	45.5	1.0938	42	450	0.56			fast	
Final	4	886.2	751.7	1.0905	689	457	0.35				6.6%
Wash			1254.9	1.0056	1248	499	2.07	95.80	58.20		

Free Acid Data:

Sample #	Aliquot	Titrant		Which Acid	Stoich mol/mol	MW g/mole	g/L	g acid
	mL	N	mL				acid	
1 h	2	0.2	4.24	H2SO4	2	98.1	21	0.8
2 h	2	0.2	4.40	H2SO4	2	98.1	22	0.9
Final	2	0.2	4.21	H2SO4	2	98.1	21	14.2
Wash	10	0.2	1.34	H2SO4	2	98.1	1	1.6

Final Filtration/Washing:

Diameter of filtration paper:	150	mm
Type of Paper (Whatman #):	3	
Filtration Time:	7	minutes
Washing Time:		minutes
Cake Moisture:	39%	
Weight Loss:	26%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	dark orange
Clarity of Wash:	clear
Colour of Wash:	very pale yellow
Colour of Residue:	grey

Residue: Total

Tare:	18.7 g
Tare + Wet:	114.5 g
Tare + Dry:	76.9 g

Acid Addition	1252	kg/t
Acid Remaining	176	kg/t
Acid Consumed	1076	kg/t

Comments:

Project: 18299-02
 Client: Lofdal (NCM)

Date: 03-Jun-21
 Technologist: A. Rashleigh

Test: WL2-AB9

Metallurgical Balance

Sample & Quant.	Assay Units	AB9 Feed	1 h Filtrate	2 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		99	41	42	689	1248	58
La	mg/L, g/t	1950			174		93
Ce	mg/L, g/t	3980			290		164
Pr	mg/L, g/t	338			36.6		22.4
Nd	mg/L, g/t	1330	126	126	124	7.71	103
Sm	mg/L, g/t	725			62.6		72.1
Eu	mg/L, g/t	435			37.0		39.1
Gd	mg/L, g/t	2380			220		183
Tb	mg/L, g/t	729			59.2		42.2
Dy	mg/L, g/t	4860			501		304
Ho	mg/L, g/t	1150			109		63.2
Y	mg/L, g/t	38900	3720	3830	3780	176	1181
Er	mg/L, g/t	3560			329		178
Tm	mg/L, g/t	494			46.8		24.9
Yb	mg/L, g/t	3100			257		143
Lu	mg/L, g/t	455			39.5		18.9
Sc	mg/L, g/t	38			4.87		
Th	mg/L, g/t	8240			527		3310
U	mg/L, g/t	154			14.5		27.0
Si	mg/L, %	10.0					14.9
Al	mg/L, %	1.66			589		1.73
Fe	mg/L, %	22.8	16100	15400	16100	632	12.9
Mg	mg/L, %	0.25			219		0.04
Ca	mg/L, %	6.58			1000		6.96
Na	mg/L, %	0.979			59		1.38
K	mg/L, %	0.15			211		0.083
Ti	mg/L, %	3.84			1880		3.30
P	mg/L, %	2.03			1190		1.55
Mn	mg/L, %	0.39			371		0.06
Cr	mg/L, %	0.03			22.3		<0.007
V	mg/L, %	0.10			80.0		0.04
S / SO4	mg/L, %	0.11			85000		6.64
Cl	mg/L, g/t	62			34		24

Extract.	Account. out/in	Calc Head	Si-tie
%	%		%
96	65	1264	97%
95	53	2112	97%
95	79	268	96%
95	85	1124	95%
91	66	477	93%
92	64	280	94%
93	69	1637	95%
94	60	436	96%
95	75	3661	96%
95	69	795	96%
98	83	32319	98%
96	67	2391	97%
96	69	340	97%
96	60	1870	97%
96	63	286	97%
-	89	34	
65	68	5606	73%
86	76	117	88%
0	87	9	
29	86	1	30%
64	92	21	62%
88	70	0	90%
15	73	5	29%
5	87	1	5%
75	131	0	63%
40	84	3	42%
48	86	2	49%
88	76	0	89%
			83%
71	78	0	74%
84	58	24	
94	404	250	

6.07 g/L TREE in final filtrate

Project: 18299-02
Client: Lofdal (NCM)

Date:	03-Jun-21
Technologist:	A. Rashleigh

Test: WL3-AB9

Purpose: To water leach an acid bake calcine

Sample: AB9 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements	Streams
3	liquor samples - Y, Nd, Fe	partial liquor samples, final wash
1	liquor samples - ICP, REE, Cl, SO4 (Enviro)	Final PLS, separate cut for SO4
1	residue sample - WRA+Y2O3, REE by ICP, S, Cl	washed residue

Conditions:

AB9 Feed:	100	g
H2SO4 added:	125.216	g
Net Calcine Weight:	156.0333	g
Calcine Feed to Test:	154.7333	g
Calculated Equivalent Feed:	99	g
Calculated Equivalent H2SO4:	124	g
Target % Solids (vs. Feed):	20%	
Calc. Pulp Weight:	496	g
DI Water to add:	341	g
Resultant Pulp Density:	31%	(relative to calcine)
Test Time:	4	h
Temperature:	25	°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 03-Jun-21
Technologist: A. Rashleigh

Test: WL3-AB9

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
15:02		21.9	1.64	435	155	341	550 RPM
15:10	0.00	27.2	1.12	474			
15:40	0.50	33.8	0.77	512			
16:10	1.00	33.7	0.70	514			Sample
16:40	1.50	31.7	0.63	513			
17:10	2.00	30.2	0.59	512			Sample
18:10	3.00	27.9	0.54	510			
19:10	4.00	26.5	0.51	509			End test - filter
Totals/Avg.		29.1	0.81	497.38	155	341	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h	1	67	46.8	1.1991	39	449	0.38			slow	
2 h	2	67	48.6	1.2064	40	448	0.34			slow	
Final	4	392.9	263.4	1.2087	218	450	0.22			slow	15.1%
Wash			1140.1	1.0113	1127	490	1.82	92.90	59.20		

Free Acid Data:

Sample #	Aliquot	Titrant		Which Acid	Stoich mol/mol	MW g/mole	g/L	g acid
	mL	N	mL				acid	
1 h	2	0.2	8.47	H2SO4	2	98.1	42	1.6
2 h	2	0.2	8.68	H2SO4	2	98.1	43	1.7
Final	2	0.2	8.88	H2SO4	2	98.1	44	9.5
Wash	10	0.2	2.46	H2SO4	2	98.1	2	2.7

Final Filtration/Washing:

Diameter of filtration paper:	150	mm
Type of Paper (Whatman #):	3	
Filtration Time:	40	minutes
Washing Time:		minutes
Cake Moisture:	36%	
Weight Loss:	26%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	dark orange
Clarity of Wash:	slightly cloudy
Colour of Wash:	orange
Colour of Residue:	grey

Residue: Total

Tare:	18.9 g
Tare + Wet:	111.8 g
Tare + Dry:	78.1 g

Acid Addition	1252	kg/t
Acid Remaining	155	kg/t
Acid Consumed	1097	kg/t

Comments:

Project: 18299-02
Client: Lofdal (NCM)

Date: 03-Jun-21
Technologist: A. Rashleigh

Test: WL3-AB9

Metallurgical Balance

Sample & Quant.	Assay Units	AB9 Feed	1 h Filtrate	2 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		99	39	40	218	1127	59
La	mg/L, g/t	1950			467		89
Ce	mg/L, g/t	3980			809		165
Pr	mg/L, g/t	338			104		24.0
Nd	mg/L, g/t	1330	338	356	352	17.0	112
Sm	mg/L, g/t	725			153		85.2
Eu	mg/L, g/t	435			110		43.7
Gd	mg/L, g/t	2380			560		188
Tb	mg/L, g/t	729			143		46.8
Dy	mg/L, g/t	4860			1170		329
Ho	mg/L, g/t	1150			306		69.5
Y	mg/L, g/t	38900	8490	8300	8200	424	1496
Er	mg/L, g/t	3560			877		202
Tm	mg/L, g/t	494			126		29.8
Yb	mg/L, g/t	3100			694		201
Lu	mg/L, g/t	455			101		30.2
Sc	mg/L, g/t	38			8.03		
Th	mg/L, g/t	8240			1290		2840
U	mg/L, g/t	154			33.1		26.7
Si	mg/L, %	10.0					14.7
Al	mg/L, %	1.66			1160		1.78
Fe	mg/L, %	22.8	33500	35900	37100	1550	12.4
Mg	mg/L, %	0.25			436		0.066
Ca	mg/L, %	6.58			1170		7.43
Na	mg/L, %	0.979			129		1.36
K	mg/L, %	0.15			235		0.07
Ti	mg/L, %	3.84			5750		2.44
P	mg/L, %	2.03			2300		1.51
Mn	mg/L, %	0.39			714		0.12
Cr	mg/L, %	0.03			36.7		0.01
V	mg/L, %	0.10			173		0.05
S / SO4	mg/L, %	0.11			180000		7.94
Cl	mg/L, g/t	62			8		123

Extract.	Account. out/in	Calc Head	Si-tie
%	%		%
95	55	1079	97%
95	47	1876	97%
94	72	243	95%
95	99	1311	94%
87	53	387	92%
90	62	268	93%
92	56	1343	95%
92	47	342	96%
93	57	2767	95%
94	62	714	96%
97	78	30446	97%
94	58	2048	96%
94	60	295	96%
93	53	1645	96%
92	53	240	95%
-	46	18	
63	55	4530	77%
82	58	89	88%
0	88	9	
19	79	1	27%
63	88	20	63%
71	55	0	82%
5	71	5	23%
3	86	1	6%
54	64	0	66%
46	71	3	57%
36	69	1	49%
68	60	0	78%
			66%
56	68	0	66%
74	44	18	
19	147	91	

14.17 g/L TREE in final filtrate

Project: 18299-02
Client: Lofdal (NCM)

Date: 02-Jun-21
Technologist: A. Rashleigh

Test: AB10

Purpose: To static acid bake REE concentrate prior to water leaching - heated ore and acid contact

Sample: Composite REE Concentrate (May 14, 2021) (Conc 2)

H&S: Review MSDS for H₂SO₄, hot surfaces, use heat protective PPE

Procedure:

1. Mix feed sample with the required amount of reagent in a tared crucible. Once blended, allow to cure.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature (if required).
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 h, rabbling every 60 min, before removing from the furnace while hot.
6. Allow the sample to cool to room temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.

Conditions:	Target	Actual	
Feed Weight:	200	200.20	
Reagent:	H ₂ SO ₄	H ₂ SO ₄	
Reagent Strength:	96%	96%	
Reagent Dosage:	1000	998	kg/t
Reagent to Add:	208	208.2	g
Total Weight:	408	408.4	g
Cure Time:	0		h
Preheat Temperature:	-	-	°C
Preheat Time:	-	-	h
Target Temperature:	300		°C
Test Time:	3		h
Sample Removal Temperature:	<100		°C

Results:

Crucible Tare:	706.6	g
Starting Gross Weight:	1104.6	g
Starting Net Weight:	398	g
Calcine Gross Weight:	1007.7	g
Calcine Net Weight:	301.10	g
Weight Loss:	24%	
Colour:	dark grey	
Consistency:	very hard, chunky	
Pulverized (yes/no):	no	

197.80 g acid effective dose

948.4915 kg/t effective dose

26% overall weight loss

Observations:

Stage	Time	Comments:
mix		Very foamy, very hot >50C, brown to dark grey when foam dissipated
heat	16:46	In room temperature oven
roast	17:16	At temperature - oven at 300 C
roast	18:16	Rabble - difficult to rabble
roast	19:16	Rabble - difficult to rabble
cool	20:16	Oven off - sample removed to fume hood
out	20:18	No fumes when removed - sample was baked to crucible and hard to move

Project: 18299-02
Client: Lofdal (NCM)

Date:	04-Jun-21
Technologist:	A. Rashleigh

Test: WL-AB10

Purpose: To water leach an acid bake calcine

Sample: AB10 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements	Streams
3	liquor samples - Y, Nd, Fe	partial liquor samples, final wash
1	liquor samples - ICP, REE, Cl, SO4 (Enviro)	Final PLS, separate cut for SO4
1	residue sample - WRA+Y2O3, REE by ICP, S, Cl	washed residue

Conditions:

AB10 Feed:	200	g
H2SO4 added:	199.872	g
Net Calcine Weight:	301.1	g
Calcine Feed to Test:	296.9	g
Calculated Equivalent Feed:	197	g
Calculated Equivalent H2SO4:	197	g
Target % Solids (vs. Feed):	10%	
Calc. Pulp Weight:	1972	g
DI Water to add:	1675	g
Resultant Pulp Density:	15%	(relative to calcine)
Test Time:	4	h
Temperature:	25	°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 04-Jun-21
Technologist: A. Rashleigh

Test: WL-AB10

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
14:42		22.0	1.71	525	297	1676	675 RPM
14:47	0.00	26.4	1.06	548			
15:17	0.50	29.1	0.82	556			
15:47	1.00	30.4	0.69	566			Sample
16:17	1.50	30.9	0.61	570			
16:47	2.00	31.0	0.57	571			Sample
17:47	3.00	30.5	0.54	570			
18:47	4.00	30.0	0.53	569			End test - filter
Totals/Avg.		28.8	0.82	559.38	297	1676	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h	1	62.4	53.1	1.0575	50	490	1.53			fast	
2 h	2	60.5	47.2	1.073	44	492	1.43			fast	
Final	4	1860.4	1606.5	1.0785	1490	490	1.28			slow	7.4%
Wash			2082.2	1.0049	2072	496	2.19	217.00	136.90	slow	

Free Acid Data:

Sample #	Aliquot	Titrant		Which Acid	Stoich mol/mol	MW g/mole	g/L	g acid
	mL	N	mL				acid	
1 h	2	0.2	1.39	H2SO4	2	98.1	7	0.3
2 h	2	0.2	1.75	H2SO4	2	98.1	9	0.4
Final	2	0.2	1.96	H2SO4	2	98.1	10	14.3
Wash	10	0.2	0.78	H2SO4	2	98.1	1	1.6

Final Filtration/Washing:

Diameter of filtration paper:	150	mm
Type of Paper (Whatman #):	3	
Filtration Time:	52	minutes
Washing Time:	60	minutes
Cake Moisture:	37%	
Weight Loss:	20%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	dark orange
Clarity of Wash:	clear
Colour of Wash:	pale yellow
Colour of Residue:	grey

Residue: Total

Tare:	18.9 g
Tare + Wet:	235.9 g
Tare + Dry:	155.8 g

Acid Addition	999	kg/t
Acid Remaining	83	kg/t
Acid Consumed	916	kg/t

Comments:

Project: 18299-02
Client: Lofdal (NCM)

Date: 04-Jun-21
Technologist: A. Rashleigh

Test: WL-AB10

Metallurgical Balance

Sample & Quant.	Assay Units	AB10 Feed	1 h Filtrate	2 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		197	50	44	1490	2072	137
La	mg/L, g/t	1950			176		122
Ce	mg/L, g/t	3980			296		240
Pr	mg/L, g/t	338			39.6		42.6
Nd	mg/L, g/t	1330	133	131	131	9.19	179
Sm	mg/L, g/t	725			60.9		112
Eu	mg/L, g/t	435			36.8		69.9
Gd	mg/L, g/t	2380			217		358
Tb	mg/L, g/t	729			57.1		91.5
Dy	mg/L, g/t	4860			508		676
Ho	mg/L, g/t	1150			110		152
Y	mg/L, g/t	38900	3560	3680	3660	182.0	3701
Er	mg/L, g/t	3560			331		446
Tm	mg/L, g/t	494			45.6		63.0
Yb	mg/L, g/t	3100			251		397
Lu	mg/L, g/t	455			37.7		54.3
Sc	mg/L, g/t	38			4.15		
Th	mg/L, g/t	8240			484		3960
U	mg/L, g/t	154			13.1		39.6
Si	mg/L, %	10.0					13.8
Al	mg/L, %	1.66			499		1.73
Fe	mg/L, %	22.8	8560	12000	14500	593	14.5
Mg	mg/L, %	0.25			208		0.07
Ca	mg/L, %	6.58			1200		6.53
Na	mg/L, %	0.979			39		1.27
K	mg/L, %	0.15			90		0.10
Ti	mg/L, %	3.84			2260		2.68
P	mg/L, %	2.03			439		1.99
Mn	mg/L, %	0.39			352		0.11
Cr	mg/L, %	0.03			17.3		0.01
V	mg/L, %	0.10			61.3		0.062
S / SO4	mg/L, %	0.11			62000		6.53
Cl	mg/L, g/t	62			2		<10

5.96 g/L TREE in final filtrate

Extract.	Account. out/in	Calc Head	Si-tie
%	%		%
94	73	1414	95%
93	60	2402	96%
91	97	329	91%
90	96	1273	90%
86	74	538	89%
85	75	326	88%
87	79	1888	89%
87	68	495	91%
89	89	4306	90%
89	81	936	90%
92	87	33853	93%
89	79	2810	91%
89	79	388	91%
87	70	2171	91%
88	71	322	91%
-	82	31	
57	78	6405	65%
78	82	126	81%
0	96	10	
24	95	2	25%
55	97	22	54%
77	82	0	81%
17	83	5	28%
3	93	1	6%
50	92	0	52%
48	93	4	49%
19	84	2	29%
78	88	0	80%
52	88	0	64%
78	62	20	56%
69	36	22	

Project: 18299-02
Client: Lofdal (NCM)

Date: 02-Jun-21
Technologist: A. Rashleigh

Test: AB11

Purpose: To static acid bake REE concentrate prior to water leaching - heated ore and acid contact

Sample: Composite REE Concentrate (May 14, 2021) (Conc 2)

H&S: Review MSDS for H₂SO₄, hot surfaces, use heat protective PPE

Procedure:

1. Mix feed sample with the required amount of reagent in a tared crucible. Once blended, allow to cure.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature (if required).
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 h, rabbling every 60 min, before removing from the furnace while hot.
6. Allow the sample to cool to room temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.

Conditions:	Target	Actual	
Feed Weight:	200	200.30	
Reagent:	H ₂ SO ₄	H ₂ SO ₄	
Reagent Strength:	96%	96%	
Reagent Dosage:	750	749	kg/t
Reagent to Add:	156	156.3	g
Total Weight:	356	356.6	g
Cure Time:	0		h
Preheat Temperature:	-	-	°C
Preheat Time:	-	-	h
Target Temperature:	300		°C
Test Time:	3		h
Sample Removal Temperature:	<100		°C

Results:

Crucible Tare:	730.8	g
Starting Gross Weight:	1073.1	g
Starting Net Weight:	342.3	g
Calcine Gross Weight:	1017.4	g
Calcine Net Weight:	286.60	g
Weight Loss:	16%	
Colour:	dark grey	
Consistency:	hard, clumpy	
Pulverized (yes/no):	no	

142.00 g acid effective dose
 680.5791 kg/t effective dose
 20% overall weight loss

Observations:

Stage	Time	Comments:
mix		Very foamy, very hot >50C, brown to grey once foam dissipated
heat	16:46	In room temperature oven
roast	17:16	At temperature - oven at 300 C
roast	18:16	Rabble - easy to rabble
roast	19:16	Rabble - easy to rabble
cool	20:16	Oven off - sample removed to fumehood
out	20:18	No fumes when removed - sample easy to move around

Project: 18299-02
Client: Lofdal (NCM)

Date:	04-Jun-21
Technologist:	A. Rashleigh

Test: WL-AB11

Purpose: To water leach an acid bake calcine

Sample: AB11 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements	Streams
3	liquor samples - Y, Nd, Fe	partial liquor samples, final wash
1	liquor samples - ICP, REE, Cl, SO4 (Enviro)	Final PLS, separate cut for SO4
1	residue sample - WRA+Y2O3, REE by ICP, S, Cl	washed residue

Conditions:

AB11 Feed:	200	g
H2SO4 added:	150.048	g
Net Calcine Weight:	286.6	g
Calcine Feed to Test:	282.1	g
Calculated Equivalent Feed:	197	g
Calculated Equivalent H2SO4:	148	g
Target % Solids (vs. Feed):	10%	
Calc. Pulp Weight:	1969	g
DI Water to add:	1686	g
Resultant Pulp Density:	14%	(relative to calcine)
Test Time:	4	h
Temperature:	25	°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 04-Jun-21
Technologist: A. Rashleigh

Test: WL-AB11

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
14:54		21.8	1.79	515	282	1687	675 RPM
15:00	0.00	25.0	1.45	517			
15:30	0.50	31.4	1.20	550			
16:00	1.00	32.6	1.04	561			Sample
16:30	1.50	32.7	0.95	563			
17:00	2.00	32.0	0.89	562			Sample
18:00	3.00	30.9	0.86	560			
19:00	4.00	29.5	0.85	558			End test - filter
Totals/Avg.		29.5	1.13	548.25	282	1687	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h	1	59.6	52	1.0527	49	491	1.57			fast	
2 h	2	57.2	49.5	1.0662	46	491	1.43			fast	
Final	4	1860.3	1634.3	1.0695	1528	491	1.37			slow	7.5%
Wash			1995.4	1.0041	1987	497	2.29	199.00	138.80	slow	

Free Acid Data:

Sample #	Aliquot	Titrant		Which Acid	Stoich mol/mol	MW g/mole	g/L acid	g acid
	mL	N	mL					
1 h	2	0.2	1.05	H2SO4	2	98.1	5	0.3
2 h	2	0.2	1.31	H2SO4	2	98.1	6	0.3
Final	2	0.2	1.49	H2SO4	2	98.1	7	11.2
Wash	10	0.2	0.59	H2SO4	2	98.1	1	1.1

Final Filtration/Washing:

Diameter of filtration paper:	150	mm
Type of Paper (Whatman #):	3	
Filtration Time:	57	minutes
Washing Time:	87	minutes
Cake Moisture:	30%	
Weight Loss:	21%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	dark orange
Clarity of Wash:	clear
Colour of Wash:	pale yellow
Colour of Residue:	grey

Residue: Total

Tare:	18.7 g
Tare + Wet:	217.7 g
Tare + Dry:	157.5 g

Acid Addition	750	kg/t
Acid Remaining	64	kg/t
Acid Consumed	686	kg/t

Comments:

Project: 18299-02
 Client: Lofdal (NCM)

Date: 04-Jun-21
 Technologist: A. Rashleigh

Test: WL-AB11

Metallurgical Balance

Sample & Quant.	Assay Units	AB11 Feed	1 h Filtrate	2 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		197	49	46	1528	1987	139
La	mg/L, g/t	1950			174		163
Ce	mg/L, g/t	3980			321		362
Pr	mg/L, g/t	338			31.9		47.8
Nd	mg/L, g/t	1330	114	115	112	7.57	226
Sm	mg/L, g/t	725			47.7		309
Eu	mg/L, g/t	435			27.5		222
Gd	mg/L, g/t	2380			173		1420
Tb	mg/L, g/t	729			41.3		387
Dy	mg/L, g/t	4860			343		3030
Ho	mg/L, g/t	1150			70.3		718
Y	mg/L, g/t	38900	2370	2460	2450	103	21025
Er	mg/L, g/t	3560			225		2140
Tm	mg/L, g/t	494			32.1		303
Yb	mg/L, g/t	3100			194		1800
Lu	mg/L, g/t	455			24.9		242
Sc	mg/L, g/t	38			3.50		
Th	mg/L, g/t	8240			362		4870
U	mg/L, g/t	154			9.94		60.3
Si	mg/L, %	10.0					13.4
Al	mg/L, %	1.66			473		1.71
Fe	mg/L, %	22.8	9300	12500	14300	439	14.8
Mg	mg/L, %	0.25			215		0.072
Ca	mg/L, %	6.58			1140		6.40
Na	mg/L, %	0.979			<20		1.28
K	mg/L, %	0.15			78		0.11
Ti	mg/L, %	3.84			1640		3.41
P	mg/L, %	2.03			531		1.93
Mn	mg/L, %	0.39			381		0.077
Cr	mg/L, %	0.03			13.9		0.01
V	mg/L, %	0.10			48.7		0.078
S / SO4	mg/L, %	0.11			53000		5.91
Cl	mg/L, g/t	62			2		<10

4.27 g/L TREE in final filtrate

Extract.	Account. out/in	Calc Head	Si-tie
%	%		%
92	75	1466	94%
91	69	2747	93%
88	83	281	89%
86	87	1161	87%
63	81	588	68%
58	85	370	62%
57	98	2344	56%
54	81	593	60%
55	99	4799	54%
52	91	1052	53%
59	93	36056	60%
54	91	3255	55%
54	94	463	54%
54	90	2775	57%
53	80	364	60%
-	71	27	
45	76	6244	56%
64	78	120	71%
0	95	9	
23	95	2	23%
54	99	22	52%
77	88	0	78%
16	82	5	28%
0	92	1	2%
44	91	0	46%
35	96	4	34%
23	87	2	29%
84	90	0	85%
			63%
41	92	0	42%
77	73	18	
69	36	23	

Project: 18299-02
Client: Lofdal (NCM)

Date: 10-Jun-21
Technologist: M Rosborough

Test: AB12 (Bulk)

Purpose: To static acid bake REE concentrate prior to water leaching - heated ore and acid contact

Sample: Composite REE Concentrate (May 14, 2021) **(Conc 2)**

H&S: Review MSDS for H₂SO₄, hot surfaces, use heat protective PPE

Procedure:

1. Mix feed sample with the required amount of reagent in a tared crucible. Once blended, allow to cure.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature (if required).
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 h, rabbling every 60 min, before removing from the furnace while hot.
Repeat as needed for remaining batches.
6. Allow the sample to cool to room temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.

Conditions:	Target	Each Batch	
Feed Weight:	720	240.00	
Reagent:	H ₂ SO ₄	H ₂ SO ₄	
Reagent Strength:	96%	96%	
Reagent Dosage:	1250	1250	kg/t
Reagent to Add:	938	312.5	g
Total Weight:	1658	1657.5	g
Cure Time:	0	0.00	h
Preheat Temperature:	-	-	°C
Preheat Time:	-	-	h
Target Temperature:	300	300	°C
Test Time:	3	3	h
Sample Removal Temperature:	<100		°C

Results:	1	2	3
Crucible Tare:	669.42	651.49	554.16
Starting Gross Weight:	1207.01	1189.98	1093.36
Starting Net Weight:	537.59	538.49	539.2
Calcine Gross Weight:	1089.09	1027.48	934.28
Calcine Net Weight:	419.67	375.99	380.12
Weight Loss:	27%		
Colour:	grey		
Consistency:	crumbly		
Pulverized (yes/no):	no		

895.28 g acid effective dose
 1193.707 kg/t effective dose
 29% overall weight loss

Observations:

Stage	Time	Comments:
mix	7:30	significant foaming+heat on acid addition, brown turn to grey
heat	7:36	furnace SP to 300°C sample in
roast	8:10	furnace @ 300°C
roast	9:10	rabble, significant fuming
roast	10:00	rabble, significant fuming
cool	11:10	heat off
out	11:55	sample out

Project: 18299-02
Client: Lofdal (NCM)

Date:	11-Jun-21
Technologist:	M Rosborough

Test: WL-AB12

Purpose: To water leach an acid bake calcine

Sample: AB12 (Bulk) Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect slurry samples at the indicated times. Filter, wash solids, and submit both PLS and residue for assay.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements	Streams
2	liquor samples - Y, Nd, Fe	partial liquor samples, final wash
1	liquor samples - ICP, REE, Cl, SO ₄ (Enviro)	Final PLS, separate cut for SO ₄
1	residue sample - WRA+Y ₂ O ₃	partial residue
1	residue sample - WRA+Y ₂ O ₃ , REE by ICP, S, Cl	washed residue

Conditions:

AB12 (Bulk) Feed:	720	g
H ₂ SO ₄ added:	900	g
Net Calcine Weight:	1175.78	g
Calcine Feed to Test:	1175.75	g
Calculated Equivalent Feed:	720	g
Calculated Equivalent H ₂ SO ₄ :	900	g
Target % Solids (vs. Feed):	20%	
Calc. Pulp Weight:	3600	g
DI Water to add:	2424	g
Resultant Pulp Density:	33%	(relative to calcine)
Test Time:	2	h
Temperature:	25	°C

Project: 18299-02
 Client: Lofdal (NCM)

Date: 11-Jun-21
 Technologist: M Rosborough

Test: WL-AB12

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
7:47	0.00	37.6	1.04	556	1175	2424	675 RPM
8:02	0.25	39.2	0.75	578			agitation on, probes calibrated
8:17	0.50	39.0	0.67	581			
8:47	1.00	37.5	0.60	581			sample 1
9:17	1.50	35.6	0.56	579			
9:47	2.00	34.5	0.53	577			end test, filter
Totals/Avg.		37.2	0.69	575.33	1175	2424	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h	1	84.59	59.72	1.2118	49	536	0.91	18.55	11.7	slow	13.8%
Final	4	3513.61	2464.41	1.2249	2012	532	0.81	763.2	454.3	very slow	12.9%
Wash			3244.08	1.0315	3145	535	1.48	763.20	454.29	very slow	

Free Acid Data:

Sample #	Aliquot mL	Titration		Which Acid	Stoich mol/mol	MW g/mole	g/L acid	g acid
		N	mL					
1 h	2	0.2	8.74	H2SO4	2	98.1	43	2.1
Final	2	0.2	10.65	H2SO4	2	98.1	52	105.1
Wash	5	0.2	3.93	H2SO4	2	98.1	8	24.2

Final Filtration/Washing:

Diameter of filtration paper:	185	mm
Type of Paper (Whatman #):	3	
Filtration Time:	overnight	minutes
Washing Time:	overnight	minutes
Cake Moisture:	40%	
Weight Loss:	23%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	brown
Clarity of Wash:	clear
Colour of Wash:	brown
Colour of Residue:	grey

Residue: Total

Tare:	12.8 g
Tare + Wet:	776.0 g
Tare + Dry:	467.1 g

Acid Addition	1250	kg/t
Acid Remaining	183	kg/t
Acid Consumed	1067	kg/t

Comments:

Project: 18299-02
Client: Lofdal (NCM)

Date: 11-Jun-21
Technologist: M Rosborough

Test: WL-AB12

Metallurgical Balance

Sample & Quant.	Assay Units	AB12 Feed	1 h Filtrate	Final Filtrate	Wash	1 h Residue	Final Residue	Distribution of REE in Feed	Extract. %	Account. out/in %	Calc Head	Si-tie %
(mL or g)		720	49	2012	3145	12	454					
La	mg/L, g/t	1950		487			107	3.0%	95	73	1428	96%
Ce	mg/L, g/t	3980		922			180	6.2%	96	68	2690	97%
Pr	mg/L, g/t	338		107			22	0.5%	96	92	313	96%
Nd	mg/L, g/t	1330	373	381	55.5		107	2.1%	95	105	1400	95%
Sm	mg/L, g/t	725		188			77.0	1.1%	92	79	574	93%
Eu	mg/L, g/t	435		121			38.8	0.7%	93	83	363	94%
Gd	mg/L, g/t	2380		656			185	3.7%	94	82	1950	95%
Tb	mg/L, g/t	729		183			43.3	1.1%	95	74	539	96%
Dy	mg/L, g/t	4860		1340			316	7.5%	95	81	3944	96%
Ho	mg/L, g/t	1150		326			61.2	1.8%	96	83	950	96%
Y	mg/L, g/t	38900	10400	10400	742	1260	1417	60%	97	87	33930	98%
Er	mg/L, g/t	3560		961			193	5.5%	96	79	2807	96%
Tm	mg/L, g/t	494		147			24.2	0.8%	96	86	426	97%
Yb	mg/L, g/t	3100		833			145	4.8%	96	78	2419	97%
Lu	mg/L, g/t	455		119			20.7	0.7%	96	76	346	97%
Sc	mg/L, g/t	38		7.36			<25		57	96	36	55%
Th	mg/L, g/t	8240		1550	198		2740		75	84	6925	77%
U	mg/L, g/t	154		39.4			<24		88	81	125	89%
Si	mg/L, %	10.0				13.9	14.7		2	95	10	
Al	mg/L, %	1.66		1390		1.71	1.73		28	91	2	29%
Fe	mg/L, %	22.8	36100	39100	128	12.7	12.4		59	84	19	63%
Mg	mg/L, %	0.25		517		0.07	0.05		83	71	0	87%
Ca	mg/L, %	6.58		1370		7.50	8.72		8	91	6	10%
Na	mg/L, %	0.979		130		1.27	1.35		6	93	1	6%
K	mg/L, %	0.15		269		0.07	0.07		62	83	0	66%
Ti	mg/L, %	3.84		7350		2.31	1.83		64	85	3	68%
P	mg/L, %	2.03		3670		1.64	1.12		60	87	2	62%
Mn	mg/L, %	0.39		839		0.14	0.10		79	78	0	82%
Cr	mg/L, %	0.03		40.1		0.007	<0.007					83%
V	mg/L, %	0.10		162		0.04	0.04		65	70	0	74%
S / SO4	mg/L, %	0.11					8.44		0	13	5	
Cl	mg/L, g/t	62		<1			20		18	25	15	

64385.53 17.17 g/L TREE in final filtrate

Project: 18299-02
Client: Lofdal (NCM)

Date: 27-Apr-21
Technologist: M Rosborough

Test: IR-1

Purpose: Impurity Removal test on combined Acid Bake-Water Leach Liquor (AB1 through AB5) with MgCO₃

Sample: Combined ABWL PLS (combine PLS from WL-AB1 through WL-AB5)

H&S: Refer to H₂SO₄ and magnesium carbonate MSDS
 Feed solution is corrosive, avoid contact with skin and clean all spills accordingly

Procedure:

1. Secure feed solutions, decant if there are visible solids in the container. Filter if required.
2. Combine PLS from WL-AB1 through WL-AB5. Measure and record the solution density. Prepare reagent to 20% slurry.
3. Set up a pyrex reactor with overhead agitator and heated to 50°C
4. ORP and pH probes were placed to contact the reaction mixture and moved up as required during the test.
5. Solution was added to the reactor manually, and pH adjusted to the first pH target gradually by addition of MgCO₃ slurry to the vortex.
6. All reagent additions were recorded.
7. After achieving the first pH target the reactor was allowed to mix for one hour before sampling. Filter and collect a sample of PLS, return solids to reactor
8. Repeat steps 5-7 for remaining pH targets, including a well-washed residue assay at the final pH target.
9. The residue was dried and submitted for analysis.
10. The filtrates were assayed per the table below.

Assays:

#	Elements	Streams
7	liquor samples - Al, Fe, Th, Y, Nd	partial liquor samples (pH 2.0, 2.5, 2.8, 3.0, 3.3, 3.5), final wash
2	liquor samples - ICP, REE	Combined Feed, Final (pH 4.0) PLS
1	residue sample - WRA, REE by XRF, S, Cl	washed residue

Conditions:

Feed to Add:	1098.6	g
Solution Density:	1.0986	g/mL
Feed to Add:	1000	mL
Reagent	MgCO₃	
Reagent Strength:	20	%
Retention Time:	1	h at each pH target
Temperature:	50	°C
Target pH:	variable	pH

pH Targets:

Target #	pH Target	100% MgCO ₃					
		g/L Feed	kg/t Con	g / g Fe	t/t Fe in Cd	kg/t Con	
1	2.0	51.1	324.5	3.24	1.9	363.1	
2	2.5	73.8	468.2	4.67	2.8	524.0	
3	2.8	76.8	487.1	4.86	2.9	545.2	
4	3.0	78.7	499.1	4.98	3.0	558.6	
5	3.3	80.6	511.2	5.10	3.1	572.1	
6	3.5	81.8	519.2	5.18	3.1	581.1	
(final)	4.0	83.2	528.0	5.27	3.2	590.9	

Project: 18299-02
Client: Lofdal (NCM)

Date: 27-Apr-21
Technologist: M Rosborough

Test: IR-1

Metallurgical Balance:

Sample & Quant. (mL or g)	Assay Units	Feed	pH 2 Filt	pH 2.5 Filt	pH 2.8 Filt	pH 3 Filt	pH 3.3 Filt	pH 3.5 Filt	Final Filt	Final Wash	Final PPT	PPT % Res/Feed
		996	51	41	36	38	36	43	740.7	1474.9	38.6	
La	mg/L, g/t	29.9							18.6		<0.01	--
Ce	mg/L, g/t	56.9							31.0		0.02	11.6
Pr	mg/L, g/t	5.80							2.95		<0.03	--
Nd	mg/L, g/t	20.8	19.9	17.6	21.2	16.5	16.1	12.4	11.0	2.78	<0.02	--
Sm	mg/L, g/t	12.0							5.76		<0.04	--
Eu	mg/L, g/t	7.91							3.79			0.0
Gd	mg/L, g/t	53.4							29.9			0.0
Tb	mg/L, g/t	13.3							7.34			0.0
Dy	mg/L, g/t	111							57.8			0.0
Ho	mg/L, g/t	24.3							13.0			0.0
Y	mg/L, g/t	824	719	644	678	689	662	521	496	122	0.36	17.1
Er	mg/L, g/t	78.9							40.8			0.0
Tm	mg/L, g/t	10.6							5.00			0.0
Yb	mg/L, g/t	65.4							27.3			0.0
Lu	mg/L, g/t	8.90							3.8			0.0
Sc	mg/L, g/t	2.64							<0.07			0.0
Th	mg/L, g/t	144	123	31.2	6.47	2.44	0.69	0.18	0.05	<0.03	0.35	94.7
U	mg/L, g/t	3.77							0.12		0.02	174.5
Si	mg/L, %										0.21	--
Al	mg/L, %	1870	1550	1270	890	693	381	171	22.7	9.6	3.97	82.5
Fe	mg/L, %	15800	10900	677	472	487	454	412	309	46.7	34.9	85.7
Mg	mg/L, %	1160							16300		0.17	--
Ca	mg/L, %	990							672		0.743	29.1
Na	mg/L, %	20							301		0.03	57.6
K	mg/L, %	735							538		0.04	2.2
Ti	mg/L, %	613							0.10		1.47	93.3
P	mg/L, %	437							<5		0.991	87.9
Mn	mg/L, %	175							129		<0.008	1.7
Cr	mg/L, %	10.1							<0.1		0.03	105.1
V	mg/L, %	31.2							<0.2		0.078	97.5
Cu	mg/L, %	<10							<5			--
Pb	mg/L, %	<2							<2			--
Zn	mg/L, %	4							2.6			--
TREE	mg/L, %	1474	739	662	699	706	678	533	754	125	0.0	0.001247
LREE	mg/L, %	125	20	18	21	17	16	12	69	3	0.0	0.003449
HREE	mg/L, %	1198	719	644	678	689	662	521	685	122	0.0	0.001173

Project: 18299-02
Client: Lofdal (NCM)

Date: 28-Apr-21
Technologist: M Rosborough

Test: IR-2

Purpose: Impurity Removal test on combined Acid Bake-Water Leach Liquor (AB1 through AB5) with CaCO₃

Sample: Combined ABWL PLS

H&S: Refer to H₂SO₄ and limestone MSDS
 Feed solution is corrosive, avoid contact with skin and clean all spills accordingly

Procedure:

1. Secure feed solution (same as Test IR-1).
2. ~~Combine PLS from WL-AB1 through WL-AB5. Measure and record the solution density.~~ Prepare reagent to 20% slurry.
3. Set up a pyrex reactor with overhead agitator and heated to 50°C
4. ORP and pH probes were placed to contact the reaction mixture and moved up as required during the test.
5. Solution was added to the reactor manually, and pH adjusted to the first pH target gradually by addition of CaCO₃ slurry to the vortex.
6. All reagent additions were recorded.
7. After achieving the first pH target the reactor was allowed to mix for one hour before sampling. Filter and collect a sample of PLS, return solids to reactor
8. Repeat steps 5-7 for remaining pH targets, including a well-washed residue assay at the final pH target.
9. The residue was dried and submitted for analysis.
10. The filtrates were assayed per the table below.

Assays:

#	Elements	Streams
6	liquor samples - Al, Fe, Th, Y, Nd	partial liquor samples (pH 2.0, 2.5, 2.8, 3.0, 3.3, 3.5), final wash
2	liquor samples - ICP, REE	Combined Feed, Final (pH 4.0) PLS
1	residue sample - WRA, REE by XRF, S, Cl	washed residue

Conditions:

Feed to Add:	1098.6	g
Solution Density:	1.0986	g/mL
Feed to Add:	1000	mL
Reagent	CaCO₃	
Reagent Strength:	20	%
Retention Time:	1	h at each pH target
Temperature:	50	°C
Target pH:	variable	pH

pH Targets:

Target #	pH Target
1	2.0
2	2.5
3	2.8
4	3.0
5	3.3
6	3.5
(final)	4.0

Project: 18299-02
Client: Lofdal (NCM)

Date: 28-Apr-21
Technologist: M Rosborough

Test: IR-2

Metallurgical Balance:

Sample & Quant.	Assay Units	Feed	pH 2 Filt	pH 2.5 Filt	pH 2.8 Filt	pH 3 Filt	pH 3.3 Filt	pH 3.5 Filt	Final Filt	Final Wash	Final PPT	PPT % Res/Feed
(mL or g)		1000	39	41	34	34	38	39	705.1	1408.8	158.7	
La	mg/L, g/t	29.9							2.72		0.02	90.5
Ce	mg/L, g/t	56.9							2.42		0.03	95.2
Pr	mg/L, g/t	5.80							0.23		<0.03	--
Nd	mg/L, g/t	20.8	3.41	2.91	1.71	1.77	1.39	1.49	0.84	0.47	<0.02	--
Sm	mg/L, g/t	12.0							0.71		<0.04	--
Eu	mg/L, g/t	7.91							0.70			0.0
Gd	mg/L, g/t	53.4							8.56			0.0
Tb	mg/L, g/t	13.3							2.47			0.0
Dy	mg/L, g/t	111							21.9			0.0
Ho	mg/L, g/t	24.3							5.34			0.0
Y	mg/L, g/t	824	474	520	503	494	496	473	257	125	0.21	40.9
Er	mg/L, g/t	78.9							17.4			0.0
Tm	mg/L, g/t	10.6							2.17			0.0
Yb	mg/L, g/t	65.4							10.9			0.0
Lu	mg/L, g/t	8.90							1.53			0.0
Sc	mg/L, g/t	2.64							<0.07			0.0
Th	mg/L, g/t	144	83.5	26.1	6.41	2.36	0.46	0.16	<0.03	<0.03	0.088	96.8
U	mg/L, g/t	3.77							0.05		<0.008	--
Si	mg/L, %										0.03	--
Al	mg/L, %	1870	1510	1250	893	614	281	133	11.2	8.1	1.01	85.8
Fe	mg/L, %	15800	5230	687	496	466	461	458	318	88.8	8.81	88.5
Mg	mg/L, %	1160							1020		0.006	0.8
Ca	mg/L, %	990							561		20.2	--
Na	mg/L, %	20							14		0.01	117.7
K	mg/L, %	735							521		0.02	3.6
Ti	mg/L, %	613							<0.1		0.37	94.6
P	mg/L, %	437							<5		0.25	91.9
Mn	mg/L, %	175							129		<0.008	7.0
Cr	mg/L, %	10.1							<0.1		0.007	107.5
V	mg/L, %	31.2							<0.2		0.02	113.9
Cu	mg/L, %	<10							<0.4			--
Pb	mg/L, %	<2							<2			--
Zn	mg/L, %	4							2.5			--
TREE	mg/L, %	1323	477	523	505	496	497	474	335	125	0.0	0.004193
LREE	mg/L, %	125	3	3	2	2	1	1	7	0	0.0	0.017343
HREE	mg/L, %	1198	474	520	503	494	496	473	328	125	0.0	0.002816

Project: 18299-02
Client: Lofdal (NCM)

Date: 19-May-21
Technologist: M Rosborough

Test: IR-3

Purpose: Impurity Removal test on combined Acid Bake Water Leach Liquor (AB₁ through AB₅) with MgCO₃ and phosphoric acid

Sample: Combined ABWL PLS (combine PLS from WL-AB1 through WL-AB5)

H&S: Refer to H₂SO₄, H₂O₂, H₃PO₄, and magnesium carbonate MSDS
 Feed solution is corrosive, avoid contact with skin and clean all spills accordingly

Procedure:

1. Secure feed solution (same as Test IR-1). Ensure feed assays are input into IR-1 to calculate H₃PO₄ addition.
2. Combine PLS from WL-AB1 through WL-AB5. Measure and record the solution density. Prepare reagent to 20% slurry.
3. Set up a pyrex reactor with overhead agitator and heated to 50°C.
4. ORP and pH probes were placed to contact the reaction mixture and moved up as required during the test.
5. Solution was added to the reactor manually, and the ORP was adjusted to greater than 550 mV using hydrogen peroxide. Target dosage of phosphoric acid was added and then the pH was adjusted to the first pH target by addition of MgCO₃ slurry.
6. All reagent additions were recorded.
7. After achieving the first pH target the reactor was allowed to mix for one hour before sampling. Filter and collect a sample of PLS, return solids to reactor
8. Repeat steps 5-7 for remaining pH targets, including a well-washed residue assay at the final pH target.
9. The residue was dried and submitted for analysis.
10. The filtrates were assayed per the table below.

Assays:

#	Elements	Streams
7	liquor samples - Al, Fe, Th, Y, Nd	partial liquor samples (pH 2.0, 2.5, 2.8, 3.0, 3.3, 3.5), final wash
2	liquor samples - ICP, REE	Combined Feed, Final (pH 4.0) PLS
1	residue sample - WRA, REE by XRF, S, Cl	washed residue

Conditions:

Feed to Add:	1098.6	g
Solution Density:	1.0986	g/mL
Feed to Add:	1000	mL
Reagent	MgCO₃	
Reagent Strength:	20	%
Retention Time:	1	h at each pH target
Temperature:	50	°C
Target pH:	variable	pH

pH Targets:

Target #	pH Target
1	2.0
2	2.5
3	2.8
4	3.0
5	3.3
6	3.5
(final)	4.0

Phosphoric Acid: 85% reagent

Fe (mg/L)	P (mg/L)	H ₃ PO ₄ (g)
15800	437	31.0

Project: 18299-02
Client: Lofdal (NCM)

Date: 19-May-21
Technologist: M Rosborough

Test: IR-3

Metallurgical Balance:

Sample & Quant.	Assay Units	Feed	pH 2 Filt	pH 2.5 Filt	pH 2.8 Filt	pH 3 Filt	pH 3.3 Filt	pH 3.5 Filt	Final Filt	Final Wash	Final PPT	PPT % Res/Feed
(mL or g)		1001	49	44	48	48	48	46	865.1	1469.1	63.3	
La	mg/L, g/t	29.9							5.41		0.03	54.1
Ce	mg/L, g/t	56.9							5.34		0.06	66.4
Pr	mg/L, g/t	5.80							0.44		<0.03	--
Nd	mg/L, g/t	20.8	15.9	15.4	15.1	13.9	13.3	12.5	1.66	0.33	0.03	78.2
Sm	mg/L, g/t	12.0							0.88		<0.04	--
Eu	mg/L, g/t	7.91							0.72			0.0
Gd	mg/L, g/t	53.4							7.57			0.0
Tb	mg/L, g/t	13.3							1.93			0.0
Dy	mg/L, g/t	111							16.7			0.0
Ho	mg/L, g/t	24.3							4.77			0.0
Y	mg/L, g/t	824	671	650	649	620	599	633	241	37.3	0.65	49.5
Er	mg/L, g/t	78.9							15.5			0.0
Tm	mg/L, g/t	10.6							2.03			0.0
Yb	mg/L, g/t	65.4							10.8			0.0
Lu	mg/L, g/t	8.90							1.69			0.0
Sc	mg/L, g/t	2.64							<0.07			0.0
Th	mg/L, g/t	144	0.44	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	0.21	92.6
U	mg/L, g/t	3.77							0.20		0.008	142.2
Si	mg/L, %										0.16	--
Al	mg/L, %	1870	1070	923	797	584	287	136	24.7	9.9	2.41	81.6
Fe	mg/L, %	15800	725	456	382	338	342	356	330	47.7	21.5	85.9
Mg	mg/L, %	1160							15900		0.084	4.6
Ca	mg/L, %	990							648		0.33	21.0
Na	mg/L, %	20							267		0.04	117.3
K	mg/L, %	735							265		0.50	42.8
Ti	mg/L, %	613							<0.03		0.911	94.0
P	mg/L, %	437							<5		12.7	91.8
Mn	mg/L, %	175							114		<0.008	2.8
Cr	mg/L, %	10.1							<0.1		0.02	128.5
V	mg/L, %	31.2							<0.2		0.04	90.8
Cu	mg/L, %	<10							<0.5			--
Pb	mg/L, %	<2							<2			--
Zn	mg/L, %	4							1.9			--
TREE	mg/L, %	1323	687	665	664	634	612	646	316	38	0.0	0.003945
LREE	mg/L, %	125	16	15	15	14	13	13	14	0	0.0	0.009066
HREE	mg/L, %	1198	671	650	649	620	599	633	303	37	0.0	0.003408

Project: 18299-02
Client: Lofdal (NCM)

Date: 15-Jun-21
Technologist: M Rosborough

Test: IR-4

Purpose: Impurity Removal test on combined Acid Bake-Water Leach Liquor (from Conc 2) with MgCO₃

Sample: Conc 2 Combined ABWL PLS (combine PLS from WL-AB6 through WL-AB10, WL-AB12)

H&S: Refer to H₂SO₄, H₂O₂ and magnesium carbonate MSDS
 Feed solution is corrosive, avoid contact with skin and clean all spills accordingly

Procedure:

1. Secure feed solutions, decant if there are visible solids in the container. Filter if required.
2. Combine PLS from WL-AB6 through WL-AB10 and WL-AB12. Measure and record the solution density. Prepare reagent to 20% slurry.
3. Set up a pyrex reactor with overhead agitator and heated to 50°C
4. ORP and pH probes were placed to contact the reaction mixture and moved up as required during the test.
5. Solution was added to the reactor manually, and the ORP was adjusted to greater than 650 mV using hydrogen peroxide. The pH was adjusted to the first pH target by addition of MgCO₃ slurry.
6. All reagent additions were recorded.
7. After achieving the first pH target the reactor was allowed to mix for one hour before sampling. Filter and collect a sample of PLS, return solids to reactor
8. Repeat steps 5-7 for remaining pH targets, including a well-washed residue assay at the final pH target.
9. The residue was dried and submitted for analysis.
10. The filtrates were assayed per the table below.

Assays:

#	Elements	Streams
7	liquor samples - Al, Fe, Th, Y, Nd	partial liquor samples (pH 2.0, 2.5, 2.8, 3.0, 3.3, 3.5), final wash
2	liquor samples - ICP, REE	Combined Feed, Final (pH 4.0) PLS
1	residue sample - WRA, REE by XRF, S, Cl	washed residue

Conditions:

Feed to Add:	844.3	g
Solution Density:	1.126	g/mL
Feed to Add:	750	mL
Reagent	MgCO₃	
Reagent Strength:	20	%
Retention Time:	1	h at each pH target
Temperature:	50	°C
Target pH:	variable	pH

pH Targets:

Target #	pH Target	100% MgCO ₃				
		g/L Feed	kg/t Con	g / g Fe	t/t Fe in Cd	kg/t Con
1	2.0	40.9	215.1	1.87	1.2	272.4
2	2.5	66.8	351.3	3.05	2.0	445.0
3	2.8	70.0	368.1	3.20	2.0	466.3
4	3.0	71.4	375.5	3.26	2.1	475.7
5	3.3	73.7	387.5	3.36	2.2	490.9
6	3.5	75.1	395.2	3.43	2.2	500.7
(final)	4.0	77.9	409.9	3.56	2.3	519.2

Project: 18299-02
Client: Lofdal (NCM)

Date: 15-Jun-21
Technologist: M Rosborough

Test: IR-4

Metallurgical Balance:

Sample & Quant.	Assay Units	Feed	pH 2 Filt	pH 2.5 Filt	pH 2.8 Filt	pH 3 Filt	pH 3.3 Filt	pH 3.5 Filt	Final Filt	Final Wash	Final PPT	PPT % Res/Feed
(mL or g)		750	37	29	29	28	29	29	475.2	1494.5	47.0	
La	mg/L, g/t	288							168		0.08	--
Ce	mg/L, g/t	513							274		0.19	22.9
Pr	mg/L, g/t	60.4							28.5		<0.03	--
Nd	mg/L, g/t	215	183	166	159	160	160	151	102	21.6	0.08	--
Sm	mg/L, g/t	103							42.7		0.05	--
Eu	mg/L, g/t	65.6							30.0			0.0
Gd	mg/L, g/t	367							180			0.0
Tb	mg/L, g/t	99.6							46.7			0.0
Dy	mg/L, g/t	724							355			0.0
Ho	mg/L, g/t	181							84.5			0.0
Y	mg/L, g/t	5760	4980	4560	4410	4420	4520	4280	3190	1390	1.73	18.8
Er	mg/L, g/t	522							260			0.0
Tm	mg/L, g/t	79.7							34.5			0.0
Yb	mg/L, g/t	480							191			0.0
Lu	mg/L, g/t	64.8							26.9			0.0
Sc	mg/L, g/t	4.33							<0.07			0.0
Th	mg/L, g/t	840	718	128	14.0	4.51	0.92	0.40	0.20	<0.03	1.02	76.0
U	mg/L, g/t	22.1							0.29		0.03	72.1
Si	mg/L, %										0.28	--
Al	mg/L, %	733	732	528	383	278	98	37	12.7	225	0.969	82.7
Fe	mg/L, %	21900	17900	1730	997	985	989	1060	837	4910	28.7	82.2
Mg	mg/L, %	290							14400		0.28	--
Ca	mg/L, %	1170							1310		0.071	3.8
Na	mg/L, %	75							317		0.04	37.2
K	mg/L, %	144							121		0.02	7.2
Ti	mg/L, %	3640							4.18		5.21	89.7
P	mg/L, %	1550							<5		2.18	88.0
Mn	mg/L, %	466							343		<0.008	1.0
Cr	mg/L, %	24.8							<0.1		0.04	103.7
V	mg/L, %	94.0							<0.2		0.15	100.8
Cu	mg/L, %	<40							<40			--
Pb	mg/L, %	<8							<8			--
Zn	mg/L, %	<5							<5			--
TREE	mg/L, %	10390	5163	4726	4569	4580	4680	4431	5014	1412	0.0	0.001297
LREE	mg/L, %	1179	183	166	159	160	160	151	615	22	0.0	0.002225
HREE	mg/L, %	8344	4980	4560	4410	4420	4520	4280	4399	1390	0.0	0.0013

Project: 18299-02
Client: Lofdal (NCM)

Date: 21-Jun-21
Technologist: M Rosborough

Test: IR-5

Purpose: Bulk Endpoint Impurity Removal test on combined Acid Bake-Water Leach Liquor (from Conc 2) with MgCO₃

Sample: Conc 2 Combined ABWL PLS (combine PLS from WL-AB6 through WL-AB10, WL-AB12)

H&S: Refer to H₂SO₄, H₂O₂ and magnesium carbonate MSDS
 Feed solution is corrosive, avoid contact with skin and clean all spills accordingly

Procedure:

1. Secure feed solutions, decant if there are visible solids in the container. Filter if required. 4000 g MgCO₃ 20% Slurry
2. Combine PLS from WL-AB6 through WL-AB10 and WL-AB12. Measure and record the solution density. Prepare reagent to 20% slurry.
3. Set up a pyrex reactor with overhead agitator and heated to 50°C
4. ORP and pH probes were placed to contact the reaction mixture and moved up as required during the test.
5. Solution was added to the reactor manually, and 23.2 g 30% peroxide was added.
 The pH was adjusted to the first pH target by addition of MgCO₃ slurry.
6. All reagent additions were recorded.
7. After achieving the target pH the reactor was allowed to mix for one hour before final filtration.
8. The residue was washed thoroughly (1 repulp at similar total volume + 2 × 2000 mL DI) dried and submitted for analysis.
9. The filtrates were assayed per the table below.

Assays:

#	Elements	Streams
1	liquor samples - Al, Fe, Th, Y, Nd	final wash
1	liquor samples - ICP, REE	Final (pH 3.0) PLS
1	residue sample - WRA, REE by XRF, S, Cl	washed residue

Conditions:

Feed to Add:	8823.5	g
Solution Density:	1.126	g/mL
Feed to Add:	7838	mL
Reagent	H₂O₂	
Reagent Strength:	30	%
Add:	23	g 30% H ₂ O ₂
Reagent	MgCO₃	
Reagent Strength:	20	%
Retention Time:	1	h
Temperature:	50	°C
Target pH:	2.9-3.0	pH

Peroxide Addition calculates after Feed Volume entered.

MgCO₃ Slurry Mass calculates after Feed Volume entered.

Project: 18299-02
Client: Lofdal (NCM)

Date: 21-Jun-21
Technologist: M Rosborough

Test: IR-5

Metallurgical Balance:

Sample & Quant. (mL or g)	Assay Units	Feed	Final Filt	Final Wash	Final PPT	PPT % Res/Feed	Calc Head	Acc %
		7838	6880.7	10224.5	561.0			
La	mg/L, g/t	288	208		0.04	--	213	74
Ce	mg/L, g/t	513	382		0.08	10.7	390	76
Pr	mg/L, g/t	60.4	40.3		<0.03	--	54	89
Nd	mg/L, g/t	215	145	39.3	0.03	11.4	203	94
Sm	mg/L, g/t	103	68.7		<0.04	--	91	89
Eu	mg/L, g/t	65.6	41.7			--	37	56
Gd	mg/L, g/t	367	278			--	244	66
Tb	mg/L, g/t	99.6	71.4			--	63	63
Dy	mg/L, g/t	724	553			--	485	67
Ho	mg/L, g/t	181	126			--	111	61
Y	mg/L, g/t	5760	4220	1120	0.71	8.8	5673	98
Er	mg/L, g/t	522	399			--	350	67
Tm	mg/L, g/t	79.7	54.8			--	48	60
Yb	mg/L, g/t	480	333			--	292	61
Lu	mg/L, g/t	64.8	44.2			--	39	60
Sc	mg/L, g/t	4.33	0.83			--	1	17
Th	mg/L, g/t	840	7.46	<1.87	1.11	94.3	801	95
U	mg/L, g/t	22.1	7.27		0.02	54.9	19	84
Si	mg/L, %				0.18	--	130	--
Al	mg/L, %	733	326	111	0.413	40.3	726	99
Fe	mg/L, %	21900	33.3	12.3	28.6	93.5	20517	94
Mg	mg/L, %	290	13700		1.57	--	--	--
Ca	mg/L, %	1170	1120		0.50	30.6	1341	115
Na	mg/L, %	75	290		0.04	35.4	281	375
K	mg/L, %	144	108		0.02	8.3	107	74
Ti	mg/L, %	3640	2.74		4.85	95.4	3477	96
P	mg/L, %	1550	<5		1.99	91.7	1425	92
Mn	mg/L, %	466	347		0.04	5.9	332	71
Cr	mg/L, %	24.8	<0.3		0.04	118.5	30	120
V	mg/L, %	94.0	<0.2		0.14	106.6	100	107
Cu	mg/L, %	<40	11.0			0.0	10	24
Pb	mg/L, %	<8	<4			0.0	4	44
Zn	mg/L, %	<5	6			0.0	5	105
TREE	mg/L, %	10390	6965	1159	0.0	0.000641	7626.445	73.4051
LREE	mg/L, %	1179	844	39	0.0	0.00135	792.1669	67.16694
HREE	mg/L, %	8344	6121	1120	0.0	0.000608	6834.278	81.90944

Project: 18299-02
Client: Lofdal (NCM)

Date: 25-Jun-21
Technologist: M Rosborough

Test: RP-1

Purpose: REE Precipitation test on Bulk Impurity Removal Filtrate (from Flot Con 2 ABWL plus MgCO₃)

Sample: IR-5 Filtrate

H&S: Refer to H₂SO₄ and sodium carbonate MSDS
 Feed solution is mildly corrosive, avoid contact with skin and clean all spills accordingly

Procedure:

1. Secure feed solution, decant if there are visible solids in the container. Filter if required.
2. Measure and record the solution density. Prepare reagent to 100 g/L solution.
3. Set up a pyrex reactor with overhead agitator.
4. ORP and pH probes were placed to contact the reaction mixture and moved up as required during the test.
5. Solution was added to the reactor manually, and pH adjusted to the first pH target gradually by addition of Na₂CO₃ to the vortex.
6. All reagent additions were recorded.
7. After achieving the first pH target the reactor was allowed to mix for 30 min before sampling. Filter and collect a sample of PLS, return solids to reactor
8. Repeat steps 5-7 for remaining pH targets, including a well-washed residue assay at the final pH target.
9. The residue was dried and submitted for analysis.
10. The filtrates were assayed per the table below.

Assays:

#	Elements	Streams
7	liquor samples - Y	partial liquor samples (pH 6.00, 6.25, 6.50, 6.75, 7.00, 7.25), final wash
1	liquor samples - ICP, REE	Final (pH 7.5) PLS
1	residue sample - REE, WRA	washed residue

Conditions:

Feed to Add:	816.0	g
Solution Density:	1.088	g/mL
Feed to Add:	750	mL
Reagent	Na₂CO₃	
Reagent Strength:	100	g/L
Retention Time:	1	h at each pH target
Temperature:	ambient	°C
Target pH:	variable	pH

pH Targets:

Target #	pH Target	g Na ₂ CO ₃ / L IR Filt	kg/t conc
1	6.00	13.52	71.1
2	6.25	14.05	73.9
3	6.50	14.76	77.6
4	6.75	15.29	80.4
5	7.00	15.79	83.1
6	7.25	16.24	85.4
(final)	7.50	16.79	88.3

Project: 18299-02
Client: Lofdal (NCM)

Date: 25-Jun-21
Technologist: M Rosborough

Test: RP-1

Metallurgical Balance:

Sample & Quant. (mL or g)	Assay Units	Feed	pH 6 Filt	pH 6.25 Filt	pH 6.5 Filt	pH 6.75 Filt	pH 7 Filt	pH 7.25 Filt	Final Filt	Final Wash	Final PPT	PPT % Res/Feed
		754	45	43	42	43	45	50	513.8	707.4	13.2	
La	mg/L, %	208							0.04		1.16	97.9
Ce	mg/L, %	382							0.05		2.13	97.8
Pr	mg/L, %	40.3							<0.03		0.21	93.1
Nd	mg/L, %	145							<0.06		0.78	94.5
Sm	mg/L, %	68.7							<0.04		0.27	68.4
Eu	mg/L, %	41.7							<0.03			--
Gd	mg/L, %	278							0.11			--
Tb	mg/L, %	71.4							0.04			--
Dy	mg/L, %	553							0.48			--
Ho	mg/L, %	126							0.18			--
Y	mg/L, %	4220	299	151	36.3	40.1	21.7	18.1	15.3	1.64	23.0	95.7
Er	mg/L, %	399							0.86			--
Tm	mg/L, %	54.8							0.16			--
Yb	mg/L, %	333							1.30			--
Lu	mg/L, %	44.2							0.21			--
Sc	mg/L, %	0.83							<0.07			--
Th	mg/L, %	7.46							<0.03		0.04	82.8
U	mg/L, %	7.27							1.34		0.05	123.0
Si	mg/L, %										0.17	--
Al	mg/L, %	326							<0.4		1.70	91.8
Fe	mg/L, %	33.3							<0.2		0.22	114.4
Mg	mg/L, %	13700							12700		1.07	1.4
Ca	mg/L, %	1120							698		1.01	15.8
Na	mg/L, %	290							6360		0.28	--
K	mg/L, %	108							116		0.02	2.7
Ti	mg/L, %	2.74							<0.02		0.01	76.9
P	mg/L, %	<5							<5		0.004	15.3
Mn	mg/L, %	347							238		0.37	18.8
Cr	mg/L, %	<0.3							<0.1		<0.007	--
V	mg/L, %	<0.2							<0.2		<0.006	--
Cu	mg/L, %	11.0							<0.1			--
Pb	mg/L, %	<4							<2			--
Zn	mg/L, %	6							<0.7			--
TREE	mg/L, %	6981	299	151	36	40	22	18	19	2	27.5	69.3
LREE	mg/L, %	844	0	0	0	0	0	0	0	0	4.5	94.6
HREE	mg/L, %	6121	299	151	36	40	22	18	19	2	23.0	66.0

Project: 18299-02
Client: Lofdal (NCM)

Date: 06-Jul-21
Technologist: M Rosborough

Test: RP-2

Purpose: Bulk REE Precipitation test on Bulk Impurity Removal Filtrate (from Flot Con 2 ABWL plus MgCO₃)

Sample: IR-5 Filtrate

H&S: Refer to H₂SO₄ and sodium carbonate MSDS
 Feed solution is mildly corrosive, avoid contact with skin and clean all spills accordingly

Procedure:

1. Secure feed solution, decant if there are visible solids in the container. Filter if required.
2. Measure and record the solution density. Prepare reagent to 100 g/L solution.
3. Set up a pyrex reactor with overhead agitator.
4. ORP and pH probes were placed to contact the reaction mixture and moved up as required during the test.
5. Solution was added to the reactor manually, and pH adjusted to the first pH target gradually by addition of Na₂CO₃ to the vortex.
6. All reagent additions were recorded.
7. After achieving the target pH the reactor was allowed to mix for one hour before final filtration.
8. The residue was washed thoroughly (1 repulp at similar total volume + 2 × 2000 mL DI) dried and submitted for analysis.
9. The filtrates were assayed per the table below.

1400 mL 100 g/L Na₂CO₃

Assays:

#	Elements	Streams
1	liquor samples - Y	final wash
1	liquor samples - ICP, REE	Final (pH 6.5) PLS
1	residue sample - REE, WRA	washed residue

Conditions:

Feed to Add:	5748.0	g
Solution Density:	1.088	g/mL
Feed to Add:	5283	mL
Reagent	Na₂CO₃	
Reagent Strength:	100	g/L
Retention Time:	1	h
Temperature:	ambient	°C
Target pH:	6.50-6.75	pH

pH 6.6

g Na ₂ CO ₃	
/ L IR Filt	kg/t conc
17.21	90.5

Project: 18299-02
Client: Lofdal (NCM)

Date: 06-Jul-21
Technologist: M Rosborough

Test: RP-2

Metallurgical Balance:

Sample & Quant. (mL or g)	Assay Units	Feed	Final Filt	Final Wash	Final PPT	PPT % Res/Feed	Calc Head	Acc %
		5283	5599.9	5969.6	86.3			
La	mg/L, %	208	0.09		1.26	99.0	206	99
Ce	mg/L, %	382	0.09		2.3	98.4	376	98
Pr	mg/L, %	40.3	<0.03		0.244	98.9	40	99
Nd	mg/L, %	145	<0.06		0.876	98.7	143	99
Sm	mg/L, %	68.7	<0.04		0.409	97.3	67	97
Eu	mg/L, %	41.7	<0.03		0.248	97.2	41	97
Gd	mg/L, %	278	0.11		1.65	97.0	270	97
Tb	mg/L, %	71.4	0.05		0.413	94.5	68	95
Dy	mg/L, %	553	0.66		3.43	101.3	561	101
Ho	mg/L, %	126	0.26		0.746	96.7	122	97
Y	mg/L, %	4220	24.1	3.63	26.7	103.4	4392	104
Er	mg/L, %	399	1.30		2.43	99.5	398	100
Tm	mg/L, %	54.8	0.25		0.317	94.5	52	95
Yb	mg/L, %	333	1.87		1.86	91.3	306	92
Lu	mg/L, %	44.2	0.31		0.256	94.6	42	95
Sc	mg/L, %	0.83	<0.07		<0.004	78.7	1	88
Th	mg/L, %	7.46	<0.03		0.0438	95.9	7	96
U	mg/L, %	7.27	1.19		0.0334	75.1	7	92
Si	mg/L, %				0.15	--	24	--
Al	mg/L, %	326	<2		1.86	93.1	306	94
Fe	mg/L, %	33.3	0.9		0.21	103.0	35	106
Mg	mg/L, %	13700	11900		0.55	0.7	12703	93
Ca	mg/L, %	1120	683		0.69	10.0	836	75
Na	mg/L, %	290	5420		0.16	--	--	--
K	mg/L, %	108	87		<0.008	1.3	94	87
Ti	mg/L, %	2.74	<0.02		0.01	71.5	2	72
P	mg/L, %	<5	<5		<0.004	--	--	--
Mn	mg/L, %	347	259		0.27	12.8	319	92
Cr	mg/L, %	<0.3	<0.1		<0.007	--	--	--
V	mg/L, %	<0.2	<0.2		<0.006	--	--	--
Cu	mg/L, %	11.0	<0.2			--	--	2
Pb	mg/L, %	<4	<2			--	--	53
Zn	mg/L, %	6	1.9			--	--	34
TREE	mg/L, %	6981	29	4	43.1	101.0	7084	101.5
LREE	mg/L, %	844	0	0	5.1	98.5	832	98.6
HREE	mg/L, %	6121	29	4	38.1	101.6	6252	102.1

	Distribution of REE in Res
La	2.9%
Ce	5.3%
Pr	0.6%
Nd	2.0%
Sm	0.9%
Eu	0.6%
Gd	3.8%
Tb	1.0%
Dy	8.0%
Ho	1.7%
Y	62%
Er	5.6%
Tm	0.7%
Yb	4.3%
Lu	0.6%

Project: 18299-02
Client: Lofdal (NCM)

Date:	Jul 12, 2021
Technologist:	M Rosborough

Test: RL1

Purpose: Release of RP cake to elevated final pH (3.5 in pulp) to reduce advancing impurities and increase REE tenor

Sample: RP-2 Final Residue

H&S: Review MSDS for H₂SO₄, magnesium carbonate

Procedure:

1. Add the target amount of water into a suitably sized reactor, and add the target amount of solids to the reactor while mixing. Heat to the target temperature.
2. Adjust the pH of the pulp to 1 using 96% H₂SO₄ and hold for 3 hours. Collect a solution sample.
3. Add remaining feed to the pulp to reach pH adjustment target, switching to MgCO₃ if necessary. Maintain the pulp at the target pH for 60 minutes. Record any observations.
4. Once the test is complete, weigh and filter the pulp.
5. Record the total weight, density, free acid, pH, and ORP of the filtrate. Collect a sample for assay.
6. Displacement wash the solids on the filter. Collect a sample of the wash water for assay, recording the total weight, density, free acid, etc.
7. Record the wet weight of any solids before drying. Record the weight again once dry and submit for assay.

Assays:	#	Elements	Streams
	2	liquor samples - Y	partial liquor samples, wash
	1	liquor samples - ICP, REE, Si	final filtrate
	1	residue sample - WRA, REE	final precipitate

Conditions:

Target TREE Tenor:	25	g/L
Assumed Feed:	34%	TREE
Wet Feed to Add:	78.00	g
Feed Moisture:	0%	
Dry Equivalent:	78.0	g
Liquor Volume:	1060.8	mL
Water to Add:	1060.80	g
Reagent	MgCO₃	(only after all wet feed added)
Reagent Pulp Density:	20	%
Retention Time:	variable	h
Temperature:	50	°C
Target Acidity:	1	pH
Adjustment pH:	3.5	pH

Project: 18299-02
Client: Lofdal (NCM)

Date:	Jul 12, 2021
Technologist:	M Rosborough

Test: RL1

Test Data:

Time		Reactor			Reagents / Feed			Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	MgCO ₃ 30% g	H ₂ SO ₄ 96% g	
7:50		21.8	7.39	275	62.48			feed addition, agitation on, heat on
8:13		55.4	7.41	226				at temp, begin pH adjustment
8:42		61.5	1.00	1011			50.82	at initial pH 1 target
9:12		57.0	1.01	826			0.62	
9:40		53.3	1.02	802			0.30	
10:40		48.0	1.00	776			1.38	
11:42		52.4	0.96	776				collect sample 1
11:48		52.3	0.96	776				begin adjusting pH by feed addition
12:25		49.2	3.45	658	17.94			all remaining feed added
12:55		51.9	3.52	606				
13:25		50.5	3.58	579				end test
Totals/Avg.		50.3	2.85		80.4	0	53	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
pH 1		49.31	48.56	1.0655	46	735	1.49			fast	
pH 3.5		1131.66	1090.48	1.0713	1018	585	4.23			fast	1%
Wash			313.91	1.0063	312	559	4.49	24.1	6.88	slow	

Final Filtration/Washing:

Diameter of filtration paper:	90	mm
Type of Paper (Whatman #):	millipore	
Filtration Time:	10	minutes
Washing Time:	25	minutes
Cake Moisture:	71%	
Weight Loss	91%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	very slight pink
Clarity of Wash:	clear
Colour of Wash:	none
Colour of Residue:	orange/yellow

Residue: pH 3.5

Tare:	19 g
Tare + Wet:	43 g

Filter Cake Subsample:

	Tare:	18.61
tion	Gross Wet Cake:	42.73
tion	Tare:	18.61
	Gross Washed Wet Cake:	42.73
	Gross Washed Dry Cake:	25.5
	% Solids:	29%

Project: 18299-02
 Client: Lofdal (NCM)

Date: Jul 12, 2021
 Technologist: M Rosborough

Test: RL1

Metallurgical Balance

Sample & Quant.	Assay Units	RP-2 Final Residue	pH 1 Filtrate	pH 3.5 Filtrate	Wash	pH 3.5 Residue
(mL or g)		80	46	1018	312	7
La	mg/L, %	1.26		842		0.0971
Ce	mg/L, %	2.3		1540		0.278
Pr	mg/L, %	0.244		165		0.0307
Nd	mg/L, %	0.876		577		0.119
Sm	mg/L, %	0.409		270		0.0683
Eu	mg/L, %	0.248		181		0.037
Gd	mg/L, %	1.65		1090		0.202
Tb	mg/L, %	0.413		281		0.0571
Dy	mg/L, %	3.43		2210		0.547
Ho	mg/L, %	0.746		501		0.117
Y	mg/L, %	26.7	13500	17100	1930	2.47
Er	mg/L, %	2.43		1580		0.48
Tm	mg/L, %	0.317		216		0.0827
Yb	mg/L, %	1.86		1270		0.681
Lu	mg/L, %	0.256		177		0.0967
Sc	mg/L, %	<0.004		1.66		0.0278
Th	mg/L, %	0.0438		1.98		0.489
U	mg/L, %	0.0334		23.3		0.0302
Si	mg/L, %	0.15		22.2		1.49
Al	mg/L, %	1.86		210		17.4
Fe	mg/L, %	0.21		1.2		2.57
Mg	mg/L, %	0.55		324		0.03
Ca	mg/L, %	0.69		519		0.02
Na	mg/L, %	0.16		32		0.03
K	mg/L, %	<0.008		29		<0.008
Ti	mg/L, %	0.01		<5		0.12
P	mg/L, %	<0.004		<5		0.057
Mn	mg/L, %	0.27		219		0.008
Cr	mg/L, %	<0.007		<0.3		0.01
V	mg/L, %	<0.006		<0.2		<0.006
Cu	mg/L, %			45.0		
Pb	mg/L, %			<20		
Zn	mg/L, %			20.3		
S	mg/L, g/t					
TREE	mg/L, %	43.1		28000		5.36

Leached	Account.	Calc	Distribution
%	out/in	Head	of REE in RL Liq.
99	85	1.07	3.0%
99	86	1.97	5.5%
99	87	0.21	0.6%
99	85	0.74	2.1%
98	85	0.35	1.0%
99	94	0.23	0.6%
99	85	1.40	3.9%
99	87	0.36	1.0%
98	83	2.84	7.9%
98	86	0.64	1.8%
99	88	23.4	61%
98	84	2.04	5.6%
97	88	0.28	0.8%
97	90	1.67	4.5%
96	91	0.23	0.6%
47	112	0.004	
6	101	0.04	
92	96	0.03	
18	104	0.16	
15	94	1.76	
1	105	0.22	
99	75		
100	96	0.66	
94	26	0.04	
98	--	0.04	
38	138	0.02	
57	--	0.01	
100	103	0.28	
24	23	0.00	
35	13	0.00	
-	-	0.06	
-	-	0.03	
-	-	0.03	
-	-	-	
99	83	-	

Project: 18299-02
Client: Lofdal (NCM)

Date: Jul 20, 2021
Technologist: M Rosborough

Test UIX1

Objective: Perform Bulk U Removal from RL Filtrate with IX Resin A660

Feed: RL1 Filtrate

Resin Preparation:

1. Sulphate the resin by contacting with excess (i.e. 10x the resin volume) 150 g/L H₂SO₄. Then wash twice with DI water (using the same volume of DI as acid). Acid and washes can be conducted in a mildly agitated reactor/beaker

Procedure:

1. Secure feed solution.
2. Prepare 2 × 150 mL of wet settled and drained resin, measuring the volume in a glass graduated cylinder. Ensure to tap the side of the cylinder to settle and pack the resin. Any excess water in the cylinder on top of the resin should be removed.
3. Add the required volume of feed solution to reaction kettle, and transfer target amount of resin. Using paddle-type impeller, supply gentle mixing (adequate to suspend resin but no vortex) and mix for 24 hours at ambient temperature.
4. After 24 hours, separate the resin from the solution (filter). Collect the filtrate separately and wash the loaded resin with water before drying overnight at 60-70°C.
5. Repeat Steps 3 and 4 using a fresh aliquot of resin and the filtrate from the first IX contact.
6. Submit the final 24 hour solutions and loaded resin for assay as required.

Experimental Conditions:

	UIX1-1	UIX1-2	
Resin:	A660	A660	mL
Resin Volume:	133	133	
Feed/Resin Ratio:	7.2	7.1	
Feed:	RL1 Filtrate	UIX1-1 Filt	g/mL
Feed Solution Density:	1.0713	1.0619	
Feed Solution Volume:	952	939	
Feed Solution Weight:	1019.5	996.9	g
Temperature:	ambient	ambient	
Shaking Speed:			

Assays:

	Elements	Streams
2	liquor samples - U, Th	24 hour solutions
0	solid samples - hold	dried resins

Sampling Data:

Sample #		Weight		SG g/mL	Volume PLS, mL	Wet resin, g	Dry resin, g	%H ₂ O	Filtrate pH
		pulp, g	Liq., g						
UIX1-1	24 hour	1160.9	1041.7	1.0619	981	101.2	72.8	28.1	2.40
UIX1-2	24 hour	1139.9	1024.8	1.0539	972	106.1	76.4	28.0	2.06

Free Acid Data: *Fill out SG data. Enter aliquot data in weight or volume basis. Enter vol of titrant. Enter type of acid (HCl, H₂SO₄ or HNO₃)*

Sample #		Aliquot mL pipet	Titration		Which Acid	Stoich mol/mol	MW g/mole	g/L acid
			N	mL				
UIX1-1	24 hour	1	0.2	0.14	H ₂ SO ₄	2	98.1	1.4
UIX1-2	24 hour	2	0.2	0.29	H ₂ SO ₄	2	98.1	1.4

Project: 18299-02
Client: Lofdal (NCM)

Date: Jul 20, 2021
Technologist: M Rosborough

Test UIX1

Results:

		UIX1-1 (Sol/Resin=951.647531037058/133)					UIX1-2 (Sol/Resin=938.760711931444/133)				
		Feed Sol'n	24 h Solution	Loaded Resin	Extraction %	Account. out/in	Feed Sol'n	24 h Solution	Loaded Resin	Extraction %	Account. out/in
Quant (mL; g)		951.6	1007.8	72.8			938.8	1000.6	76.4		
Element	Units	Assay (mg/L, %, g/t)			%	%	Assay (mg/L, %, g/t)			%	%
Tb	mg/L, g/t	281	244		8	92	244	197		14	86
Dy	mg/L, g/t	2210	1980		5	95	1980	1650		11	89
Y	mg/L, g/t	17100	15800		2	98	15800	13100		12	88
Th	mg/L, g/t	1.98	0.46		75	25	0.46	0.11		75	25
U	mg/L, g/t	23.3	0.03		100	0	0.03	<0.02		29	71
La	mg/L, %	842	757		5	95	757	631		11	89
Ce	mg/L, %	1540	1420		2	98	1420	1160		13	87
Pr	mg/L, %	165	149		4	96	149	123		12	88
Nd	mg/L, %	577	524		4	96	524	433		12	88
Sm	mg/L, %	270	238		7	93	238	201		10	90
Eu	mg/L, %	181	150		12	88	150	120		15	85
Gd	mg/L, %	1090	997		3	97	997	785		16	84
Ho	mg/L, %	501	449		5	95	449	366		13	87
Er	mg/L, %	1580	1480		1	99	1480	1230		11	89
Tm	mg/L, %	216	195		4	96	195	163		11	89
Yb	mg/L, %	1270	1170		2	98	1170	1010		8	92
Lu	mg/L, %	177	158		5	95	158	134		10	90
Sc	mg/L, %	1.66	0.70		55	45	0.70	0.27		59	41
Th	mg/L, %	1.98	2		0	100		0.16		91	9
U	mg/L, %	23.3	22		0	100		<0.02		100	0
Si	mg/L, %	22.2	21		0	100				100	0
Al	mg/L, %	210	198		0	100		174		6	94
Fe	mg/L, %	1.2	1		0	100		<0.6		44	56
Mg	mg/L, %	324	306		0	100		274		5	95
Ca	mg/L, %	519	490		0	100		447		3	97
Na	mg/L, %	32	30		0	100		<30		-6	106
K	mg/L, %	29	27		0	100		<30		-17	117
Ti	mg/L, %	5	5		0	100		<4		10	90
P	mg/L, %	5	5		0	100		<5		-13	113
Mn	mg/L, %	219	207		0	100		181		7	93
Cr	mg/L, %	0.3	0		0	100		<0.2		25	75
V	mg/L, %	0.2	0		0	100		<0.2		-13	113
Cu	mg/L, %	45	42		0	100		<40		0	100
Pb	mg/L, %	20	19		0	100		<9		49	51
Zn	mg/L, %	20.3	19		0	100		15		17	83
TREE	mg/L, %	28000	25711		3	97	25711	21303		12	88

assumes zero extraction in UIX-1

after pH adjust for ThSX

Project: 18299-02
Client: Lofdal (NCM)

Date: 23-Jul-21
Technologist: M Rosborough

Test: ThSX-1
Type: Extraction

Purpose: To generate thorium-free raffinate for REE Precip testing

Aqueous Sample: UIX1-2 Filtrate

Organic Sample: See blend details below

H&S: Review SDS for sulphuric acid, Primene JMT, tridecanol, Aromatic 150ND

Feed Preparation:

1. Secure entire volume of UIX1-2 Filtrate. Refilter through a 0.5 µm filter if required.
2. Record the FA, pH, ORP, and density of the solution prior to use. **Adjust the acidity as specified.**
Record acid addition in the Notes section.
3. Collect a small subsample of the pH adjusted feed and submit for assay.

Organic Preparation:

1. Prepare sufficient volume of the organic blend according to the requirements. Use diluent (Aromatic 150ND) to rinse smaller cylinders used for Primene JMT and tridecanol. Prior to use, contact the organic 1/1 with pH 1 H₂SO₄, twice.

Contact Procedure:

1. In a suitable size vessel with a bottom drain, add the indicated volumes of the organic and aqueous phases.
2. Mix the two phases (using overhead mixing) to ensure that equilibrium is reached. Record the actual temperature during mixing (as close to target as possible).
3. At 10 minutes, stop mixing and allow the phases to separate. Record the time that the first clear phase appears, which phase it is, and the total time required for complete phase separation (when the emulsion layer has collapsed to less than 1 mm thick at the interface).
4. When the phases have separated, record the colour and clarity of each phase. If any emulsion persists after 20 minutes, record the location of the emulsion and the relative quantity (either thickness in mm or as a percentage of the phase it is in). Take photo of each separated contact and paste as JPEG below.
5. If any crud or precipitate is formed, note the amount (in mL) and what phase it is in.
6. Drain most of the aqueous into a cylinder.
7. Drain the remaining aqueous and organic out into a separatory funnel and remove the rest of the aqueous from the organic. Combine the aqueous with the rest already recovered and measure the volume and density of both phases. Collect an aqueous and organic sample for assay.
8. Filter the aqueous sample through Whatman #3 paper to adsorb any entrained organic. Put the filter paper cone directly into the sample cup. Do not allow the filter paper cone to drain fully to make sure that no organic passes through the paper. Remaining aqueous is stored for precipitation work.
9. Filter the organic sample through Whatman #1ps paper to separate any entrained aqueous. Put the filter paper cone directly into the sample cup.
10. Submit the aqueous and organic solutions for assay as required. Store remainder of organic sample.

Assays:	#	Elements	Streams
	2	aqueous sample - REE, ICP	pH adjusted feed, raffinate
	1	organic sample - Y, Th	loaded organic

Conditions:

Phase Ratio:	10:1	A/O
Contact Time:	10	min
Equilibrium pH:	as-is	
Temperature:	ambient	°C

Mixer Type:	overhead
Phase Continuity:	not controlled
Mixing Rate:	ALAP

Organic Blends (by volume):

Organic 1:	0.5% Primene JMT
	2.50% tridecanol
	97.0% Aromatic 150ND

300 mL total organic
 1.5 mL Primene JMT
 7.5 mL tridecanol
 291 mL Aromatic 150ND

Project: 18299-02
Client: Lofdal (NCM)

Date: 23-Jul-21
Technologist: M Rosborough

Test: ThSX-1
Type: Extraction

Test Data:

Contact			Feed	ThSX-1-1
Organic Blend				1
Aqueous Feed				UIX1-2 Filtrate
Aqueous Feed pH				1.5
Phase Ratio		A/O		10
Temperature		°C		ambient
Volumes In/Out	Org. In	mL		89
	Aq. In	mL		885
	Org. Out	mL		88
	Aq. Out	mL		880
Disengagement Time		sec*		8
First Phase to Separate				Aq
Emulsion (quantity)		mL		-
Emulsion Location				-
Organic Density		g/mL	0.881	0.884
Aqueous Density		g/mL	1.057	1.056
Phases	Aqueous	colour	lt pink	lt pink
		clarity	clear	clear
		pH	1.46	1.48
		ORP (mV)	327	350
	Organic	colour	lt yellow	lt yellow
		clarity	clear	clear
Free Acid Data	Sample		Feed	ThSX-1-1R
	Aliquot	mL	2.0	2.0
	NaOH	N	0.2	0.2
	Titration	mL	1.30	1.34
	Acid Type		H2SO4	H2SO4
	MW	g/mole	98.1	98.1
	g/L	acid	6.4	6.6

5.40 g conc H2SO4 added to adjust Feed pH

25.4 °C contact temperature

Notes:

This image shows a blank sheet of white paper with horizontal ruling lines. The lines are evenly spaced and run across the width of the page. There are no margins, text, or other markings on the paper.

Project: 18299-02
Client: Lofdal (NCM)

Date: 30-Jul-21
Technologist: M Rosborough

Test: RP-3

Purpose: Final REE Precipitation test on ThSX Raffinate

Sample: ThSX Raff

H&S: Refer to H₂SO₄ and oxalic acid MSDS
 Feed solution is mildly corrosive, avoid contact with skin and clean all spills accordingly

Procedure:

1. Secure feed solution, decant if there are visible solids in the container. Filter if required.
2. Measure and record the solution density. Prepare reagent to 10% strength.
3. Set up a pyrex reactor with overhead agitator.
4. ORP and pH probes were placed to contact the reaction mixture and moved up as required during the test.
5. Solution was added to the reactor manually and heatup started, followed by the target dosage of oxalic acid solution once at temperature.
6. All reagent additions were recorded.
7. After target dosage of oxalic acid the reactor was allowed to mix for two hours before final filtration.
8. The residue was washed thoroughly (3 × 100 mL DI displacement) and dried. Once dried, sample will be directed to calcination.
9. The filtrates were assayed per the table below. Final Res assays will be calculated from calcined material.

Assays:

#	Elements	Streams
1	liquor samples - Y	final wash
1	liquor samples - ICP, REE	Final PLS

Conditions:

Feed to Add:	422.5	g
Solution Density:	1.056	g/mL
Feed to Add:	400	mL
Reagent:	oxalic acid	
Reagent Strength:	10	%
Retention Time:	2	h
Temperature:	50	°C
Oxalic Dose Target:	110%	stoichiometric
10% Oxalic to Add:	119.2	g

Oxalic Acid Make Up:

Target Strength:	10%	
Weight To Make:	150	g
Reagent Oxalic Acid :	71.4%	
Oxalic Reagent to Add:	21.0	g
DI Water to Add:	129.0	g

Project: 18299-02
Client: Lofdal (NCM)

Date: 30-Jul-21
Technologist: M Rosborough

Test: RP-3

Metallurgical Balance:

Sample & Quant. (mL or g)	Assay Units	Feed	Final Filt	Final Wash	Final PPT	PPT % Res/Feed	Calc Head	Acc %
		400	465.5	401.5	21.7			
La	mg/L, %	637	14.1		11002	93.9	614	96
Ce	mg/L, %	1160	8.52		19924	93.4	1093	94
Pr	mg/L, %	124	0.42		2094	91.8	114	92
Nd	mg/L, %	434	0.92		7489	93.8	408	94
Sm	mg/L, %	198	0.23		3467	95.2	189	95
Eu	mg/L, %	120	0.14		2140	96.9	116	97
Gd	mg/L, %	800	1.18		13961	94.8	760	95
Tb	mg/L, %	200	0.49		3587	97.5	196	98
Dy	mg/L, %	1630	5.94		28985	96.6	1582	97
Ho	mg/L, %	369	2.11		6703	98.7	367	99
Y	mg/L, %	13200	212	8.51	226056	93.1	12541	95
Er	mg/L, %	1210	10.7		21404	96.1	1176	97
Tm	mg/L, %	164	2.22		2922	96.8	161	98
Yb	mg/L, %	993	15.8		17243	94.4	956	96
Lu	mg/L, %	135	2.40		2325	93.6	129	96
Sc	mg/L, %	0.14	<0.07		<12	448.7	1	507
Th	mg/L, %	0.04	<0.03		0.51	69.1	0	156
U	mg/L, %	<0.02	<0.02		<0.2	62.8	0	179
Si	mg/L, %				0.088	--	48	--
Al	mg/L, %	171	147		<0.002	0.7	172	101
Fe	mg/L, %	<0.6	1.9		0.003	--	--	--
Mg	mg/L, %	268	217		0.14	28.1	328	122
Ca	mg/L, %	439	328		0.074	9.2	422	96
Na	mg/L, %	<30	14		0.14	--	--	--
K	mg/L, %	<30	34		<0.004	--	--	--
Ti	mg/L, %	<4	0.42		<0.003	--	--	--
P	mg/L, %	<5	<5		0.002	--	--	--
Mn	mg/L, %	178	145		<0.004	1.1	171	96
Cr	mg/L, %	<0.2	0.2		--	--	--	--
V	mg/L, %	<0.2	<0.2		--	--	--	--
Cu	mg/L, %	<40	18.7		--	--	--	--
Pb	mg/L, %	<9	<2		--	--	--	--
Zn	mg/L, %	15	14.4		<20	73700	11072	73812
TREE	mg/L, %	21374	277	9	369303	93.9	20403	95.5
LREE	mg/L, %	2553	24	0	43977	93.6	2418	94.7
HREE	mg/L, %	18821	253	9	325326	93.9	17984	95.6

	Distribution of REE in Res
La	3.0%
Ce	5.4%
Pr	0.6%
Nd	2.0%
Sm	0.9%
Eu	0.6%
Gd	3.8%
Tb	1.0%
Dy	7.8%
Ho	1.8%
Y	61%
Er	5.8%
Tm	0.8%
Yb	4.7%
Lu	0.6%

Project: 18299-02
Client: Lofdal (NCM)

Date: 30-Jul-21
Technologist: M Rosborough

Test: RP-4

Purpose: Final REE Precipitation test on ThSX Raffinate

Sample: ThSX Raff

H&S: Refer to H₂SO₄ and sodium carbonate MSDS
 Feed solution is mildly corrosive, avoid contact with skin and clean all spills accordingly

Procedure:

1. Secure feed solution, decant if there are visible solids in the container. Filter if required.
2. Measure and record the solution density. Prepare reagent to 100 g/L solution.
3. Set up a pyrex reactor with overhead agitator.
4. ORP and pH probes were placed to contact the reaction mixture and moved up as required during the test.
5. Solution was added to the reactor manually and heatup started, then pH adjusted to the first pH target gradually by addition of Na₂CO₃ to the vortex once at temperature.
6. All reagent additions were recorded.
7. After achieving the target pH the reactor was allowed to mix for one hour before final filtration.
8. The residue was washed thoroughly (3 × 100 mL DI displacement) and dried. Once dried, sample will be directed to calcination.
9. The filtrates were assayed per the table below. Final Res assays will be calculated from calcined material.

500 mL 100 g/L Na₂CO₃

Assays:

#	Elements	Streams
1	liquor samples - Y	final wash
1	liquor samples - ICP, REE	Final (pH 6.5) PLS

Conditions:

Feed to Add:	422.5	g
Solution Density:	1.056	g/mL
Feed to Add:	400	mL
Reagent	Na₂CO₃	
Reagent Strength:	100	g/L
Retention Time:	1	h
Temperature:	50	°C
Target pH:	6.50-6.60	pH

Project: 18299-02
Client: Lofdal (NCM)

Date: 30-Jul-21
Technologist: M Rosborough

Test: RP-4

Metallurgical Balance:

Sample & Quant. (mL or g)	Assay Units	Feed	Final Filt	Final Wash	Final PPT	PPT % Res/Feed	Calc Head	Acc %
		400	432.4	371.7	17.8			
La	mg/L, %	637	0.24		13129	91.9	586	92
Ce	mg/L, %	1160	0.12		23691	91.1	1057	91
Pr	mg/L, %	124	<0.03		2457	88.4	110	88
Nd	mg/L, %	434	<0.05		8928	91.7	398	92
Sm	mg/L, %	198	<0.04		4038	91.0	180	91
Eu	mg/L, %	120	<0.03		2521	93.7	112	94
Gd	mg/L, %	800	0.16		16689	93.0	744	93
Tb	mg/L, %	200	0.05		4266	95.1	190	95
Dy	mg/L, %	1630	0.55		34428	94.2	1536	94
Ho	mg/L, %	369	0.17		7994	96.6	357	97
Y	mg/L, %	13200	16.1	3.68	280090	94.6	12513	95
Er	mg/L, %	1210	0.72		25383	93.6	1133	94
Tm	mg/L, %	164	0.12		3542	96.3	158	96
Yb	mg/L, %	993	0.73		20948	94.1	935	94
Lu	mg/L, %	135	0.11		2824	93.3	126	93
Sc	mg/L, %	0.14	<0.07		<15	464.7	1	519
Th	mg/L, %	0.04	<0.03		1.1	123.6	0	205
U	mg/L, %	<0.02	<0.02		<0.3	65.1	0	173
Si	mg/L, %				0.992	--	442	--
Al	mg/L, %	171	<0.2		0.337	87.8	150	88
Fe	mg/L, %	<0.6	<0.2		0.01	--	--	--
Mg	mg/L, %	268	167		0.21	35.0	274	102
Ca	mg/L, %	439	153		0.429	43.6	357	81
Na	mg/L, %	<30	12500		0.34	--	--	--
K	mg/L, %	<30	14		<0.005	--	--	--
Ti	mg/L, %	<4	<0.02		0.006	--	--	--
P	mg/L, %	<5	<5		0.01	--	--	--
Mn	mg/L, %	178	51.3		0.14	35.1	118	66
Cr	mg/L, %	<0.2	<0.1			--	--	--
V	mg/L, %	<0.2	<0.2			--	--	--
Cu	mg/L, %	<40	<0.1			--	--	--
Pb	mg/L, %	<9	<2			--	--	--
Zn	mg/L, %	15	0.8		0.00	0.0	1	6
TREE	mg/L, %	21374	19	4	450927	94.1	20136	94.2
LREE	mg/L, %	2553	0	0	52243	91.3	2331	91.3
HREE	mg/L, %	18821	19	4	398684	94.5	17805	94.6

	Distribution of REE in Res
La	2.9%
Ce	5.3%
Pr	0.5%
Nd	2.0%
Sm	0.9%
Eu	0.6%
Gd	3.7%
Tb	0.9%
Dy	7.6%
Ho	1.8%
Y	62%
Er	5.6%
Tm	0.8%
Yb	4.6%
Lu	0.6%

Project: 18299-02
Client: Lofdal (NCM)

Date: 05-Aug-21
Technologist: M Rosborough

Test: C-RP3

Purpose: To convert REE oxalate into REE oxide by calcination

Sample: RP-3 Precip

H&S: Review MSDS for hot surfaces, use heat protective PPE.

Procedure:

1. Weigh feed sample and place in a tared crucible. Record the gross weight.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature.
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 hours before turning off the heat and allowing to cool within the furnace.
6. Allow the sample to cool to a safe temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.
8. Using a mortar and pestle, pulverize the calcine. Carefully discharge onto a piece of white paper and create a cone. Take a clear close-up photo of the material with the test ID in frame.

Conditions:	Target	Actual	
Feed Weight:	19.15	21.74	g
Total Weight:	19.15	21.74	
Cure Time:	-		
Preheat Temperature:	200	200	°C
Preheat Time:	1	1	h
Target Temperature:	1200	1200	°C
Test Time:	3	3	h
Sample Removal Temperature:	<100	Room Temp	°C

Results:		
Crucible Tare:	669.67	g
Starting Gross Weight:	691.41	g
Starting Net Weight:	21.74	g
Calcine Gross Weight:	**	g
Calcine Net Weight:	10.05	g
Weight Loss:	54%	
Colour:	brown	
Consistency:	powder	
Pulverized (yes/no):	no	

Assays:

#	Elements	Streams
1	calcine sample - REE, ICP, S	calcine

Observations:

Stage	Time	Comments:	Temp (°C)
heat	7:59	Heat on	RT
preheat	8:15	at preheat target	199
heat	9:15	preheat complete, set target for 1200°C	202
roast	10:10	at target temp, time zero	1198
roast	11:10		1200
cool	13:10	heat off, cool overnight with door closed	1200
out	7:50	sample out, bottom of tray flaking off unable to get full gross weight	RT

Project: 18299-02
Client: Lofdal (NCM)

Date: 05-Aug-21
Technologist: M Rosborough

Test: C-RP3

Assays:

Sample & Quant.	RP-3 Precip	C-RP3 Calcine
(mL or g)	22	10
La	11002	23800
Ce	19924	43100
Pr	2094	4530
Nd	7489	16200
Sm	3467	7500
Eu	2140	4630
Gd	13961	30200
Tb	3587	7760
Dy	28985	62700
Ho	6703	14500
Y	226056	489000
Er	21404	46300
Tm	2922	6320
Yb	17243	37300
Lu	2325	5030
Sc	<12	<25
Th	0.51	1.1
U	<0.2	<0.5
Si	878	1900
Al	<23	<50
Fe	32	70
Mg	1387	3000
Ca	740	1600
Na	1433	3100
K	<37	<80
Ti	<28	<60
P	18	40
Mn	<37	<80
Zn	20	44
S (%)		0.01
TREE (%)	36.9	79.9

Direct Assays

TREE 79.9 %
TREG 98.1 %

100% - Impurities

TREG 98.8 % (direct elemental minus)
TREG 98.1 % (converted impurities to oxides)
TREG 98.2 % (only above detection converted)

0.776

6.27

48.9

Add'n Elements (g/t)	
Ag	<50
As	<200
Ba	15
Be	0.90
Bi	105
Cd	<3
Co	<200
Cr	100
Li	<30
Mo	381
Nb	
Ni	99
Pb	<200
Sb	35
Se	<50
Sn	40
Sr	16.7
Ta	
Tl	<50
V	<60
Zr	
LOI (%)	0.49

Project: 18299-02
Client: Lofdal (NCM)

Date: 06-Aug-21
Technologist: M Rosborough

Test: C-RP4

Purpose: To convert REE carbonate into REE oxide by calcination

Sample: RP-4 Precip

H&S: Review MSDS for hot surfaces, use heat protective PPE.

Procedure:

1. Weigh feed sample and place in a tared crucible. Record the gross weight.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature.
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 hours before turning off the heat and allowing to cool within the furnace.
6. Allow the sample to cool to a safe temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.
8. Using a mortar and pestle, pulverize the calcine. Carefully discharge onto a piece of white paper and create a cone. Take a clear close-up photo of the material with the test ID in frame.

Conditions:	Target	Actual	
Feed Weight:	15.77	17.84	g
Total Weight:	15.77	17.84	
Cure Time:	-		
Preheat Temperature:	200	200	°C
Preheat Time:	1	1	h
Target Temperature:	1200	1200	°C
Test Time:	3	3	h
Sample Removal Temperature:	<100	RT	°C

Results:		
Crucible Tare:	651.78	g
Starting Gross Weight:	669.62	g
Starting Net Weight:	17.84	g
Calcine Gross Weight:	662.19	g
Calcine Net Weight:	10.4	g
Weight Loss:	42%	
Colour:	grey	
Consistency:	granular+powder	
Pulverized (yes/no):	no	

Assays:

#	Elements	Streams
1	calcine sample - REE, ICP, S	calcine

Observations:

Stage	Time	Comments:	Temp (°C)
heat	8:35	Heat on	RT
preheat	8:50	at preheat target	199
heat	9:50	preheat complete, set target for 1200°C	202
roast	10:47	at target temp, time zero	1197
cool	13:55	heat off, cool overnight with door closed	1201
out	8:50	sample out	RT

Project: 18299-02
Client: Lofdal (NCM)

Date: 06-Aug-21
Technologist: M Rosborough

Test: C-RP4

Assays:

Sample & Quant.	RP-4 Precip	C-RP4 Calcine
(mL or g)	18	10
La	13129	22500
Ce	23691	40600
Pr	2457	4210
Nd	8928	15300
Sm	4038	6920
Eu	2521	4320
Gd	16689	28600
Tb	4266	7310
Dy	34428	59000
Ho	7994	13700
Y	280090	480000
Er	25383	43500
Tm	3542	6070
Yb	20948	35900
Lu	2824	4840
Sc	14.588	<25
Th	1.1	1.9
U	<0.3	<0.5
Si	9920	17000
Al	3367	5770
Fe	117	200
Mg	2101	3600
Ca	4295	7360
Na	3384	5800
K	<47	<80
Ti	58	100
P	117	200
Mn	1400	2400
Zn	0	
S (%)		0.03
TREE (%)	45.1	77.3

Direct Assays

TREE 77.3 %
TREG 95.0 %

100% - Impurities

TREG 95.5 % (direct elemental minus)
TREG 92.3 % (converted impurities to oxides)
TREG 92.4 % (only above detection converted)

Add'n Elements (g/t)	
Ag	<50
As	<200
Ba	158
Be	18.9
Bi	151
Cd	<3
Co	<200
Cr	<70
Li	<30
Mo	225
Nb	
Ni	232
Pb	<200
Sb	41
Se	<50
Sn	51
Sr	214
Ta	
Tl	<50
V	<60
Zr	
LOI (%)	0.69

Appendix G:

An Investigation into Hydrometallurgical Recovery of Rare Earth Elements from the Lofdal Deposit
prepared for Namibia Critical Metals and dated October 7, 2022 from SGS Canada Inc.



An Investigation into
THE HYDROMETALLURGICAL RECOVERY OF RARE EARTH ELEMENTS FROM THE
LOFDAL DEPOSIT

prepared for

NAMIBIA CRITICAL METALS INC.

Project 18299-02 – Report 2
October 7, 2022

NOTES

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Executive Summary

One additional flotation sample produced from the Lofdal Deposit was tested following the first three process steps of a partially optimized flowsheet reported on January 20th, 2022 under SGS project number 18299-02. This new sample was subjected to acid baking at the optimum target acid dosage of 1250 kg/t as well as +/- 20% acid dosage (1500 kg/t and 1000 kg/t) under otherwise identical conditions, followed by water leaching. The resultant water leach liquors were combined for a single impurity removal test followed by a single crude rare earth precipitation test.

The goal of this test series was to confirm the conditions developed previously and to measure the reagent requirements associated with this new concentrate. The variability in assay results for the new concentrate and the two previous concentrates is shown in Table I.

Table I: Flotation Concentrate Assays

Sample ID	Date Rec'd	La, g/t	Ce, g/t	Pr, g/t	Nd, g/t	Sm, g/t	Eu, g/t	Gd, g/t	Tb, g/t	Dy, g/t
Conc 1	Mar-21	426	683	< 256	257	<431	--	--	145	1160
Conc 2	May-21	1950	3980	338	1330	725	435	2380	729	4860
Conc 3	Jun-22	3000	5540	582	2030	588	289	1510	367	2740
		Ho, g/t	Y, g/t	Er, g/t	Tm, g/t	Yb, g/t	Lu, g/t	Sc, g/t	Th, g/t	U, g/t
Conc 1	Mar-21	--	9056	--	--	--	--	--	1494	< 85
Conc 2	May-21	1150	38900	3560	494	3100	455	38	8240	154
Conc 3	Jun-22	601	18347	1750	247	1500	198	--	2970	88.4
		Si, %	Al, %	Fe, %	Mg, %	Ca, %	Na, %	K, %	Ti, %	P, %
Conc 1	Mar-21	12.0	2.99	18.7	1.00	12.4	1.12	0.81	0.935	0.450
Conc 2	May-21	10.0	1.66	22.8	0.25	6.58	0.979	0.15	3.84	2.03
Conc 3	Jun-22	9.82	2.25	21.7	0.838	6.64	1.40	0.14	5.41	1.29
		Mn, %	Cr, %	V, %	Ag, g/t	As, g/t	Ba, g/t	Be, g/t	Bi, g/t	Cd, g/t
Conc 1	Mar-21	0.24	0.01	0.04	< 2	< 40	405	7.35	< 20	< 2
Conc 2	May-21	0.39	0.03	0.10	--	--	--	--	--	--
Conc 3	Jun-22	0.51	0.089	0.067	--	--	--	--	--	--
		Co, g/t	Cu, g/t	Li, g/t	Mo, g/t	Ni, g/t	Pb, g/t	Sb, g/t	Se, g/t	Sn, g/t
Conc 1	Mar-21	118	42	27	< 40	< 70	< 30	< 50	< 30	< 20
Conc 2	May-21	--	--	--	--	--	--	--	--	--
Conc 3	Jun-22	--	--	--	--	--	--	--	--	--
		Sr, g/t	Tl, g/t	Zn, g/t	S, %	Cl, g/t				
Conc 1	Mar-21	179	< 30	38	0.03	--				
Conc 2	May-21	--	--	--	0.11	62				
Conc 3	Jun-22	--	--	--	0.09	--				

Rare earth element extraction in the acid bake with water leach testing was slightly lower with the newest flotation concentrate (Conc 3) compared to Conc 2 (96% vs. 98%) under the same test conditions. Increased acid addition (+20%) partially made up the difference, increasing REE dissolution to approximately 97%, but this would lead to higher reagent requirements in the acid bake and impurity removal steps.

Impurity removal and crude REE precipitation performance were similar to the optimized endpoint tests completed in the earlier test series. Reduced reagent requirements were observed with the new sample:

- MgCO_3 dosage in IR-5 = 381 kg/t concentrate basis; MgCO_3 dosage in IR-6 = 324 kg/t.
- H_2O_2 dosage in IR-5 = 4.7 kg/t concentrate basis; H_2O_2 dosage in IR-6 = 2.6 kg/t.
- Na_2CO_3 dosage in RP-2 = 90.5 kg/t concentrate basis; Na_2CO_3 dosage in RP-5 = 39.1 kg/t.

Crude REE precipitation generated an intermediate product assaying at 46% total REE with 3.44% Al, 0.34% Fe, 425 g/t Th, and 334 g/t U. Further optimization of the Impurity Removal and REE Precipitation steps along with integrated flowsheet recycle streams are expected to result in >95% recovery of dissolved REE into the crude REE precipitate.

These observations and measurements are the direct result of applying optimized conditions developed using a different flotation concentrate (Conc 2) and may not represent the optimum values for the current concentrate (Conc 3). Optimization testwork using the expected flotation concentrate through the entire conceptual flowsheet is recommended to refine these results.

Introduction

SGS Canada Inc. in Lakefield, Ontario was requested to expand upon an existing investigation into the recovery of rare earth elements (especially terbium and dysprosium) from flotation concentrates generated from samples originating from the Lofdal Deposit in Namibia by Ms. Barbara Mulcahy on behalf of Namibia Critical Metals Inc.

The program outlined in this report included concentrate receipt and characterization, acid bake with water leaching, acidic leach liquor impurity removal, and crude rare earth precipitation. The objective was to evaluate REE recovery from a new concentrate (with different REE distribution) employing test conditions developed in the previous phase of the project, reported in January 2022.

Throughout the test program, test proposals and results were shared with Ms. Barbara Mulcahy and Mr. Rainer Ellmies for review and comment.



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Testwork Summary

On the basis of results obtained during the optimization testing of the earlier test program reported on January 20, 2022, a third flotation concentrate generated from Lofdal material (“Conc 3”) was subjected to a series of three Acid Bake with Water Leach tests. The conditions for these three tests were identical save for acid dosage, which straddled the optimized acid dosage from previous testing (1250 kg/t).

The analytical assay results for the third concentrate are shown in Table 1, comparing these results to the first two flotation concentrates. Only the original concentrate was assayed by ICP-AES (Ag through Zn in the table).

Table 1: Flotation Concentrate Assay Summary

Sample ID	Date Rec'd	La, g/t	Ce, g/t	Pr, g/t	Nd, g/t	Sm, g/t	Eu, g/t	Gd, g/t	Tb, g/t	Dy, g/t
Conc 1	Mar-21	426	683	< 256	257	<431	--	--	145	1160
Conc 2	May-21	1950	3980	338	1330	725	435	2380	729	4860
Conc 3	Jun-22	3000	5540	582	2030	588	289	1510	367	2740
		Ho, g/t	Y, g/t	Er, g/t	Tm, g/t	Yb, g/t	Lu, g/t	Sc, g/t	Th, g/t	U, g/t
Conc 1	Mar-21	--	9056	--	--	--	--	--	1494	< 85
Conc 2	May-21	1150	38900	3560	494	3100	455	38	8240	154
Conc 3	Jun-22	601	18347	1750	247	1500	198	--	2970	88.4
		Si, %	Al, %	Fe, %	Mg, %	Ca, %	Na, %	K, %	Ti, %	P, %
Conc 1	Mar-21	12.0	2.99	18.7	1.00	12.4	1.12	0.81	0.935	0.450
Conc 2	May-21	10.0	1.66	22.8	0.25	6.58	0.979	0.15	3.84	2.03
Conc 3	Jun-22	9.82	2.25	21.7	0.838	6.64	1.40	0.14	5.41	1.29
		Mn, %	Cr, %	V, %	Ag, g/t	As, g/t	Ba, g/t	Be, g/t	Bi, g/t	Cd, g/t
Conc 1	Mar-21	0.24	0.01	0.04	< 2	< 40	405	7.35	< 20	< 2
Conc 2	May-21	0.39	0.03	0.10	--	--	--	--	--	--
Conc 3	Jun-22	0.51	0.089	0.067	--	--	--	--	--	--
		Co, g/t	Cu, g/t	Li, g/t	Mo, g/t	Ni, g/t	Pb, g/t	Sb, g/t	Se, g/t	Sn, g/t
Conc 1	Mar-21	118	42	27	< 40	< 70	< 30	< 50	< 30	< 20
Conc 2	May-21	--	--	--	--	--	--	--	--	--
Conc 3	Jun-22	--	--	--	--	--	--	--	--	--
		Sr, g/t	Tl, g/t	Zn, g/t	S, %	Cl, g/t				
Conc 1	Mar-21	179	< 30	38	0.03	--				
Conc 2	May-21	--	--	--	0.11	62				
Conc 3	Jun-22	--	--	--	0.09	--				

1. Acid Bake and Water Leach (AB) Testwork

1.1. AB Test Procedure

Three additional acid bake and water leach (AB) tests were completed in addition to the original 12 tests reported on January 20, 2022. These tests were designed to investigate the dissolution of rare earth elements (REE) and the behaviour of gangue minerals. Acid baking was conducted with sulphuric acid at elevated temperature (300°C). Acid was added to the feed solids and manually homogenized. Once blended, the crucible containing the mixture was placed in a furnace at ambient temperature and the furnace was gradually heated to the target temperature. After the furnace reached the operating temperature, the contents of the crucible were mixed (“rabbled”) once per hour during the three hours at

temperature. After three hours, the furnace was allowed to cool slightly before the baked calcine was removed and cooled to ambient temperature in preparation for water leaching. A summary of the acid bake conditions is presented in Table 2, including the bulk acid bake (AB12) conditions from the previous test program.

Table 2: Acid Bake Test Conditions Summary

Test ID	AB12 (Bulk)	AB13	AB14	AB15
Feed	Composite REE Concentrate (May 14, 2021)	Composite REE Concentrate (June 2022)	Composite REE Concentrate (June 2022)	Composite REE Concentrate (June 2022)
Reagent	H ₂ SO ₄	H ₂ SO ₄	H ₂ SO ₄	H ₂ SO ₄
Reagent Strength	96%	96%	96%	96%
Target Reagent Dose (kg/t)	1250	1250	1500	1000
Effective Reagent Dose (kg/t)	1194	1176	1430	921
Target Temperature (°C)	300	300	300	300
Test Time (h)	3	3	3	3

In Table 2, “effective dose” is calculated by assuming any weight loss during the initial mixing process of concentrate and acid is the result of acid vaporization prior to inserting the sample into the furnace.

The calcines were each subjected to water leaching under optimized conditions: a target of 20% solids (on acid bake feed basis) and agitated for two hours at ambient temperature. An intermediate liquor sample was taken from each test after one hour before final filtration after two hours. Intermediate solution samples and final wash liquors were analyzed for yttrium, neodymium, and iron. Final solution samples were analyzed by ICP-AES for a full scan, ICP-MS for a REE scan, and also analyzed for chloride and sulphate content. Final residues were analyzed for whole rock analysis (WRA) plus yttrium by X-Ray Fluorescence (XRF), chlorine and sulphur content, and REE scan by ICP-MS.

1.2. AB Test Results

The three new AB tests sought to confirm the optimized conditions determined in the previous test program and applied in test AB12, as reported on January 20, 2022. The current three tests varied only the acid dosage, with AB13 applying the optimized conditions, AB14 increasing the acid dosage, and AB15 decreasing the acid dosage.

The water-leach residue samples from these tests were assayed for rare earth elements by ICP-MS. Both light and heavy rare earth dissolution increased with increasing acid dosage. This trend was only observed with the heavy rare earth elements in the previous test program. When compared to the previous results obtained in AB12, dysprosium and terbium dissolution were similar at both the optimized acid dosage as well as the elevated acid dosage.

In addition to the REE, sulphuric acid-soluble impurities also dissolved during the water leach (Th, U, Al, Fe, Mg, K, Ti, P, Mn, V) while silicon, calcium, and sodium remained in the solids phase. These dissolved impurities can be removed through pH adjustment. Metal extractions for these three tests and AB12 are summarized in Table 3 and liquor assays are summarized in Figure 1 and Table 4.

Table 3: Acid Bake and Water Leach Extraction Summary

Element	WL-AB12	WL-AB13	WL-AB14	WL-AB15
La	95	96	97	94
Ce	96	96	96	94
Pr	96	94	95	92
Nd	95	94	96	92
Sm	92	88	91	84
Eu	93	88	91	84
Gd	94	91	93	87
Tb	95	92	94	89
Dy	95	93	94	89
Ho	96	92	94	90
Y	97	96	98	94
Er	96	93	95	90
Tm	96	92	94	90
Yb	96	92	94	89
Lu	96	91	93	89
Sc	57	-	-	-
Th	75	86	89	81
U	88	91	93	82
Si	2	0	0	0
Al	28	15	16	17
Fe	59	71	74	64
Mg	83	93	94	92
Ca	8	7	6	6
Na	6	1	1	3
K	62	68	68	67
Ti	64	38	51	25
P	60	39	54	34
Mn	79	78	81	74
Cr	0	71	80	59
V	65	57	61	46



Figure 1: Acid Bake and Water Leach Extraction

Table 4: Acid Bake and Water Leach Filtrate Assay Summary

Element	WL-AB12	WL-AB13	WL-AB14	WL-AB15
La	487	739	792	709
Ce	922	1360	1440	1300
Pr	107	141	150	136
Nd	381	489	506	461
Sm	188	148	158	136
Eu	121	68.5	73.4	63.3
Gd	656	374	404	351
Tb	183	90.2	97.2	86
Dy	1340	685	725	633
Ho	326	147	158	141
Y	10400	4840	5110	4590
Er	961	444	466	419
Tm	147	59.9	65.9	56.9
Yb	833	357	398	340
Lu	119	44	50.2	41.7
Sc	7.36	7.69	9.45	6.89
Th	1550	749	810	677
U	39.4	22.1	24.5	20.1
Al	1390	1170	1410	1240
Fe	39100	39800	45200	37000
Mg	517	1740	2080	1830
Ca	1370	1520	1310	1160
Na	130	74	83	137
K	269	253	266	242
Ti	7350	5670	8490	4070
P	3670	1340	2110	1310
Mn	839	819	1070	876
Cr	40.1	160	215	142
V	162	108	134	92
S	--	150000	200000	120000
Cl	1	2	5	4

Solutions from the three current tests (AB13-AB15) were combined for one confirmatory impurity removal (IR) test.

2. Impurity Removal (IR) Testwork

2.1. IR Test Procedure

One impurity removal test was completed on combined AB leach liquor following the same conditions as test IR-5 reported on January 20, 2022 (50°C, 120% stoichiometric peroxide addition, pH 2.9 with 20% magnesium carbonate, one hour). This test was a bulk “endpoint” test including a stoichiometric addition of hydrogen peroxide to oxidize an estimated 900 mg/L ferrous iron in solution to ferric (based on early testing with previous concentrates), in order for it to precipitate. In this test, the magnesium carbonate reagent was added to achieve a target of pH 2.9 and then maintained for one hour before filtration.

2.2. IR Test Results

Test IR-6 was designed as a confirmatory bulk endpoint test to produce liquor for further downstream testwork. This test aimed to achieve a final target of pH 2.9-3.0 with magnesium carbonate, along with the

addition of hydrogen peroxide to precipitate iron. In this test, all of the iron and thorium precipitated, along with 52% of the aluminum. Approximately 21% of the neodymium and 11% of the yttrium co-precipitated, with is greater than what was seen in test IR-5 (indicating that there is still room to optimize in order to achieve low REE losses in the range of the 3% observed at pH 2.9 in tests IR-1 and IR-4). The metallurgical balance for IR-6 can be seen in Table 5. The magnesium carbonate dosage needed to achieve the target pH was 324 kg/t (based on flotation concentrate), a value that was slightly lower than the value reported in test IR-5. Manganese-tie calculations in which manganese is assumed to remain fully in solution allow for an alternative method of determining the extent of precipitation of other metals. This calculation assumes that the change in manganese concentration between the feed and final liquors is the result of dilution and applies the same dilution factor to all other assayed elements with any variance from that calculated dilution deemed to be the result of precipitation. In test IR-6, manganese-tie calculations showed a 4% precipitation of total REE.

Table 5: IR-6 Summary Metallurgical Balance

Sample & Quant.	Assay Units	Feed	Final Filt	Final Wash	Final PPT	PPT % Res/Feed	Calc Head	Acc %
(mL or g)		1197	1022.5	1849.8	194.6			
La	mg/L, g/t	670	452		0.09	22.8	539	80
Ce	mg/L, g/t	1280	823		0.20	24.9	1022	80
Pr	mg/L, g/t	133	84.8		<0.03	--	114	86
Nd	mg/L, g/t	458	289	91.8	0.06	21.3	486	106
Sm	mg/L, g/t	139	87.2		<0.04	--	145	104
Eu	mg/L, g/t	62.7	40.6			--	35	55
Gd	mg/L, g/t	363	244			--	208	57
Tb	mg/L, g/t	89.4	59.6			--	51	57
Dy	mg/L, g/t	624	410			--	350	56
Ho	mg/L, g/t	146	99.2			--	85	58
Y	mg/L, g/t	4640	3090	966	0.31	11.0	4644	100
Er	mg/L, g/t	434	296			--	253	58
Tm	mg/L, g/t	58.7	39.2			--	33	57
Yb	mg/L, g/t	343	228			--	195	57
Lu	mg/L, g/t	44.2	29.3			--	25	57
Sc	mg/L, g/t	9.32	1.74			--	1	16
Th	mg/L, g/t	664	6.31	2.14	0.50	122.6	823	124
U	mg/L, g/t	20.7	6.20		0.02	133.2	33	159
Si	mg/L, %				0.25	--	403	--
Al	mg/L, %	1260	508	151	0.402	51.9	1321	105
Fe	mg/L, %	40700	44.8	15.8	27.1	108.4	44176	109
Mg	mg/L, %	1850	21400		1.69	--	--	--
Ca	mg/L, %	1030	861		0.18	28.2	1026	100
Na	mg/L, %	91	62		0.03	53.0	101	111
K	mg/L, %	272	165		0.03	19.8	195	72
Ti	mg/L, %	6090	10.2		4.23	112.9	6887	113
P	mg/L, %	1130	<5		1.06	153.2	1735	154
Mn	mg/L, %	902	622		0.05	8.4	607	67
Cr	mg/L, %	169	1.1		0.11	105.3	179	106
V	mg/L, %	109	<0.2		0.08	117.0	128	117
Cu	mg/L, %	<40	<40			0.0	34	85
Pb	mg/L, %	<6	<6			0.0	5	85
Zn	mg/L, %	7.6	6.5			0.0	6	73
TREE	mg/L, %	10179	6274	1058	0.0	--	6993	69
LREE	mg/L, %	2680	1736	92	0.0	--	1625	61
HREE	mg/L, %	6805	4536	966	0.0	--	5367	79

3. Crude Rare Earth Precipitation Testwork

3.1. Rare Earth Precipitation (RP) Test Procedure

One crude rare earth precipitation test (RP-5) was completed on IR-6 filtrate following the same conditions as test RP-2 reported on January 20, 2022 (ambient temperature, pH 6.5 with 100 g/L sodium carbonate, one hour).

3.2. Rare Earth Precipitation (RP) Test Results

The bulk endpoint test (RP-5) successfully precipitated the rare earth elements along with impurity elements (Th, U, Al, Fe). The final residue assayed 46% total REE and 3.44% aluminum. The precipitate also contained 0.34% Fe, 425 g/t Th, and 334 g/t U (similar to the results of RP-2). The metallurgical balance for this test is included in Table 6. Sodium carbonate dosage to achieve the target pH was 39.1 kg/t (based

on flotation concentrate), a value that was slightly higher than the calculated estimate of 34 kg/t. Magnesium-tie calculations (similar to manganese-tie calculations outlined in the IR Test Results section) showed a 99% precipitation of total REE in test RP-5.

Table 6: RP-5 Summary Metallurgical Balance

Sample & Quant.	Assay Units	Feed	Final Filt	Final Wash	Final PPT	PPT % Res/Feed	Calc Head	Acc %
(mL or g)		975	1096.7	1885.8	12.6			
La	mg/L, %	452	0.86		3.33	95.2	431	95
Ce	mg/L, %	823	1.00		5.9	92.6	764	93
Pr	mg/L, %	84.8	0.10		0.604	92.0	78	92
Nd	mg/L, %	289	0.35		2.11	94.3	273	94
Sm	mg/L, %	87.2	0.15		0.631	93.5	82	94
Eu	mg/L, %	40.6	0.06		0.292	92.9	38	93
Gd	mg/L, %	244	0.59		1.69	89.5	219	90
Tb	mg/L, %	59.6	0.19		0.391	84.8	51	85
Dy	mg/L, %	410	1.92		3.05	96.1	396	97
Ho	mg/L, %	99.2	0.66		0.658	85.7	86	86
Y	mg/L, %	3090	50.7	2.96		98.0	3090	
Er	mg/L, %	296	2.61		1.97	86.0	257	87
Tm	mg/L, %	39.2	0.45		0.26	85.7	34	87
Yb	mg/L, %	228	2.94		1.53	86.7	201	88
Lu	mg/L, %	29.3	0.44		0.198	87.3	26	89
Sc	mg/L, %	1.74	<0.07			89.5	2	
Th	mg/L, %	6.31	<0.03		0.0425	87.0	6	88
U	mg/L, %	6.20	1.15		0.0334	69.6	6	90
Si	mg/L, %				0.19	--	25	--
Al	mg/L, %	508	0.2		3.44	87.5	445	88
Fe	mg/L, %	44.8	0.5		0.34	96.8	44	98
Mg	mg/L, %	21400	19400		0.25	0.2	21853	102
Ca	mg/L, %	861	705		0.21	3.2	821	95
Na	mg/L, %	62	4960		0.18	--	--	--
K	mg/L, %	165	146		0.008	0.7	165	100
Ti	mg/L, %	10.2	<0.02		0.06	75.9	8	76
P	mg/L, %	<5	<5		0.013	--	--	--
Mn	mg/L, %	622	513		0.17	3.5	599	96
Cr	mg/L, %	1.1	<0.1		<0.007	--	--	--
V	mg/L, %	<0.2	<0.2		<0.006	--	--	--
Cu	mg/L, %	<40	0.2			--	--	1
Pb	mg/L, %	<6	<2			--	--	37
Zn	mg/L, %	6.5	2.4			--	--	42
TREE	mg/L, %	6272	63	3	46.0	94.9	6026	96.1
LREE	mg/L, %	1736	2	0	12.6	93.6	1628	93.8
HREE	mg/L, %	4536	61	3	33.5	95.3	4398	97.0

Conclusions and Recommendations

Based on the testwork results reported herein, the following conclusions can be made:

- The flowsheet developed in previous testing was capable of extracting 95% REE (93% Dy and 92% Tb) from the new concentrate, with the demonstrated ability to increase to at least 96% REE (96% Tb and Dy) by increasing acid dosage by 20% (from baseline 1250 kg/t to 1500 kg/t on a concentrate basis).
- The leach liquor contained approximately 10 g/L TREE and 37-45 g/L iron, along with lower levels of other impurities.
- Neutralization of the leach liquor with magnesium carbonate at pH 2.9 along with stoichiometric peroxide to oxidize the dissolved iron to the ferric state, precipitated most of the iron and thorium, along with 52% of the aluminum. However, approximately 21% of the neodymium and 11% of the yttrium co-precipitated at this pH. Further optimization of this step in the process is required, such as an examination of a two-stage counter-current precipitation process since this may allow for more efficient impurity precipitation (particularly aluminum) with lower REE losses.
- Raising the pH of the liquor after impurity removal to 6.5 with sodium carbonate successfully precipitated all of the REE from solution. Residual impurities (Sc, Th, U, Al, Fe, Ti) also precipitated under these conditions. Soda ash consumption was ~40 kg/t.
- A crude rare earth precipitate was produced containing 46% REE (estimated 23% Y, assayed 0.4% Tb and 3.1% Dy). Key impurities included 425 g/t Th, 334 g/t U, 3.44% Al, 0.25% Mg, 0.19% Si, 0.21% Ca, 0.34% Fe, and 0.18% Na. These impurities matched those measured in the crude REE precipitate generated previously, with the exception of aluminum, which was higher. Previous testing demonstrated that most of the precipitated Th, Al, Si, and Fe can be removed by re-leaching, while U and Th can be removed by ion exchange and solvent extraction (respectively), and the remaining impurities can largely be removed by oxalic acid precipitation.

Results from these tests can be used as a general guideline to estimate the performance of each step and the final rare earth precipitate purity. A much larger sample of the expected flotation concentrate would allow for optimization of every process step and would provide better estimates of reagent demands at each step as well as meaningful production of final rare earth precipitate/calcine.

Appendix A – Test Sheets

Project: 18299-02
Client: Lofdal (NCM)

Date: 29-Jun-22
Technologist: M Rosborough

Test: AB13

Purpose: To static acid bake REE concentrate prior to water leaching - heated ore and acid contact

Sample: Composite REE Concentrate (June 2022) **(Conc 3)**

H&S: Review MSDS for H₂SO₄, hot surfaces, use heat protective PPE

Procedure:

1. Mix feed sample with the required amount of reagent in a tared crucible. Once blended, allow to cure.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature (if required).
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 h, rabbling every 60 min, before removing from the furnace while hot.
6. Allow the sample to cool to room temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.

Conditions:	Target	Actual	
Feed Weight:	200	200.70	
Reagent:	H ₂ SO ₄	H ₂ SO ₄	
Reagent Strength:	96%	96%	
Reagent Dosage:	1250	1248	kg/t
Reagent to Add:	260	260.9	g
Total Weight:	460	461.6	g
Cure Time:	0	0.17	h
Preheat Temperature:	-	-	°C
Preheat Time:	-	-	h
Target Temperature:	300	300	°C
Test Time:	3	3	h
Sample Removal Temperature:	<100	58	°C

Results:

Crucible Tare:	666.98	g
Starting Gross Weight:	1113.6	g
Starting Net Weight:	446.62	g
Calcine Gross Weight:	987.8	g
Calcine Net Weight:	320.82	g
Weight Loss:	28%	
Colour:		
Consistency:		
Pulverized (yes/no):	no	

245.92 g acid effective dose

1176.299 kg/t effective dose

30% overall weight loss

Observations:

Stage	Time	Comments:
mix	10:05	Significant fuming/foaming/heat generation during addition
heat	10:15	furnace on to 300°C sample in
roast	10:49	Furnace at 300°C
roast	11:55	Significant fuming
roast	13:00	Significant fuming
cool	13:49	Furnace SP down to 0°C
out	8:45	Sample out

Project: 18299-02
Client: Lofdal (NCM)

Date: 30-Jun-22
Technologist: M Rosborough

Test: WL-AB13

Purpose: To water leach an acid bake calcine

Sample: AB13 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements
3	liquor samples - Y, Nd, Fe GC_SOL84T_Fe, GC_SOL94T_Nd, GC_SOL94T-AE_Y
1	liquor samples - ICP, REE, Cl, SO4 (Enviro) GC_SOL91T, GC_SOL94T, GC_CLA80T, ENV SO4
1	residue sample - WRA+Y2O3, REE by ICP, S, Cl GC_XRF76V, GC_XRF76V_Y2O3, GC_IMS93A, GC_CSA06V (total), GC_CLA27E

Conditions:

AB13 Feed:	200	g
H2SO4 added:	250.464	g
Net Calcine Weight:	320.82	g
Calcine Feed to Test:	320	g
Calculated Equivalent Feed:	199	g
Calculated Equivalent H2SO4:	250	g
Target % Solids (vs. Feed):	20%	
Calc. Pulp Weight:	997	g
DI Water to add:	677	g
Resultant Pulp Density:	32%	(relative to calcine)
Test Time:	2	h
Temperature:	25	°C

Project: 18299-02
 Client: Lofdal (NCM)

Date: 30-Jun-22
 Technologist: M Rosborough

Test: WL-AB13

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
11:28	0.0	22.1	0.99	507	321	682	450 RPM
11:40	0.2	24.6	-0.36	537			mixing on, feed addition
11:55	0.4	28.3	-0.62	568			
12:28	1.0	31.1	-0.87	588			sample 1
12:58	1.5	31.7	-0.94	591			
13:28	2.0	31.3	-0.88	591			end test, filter
Totals/Avg.		28.2	-0.45	563.67	321	682	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h	1	73.2	51.3	1.1672	44	562	1.12			slow	
Final	2	948.1	625.7	1.2019	521	560	1.04			slow	13.3%
Wash			1409.8	1.0338	1364	555	1.56	185.99	126.02	slow	

Free Acid Data:

Sample #	Aliquot mL	Titration		Which Acid	Stoich mol/mol	MW g/mole	g/L acid	g acid
		N	mL					
1 h	2	0.2	5.52	H2SO4	2	98.1	27	1.2
Final	2	0.2	6.67	H2SO4	2	98.1	33	17.0
Wash	10	0.2	6.33	H2SO4	2	98.1	6	8.5

Final Filtration/Washing:

Diameter of filtration paper:	150	mm
Type of Paper (Whatman #):	3	
Filtration Time:	overnight	minutes
Washing Time:	95	minutes
Cake Moisture:	32%	
Weight Loss:	23%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	brown
Clarity of Wash:	hazy
Colour of Wash:	light brown
Colour of Residue:	greyish brown

Residue: Total

Tare:	10.5 g
Tare + Wet:	196.5 g
Tare + Dry:	136.6 g

Acid Addition	1252	kg/t
Acid Remaining	133	kg/t
Acid Consumed	1119	kg/t

Comments:

close vacuum valve to flask overnight, filtration complete by AM

Project: 18299-02
Client: Lofdal (NCM)

Date: 30-Jun-22
Technologist: M Rosborough

Test: WL-AB13

Metallurgical Balance

Sample & Quant.	Assay Units	AB13 Feed	1 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		199	44	521	1364	126
La	mg/L, g/t	3000		739		123
Ce	mg/L, g/t	5540		1360		264
Pr	mg/L, g/t	582		141		36.5
Nd	mg/L, g/t	2030	498	489	57.2	180
Sm	mg/L, g/t	588		148		86.0
Eu	mg/L, g/t	289		68.5		37.0
Gd	mg/L, g/t	1510		374		159
Tb	mg/L, g/t	367		90.2		32.4
Dy	mg/L, g/t	2740		685		228
Ho	mg/L, g/t	601		147		49.7
Y	mg/L, g/t	18300	4820	4840	579	1102
Er	mg/L, g/t	1750		444		140
Tm	mg/L, g/t	247		59.9		20.4
Yb	mg/L, g/t	1500		357		125
Lu	mg/L, g/t	198		44.0		17.2
Sc	mg/L, g/t			7.69		
Th	mg/L, g/t	2970		749		509
U	mg/L, g/t	88		22.1		9.3
Si	mg/L, %	9.82				14.9
Al	mg/L, %	2.25		1170		2.69
Fe	mg/L, %	21.7	32200	39800	5670	9.72
Mg	mg/L, %	0.838		1740		0.054
Ca	mg/L, %	6.64		1520		8.50
Na	mg/L, %	1.40		74		2.14
K	mg/L, %	0.14		253		0.05
Ti	mg/L, %	5.41		5670		3.85
P	mg/L, %	1.29		1340		0.877
Mn	mg/L, %	0.51		819		0.093
Cr	mg/L, %	0.089		160		0.03
V	mg/L, %	0.067		108		0.03
S / SO4	mg/L, %	0.09		150000		8.10
Cl	mg/L, g/t	55		2		33

Extract.	Account. out/in	Calc Head	Si-tie
%	%		%
96	67	2006	97%
96	67	3716	97%
94	67	391	96%
94	93	1891	94%
88	75	441	90%
88	70	202	92%
91	71	1076	93%
92	70	256	94%
93	70	1932	95%
92	69	415	95%
96	100	18347	96%
93	71	1247	95%
92	69	169	95%
92	67	1011	94%
91	63	126	94%
-	-	20	
86	77	2276	89%
91	72	64	93%
0	96	9	
15	89	2	21%
71	97	21	70%
93	58	0	96%
7	87	6	15%
1	98	1	-1%
68	69	0	77%
38	72	4	53%
39	70	1	55%
78	53	0	88%
71	66	0	80%
57	74	0	67%
72	44	18	
20	47	26	

9.99 g/L TREE in final filtrate

Project: 18299-02
Client: Lofdal (NCM)

Date: 29-Jun-22
Technologist: M Rosborough

Test: AB14

Purpose: To static acid bake REE concentrate prior to water leaching - heated ore and acid contact

Sample: Composite REE Concentrate (June 2022) **(Conc 3)**

H&S: Review MSDS for H₂SO₄, hot surfaces, use heat protective PPE

Procedure:

1. Mix feed sample with the required amount of reagent in a tared crucible. Once blended, allow to cure.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature (if required).
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 h, rabbling every 60 min, before removing from the furnace while hot.
6. Allow the sample to cool to room temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.

Conditions:	Target	Actual	
Feed Weight:	200	200.31	
Reagent:	H ₂ SO ₄	H ₂ SO ₄	
Reagent Strength:	96%	96%	
Reagent Dosage:	1500	1500	kg/t
Reagent to Add:	313	312.9	g
Total Weight:	513	513.21	g
Cure Time:	0	0.12	h
Preheat Temperature:	-	-	°C
Preheat Time:	-	-	h
Target Temperature:	300	300	°C
Test Time:	3	3	h
Sample Removal Temperature:	<100	58	°C

Results:

Crucible Tare:	719.28	g
Starting Gross Weight:	1218	g
Starting Net Weight:	498.72	g
Calcine Gross Weight:	1055.8	g
Calcine Net Weight:	336.52	g
Weight Loss:	33%	
Colour:		
Consistency:		
Pulverized (yes/no):	no	

298.41 g acid effective dose

1430.151 kg/t effective dose

34% overall weight loss

Observations:

Stage	Time	Comments:
mix	10:08	Significant fuming/foaming/heat generation during addition
heat	10:15	furnace on to 300°C sample in
roast	10:49	Furnace at 300°C
roast	11:55	Significant fuming
roast	13:00	Significant fuming
cool	13:49	Furnace SP down to 0°C
out	8:45	Sample out

Project: 18299-02
Client: Lofdal (NCM)

Date: 30-Jun-22
Technologist: M Rosborough

Test: WL-AB14

Purpose: To water leach an acid bake calcine

Sample: AB14 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements
3	liquor samples - Y, Nd, Fe GC_SOL84T_Fe, GC_SOL94T_Nd, GC_SOL94T-AE_Y
1	liquor samples - ICP, REE, Cl, SO4 (Enviro) GC_SOL91T, GC_SOL94T, GC_CLA80T, ENV SO4
1	residue sample - WRA+Y2O3, REE by ICP, S, Cl GC_XRF76V, GC_XRF76V_Y2O3, GC_IMS93A, GC_CSA06V (total), GC_CLA27E

Conditions:

AB14 Feed:	200	g
H2SO4 added:	300.384	g
Net Calcine Weight:	336.52	g
Calcine Feed to Test:	336	g
Calculated Equivalent Feed:	200	g
Calculated Equivalent H2SO4:	300	g
Target % Solids (vs. Feed):	20%	
Calc. Pulp Weight:	998	g
DI Water to add:	662	g
Resultant Pulp Density:	34%	(relative to calcine)
Test Time:	2	h
Temperature:	25	°C

Project: 18299-02
 Client: Lofdal (NCM)

Date: 30-Jun-22
 Technologist: M Rosborough

Test: WL-AB14

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
11:40	0.0	23.2	1.18	498	335	663	450 RPM
11:55	0.2	28.8	0.51	569			mixing on, feed addition
12:10	0.5	31.8	0.32	587			
12:40	1.0	34.4	0.12	594			sample 1
13:10	1.5	33.9	0.07	592			
13:40	2.0	33.5	0.04	589			end test, filter
Totals/Avg.		30.9	0.37	571.50	335	663	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h	1	72.4	48.3	1.1999	40	558	0.96			slow	
Final	2	941.7	575.4	1.244	463	545	0.88			slow	12.6%
Wash			1430.5	1.0327	1385	550	1.53	180.92	118.82	slow	

Free Acid Data:

Sample #	Aliquot mL	Titration		Which Acid	Stoich mol/mol	MW g/mole	g/L acid	g acid
		N	mL					
1 h	2	0.2	8.52	H2SO4	2	98.1	42	1.7
Final	2	0.2	11.14	H2SO4	2	98.1	55	25.3
Wash	10	0.2	6.69	H2SO4	2	98.1	7	9.1

Final Filtration/Washing:

Diameter of filtration paper:	150	mm
Type of Paper (Whatman #):	3	
Filtration Time:	overnight	minutes
Washing Time:	120	minutes
Cake Moisture:	34%	
Weight Loss:	24%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	brown
Clarity of Wash:	hazy
Colour of Wash:	light brown
Colour of Residue:	greyish brown

Residue: Total

Tare:	10.6 g
Tare + Wet:	191.5 g
Tare + Dry:	129.4 g

Acid Addition	1502	kg/t
Acid Remaining	180	kg/t
Acid Consumed	1322	kg/t

Comments:

close vacuum valve to flask overnight, filtration almost complete restarted in AM for ~1hr more

Project: 18299-02
Client: Lofdal (NCM)

Date: 30-Jun-22
Technologist: M Rosborough

Test: WL-AB14

Metallurgical Balance

Sample & Quant.	Assay Units	AB14 Feed	1 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		200	40	463	1385	119
La	mg/L, g/t	3000		792		105
Ce	mg/L, g/t	5540		1440		212
Pr	mg/L, g/t	582		150		29.0
Nd	mg/L, g/t	2030	492	506	64.4	132
Sm	mg/L, g/t	588		158		63
Eu	mg/L, g/t	289		73.4		27.8
Gd	mg/L, g/t	1510		404		121
Tb	mg/L, g/t	367		97.2		24.8
Dy	mg/L, g/t	2740		725		185
Ho	mg/L, g/t	601		158		38.2
Y	mg/L, g/t	18300	5000	5110	624	709
Er	mg/L, g/t	1750		466		105
Tm	mg/L, g/t	247		65.9		16.2
Yb	mg/L, g/t	1500		398		94.8
Lu	mg/L, g/t	198		50.2		13.6
Sc	mg/L, g/t			9.45		
Th	mg/L, g/t	2970		810		396
U	mg/L, g/t	88		24.5		7.1
Si	mg/L, %	9.82				16.2
Al	mg/L, %	2.25		1410		2.78
Fe	mg/L, %	21.7	36400	45200	5320	8.74
Mg	mg/L, %	0.838		2080		0.054
Ca	mg/L, %	6.64		1310		8.72
Na	mg/L, %	1.40		83		2.23
K	mg/L, %	0.14		266		0.05
Ti	mg/L, %	5.41		8490		3.16
P	mg/L, %	1.29		2110		0.694
Mn	mg/L, %	0.51		1070		0.101
Cr	mg/L, %	0.089		215		0.02
V	mg/L, %	0.067		134		0.03
S / SO4	mg/L, %	0.09		200000		8.11
Cl	mg/L, g/t	55		5		50

Extract.	Account. out/in	Calc Head	Si-tie
%	%		%
97	63	1897	98%
96	62	3462	98%
95	63	365	97%
96	88	1796	96%
91	69	404	93%
91	65	187	94%
93	67	1008	95%
94	65	240	96%
94	65	1789	96%
94	65	389	96%
98	96	17594	98%
95	65	1142	96%
94	66	162	96%
94	65	978	96%
93	63	124	96%
-	-	22	
89	71	2112	92%
93	69	61	95%
0	98	10	
16	88	2	25%
74	93	20	76%
94	61	1	96%
6	83	5	21%
1	96	1	4%
68	65	0	79%
51	71	4	65%
54	70	1	67%
81	60	0	88%
80	70	0	86%
61	76	0	70%
76	41	20	
28	75	41	

10.59 g/L TREE in final filtrate

Project: 18299-02
Client: Lofdal (NCM)

Date: 29-Jun-22
Technologist: M Rosborough

Test: AB15

Purpose: To static acid bake REE concentrate prior to water leaching - heated ore and acid contact

Sample: Composite REE Concentrate (June 2022) **(Conc 3)**

H&S: Review MSDS for H₂SO₄, hot surfaces, use heat protective PPE

Procedure:

1. Mix feed sample with the required amount of reagent in a tared crucible. Once blended, allow to cure.
2. Place the sample into the furnace at room temperature, and heat up to the preheat temperature (if required).
3. Keep at the preheat temperature for the prescribed time.
4. Increase the temperature of the furnace to the target, recording the time required to reach target.
5. Hold the furnace at temperature for 3 h, rabbling every 60 min, before removing from the furnace while hot.
6. Allow the sample to cool to room temperature before handling (place in a bag if left overnight).
7. Record the crucible gross weight, remove the sample, and hold for further work.

Conditions:	Target	Actual	
Feed Weight:	200	200.07	
Reagent:	H ₂ SO ₄	H ₂ SO ₄	
Reagent Strength:	96%	96%	
Reagent Dosage:	1000	998	kg/t
Reagent to Add:	208	207.9	g
Total Weight:	408	407.97	g
Cure Time:	0	0.05	h
Preheat Temperature:	-	-	°C
Preheat Time:	-	-	h
Target Temperature:	300	300	°C
Test Time:	3	3	h
Sample Removal Temperature:	<100	58	°C

Results:

Crucible Tare:	653.89	g
Starting Gross Weight:	1046	g
Starting Net Weight:	392.11	g
Calcine Gross Weight:	955.5	g
Calcine Net Weight:	301.61	g
Weight Loss:	23%	
Colour:		
Consistency:		
Pulverized (yes/no):	no	

192.04 g acid effective dose

921.4695 kg/t effective dose

26% overall weight loss

Observations:

Stage	Time	Comments:
mix	10:12	Significant fuming/foaming/heat generation during addition
heat	10:15	furnace on to 300°C sample in
roast	10:49	Furnace at 300°C
roast	11:55	Significant fuming
roast	13:00	Significant fuming
cool	13:49	Furnace SP down to 0°C
out	8:45	Sample out

Project: 18299-02
Client: Lofdal (NCM)

Date: 30-Jun-22
Technologist: M Rosborough

Test: WL-AB15

Purpose: To water leach an acid bake calcine

Sample: AB15 Calcine

Procedure:

1. Add the calculated amount of lixiviant to a suitably sized reactor and heat to the target temperature. While mixing add the target amount of feed. Ensure that mixing is vigorous to suspend the solids (>500 rpm is expected).
2. Note the time when all feed has been charged to the reactor; this is time zero. No further acid additions are to be made.
3. Collect solution samples at the indicated times. Return solids to reactor.
4. After the required time at temperature, stop the test and weigh the pulp before filtering. Collect the filtrate and submit a sample for assay. Measure the pH, ORP, and density of the filtrate.
5. Repulp wash the filter cake for 20 minutes in a volume of water similar to the amount initially used in the test.
6. Filter again, and displacement wash three times. Combine all washes, weigh, and submit a sample for assay. Measure the pH, ORP, and density of the combined wash.
7. Record the wet weight of the solids and collect a cut for moisture determination and assay. Record the wet and dry weights of the assay cut before submitting.

Assays:

#	Elements
3	liquor samples - Y, Nd, Fe GC_SOL84T_Fe, GC_SOL94T_Nd, GC_SOL94T-AE_Y
1	liquor samples - ICP, REE, Cl, SO4 (Enviro) GC_SOL91T, GC_SOL94T, GC_CLA80T, ENV SO4
1	residue sample - WRA+Y2O3, REE by ICP, S, Cl GC_XRF76V, GC_XRF76V_Y2O3, GC_IMS93A, GC_CSA06V (total), GC_CLA27E

Conditions:

AB15 Feed:	200	g
H2SO4 added:	199.584	g
Net Calcine Weight:	301.61	g
Calcine Feed to Test:	301	g
Calculated Equivalent Feed:	200	g
Calculated Equivalent H2SO4:	199	g
Target % Solids (vs. Feed):	20%	
Calc. Pulp Weight:	998	g
DI Water to add:	697	g
Resultant Pulp Density:	30%	(relative to calcine)
Test Time:	2	h
Temperature:	25	°C

Project: 18299-02
Client: Lofdal (NCM)

Date: 30-Jun-22
Technologist: M Rosborough

Test: WL-AB15

Test Data:

Time		Reactor			Reagents / Feed		Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	H2O g	
11:55	0.0	22.2	1.77	531	296	695	675 RPM
12:10	0.3	25.5	1.22	526			mixing on, feed addition
12:28	0.6	27.6	1.06	549			
12:55	1.0	31.3	0.85	574			sample 1
13:25	1.5	32.6	0.72	584			
13:55	2.0	32.6	0.66	586			end test, filter
Totals/Avg.		28.6	1.05	558.33	296	695	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration fast/slow	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
1 h	1	71.5	51.4	1.134	45	565	1.13			slow	
Final	2	934.8	654.7	1.1755	557	567	1.05			slow	14.4%
Wash			1233.6	1.0242	1204	568	1.59	194.59	135.06	slow	

Free Acid Data:

Sample #	Aliquot mL	Titration		Which Acid	Stoich mol/mol	MW g/mole	g/L acid	g acid
		N	mL					
1 h	2	0.2	2.63	H2SO4	2	98.1	13	0.6
Final	2	0.2	3.85	H2SO4	2	98.1	19	10.5
Wash	10	0.2	4.31	H2SO4	2	98.1	4	5.1

Final Filtration/Washing:

Diameter of filtration paper:	150	mm
Type of Paper (Whatman #):	3	
Filtration Time:	33	minutes
Washing Time:	overnight	minutes
Cake Moisture:	31%	
Weight Loss:	22%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	brown
Clarity of Wash:	clear
Colour of Wash:	light brown
Colour of Residue:	greyish brown

Residue: Total

Tare:	10.5 g
Tare + Wet:	205.1 g
Tare + Dry:	145.5 g

Acid Addition	998	kg/t
Acid Remaining	81	kg/t
Acid Consumed	917	kg/t

Comments:

Repulp wash on in PM vacuum shut off overnight, complete displacement washes in AM

Project: 18299-02
Client: Lofdal (NCM)

Date: 30-Jun-22
Technologist: M Rosborough

Test: WL-AB15

Metallurgical Balance

Sample & Quant.	Assay Units	AB15 Feed	1 h Filtrate	Final Filtrate	Wash	Final Residue
(mL or g)		200	45	557	1204	135
La	mg/L, g/t	3000		709		176
Ce	mg/L, g/t	5540		1300		369
Pr	mg/L, g/t	582		136		51.8
Nd	mg/L, g/t	2030	464	461	49.6	230
Sm	mg/L, g/t	588		136		110
Eu	mg/L, g/t	289		63.3		49.5
Gd	mg/L, g/t	1510		351		209
Tb	mg/L, g/t	367		86.0		45.5
Dy	mg/L, g/t	2740		633		315
Ho	mg/L, g/t	601		141		66.3
Y	mg/L, g/t	18300	4460	4590	479	1575
Er	mg/L, g/t	1750		419		194
Tm	mg/L, g/t	247		56.9		27.0
Yb	mg/L, g/t	1500		340		169
Lu	mg/L, g/t	198		41.7		22.3
Sc	mg/L, g/t			6.89		
Th	mg/L, g/t	2970		677		655
U	mg/L, g/t	88		20.1		18.1
Si	mg/L, %	9.82				13.6
Al	mg/L, %	2.25		1240		2.49
Fe	mg/L, %	21.7	26200	37000	3880	10.9
Mg	mg/L, %	0.838		1830		0.066
Ca	mg/L, %	6.64		1160		7.86
Na	mg/L, %	1.40		137		1.95
K	mg/L, %	0.14		242		0.05
Ti	mg/L, %	5.41		4070		5.05
P	mg/L, %	1.29		1310		1.05
Mn	mg/L, %	0.51		876		0.124
Cr	mg/L, %	0.089		142		0.04
V	mg/L, %	0.067		92		0.04
S / SO4	mg/L, %	0.09		120000		7.59
Cl	mg/L, g/t	55		4		25

Extract.	Account. out/in	Calc Head	Si-tie
%	%		%
94	70	2097	96%
94	70	3877	95%
92	71	415	94%
92	91	1847	92%
84	77	454	86%
84	73	210	88%
87	74	1121	90%
89	74	271	91%
89	72	1979	92%
90	73	438	92%
94	97	17777	94%
90	74	1300	92%
90	72	177	92%
89	71	1063	92%
89	66	131	92%
-	-	19	
81	79	2332	84%
82	77	68	85%
0	94	9	
17	90	2	20%
64	95	21	64%
92	66	1	94%
6	85	6	15%
3	97	1	0%
67	72	0	75%
25	84	5	33%
34	83	1	41%
74	64	0	83%
59	76	0	67%
46	83	0	52%
69	50	16	
40	51	28	

9.46 g/L TREE in final filtrate

Project: 18299-02
Client: Lofdal (NCM)

Date: 4-Aug-22
Technologist: Ella Mihailescu

Test: IR-6

Purpose: Bulk Endpoint Impurity Removal test on combined Acid Bake-Water Leach Liquor (from Conc 3) with MgCO₃

Sample: Conc 3 Combined ABWL PLS (combine PLS from WL-AB13 through WL-AB15)
 CC000129811, NORM Lab Shelf 1-A

H&S: Refer to H₂SO₄, H₂O₂ and magnesium carbonate MSDS
 Feed solution is corrosive, avoid contact with skin and clean all spills accordingly

Procedure:

1. Secure feed solutions, decant if there are visible solids in the container. Filter if required. 600 g MgCO₃ 20% Slurry
2. Combine PLS from WL-AB6 through WL-AB10 and WL-AB12. Measure and record the solution density. Prepare reagent to 20% slurry.
3. Set up a pyrex reactor with overhead agitator and heated to 50°C
4. ORP and pH probes were placed to contact the reaction mixture and moved up as required during the test.
5. Solution was added to the reactor manually, and 3.5 g 30% peroxide was added.
 The pH was adjusted to the first pH target by addition of MgCO₃ slurry.
6. All reagent additions were recorded.
7. After achieving the target pH the reactor was allowed to mix for one hour before final filtration.
8. The residue was washed thoroughly (1 repulp at similar total volume + 2 × 2000 mL DI) dried and submitted for analysis.
9. The filtrates were assayed per the table below.

Assays:

#	Elements	
1	liquor samples - Al, Fe, Th, Y, Nd	GC_SOL84T_AI, GC_SOL84T-AE_Fe, GC_SOL94T_Nd, GC_SOL94T-AE_Th, GC_SOL94T-AE_Y
2	liquor samples - ICP, REE	GC_SOL91T, GC_SOL94T
1	residue sample - WRA, REE by XRF, S, Cl	GC_XRF72MET, GC_XRF72MET-REE, GC_CSA06V (total), GC_CLA27E

Conditions:

Feed to Add:	1445.0	g
Solution Density:	1.207	g/mL
Feed to Add:	1197	mL
Reagent	H₂O₂	
Reagent Strength:	30	%
Add:	4	g 30% H ₂ O ₂
Reagent	MgCO₃	
Reagent Strength:	20	%
Retention Time:	1	h
Temperature:	50	°C
Target pH:	2.9-3.0	pH

Peroxide Addition calculates after Feed Volume entered.

MgCO₃ Slurry Mass calculates after Feed Volume entered.

100% MgCO ₃		
pH Target	g/L Feed	kg/t Con
3.0	126.3	324.4

Test: IR-6

Test Data:

Time		Reactor			Reagents / Feed			Comments:
(24 h)	(h) elapsed	Temp °C	pH	ORP mV	Feed g	MgCO ₃ 20% g	H ₂ O ₂ g	
7:18		22.3	1.11	550	1445			mixing, heating
7:35		52.7	0.76	615				starting H ₂ O ₂
7:37		54.4	0.76	891			4	dose in, starting MgCO ₃
7:58		53.8	1.14	895		175.10		LOTS of foaming, more addition
8:19		50.4	2.39	706		442.50		LOTS of foaming, more addition
8:32		50.0	2.81	629				pulp turned to gel, increased mixing to over 1200RPM
9:35	0.0	50.2	2.92	595		121.10		at target
9:10		49.9	2.93	577		11.00		
9:35	1.00	50.2	2.96	565		6.40		final
Totals/Avg.		51.3	2.27		1445	756	4	

Sampling Info:

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
Final		2144	1153.5	1.128	1023	545	3.06			v slw	9%
Wash			1923.8	1.040	1850			814.4	194.6	v slw	

Final Filtration/Washing:

Diameter of filtration paper:	185	mm
Type of Paper (Whatman #):	3	
Filtration Time:	overnight	minutes
Washing Time:		minutes
Cake Moisture:	76%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	lt orange
Clarity of Wash:	clear
Colour of Wash:	v lt brn
Colour of Residue:	brown

Residue (assay cut):

Tare:	8.3 g
Tare + Wet:	107.9 g
Tare + Dry:	32.1 g

Project: 18299-02
Client: Lofdal (NCM)

Date: 4-Aug-22
Technologist: Ella Mihailescu

Test: IR-6

Metallurgical Balance:

Sample & Quant. (mL or g)	Assay Units	Feed	Final Filt	Final Wash	Final PPT	PPT % Res/Feed	Calc Head	Acc %
		1197	1022.5	1849.8	194.6			
La	mg/L, g/t	670	452		0.09	22.8	539	80
Ce	mg/L, g/t	1280	823		0.20	24.9	1022	80
Pr	mg/L, g/t	133	84.8		<0.03	--	114	86
Nd	mg/L, g/t	458	289	91.8	0.06	21.3	486	106
Sm	mg/L, g/t	139	87.2		<0.04	--	145	104
Eu	mg/L, g/t	62.7	40.6			--	35	55
Gd	mg/L, g/t	363	244			--	208	57
Tb	mg/L, g/t	89.4	59.6			--	51	57
Dy	mg/L, g/t	624	410			--	350	56
Ho	mg/L, g/t	146	99.2			--	85	58
Y	mg/L, g/t	4640	3090	966	0.31	11.0	4644	100
Er	mg/L, g/t	434	296			--	253	58
Tm	mg/L, g/t	58.7	39.2			--	33	57
Yb	mg/L, g/t	343	228			--	195	57
Lu	mg/L, g/t	44.2	29.3			--	25	57
Sc	mg/L, g/t	9.32	1.74			--	1	16
Th	mg/L, g/t	664	6.31	2.14	0.50	122.6	823	124
U	mg/L, g/t	20.7	6.20		0.02	133.2	33	159
Si	mg/L, %				0.25	--	403	--
Al	mg/L, %	1260	508	151	0.402	51.9	1321	105
Fe	mg/L, %	40700	44.8	15.8	27.1	108.4	44176	109
Mg	mg/L, %	1850	21400		1.69	--	--	--
Ca	mg/L, %	1030	861		0.18	28.2	1026	100
Na	mg/L, %	91	62		0.03	53.0	101	111
K	mg/L, %	272	165		0.03	19.8	195	72
Ti	mg/L, %	6090	10.2		4.23	112.9	6887	113
P	mg/L, %	1130	<5		1.06	153.2	1735	154
Mn	mg/L, %	902	622		0.05	8.4	607	67
Cr	mg/L, %	169	1.1		0.11	105.3	179	106
V	mg/L, %	109	<0.2		0.08	117.0	128	117
Cu	mg/L, %	<40	<40			0.0	34	85
Pb	mg/L, %	<6	<6			0.0	5	85
Zn	mg/L, %	7.6	6.5			0.0	6	73
TREE	mg/L, %	9485	6272	1058	0.0	--	6991	74
LREE	mg/L, %	2680	1736	92	0.0	--	1625	61
HREE	mg/L, %	6805	4536	966	0.0	--	5367	79

Project: 18299-02
Client: Lofdal (NCM)

Date: 12-Aug-22
Technologist: Ella Mihailescu

Test: RP-5

Purpose: Bulk REE Precipitation test on Bulk Impurity Removal Filtrate (from Flot Con 3 ABWL plus MgCO₃)

Sample: IR-6 Filtrate

H&S: Refer to H₂SO₄ and sodium carbonate MSDS
 Feed solution is mildly corrosive, avoid contact with skin and clean all spills accordingly

Procedure:

1. Secure feed solution, decant if there are visible solids in the container. Filter if required. 300 mL 100 g/L Na₂CO₃
2. Measure and record the solution density. Prepare reagent to 100 g/L solution.
3. Set up a pyrex reactor with overhead agitator.
4. ORP and pH probes were placed to contact the reaction mixture and moved up as required during the test.
5. Solution was added to the reactor manually, and pH adjusted to the first pH target gradually by addition of Na₂CO₃ to the vortex.
6. All reagent additions were recorded.
7. After achieving the target pH the reactor was allowed to mix for one hour before final filtration.
8. The residue was washed thoroughly (1 repulp at similar total volume + 2 × 2000 mL DI) dried and submitted for analysis.
9. The filtrates were assayed per the table below.

Assays:

#	Elements	Streams
1	liquor samples - Y GC_SOL94T-AE_Y	final wash
1	liquor samples - ICP, REE GC_SOL91T, GC_SOL94T	Final (pH 6.5) PLS
1	residue sample - REE, WRA GC_IMS93A, GC_XRF72MET	washed residue

Conditions:

Feed to Add:	1100.0	g
Solution Density:	1.128	g/mL
Feed to Add:	975	mL
Reagent	Na₂CO₃	
Reagent Strength:	100	g/L
Retention Time:	1	h
Temperature:	ambient	°C
Target pH:	6.50-6.75	pH

pH 6.6

g Na ₂ CO ₃	
/ L IR Filt	kg/t conc
15.24	39.1

Test: RP-5

Test Data:

[illegible]**Sampling Info:**

Sample	Elapsed Time (h)	Weight (g)		Filt. Dens. g/mL	PLS Vol mL	At Ambient Temp		Wet Res. g	Dry Res. g	Filtration	Pulp % Solids
		Pulp	Filtrate			ORP	pH				
Final		1261	1173.5	1.070	1097						1%
Wash			1904.7	1.010	1886			58.2	12.6		

Final Filtration/Washing:

Diameter of filtration paper:	150	mm
Type of Paper (Whatman #):	3	
Filtration Time:	33	minutes
Washing Time:	186	minutes
Cake Moisture:	78%	

Colour and Clarity:

Clarity of Filtrate:	clear
Colour of Filtrate:	yellow/orange
Clarity of Wash:	clear
Colour of Wash:	ww
Colour of Residue:	white

Residue:

Tare:	18.4 g
Tare + Wet:	76.6 g
Tare + Dry:	31.0 g

Project: 18299-02
Client: Lofdal (NCM)

Date: 12-Aug-22
Technologist: Ella Mihailescu

Test: RP-5

Metallurgical Balance:

Sample & Quant. (mL or g)	Assay Units	Feed	Final Filt	Final Wash	Final PPT	PPT % Res/Feed	Calc Head	Acc %
		975	1096.7	1885.8	12.6			
La	mg/L, %	452	0.86		3.33	95.2	431	95
Ce	mg/L, %	823	1.00		5.9	92.6	764	93
Pr	mg/L, %	84.8	0.10		0.604	92.0	78	92
Nd	mg/L, %	289	0.35		2.11	94.3	273	94
Sm	mg/L, %	87.2	0.15		0.631	93.5	82	94
Eu	mg/L, %	40.6	0.06		0.292	92.9	38	93
Gd	mg/L, %	244	0.59		1.69	89.5	219	90
Tb	mg/L, %	59.6	0.19		0.391	84.8	51	85
Dy	mg/L, %	410	1.92		3.05	96.1	396	97
Ho	mg/L, %	99.2	0.66		0.658	85.7	86	86
Y	mg/L, %	3090	50.7	2.96		98.0	3090	
Er	mg/L, %	296	2.61		1.97	86.0	257	87
Tm	mg/L, %	39.2	0.45		0.26	85.7	34	87
Yb	mg/L, %	228	2.94		1.53	86.7	201	88
Lu	mg/L, %	29.3	0.44		0.198	87.3	26	89
Sc	mg/L, %	1.74	<0.07			89.5	2	
Th	mg/L, %	6.31	<0.03		0.0425	87.0	6	88
U	mg/L, %	6.20	1.15		0.0334	69.6	6	90
Si	mg/L, %				0.19	--	25	--
Al	mg/L, %	508	0.2		3.44	87.5	445	88
Fe	mg/L, %	44.8	0.5		0.34	96.8	44	98
Mg	mg/L, %	21400	19400		0.25	0.2	21853	102
Ca	mg/L, %	861	705		0.21	3.2	821	95
Na	mg/L, %	62	4960		0.18	--	--	--
K	mg/L, %	165	146		0.008	0.7	165	100
Ti	mg/L, %	10.2	<0.02		0.06	75.9	8	76
P	mg/L, %	<5	<5		0.013	--	--	--
Mn	mg/L, %	622	513		0.17	3.5	599	96
Cr	mg/L, %	1.1	<0.1		<0.007	--	--	--
V	mg/L, %	<0.2	<0.2		<0.006	--	--	--
Cu	mg/L, %	<40	0.2			--	--	1
Pb	mg/L, %	<6	<2			--	--	37
Zn	mg/L, %	6.5	2.4			--	--	42
TREE	mg/L, %	6272	63	3	46.0	94.9	6026	96.1
LREE	mg/L, %	1736	2	0	12.6	93.6	1628	93.8
HREE	mg/L, %	4536	61	3	33.5	95.3	4398	97.0

	Distribution of REE in Res
La	7.2%
Ce	12.8%
Pr	1.3%
Nd	4.6%
Sm	1.4%
Eu	0.6%
Gd	3.7%
Tb	0.8%
Dy	6.6%
Ho	1.4%
Y	51%
Er	4.3%
Tm	0.6%
Yb	3.3%
Lu	0.4%

Appendix H:

An Investigation into Metallurgical Testwork on Samples from The Lofdal Heavy Rare Earth Project
prepared for Namibia Critical Metals from SGS Canada Inc.



**An Investigation into
METALLURGICAL TESTWORK ON SAMPLES FROM THE
LOFDAL HEAVY RARE EARTH PROJECT**

prepared for

NAMIBIA CRITICAL METALS

Project 18299-03 – Final Report
November 11, 2022

NOTES

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Executive Summary

Fresh Run of Mine (ROM) sample taken from Namibia Critical Metals' Lofdal Heavy Rare Earth Project, as well as multiple ore sorted samples, were received at SGS Lakefield. The testing program was mostly performed on the ROM sample. A low-grade XRF sample was composited (designated as LG XRF Comp) from the XRF sorted products and sorting bypass fines and its flotation performance was evaluated.

The main objectives of the test program were to confirm the flotation performance on the fresh low-grade sample and optimize the beneficiation flowsheet, as required. This report covers head characterization, magnetic separation testwork, and batch flotation testwork.

The current testwork indicated that the previously developed flotation flowsheet worked well on the new low-grade ROM sample. With two cleaner stages, the yttrium could be upgraded from 0.1% to ~2% Y_2O_3 (~20 times upgrade) in a ~3% mass pull, with a yttrium recovery of ~70%.

More details of the testwork outcome are summarized as follows:

- The head assays of the ROM and LG XRF Comp were 0.22% and 0.27% TREO (including yttrium), respectively. The new samples tested in this program contained a lower HREO distribution than those used in previous program (SGS Project number 18299-01).
- The xenotime liberation of the ROM sample increased from 32% at +38 μm fraction to 76% in the -38 μm fraction.
- Wet high intensity magnetic separation demonstrated the potential to pre-concentrate the ROM sample, but the yttrium loss was considered to be high. At a P_{100} of 300 μm , the yttrium recovery in the combined magnetic concentrates (up to 15,000 Gauss) was 75%, grading 0.38% Y_2O_3 (~4 times upgrade) in ~18% mass pull.
- A similar flotation reagent scheme to that developed previously was applied to the ROM sample, with a reduced collector Florrea 3900 dosage and reduced Calgon dosage. Alternative collector types were investigated, but they did not perform as well as Florrea 3900 and 3000. High intensity conditioning was beneficial for producing a cleaner product. Table I summarizes the results of the three best batch tests achieved in current test program.
 - At a P_{100} of 53 μm , test F114 with 1600 g/t collector Florrea 3900 generated similar quality of 2nd cleaner concentrate and recovery to test F109 with 2000 g/t of Florrea 3900. The yttrium grade was ~2.27% Y_2O_3 (5.4-5.6% TREO) at a yttrium recovery of ~70%. The HREO+Y recovery was good

- at 68-69%, but the LREO recovery was lower, at 56-58%. The combined TREO recovery was 64-65%.
- The recovery was higher with a finer grind in test F115 (P_{100} of 38 μm), but with a slightly higher mass pull and lower grade. The F115 2nd cleaner concentrate graded 1.73% Y_2O_3 (3.85% TREO) at an yttrium recovery of 77%, HREO+Y recovery of 75%, LREO recovery of 58%, and a TREO recovery of 69%.
 - The mineralogical analysis showed that the liberation of xenotime and other rare earth minerals (REM) in the 2nd cleaner concentrate was similar at a P_{100} of 38 μm or 53 μm . But the rougher tailings were much better liberated with the finer grind, which might be the reason for a higher recovery in F115.
 - The scale up from 2 kg to 10 kg batch flotation tests was successful, demonstrating repeatability and confirmation of the flotation regime. About 1 kg of flotation concentrate was generated for hydrometallurgical testwork from four 10 kg batch flotation tests. The grade of the concentrate was 2.63% Y_2O_3 on average.
 - The ROM sample produced better flotation performance than the LG XRF Comp sample.

Table I: Selected Best Flotation Test Results of 2nd Cleaner Concentrate 1-2

Test	Cond. (3900 Dosage, Grind Size)	Wt %	Assay, %				Distribution, %			
			Y_2O_3	LREO	HREO+Y	TREO	Y_2O_3	LREO	HREO+Y	TREO
F109	2000g/t, -53 μm	2.9	2.07	1.48	3.77	5.25	70.5	57.9	68.8	65.3
F114	1600g/t, -53 μm	2.6	2.27	1.55	4.09	5.63	70.0	56.2	68.1	64.4
F115	2000g/t, -38 μm	3.9	1.73	1.16	2.69	3.85	77.4	57.9	75.2	69.0
ROM Head (Direct)			0.11	0.08	0.14	0.22				

Introduction

Bernadine Ballington from SGS South Africa contacted SGS Lakefield, on behalf of Namibia Critical Metals (NCM), for beneficiation flowsheet confirmation on fresh ore samples from the Lofdal Heavy Rare Earths Project, located in northern Namibia. SGS Lakefield developed a simplified and improved flowsheet with the trench samples in 2021 (SGS project number 18299-01). NCM requested flowsheet confirmation and optimization on the new samples.

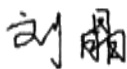
The following samples were received for testwork in the current phase of program:

- Low-grade fresh run-of-mine ore sample
- Low-grade XRF sorted products
- XRF sorting bypass fines

The main objectives of the beneficiation program are:

- 1) Confirm the previous developed flowsheet is applicable with the new sample(s)
- 2) Compare the flotation performance of the direct ore sample with the sorted product
- 3) Evaluate magnetic separation as a pre-concentration stage on the low-grade ore sample prior to flotation
- 4) Generate flotation concentrate for downstream hydrometallurgical testwork

Testwork was conducted in close consultation with Mr. Rainer Ellmies and Ms. Barbara Mulcahy of Namibia Critical Metals with progress discussed through emails and conference calls. Results were provided to Namibia Critical Metals as they became available.



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Testwork Summary

1. Sample Receipt and Preparation

One skid with two totes of samples weighing about 92 kg, and two supersacks and one skid with plastic totes of samples weighing about 1174 kg, were shipped from Namibia Critical Metals and received at SGS Lakefield, on January 14, 2022 and on March 30, 2022, respectively. Internal receipt numbers of 0127-JAN22 and 0381-MAR22 were assigned. A photograph of the second shipment is included in Figure 1.



Figure 1: Photograph of the Received Samples in the Second Shipment

The first shipment sample, expected to be the sorting bypass fines, was not used in this testwork. A total of nine samples were included in the second shipment, including the run of mine sample, various sorting products, and sorting bypass fines. The inventory of the second shipment is shown in Table 2.

Table 2: Inventory of the Second Shipment

Sample	Mass, kg
Coarse Conc 2	173.0
Coarse Conc 3	109.0
Coarse Conc 4	61.8
FC	31.0
Fines Conc 2	147.0
Fines Conc 4	56.4
FF	29.0
FCon	15.5
ROM -20 mm	279.0
ROM Sample	171.0
Total Received	1072.7

The Coarse Conc 4 and Fines Conc 4 were products of -80+40 mm and -40+20 mm, respectively, that had been sorted by XRF. The ROM sample was the run of mine sample, and ROM-20 mm was the sorting bypass fines at minus 20 mm.

For the ROM sample, a ~110 kg subsample was stage-crushed to 100% passing 3.35 mm and then split into 10 kg and 2 kg test charges. For the ROM -20 mm sample, 20 kg subsample was stage-crushed to 100% passing 3.35 mm and split into two 10 kg test charges.

The Coarse Conc 4 (~62 kg), 17 Fines Conc 4 (25 kg), and ROM -20mm (94 kg) samples were stage crushed to 100% passing 3.35 mm and then combined and homogenized to make the LG XRF Comp, which was split into 10 kg and 2 kg test charges.

Two head samples were taken from each of ROM, ROM -20mm, and LG XRF Comp; one was submitted for head analysis, and the other was submitted for mineralogy analysis. The unused samples were placed in storage for future testing.

2. Head Characterization

2.1. Head Chemical Analysis

A representative subsample was split from the crushed ROM, ROM -20mm, and LG XRF Comp and submitted for an ICP rare earth elements scan and whole-rock analysis.

As shown in Table 3, the total rare earth oxide (TREO) content (including yttrium) in the head sample of ROM was low, at ~0.2% TREO. The most abundant rare earth element was yttrium (Y), at 970 g/t Y_2O_3 , which accounted for 44% of the TREO. The second most abundant rare earth element was cerium, grading 367 g/t CeO_2 , which accounted for 17% of the TREO. Other rare earth elements with decent contents included lanthanum (190 g/t La_2O_3), neodymium (126 g/t Nd_2O_3), and dysprosium (143 g/t Dy_2O_3). The yttrium was used as a proxy element to track the performance of TREO during the beneficiation testwork, at a conversion ratio of 2.28 from Y_2O_3 to TREO.

The head assay of the ROM -20mm was similar to the ROM sample, with a slightly lower yttrium distribution (at 35% of ROM-20mm versus 44% of ROM). The TREO content in the head sample of LG XRF Comp (0.27% TREO) was slightly higher than ROM. The distribution of rare earth elements in LG XRF Comp was similar to that of ROM sample.

Compared to the samples used in the Phase 1 program (SGS Project 18299-01), the samples in the current phase all contained a lower TREO grade and a higher distribution of light REE.

Table 3: Head Assays and REO Distributions of the XRF SP and XRT SP

Element	Assays, %, g/t			TREO Distribution, %			Element	Assays, %		
	Head ROM	Head ROM - 20mm	LG XRF Comp	Head ROM	Head ROM - 20mm	LG XRF Comp		Head ROM	Head ROM - 20mm	LG XRF Comp
TREO, %	0.22	0.19	0.27	100	100	100	SiO_2 %	49.7	53.7	48.5
Y_2O_3 , g/t	969	687	1218	43.7	35.4	45.9	Al_2O_3 %	12.9	14.1	12.5
Dy_2O_3 , g/t	143	106	188	6.5	5.5	7.1	Fe_2O_3 %	6.44	6.69	7.16
La_2O_3 , g/t	190	220	211	8.6	11.4	8.0	MgO %	1.96	1.2	2.29
CeO_2 , g/t	367	415	408	16.6	21.4	15.4	CaO %	9.89	7.44	10.3
Pr_6O_{11} , g/t	39.4	47.7	40.7	1.8	2.5	1.5	Na_2O %	5.92	5.85	5.7
Nd_2O_3 , g/t	126	159	131	5.7	8.2	4.9	K_2O %	1.05	1.38	1.04
Sm_2O_3 , g/t	43.8	39.3	44.2	2.0	2.0	1.7	TiO_2 %	0.58	0.58	0.55
Eu_2O_3 , g/t	18.3	15.6	20.1	0.8	0.8	0.8	P_2O_5 %	0.25	0.19	0.24
Gd_2O_3 , g/t	86.1	80.7	106.7	3.9	4.2	4.0	MnO %	0.2	0.13	0.19
Tb_4O_7 , g/t	19.2	16.1	23.1	0.9	0.8	0.9	Cr_2O_3 %	0.01	0.01	< 0.01
Ho_2O_3 , g/t	29.4	21.5	33.8	1.3	1.1	1.3	V_2O_5 %	0.02	0.02	0.02
Er_2O_3 , g/t	86.6	65.1	111.1	3.9	3.4	4.2	LOI %	10.3	7.79	11
Tm_2O_3 , g/t	12.6	9.1	14.0	0.6	0.5	0.5	Sum %	99.2	99.1	99.5
Yb_2O_3 , g/t	74.4	49.1	91.4	3.4	2.5	3.4				
Lu_2O_3 , g/t	10.7	6.9	12.2	0.5	0.4	0.5				
Sc g/t	< 25	< 25								
U_3O_8 , g/t	9.3	7.5	12.1							
ThO_2 , g/t	217.3	168.4	242.4							

2.2. Head Sample Mineralogy

Two samples, referred to as ROM and LG XRF Comp, were received by the SGS Advanced Mineralogical Facility for mineralogical examination. The mineralogical work was conducted with a TESCAN Integrated Mineral Analyzer (TIMA-X). The ROM -20 mm was also submitted for X-Ray Diffraction analysis. The purpose of this test program was to determine the overall mineral assemblage of the samples, and the occurrence of REE minerals.

The ROM sample was stage-ground to a P_{100} of $\sim 75 \mu\text{m}$ and the LG XRF SP sample was stage-ground to a P_{100} of $\sim 106 \mu\text{m}$. The ROM sample was screened at $38 \mu\text{m}$ and both the $+38 \mu\text{m}$ and $-38 \mu\text{m}$ fractions were analyzed by TIMA. The LG XRF SP sample was analyzed un-sized. Subsamples from each fraction were riffled and submitted for chemical assay including whole rock analysis by XRF for major elements, zirconium, REE, yttrium, thorium, and uranium, which were used as quality control for the TIMA-X analysis. Subsamples were riffled from each fraction to prepare two graphite-impregnated polished epoxy grain mounts from each fraction.

A summary of the results is given below. The complete mineralogy report is included in Appendix A.

2.2.1. X-Ray Diffraction (XRD) Analysis on the ROM -20 mm Sample

XRD analysis of the ROM -20 mm sample indicated similar mineralogy to the sample investigated in the previous phase, consisting mainly of albite (50%), quartz (14%), calcite (11%) and muscovite (11%), with lesser hematite (4%), chlorite (3%), and trace amounts ($<3\%$) of microcline, magnetite, ankerite, rutile, and dolomite.

Table 4: XRD Results of ROM -20mm Head

Mineral/Compound	ROM -20mm Head APR5029-01 (wt %)
Quartz	14.0
Albite	49.7
Chlorite	3.3
Montmorillonite	0.4
Muscovite	11.1
Microcline	2.2
Hematite	4.2
Calcite	11.4
Dolomite	0.3
Ankerite	2.5
Rutile	0.6
Magnetite	0.4
TOTAL	100

2.2.2. TIMA-X Analysis

2.2.2.1. Modal Mineralogy

The modal mineralogy of the ROM and LG XRF Comp samples were similar, which consisted of quartz/feldspars (both at ~64%), carbonates (mainly calcite) (18% and 19%), Fe-oxides (mainly hematite and lesser magnetite) (6% and 7%), and biotite/chlorite/muscovite (5% and 4%) (Table 5).

The rare earth minerals (REM) consisted mainly of xenotime, at ~0.2-0.3%, as well as trace amounts (collectively ~0.1%) of synchysite/monazite, and zircon (~0.1%).

Table 5: Modal Mineralogy of ROM and LG XRF Comp Samples

Sample		ROM Head			LG XRF Comp
Fraction		Comb	+38 µm	-38 µm	As Rec'd
Mass Size Distribution (%)		100.0	58.2	41.8	100.0
Mineral Mass (%)	Xenotime	0.30	0.22	0.32	0.20
	REM	0.10	0.12	0.13	0.10
	Thorite/Th-Y-silicates	0.00	0.02	0.02	0.00
	Zr Silicates	0.30	0.19	0.32	0.30
	Apatite	0.50	0.37	0.58	0.40
	Calcite/Dolomite	18.2	16.1	21.1	18.6
	Ankerite/Siderite	1.80	1.79	1.83	1.20
	Quartz/Feldspars	63.8	67.2	58.9	64.4
	Biotite/Chlorite/Muscovite	5.10	3.45	7.40	4.10
	Amphibole/Epidote	2.80	1.69	4.22	2.20
	Fe-Oxides	6.10	7.51	4.13	6.80
	Ilmenite	0.20	0.23	0.26	0.20
	Rutile	0.60	0.77	0.45	0.50
	Other	0.30	0.29	0.35	0.90
	Total	100.0	100.0	100.0	100.0

2.2.2.2. Liberation and Association of Xenotime

Liberation (free, ≥95% of the total particle area, and liberated, ≥80% of the total particle area) for xenotime accounted for 38% in LG XRF SP Comp at a P₁₀₀ of ~100 µm and 55% in ROM at a P₁₀₀ of ~75 µm (Table 6). The remainder occurred mainly as complex particles (38% and 27%, respectively), and as binaries with quartz/feldspars (10% and 7%), and calcite (5% and 6%). Xenotime liberation increased from 32% in the -75+38 µm fraction to 76% in the -38 µm fraction of sample ROM. Based on the liberation characteristics, a fine grind size of ~38 µm is likely required for flotation.

Table 6: Mineralogy Results Summary of Xenotime Liberation and Association (in %)

Mineral Name	LG XRF Comp	ROM Head Comb	ROM +38 µm	ROM -38 µm
Pure Xenotime	22.9	41.7	15.0	66.9
Free Xenotime	5.60	3.80	6.20	1.61
Lib Xenotime	9.90	9.40	11.1	7.78
Xnt: REM	0.00	0.10	0.03	0.23
Xnt: Zr Silicates	1.50	1.50	0.35	2.53
Xnt: Apatite	1.10	0.60	0.79	0.49
Xnt: Calcite/Dolomite	4.90	6.00	7.78	4.37
Xnt: Ankerite/Siderite	0.50	0.50	0.43	0.54
Xnt: Quartz/Feldspars	9.80	6.60	6.69	6.57
Xnt: Biotite/Chlorite/Muscovite	1.10	0.90	0.41	1.29
Xnt: Fe-Oxides	4.10	1.20	2.13	0.28
Xnt: Other	0.50	0.20	0.40	0.14
Complex	38.0	27.1	48.7	6.6
Total	100.0	100.0	100.0	100.0
Pure+Free + Liberated	38.4	54.9	32.3	76.3

3. Stage-Grinding

To prepare test charges for batch-scale magnetic separation and flotation bench-scale testwork, subsamples of the -3.3 mm ROM and LG-XRF Comp samples were taken and wet stage-ground to a target grind size. A target grind size of ~100% passing 53 µm was used for most of the stage grinding batches, while target grind sizes of ~100% passing 300 µm, 150 µm, 75 µm, and 38 µm were used as required. These grind sizes were selected based on mineralogical data.

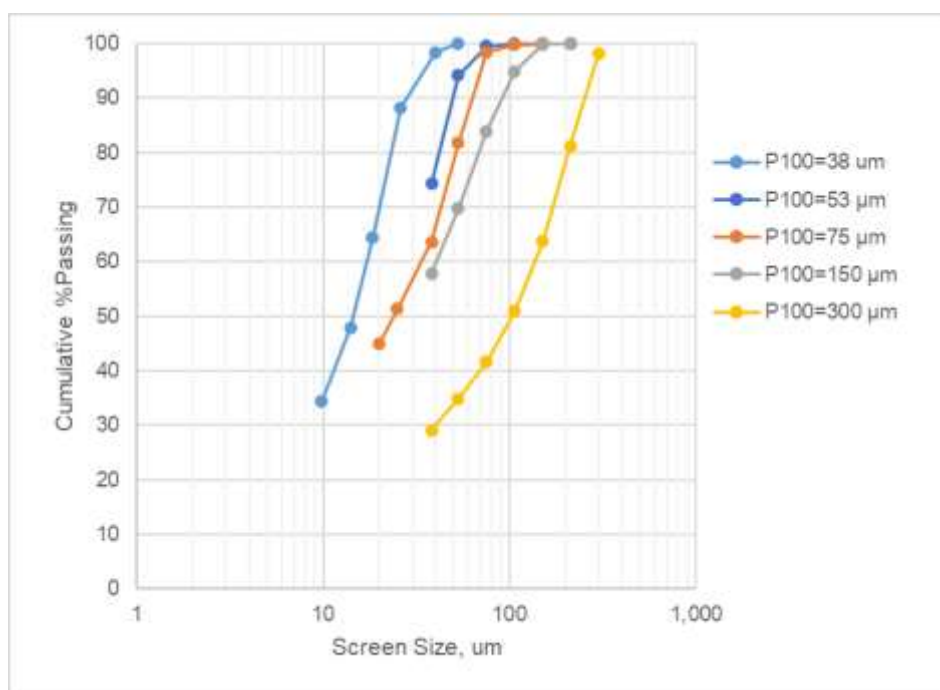
Each -3.3 mm sample was ground in a 10 kg batch rod mill, at about 60% pulp density, for a short time. To avoid over-grinding, the ground material was then screened at various target sizes (38, 53, 75, 150, 300 µm) to collect the undersize fraction generated. This screen oversize was returned to the mill and ground again followed by screening. The grinding-screening sequence was repeated until the weight of the oversize fraction dropped to ~4 kg dry equivalent. At this point, a smaller (4 kg) batch rod mill was used for grinding. Similarly, a 2 kg mill was used when the oversize weight reached ~2 kg dry equivalent. The grinding time was varied in each stage, based on the remaining oversize mass.

Stage-grinding was complete when almost all the sample (>98%) had been ground to finer than the target sizes. The ground material was then filtered, homogenized, and split into 2 kg (or smaller if necessary) charges for magnetic separation or flotation testing.

Representative subsamples of the stage-ground product from each batch were submitted for particle size analysis. Table 7 summarizes the results of the stage-ground feed particle size distribution. Figure 2 depicts the particle size distributions of typical stage-ground feed at a 100% passing 53 µm, 75 µm, and 106 µm.

Table 7: Summary of Stage-Ground Feed Particle Size Distributions

Sample	P ₁₀₀ ,µm	P ₈₀ ,µm
ROM	-38	23
	-53	42
	-75	52
	-150	69
	-300	207
LG-XRF Comp	-53	44


Figure 2: Stage-Ground Feed Particle Size Distributions

4. Wet Magnetic Separation Testwork

Magnetic separation was evaluated in batch tests to identify the potential for pre-concentrating the rare earth minerals in the ROM sample before feeding to flotation. Two batch tests, designated WHIMS-1 and WHIMS-2, were conducted at feed sizes of P₁₀₀ of ~150 µm and ~300 µm, respectively.

About 250 g of -150 µm or -300 µm ROM sample, at a solids content of ~25%, was passed three or four times through an Eriez model L-4-20 laboratory-scale Wet High-Intensity Magnetic Separator (WHIMS), using expanded metal matrix (medium size 1/3–18 mm). The WHIMS tests were performed at 1,000, 5,000, 10,000, and 15,000 Gauss. The 1,000 Gauss was only included in Test WHIMS-1. In both tests, the testwork was performed from low to high magnetic intensities, with the non-magnetic product re-passing at the subsequent higher magnetic intensity. The magnetic concentrates from each of the three or four passes

and the final non-magnetic product were filtered, weighed, and submitted for analysis. A brief flowsheet diagram is shown in Figure 3.

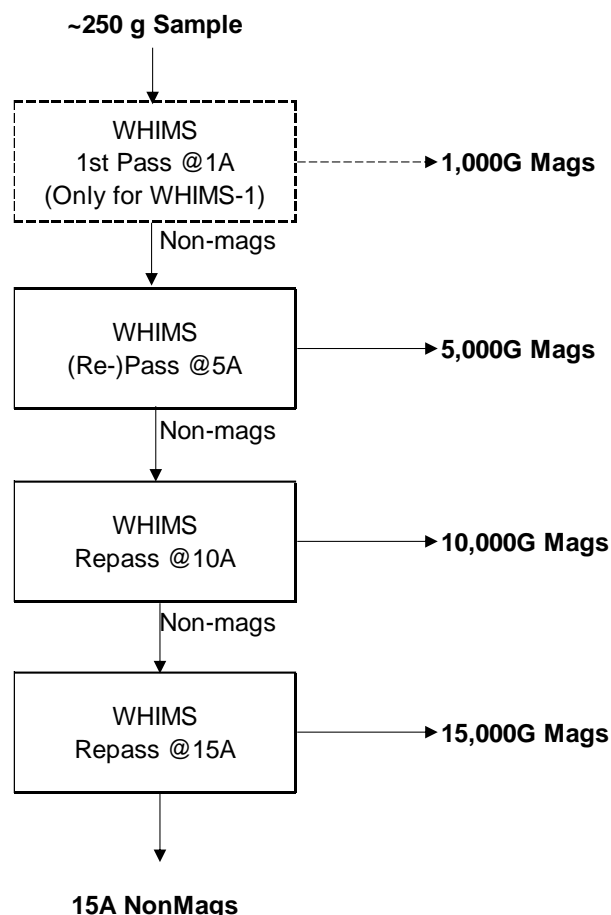


Figure 3: Block Diagram of WHIMS Tests

4.1. Eriez WHIMS Batch Test Results

The WHIMS-1 and WHIMS-2 test results are shown in Table 8 and Table 9, respectively. The yttrium recovery, mass pull to the magnetic concentrates, and yttrium grade as a function of magnetic intensity are depicted in Figure 4.

At a coarser grind size (P_{100} of 300 μm) in WHIMS-2, the cumulative yttrium recovery to 15,000 Gauss magnetic concentrates was significantly higher (75%) than at a P_{100} of 150 μm in test WHIMS-1 (60%), although the mass pull increased from 13% to 18% and the yttrium grade decreased from 0.48% to 0.38% Y_2O_3 .

A size-by-size analysis on the WHIMS-2 15 kG non-magnetic product showed losses in both the -20 μm and +20 μm fractions. The losses in +20 μm fraction are likely due to poor liberation of xenotime

and the losses in the -20 µm fraction are likely due to poor efficiency of magnetic separation at very fine particle size range.

The magnetic separation tests demonstrated some potential for upgrading the ROM sample as a pre-concentrate step, but the yttrium losses were considered to be high. Therefore, direct froth flotation was selected for further flowsheet development.

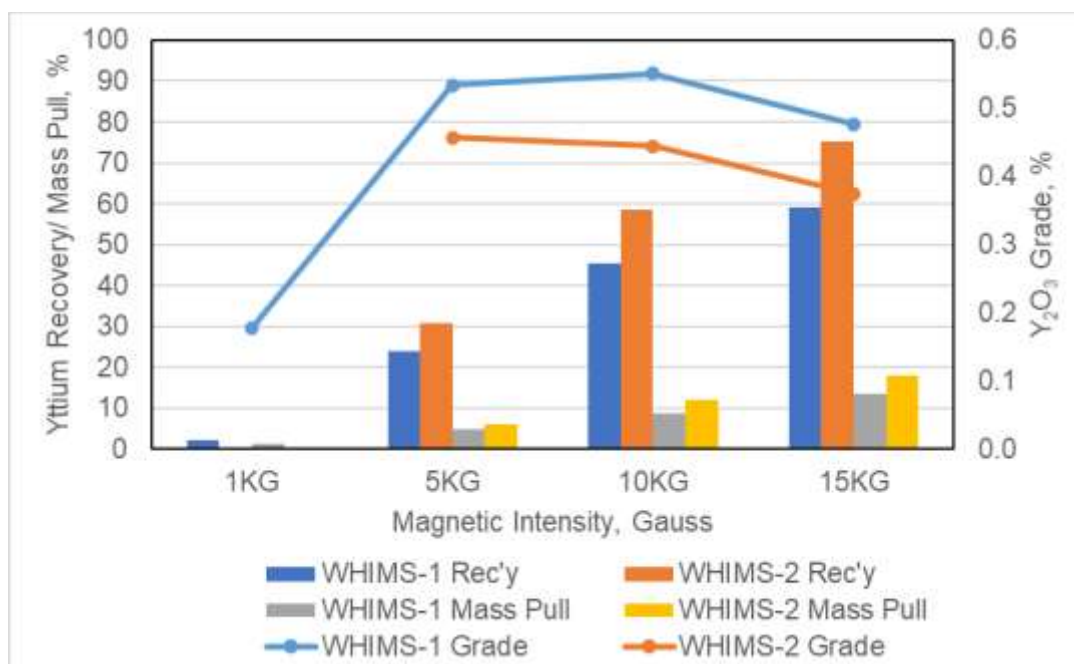


Figure 4: Test Results of WHIMS-1 and WHIMS-2

Table 8: Mass Balance Results of Test WHIMS-1 (-150µm)

Product	Mass		Assay,%								Distribution,%							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
1KG Mags	3.2	1.3	0.18	0.41	7.43	33.3	34.0	0.72	3.69	7.96	2.1	2.1	0.9	0.9	6.9	1.6	0.8	0.8
5KG Mags	9.0	3.6	0.66	1.51	12.9	19.8	31.1	1.17	1.58	5.55	21.8	21.8	4.5	1.4	17.9	7.3	1.0	1.5
10KG Mags	10.2	4.0	0.57	1.31	17.7	19.0	19.1	1.13	1.29	5.76	21.4	21.4	7.1	1.6	12.4	8.0	0.9	1.8
15KG Mags	11.4	4.5	0.33	0.76	17.2	19.7	22.5	1.01	1.36	5.79	13.8	13.8	7.7	1.8	16.4	8.0	1.1	2.0
15KG NonMags	220	86.7	0.05	0.12	9.30	53.3	3.31	0.49	6.46	13.9	40.9	40.9	79.8	94.3	46.4	75.0	96.3	93.8
Feed, calc	253	100.0	0.11	0.25	10.1	49.0	6.18	0.57	5.81	12.8	100	100	100	100	100	100	100	100
Feed, direct			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								
Combined Product																		
1KG Mags	3.2	1.3	0.18	0.41	7.43	33.3	34.0	0.72	3.69	7.96	2.1	2.1	0.9	0.9	6.9	1.6	0.8	0.8
Up to 5KG Mags	12.2	4.8	0.53	1.22	11.5	23.3	31.9	1.05	2.13	6.18	23.9	23.9	5.5	2.3	24.8	8.9	1.8	2.3
Up to 10KG Mags	22.4	8.8	0.55	1.26	14.3	21.4	26.1	1.09	1.75	5.99	45.3	45.3	12.5	3.9	37.2	17.0	2.7	4.1
Up to 15KG Mags	33.8	13.3	0.48	1.09	15.3	20.8	24.9	1.06	1.62	5.92	59.1	59.1	20.2	5.7	53.6	25.0	3.7	6.2
15KG NonMags	220	86.7	0.05	0.12	9.30	53.3	3.31	0.49	6.46	13.9	40.9	40.9	79.8	94.3	46.4	75.0	96.3	93.8

Table 9: Mass Balance Results of Test WHIMS-2 (-300µm)

Product	Mass		Assay,%								Distribution,%							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
5KG Mags	14.9	6.1	0.46	1.05	13.0	24.3	26.9	0.96	2.05	6.62	30.7	30.7	8.1	2.9	26.9	9.9	2.0	3.1
10KG Mags	14.3	5.8	0.43	0.99	17.0	22.6	17.9	0.99	1.64	6.73	27.9	27.9	10.2	2.6	17.2	9.8	1.5	3.0
15KG Mags	15.2	6.2	0.24	0.55	16.7	22.9	22.4	0.93	1.74	6.56	16.5	16.5	10.7	2.8	22.9	9.8	1.7	3.1
15KG NonMags +20 um	168.7	68.8	0.03	0.06	7.76	58.5	1.92	0.52	7.51	14.5	19.3	19.3	55.0	79.3	21.8	60.6	83.3	76.6
15KG NonMags -20 um	32.3	13.1	0.04	0.09	11.8	47.7	5.20	0.45	5.36	14.0	5.5	5.5	16.0	12.4	11.3	10.0	11.4	14.1
Feed, calc	245	100.0	0.09	0.21	9.70	50.7	6.07	0.59	6.20	13.0	100	100	100	100	100	100	100	100
Feed, direct			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								
Combined Product																		
5KG Mags	14.9	6.1	0.46	1.05	13.0	24.3	26.9	0.96	2.05	6.62	30.7	30.7	8.1	2.9	26.9	9.9	2.0	3.1
Up 10KG Mags	29.2	11.9	0.44	1.02	15.0	23.5	22.5	0.97	1.85	6.67	58.6	58.6	18.3	5.5	44.1	19.6	3.6	6.1
Up to 15KG Mags	44.4	18.1	0.38	0.86	15.6	23.3	22.5	0.96	1.81	6.63	75.1	75.1	29.0	8.3	67.0	29.4	5.3	9.2
15KG NonMags	201	81.9	0.03	0.06	8.41	56.8	2.45	0.51	7.16	14.4	24.9	24.9	71.0	91.7	33.0	70.6	94.7	90.8

5. Batch Flotation Testwork

5.1. Batch Flotation Testing Overview

The flotation flowsheet was developed for trench samples taken from the Lofdal project in the Phase 1 program (SGS Project 18299-01). The flowsheet included a magnetic separation to remove iron oxide, followed by processing of the non-magnetic product by xenotime flotation (with two stages of cleaning).

A fresh bulk sample with lower head grade was taken from the Lofdal project in 2021. The ROM and LG XRF Comp were submitted for flotation flowsheet evaluation and optimization.

The stated objectives were:

- i) Confirm the flotation flowsheet is applicable with the new lower grade samples
- ii) Evaluate the recovery and grade with the new samples
- iii) Compare the flotation performance of a ROM sample versus LG XRF Comp sample
- iv) Investigate options to reduce the OPEX costs, i.e., coarser grind size, alternative collector types, reduced collector dosages
- v) Generate flotation concentrate from the new sample for downstream hydrometallurgical testwork

The typical flowsheet used in this testing program is depicted in Figure 5. The testing feed was stage-ground to the target size (mostly at a P_{100} of 53 μm , one test at a P_{100} of minus 75 μm , and one test at a P_{100} of minus 38 μm). The ground product was passed through WHIMS at 2 Amps (~2,000 Gauss) to reject iron oxide (one cleaner stage was included to minimize REE loss) and the non-magnetic product was used as feed for flotation. The flotation feed was conditioned at pH ~9, at elevated pulp temperature ~50°C, and at a high pulp density (~50%). The conditioning stages included pH adjustment with sodium hydroxide, followed by depressants and dispersant conditioning (sodium silicate, Calgon), and lastly collector conditioning (typically Florrea 3900, Florrea 3000). The rougher flotation was performed at ~35% pulp density. The rougher concentrate would be cleaned at pH 9-9.5 in initial tests, natural pH in later tests. Two cleaners were included in most cases.

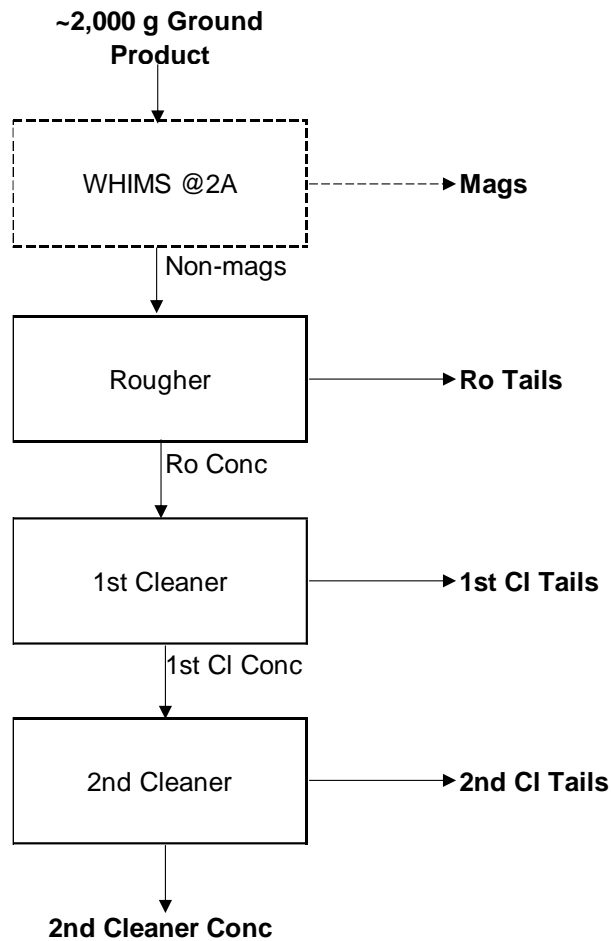


Figure 5: Typical Batch Flotation Flowsheet

Fifteen batch flotation tests (F101 to F115) were carried out on 2 kg charges of the ROM and LG-XRF samples to evaluate the flotation flowsheet for REE beneficiation. A summary of testing objectives is provided in Table 10.

Table 10: Batch Flotation Tests Objectives

Test #	Objective
ROM Sample	
F101	Initial baseline test on ROM sample, at -53 µm
F102	Similar to F101, with ~ half Calgon dosage
F103	Similar to F102, test alternative collector SM15
F104	Similar to F102, test ~50% more collector dosage of 3900 and natural pH in cleaner
F105	Similar to F101, with ~ one quarter of Calgon dosage
F106	Similar to F102, test coarser primary grind size, at -75 µm
F109	Similar to F105, test high intensity conditioning (HIC)
F110	Similar to F109, test alternative collector 8905Z
F111	Similar to F109, test ~40% reduced collector dosage
F112	Similar to F109, test ~65% reduced collector dosage
F113	Similar to F109, test alternative collector TJ-B3
F114	Similar to F109, test ~20% reduced collector dosage
F115	Similar to F109, test finer primary grind size, at -38 µm
LG-XRF Comp	
F107	Similar to F102, baseline test on LG-XRF Comp sample
F108	Similar to F105, test on LG-XRF Comp sample

Details of the flotation procedure and reagent scheme used in each test are provided in Appendix B.

Each test was carried out in a Denver D12 laboratory flotation machine, and three cell sizes (4L, 2L, 1L) were used in this program. Deionized water (DI) was used through the flotation testwork to avoid variations in performance due to water chemistry fluctuations.

5.2. Batch Flotation Test Results Summary

The flotation test results are discussed in the following sections:

- Baseline tests
- Alternative collectors
- Effect of collector dosages
- Effect of grind size
- ROM versus LG-XRF Comp Sample

All sections in the discussion used ROM sample except the above last sections, where both LG-XRF and ROM samples were included.

5.2.1. Baseline Tests

Table 11 summarizes the testing conditions for those tests (F101, F102, F105, F109) that served as baseline(s) in the subsequent sections. Table 12 presents the results summary and Figure 6 depicts the testing results as a function of yttrium grade and recovery.

Test F101 was the first baseline test, with conditions that were based on test F58 of the previous project 18299-01, which used a sorted product with higher head grade. As the yttrium losses were high in F101, lower dosages of depressant Calgon (at 100 g/t and 50 g/t) were employed in the roughers in tests F102 and F105, respectively. The yttrium rougher recovery improved significantly, from 56% in F101, to 79% in F102 and 89% in F105. However, the mass pull to the rougher concentrate increased at the reduced dosage of Calgon, along with the amounts of impurities (calcium, silicon, iron). The required amount of Calgon dosage was lower for the new sample, probably due to the lower calcium content in the new sample.

High intensity conditioning (HIC) was employed in test F109, by applying a higher agitation speed (~1800 rpm vs. ~1200 rpm normal) and 50 g/t Calgon was added in the rougher and 10 g/t of Calgon in the cleaner. With the HIC at 50 g/t Calgon, the rougher concentrate was similar to F102 with 100 g/t Calgon and normal conditioning. It seems the HIC might help to clean the mineral surfaces thus producing a cleaner concentrate with lower reagent dosage. In the cleaner, the 10 g/t Calgon rejected most of the calcium without significant yttrium loss. The 2nd cleaner concentrate in test F109 graded 2.57% Y₂O₃ (estimate ~5.87% TREO) with a recovery of ~70% and a mass pull of 2.9%. That represented a ~25 times yttrium upgrade ratio.

Table 11: Summary of Test Conditions of F101, F102, F105, F109, and Old F58 (18299-01)

Test #	P ₁₀₀	Roughers (Cleaner Feed)				Cleaners				Notes
	µm	Na ₂ SiO ₃ g/t	Calgon g/t	3900 g/t	3000 g/t	Na ₂ SiO ₃ g/t	Calgon g/t	3900 g/t	3000 g/t	
Old F58	53	250	200	1800	90	125	20	62.5	0	Baseline Test from Phase 1 on XRF SP sample
F101	53	250	200	1800	90	125	20	85	10	Baseline
F102	53	250	100	1800	80	125	20	200	10	~50% less Calgon
F105	53	250	50	1800	80	75	0	200	10	~75% less Calgon; Reduced depressant in CI
F109	53	250	50	1800	80	75	10	200	10	High Intensity Conditioning

Table 12: Flotation Test Results Summary of F101, F102, F105, and F109

Test #	Head, calc	Products	Wt	Assay, %					Distribution, %			
	Y ₂ O ₃ %		%	Y ₂ O ₃	TREO, est	CaO	SiO ₂	Fe ₂ O ₃	Y ₂ O ₃	CaO	SiO ₂	Fe ₂ O ₃
Old F58	0.43	2nd CI Con 1-2	5.8	5.21	8.75	9.2	17.8	35.5	70.6	3.0	2.5	22.2
		Ro Conc 1-5	16.5	2.00	3.37	17.2	28.7	18.6	77.5	15.9	11.5	33.2
F101	0.11	2nd CI Con 1-2	1.4	2.44	5.59	4.6	13.0	44.3	31.9	0.6	0.4	9.1
		Ro Conc 1-5	10.4	0.57	1.30	10.4	36.5	17.8	55.5	10.5	7.8	27.4
F102	0.10	2nd CI Con 1-2	1.4	3.51	8.05	5.4	14.1	38.7	47.5	0.7	0.40	7.9
		Ro Conc 1-5	12.9	0.64	1.46	11.7	37.0	15.8	79.2	14.6	9.7	29.8
F105	0.10	2nd CI Con 1-2	4.1	1.81	4.14	18.2	11.7	25.8	74.1	7.3	1.0	17.5
		Ro Conc 1-5	27.0	0.33	0.76	18.9	31.4	10.9	88.6	49.8	17.3	48.2
F109	0.11	2nd CI Con 1-2	2.9	2.57	5.87	10.4	15.3	34.1	69.5	2.9	0.9	15.0
		Ro Conc 1-5	13.6	0.63	1.44	16.6	29.7	15.2	80.0	22.0	8.3	31.4

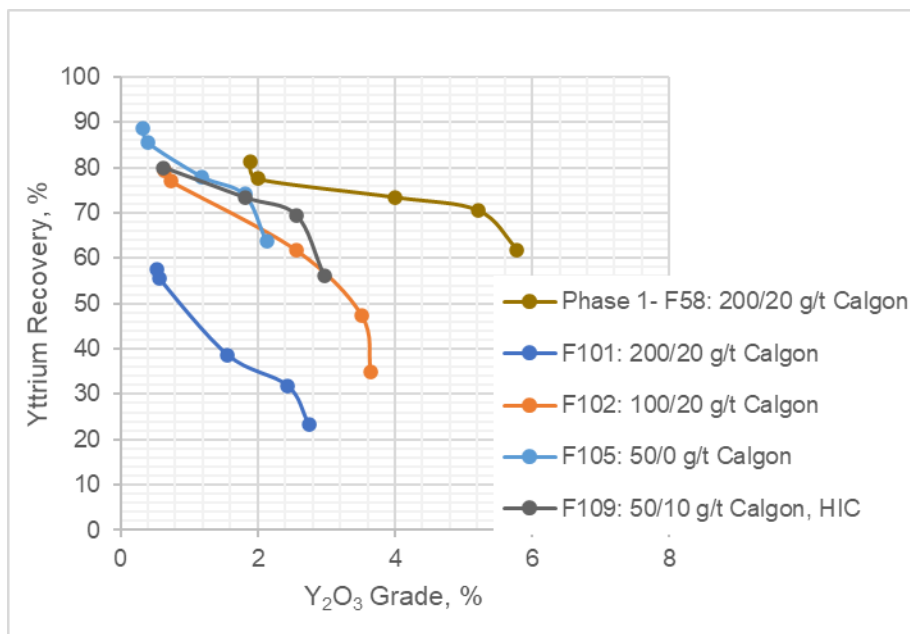


Figure 6: Flotation Results of F101, F102, F105, F109 Compared to 18299-01 F58

5.2.2. Alternative Collectors

As recognized in the previous program both the dosage and unit price of the collector Florrea 3900 were high, so alternative collectors were investigated in this program. Table 13 lists the collectors that were evaluated in this program, in tests F103, F105, and F113. The testing conditions, results summary, and plotted results are shown in Table 14, Table 15, and Figure 7, respectively.

The phosphate ester collector SM-15 (F103) is a strong collector with low selectivity to calcium. Test F103 floated more calcium than yttrium in both rougher (200 g/t Calgon) and cleaner (75 g/t Calgon).

Modified hydroxamate collector 8905Z, which was tested in F110 performed very similarly to Florrea 3900 (in F109) in the roughers, although Florrea 3900 produced a cleaner final product than 8905Z.

A modified hydroxamate collector TJ-B3, used in Test F113 did not produce as good a yttrium grade or recovery compared to Florrea 3900 or 8905Z.

Table 13: Summary of Collectors Investigated

Reagent Name	Supplier	Description
3900	Florrea	Modified hydroxamate
SM15	Clariant	Phosphate Ester
8905Z	Florrea	Modified hydroxamate
TJ-B3	Hunan Tongju Chemical	Modified hydroxamate

Table 14: Test Conditions Summary of F103, F110, and F113

Test #	P ₁₀₀	Roughers (Cleaner Feed)				Cleaners				Notes
	µm	Na ₂ SiO ₃ g/t	Calgon g/t	3900 g/t	3000 g/t	Na ₂ SiO ₃ g/t	Calgon g/t	3900 g/t	3000 g/t	
F103	53	250	200	-	-	125	75	-	-	SM15: 600 g/t in Ro 85 g/t in Cl
F110	53	250	50	-	80	75	10	-	10	8905Z: 1800 g/t in Ro; 200 g/t in Cl
F113	53	250	50	-	80	75	10	-	10	TJ-B3: 1800 g/t in Ro; 200 g/t in Cl

Table 15: Flotation Test Results Summary of F103 (vs. F101), F110, and F113 (vs. F109)

Test #	Products	Wt	Assay, %					Distribution, %			
		%	Y ₂ O ₃	TREO, est	CaO	SiO ₂	Fe ₂ O ₃	Y ₂ O ₃	CaO	SiO ₂	Fe ₂ O ₃
F101	2nd Cl Con 1-2	1.4	2.44	5.59	4.55	13.0	44.3	31.9	0.6	0.4	9.1
	Ro Conc 1-5	10.4	0.57	1.30	10.4	36.5	17.8	55.5	10.5	7.8	27.4
F103	2nd Cl Con 1-2	14.3	0.32	0.73	41.4	3.34	6.47	45.6	57.8	1.0	14.0
	Ro Conc 1-5	35.7	0.24	0.55	26.2	24.3	7.74	85.5	91.2	17.4	41.6
F109	2nd Cl Con 1-2	2.9	2.57	5.87	10.4	15.3	34.1	69.5	2.9	0.9	15.0
	Ro Conc 1-4	13.6	0.63	1.44	16.6	29.7	15.2	80.0	22.0	8.3	31.4
F110	2nd Cl Con 1-2	3.8	1.95	4.47	9.39	25.6	27.4	70.5	3.4	2.0	15.6
	Ro Conc 1-4	13.8	0.61	1.40	13.6	35.4	14.5	81.3	18.2	10.0	30.3
F113	2nd Cl Con 1-2	2.6	1.62	3.70	10.6	21.5	31.9	39.3	2.7	1.1	12.8
	Ro Conc 1-4	15.4	0.47	1.09	18.7	30.8	13.3	67.0	28.2	9.5	31.0

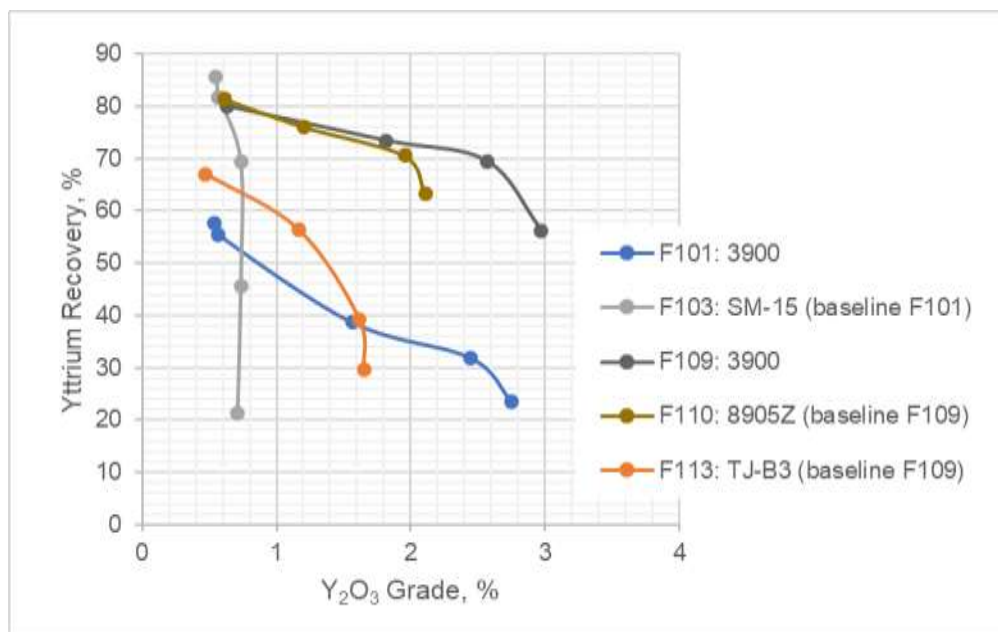


Figure 7: Flotation Test Results of Alternative Collectors: F103, F110, F113

5.2.3. Effect of Collector Dosages

As Florrea 3900 was still determined to be the most effective collector, various dosages of 3900 were investigated in tests F104, F111, F112, and F114. The testing conditions, results summary, and plotted results are shown in Table 16, Table 17, and Figure 8, respectively.

A higher Florrea 3900 dosage of 2800 g/t (vs 1800 g/t in the baseline) was evaluated in F102, but this resulted in a reduced yttrium recovery and increased calcium recovery. Therefore, in F114, the Florrea dosage in the roughers was reduced to 1400 g/t and this test produced results that were similar to those of test F109.

A further reduction in Florrea 300 dosage to 1000 g/t in F111 lowered yttrium recovery in the roughers from 80% in F109 to 68% in F111. Calcium recovery to the concentrate was also reduced from 22% to 11%. The recovery of the 2nd cleaner concentrate was also about 10% lower than F109. A lower dosage of the depressant Calgon may help to improve the yttrium recovery under test F111 conditions.

At a much lower collector dosage of 500 g/t in F112, the yttrium recovery was significantly reduced. This dosage was clearly too low.

Table 16: Test Conditions Summary of F104, F111, F112, and F114

Test #	P ₁₀₀	Roughers (Cleaner Feed)				Cleaners			
	µm	Na ₂ SiO ₃ g/t	Calgon g/t	3900 g/t	3000 g/t	Na ₂ SiO ₃ g/t	Calgon g/t	3900 g/t	3000 g/t
F104	53	250	100	2800	80	75	10	200	10
F111	53	250	50	1000	80	75	10	200	10
F112	53	250	50	500	80	75	10	200	10
F114	53	250	50	1400	80	75	10	200	10

Table 17: Flotation Test Results Summary of F104 (vs. F102), F111, F112, and F114 (vs. F109)

Test #	Products	Wt	Assay, %					Distribution, %			
		%	Y ₂ O ₃	TREO, est	CaO	SiO ₂	Fe ₂ O ₃	Y ₂ O ₃	CaO	SiO ₂	Fe ₂ O ₃
F102	2nd Cl Con 1-2	1.4	3.51	8.05	5.39	14.1	38.7	47.5	0.7	0.4	7.9
	<i>Ro Conc 1-5</i>	12.9	0.64	1.46	11.7	37.0	15.8	79.2	14.6	9.7	29.8
F104	2nd Cl Con 1-2	1.6	3.32	7.60	6.50	14.8	38.1	51.0	1.0	0.5	9.4
	<i>Ro Conc 1-5</i>	13.3	0.58	1.33	13.8	34.1	15.1	73.7	18.2	9.2	30.8
F109	2nd Cl Con 1-2	2.9	2.57	5.87	10.4	15.3	34.1	69.5	2.9	0.9	15.0
	<i>Ro Conc 1-4</i>	13.6	0.63	1.44	16.6	29.7	15.2	80.0	22.0	8.3	31.4
F111	2nd Cl Con 1-2	2.0	3.02	6.92	5.88	18.4	36.5	57.5	1.2	0.8	11.2
	<i>Ro Conc 1-4</i>	10.4	0.69	1.59	10.8	37.4	15.6	67.9	11.0	8.0	24.7
F112	2nd Cl Con 1-2	0.7	2.07	4.75	4.21	10.4	32.9	13.9	0.3	0.1	3.5
	<i>Ro Conc 1-4</i>	7.0	0.61	1.41	10.8	36.7	16.3	41.5	7.4	5.1	17.5
F114	2nd Cl Con 1-2	2.6	2.70	6.18	8.02	17.7	35.6	68.7	2.1	0.9	14.5
	<i>Ro Conc 1-4</i>	12.4	0.63	1.44	12.1	36.4	15.0	75.3	14.8	9.2	28.7

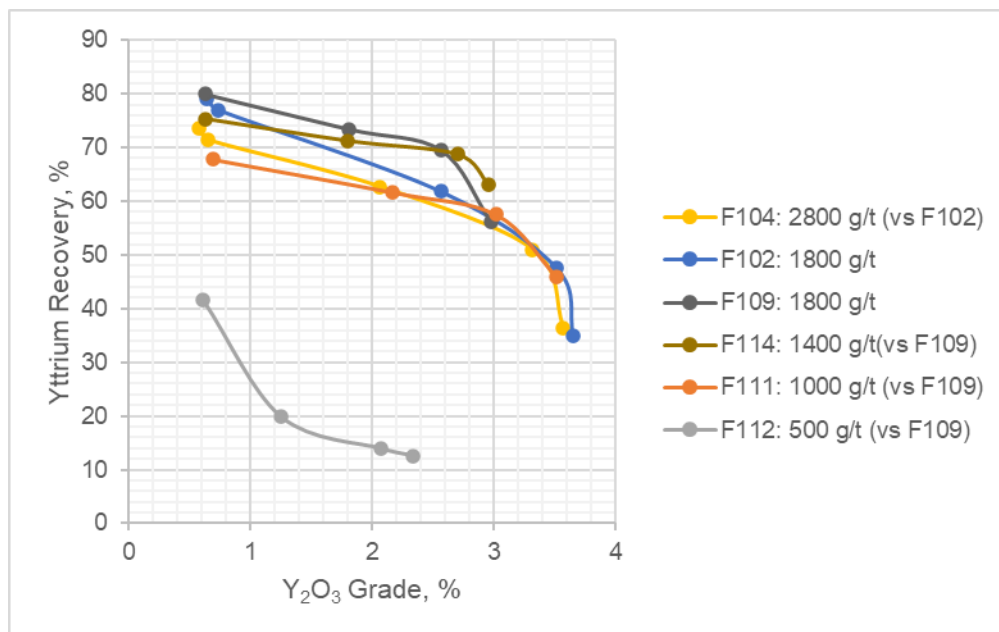


Figure 8: Flotation Test Results of Various Collector Dosages: F104, F111, F112, and F114

5.2.4. Effect of Grind Size

The effect of primary grind size was investigated in tests F106 and F105, at -75 μm and -38 μm , respectively. The testing conditions, results summary, and plotted results are shown in Table 18, Table 19, and Figure 9, respectively.

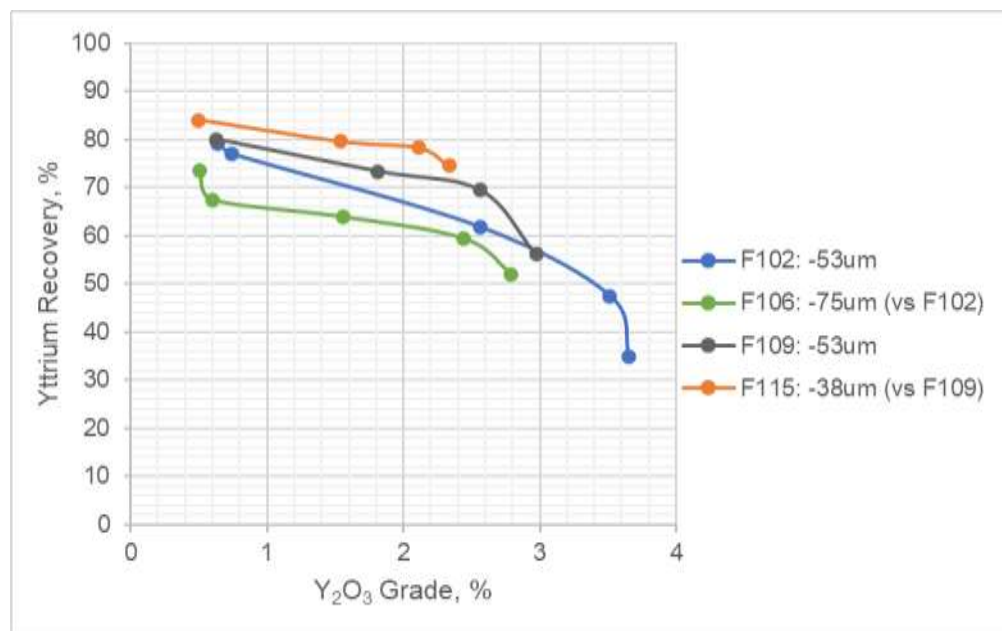
At -75 μm , in test F106, the yttrium recovery was slightly lower than the baseline test F102 at a similar concentrate grade. At -38 μm , in test F115, the yttrium recovery was higher with a slightly higher mass pull. Based on the yttrium grade vs recovery curves, the finer grind at -38 μm seemed to achieve better metallurgical results, which is consistent with the mineralogy investigation. The F115 2nd cleaner concentrate graded 2% Y₂O₃ (estimate ~4.9% TREO) with an yttrium recovery of 78% and a mass pull of 3.9%.

Table 18: Test Conditions Summary of F106, F109, and F115

Test #	P ₁₀₀	Roughers (Cleaner Feed)				Cleaners			
	μm	Na ₂ SiO ₃ g/t	Calgon g/t	3900 g/t	3000 g/t	Na ₂ SiO ₃ g/t	Calgon g/t	3900 g/t	3000 g/t
F106	75	250	100	1800	80	75	0	200	10
F109	53	250	50	1800	80	75	10	200	10
F115	38	250	50	1800	80	75	10	200	10

Table 19: Flotation Test Results Summary of F106 (vs. F102), F115 (vs. F109)

Test #	Products	Wt	Assay, %					Distribution, %			
		%	Y ₂ O ₃	TREO _{est}	CaO	SiO ₂	Fe ₂ O ₃	Y ₂ O ₃	CaO	SiO ₂	Fe ₂ O ₃
F102	2nd Cl Con 1-2	1.4	3.51	8.05	5.39	14.1	38.7	47.5	0.7	0.4	7.9
	Ro Conc 1-5	12.9	0.64	1.46	11.7	37.0	15.8	79.2	14.6	9.7	29.8
F106	2nd Cl Con 1-2	2.5	2.44	5.59	7.09	23.7	31.3	59.4	1.8	1.2	11.1
	Ro Conc 1-5	15.1	0.51	1.16	12.0	37.9	13.6	73.4	17.9	11.7	28.8
F109	2nd Cl Con 1-2	2.9	2.57	5.87	10.4	15.3	34.1	69.5	2.9	0.9	15.0
	Ro Conc 1-4	13.6	0.63	1.44	16.6	29.7	15.2	80.0	22.0	8.3	31.4
F115	2nd Cl Con 1-2	3.9	2.12	4.85	10.7	17.0	33.4	78.3	4.0	1.37	20.1
	Ro Conc 1-4	17.9	0.50	1.14	14.5	35.6	13.6	84.1	24.6	13.0	37.1


Figure 9: Effect of Grind Size: Flotation Test Results of F106 and F115

5.2.5. LG-XRF Sample vs ROM

The low-grade sorting composite (LG-XRF Comp) was evaluated using the developed flowsheet, in tests F107 and F108. The testing conditions, results summary, and plotted results are shown in Table 20, Table 21, and Figure 10, respectively. The LG-XRF Comp showed no improvement in concentrate grade or recovery in both tests. The results suggested that the sorting process might not be beneficial. The compounding effect of the recovery loss over the sorting stage is excluded from the flotation data.

Table 20: Test Conditions Summary of F107 and F108 on LG-XRF Comp

Test #	P ₁₀₀	Roughers (Cleaner Feed)				Cleaners			
	µm	Na ₂ SiO ₃ g/t	Calgon g/t	3900 g/t	3000 g/t	Na ₂ SiO ₃ g/t	Calgon g/t	3900 g/t	3000 g/t
F107	53	250	100	1800	80	75	0	200	10
F108	53	250	50	1800	80	75	0	200	10

Table 21: Flotation Test Results Summary of F107 (vs. F102), F108 (vs. F105)

Test #	Comp ID	Head, calc	Products	Wt	Assay, %					Distribution, %			
		Y ₂ O ₃ %		%	Y ₂ O ₃	TREO, est	CaO	SiO ₂	Fe ₂ O ₃	Y ₂ O ₃	CaO	SiO ₂	Fe ₂ O ₃
F102	ROM	0.10	2nd Cl Con 1-2	1.4	3.51	8.05	5.39	14.1	38.7	47.5	0.7	0.4	7.9
			Ro Conc 1-5	12.9	0.64	1.46	11.7	37.0	15.8	79.2	14.6	9.7	29.8
F107	LG-XRF	0.11	2nd Cl Con 1-2	3.0	2.09	4.56	8.89	21.2	34.0	59.9	2.6	1.3	14.8
			Ro Conc 1-5	14.0	0.57	1.23	13.3	33.8	16.8	74.5	17.7	9.8	33.7
F105	ROM	0.10	2nd Cl Con 1-2	4.1	1.81	4.14	18.2	11.7	25.8	74.1	7.3	1.0	17.5
			Ro Conc 1-5	27.0	0.33	0.76	18.9	31.4	10.9	88.6	49.8	17.3	48.2
F108	LG-XRF	0.11	2nd Cl Con 1-2	2.8	2.22	5.07	14.3	12.8	32.4	56.8	3.8	0.73	11.6
			Ro Conc 1-5	22.4	0.39	0.90	21.4	25.5	13.3	81.6	46.5	11.8	38.6

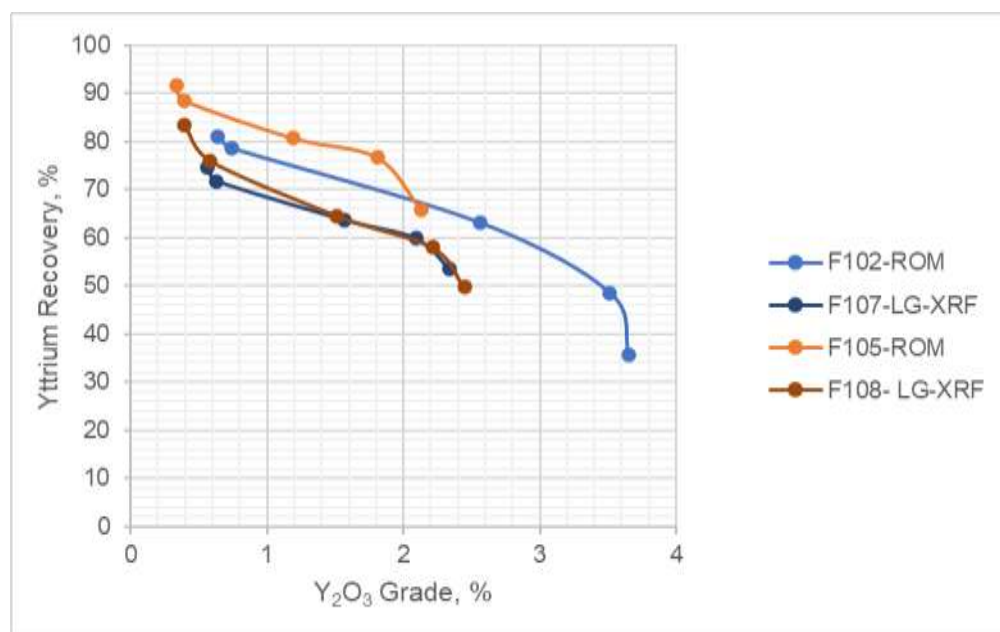


Figure 10: Flotation Test Results: Effect of Grind Size: F107 and F108

5.3. Individual Rare Earth Element Distribution

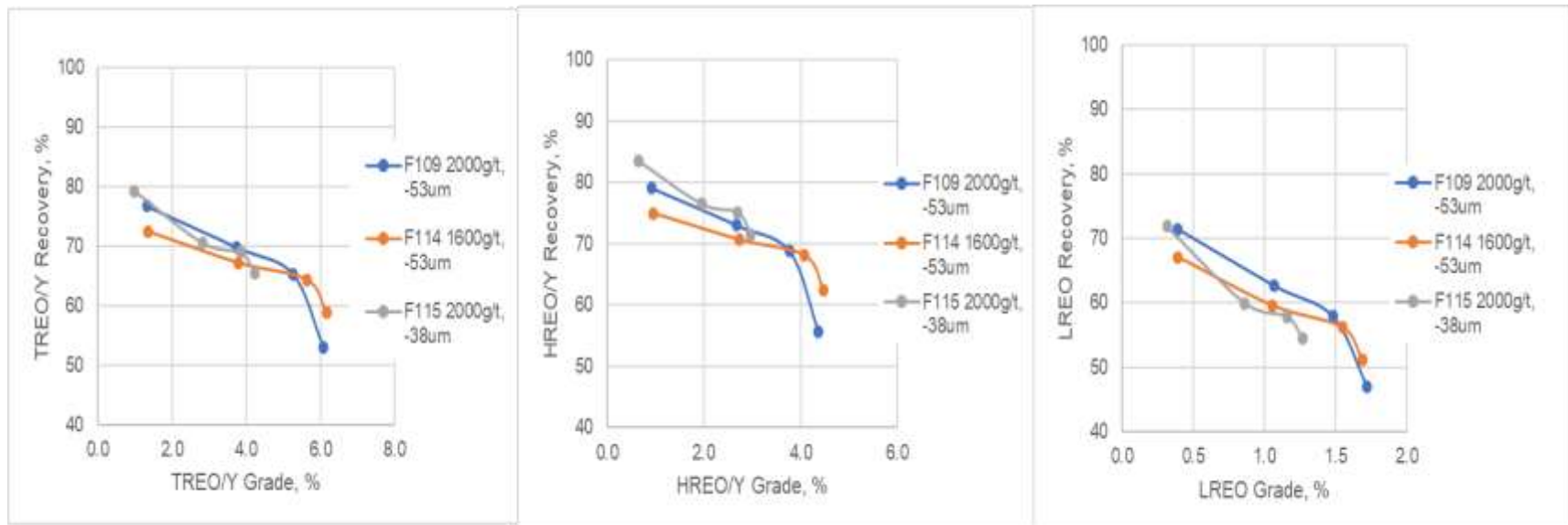
The flotation products from the most promising tests F109, F114, and F115 were submitted for a rare earth element scan by ICP-MS, to understand the distribution of individual rare earth elements. The results are summarized in Table 22 and depicted in Figure 11.

In general, the heavy rare earth elements (Eu to Lu) showed a similar trend to yttrium. The recovery of HREO (without yttrium) was in the 64 to 72% range. However, recovery of the light rare earth elements (La to Sm) was lower in the 56 to 58% range.

The measured TREO grade and recovery in F115 (-38 µm) was similar to that of F109 (-53 µm). The HREO+Y performance was slightly better at the -38 µm grind, while the LREO seemed to float better at the -53 µm grind.

Table 22: Results Summary of REE Distribution on the 2nd Cleaner Concentrate 1-2

Test	Cond.	Wt %	Assay, % g/t																			Distribution, %																				
			Y ₂ O ₃	La g/t	Ce g/t	Pr g/t	Nd g/t	Sm g/t	Eu g/t	Gd g/t	Tb g/t	Dy g/t	Ho g/t	Er g/t	Tm g/t	Yb g/t	Lu g/t	U g/t	Th g/t	LREO	HREO/Y	TREO	Y ₂ O ₃	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	U	Th	LREO	HREO/Y	TREO
F109	2000g/t, -53um	2.9	2.07	3191	5794	588	2105	654	299	1617	388	2915	639	1906	259	1638	209	102	3101	1.5	3.8	5.2	70.5	57.5	59.4	55.0	56.0	55.7	56.7	59.2	63.3	67.0	67.2	67.3	65.9	67.9	61.8	37.3	43.5	57.9	68.8	65.3
F114	1600g/t, -53um	2.6	2.27	3243	5862	608	2205	689	317	1780	413	3259	693	2137	283	1706	228	99	3223	1.5	4.1	5.6	70.0	56.5	56.3	56.6	56.9	51.7	55.5	58.3	61.0	64.4	69.5	65.1	66.8	66.9	63.9	64.9	42.8	56.2	68.1	64.4
F115	2000g/t, -38um	3.9	1.73	2467	4554	470	1676	500	254	1387	319	2531	531	1658	220	1308	176	84	2614	1.2	2.7	3.9	77.4	57.7	58.9	58.2	56.3	54.2	62.5	66.6	69.3	72.9	74.4	73.5	72.2	72.7	73.3	50.2	48.2	57.9	75.2	69.0
Head (Direct)			0.11	162	299	33	108	38	16	75	16	125	26	76	11	65	9	8	191	0.08	0.14	0.22																				


Figure 11: Flotation Results of Measured TREO, HREO, and LREO.

5.4. Mineralogical Analysis on Flotation Products

The following flotation products were submitted for TIMA-X analysis. The flotation feed particle sizes were at P₁₀₀ of 53 µm for F114 and P₁₀₀ of 38 µm for F115. The objective was to identify the gangue minerals and to understand the mineral liberation/association information in various flotation products. Mineralogical results are summarized in Table 23, Table 24, and Table 25. The complete mineralogy results are included in Appendix C.

- F114 2nd CI Conc -A
- F114 Ro Tails
- F115 2nd CI Conc -A
- F115 Ro Tails

Noticeable amounts of rare earth minerals other than xenotime were identified in the 2nd cleaner concentrates, mostly synchysite and lesser amounts of monazite. The gangue minerals in the 2nd cleaner concentrates were mostly iron-oxides, quartz/feldspar, calcite/dolomite, rutile/ilmenite, and a minor amount of Zr silicates, apatite, biotite/chlorite/mica, and amphibole/epidote.

Table 23: Summary of Mineral Modals

Sample		F114 Ro Tails	F114 2 nd CI Conc - A	F115 Ro Tails	F115 2 nd CI Conc - A
Mineral Mass (%)	Xenotime	0.06	7.00	0.05	5.54
	Other REM	0.03	3.15	0.03	2.53
	Thorite/Th-Y-silicates	0.01	0.44	0.00	0.33
	Zr Silicates	0.08	3.28	0.06	3.30
	Apatite	0.21	2.33	0.12	4.67
	Calcite/Dolomite	19.8	8.45	19.1	11.7
	Ankerite/Siderite	1.30	3.15	1.30	4.58
	Quartz/Feldspars	66.3	12.0	66.6	12.5
	Biotite/Chlorite/Mica	5.55	1.32	6.23	1.38
	Amphibole/Epidote	3.25	1.48	3.16	1.68
	Fe-Oxides	2.04	38.7	1.79	35.5
	Ilmenite	0.11	4.63	0.08	3.90
	Rutile	0.21	10.3	0.18	8.37
	Other	1.07	3.79	1.29	4.02
	Total	100.0	100.0	100.0	100.0

The xenotime and other REM were reasonably liberated in both 2nd cleaner concentrates, at ~70-75%. The unliberated minerals were mostly associated with complex mineralogy (more than two types of minerals).

The xenotime and other REM of F115 2nd cleaner concentrate at P₁₀₀ of 53 µm was only slightly better liberated than those of F114 2nd cleaner concentrate at P₁₀₀ of 38 µm, but their liberation in F115 rougher tails was much better than F114 rougher tails.

Table 24: Xenotime Liberation and Association Summary

Mineral Name	F114 Ro Tails	F114 2nd CI Conc - A	F115 Ro Tails	F115 2nd CI Conc - A
Liberated Xenotime	14.7	73.5	53.1	75.1
Xnt: REM	0.00	0.26	0.00	0.29
Xnt: Zr Silicates	0.31	1.62	3.08	1.67
Xnt: Apatite	2.19	0.67	1.25	1.15
Xnt: Calcite/Dolomite	8.84	3.42	9.25	2.92
Xnt: Ankerite/Siderite	0.46	0.58	0.10	0.75
Xnt: Quartz/Feldspars	6.89	1.76	11.7	1.87
Xnt: Biotite/Chlorite/Mica	0.84	0.28	0.68	0.27
Xnt: Fe-Oxides	0.35	3.53	0.31	3.66
Xnt: Other	1.00	1.75	0.00	1.73
Complex	64.4	12.6	20.5	10.6
Total	100.0	100.0	100.0	100.0

Table 25: Other REM Liberation and Association Summary

Mineral Name	F114 Ro Tails	F114 2nd CI Conc - A	F115 Ro Tails	F115 2nd CI Conc - A
Liberated REM	29.7	67.3	42.4	70.7
REM: Xenotime	0.00	0.55	0.18	0.55
REM: Apatite	3.02	0.36	0.00	0.58
REM: Calcite/Dolomite	5.77	2.83	12.4	2.33
REM: Ankerite/Siderite	0.00	0.35	0.00	0.75
REM: Quartz/Feldspars	3.11	1.93	7.59	1.85
REM: Biotite/Chlorite/Mica	2.76	0.59	0.64	0.52
REM: Amphibole/Epidote	0.00	0.56	0.24	0.39
REM: Fe-Oxides	3.79	6.92	8.61	6.80
REM: Other	0.39	2.75	1.14	3.08
Complex	51.4	15.9	26.8	12.5
Total	100.0	100.0	100.0	100.0

6. Bulk Concentration Production by Flotation

Bulk flotation concentrates were produced for downstream hydrometallurgical testing utilizing 10 kg test charges of the ROM sample. Due to the tight timeline, bulk concentrate production was performed while the optimization of the batch flotation flowsheet was still in progress. Tests F102 and F105 were chosen as the baseline conditions for bulk concentration production tests. Four bulk concentrate production tests were performed. The testing conditions summary, testing results summary, and plotted results are shown in Table 26, Table 27, and Figure 12, respectively. Details of these tests are provided in Appendix D.

Based on experience from previous testwork (SGS Project 18299-01), a 1.3 times scale-up factor for the residence time was used to scale up from the 2 kg tests to 10 kg bulk flotation tests. Iron removal by WHIMS was performed on the 2nd cleaner concentrate, as the flotation feed sample volume was too high for the laboratory batch WHIMS unit. This change apparently had minimal impact on the quality of the flotation concentrate.

The testing conditions for all four bulk flotation tests were similar. Tests CP102 and CP104 used 5 g/t more Calgon depressant in the cleaner stage; this increased the yttrium concentrate grade slightly, but at the cost of reduced recovery. Overall, the bulk flotation tests demonstrated good repeatability and potential for scale up (to 10 kg batch) to produce a similar quality of concentrate from the 2 kg batch tests. The 2nd Cleaner Concentrate Non-Mags produced in CP101 to CP104 graded at 2.55% Y₂O₃ (estimate ~5.9% TREO) at an yttrium recovery of 67% and a mass pull of 3% on average. The four bulk flotation tests produced ~1 kg REO concentrate, grading 2.6% Y₂O₃ (estimate ~6% TREO) (Table 28).

Table 26: Test Conditions Summary of Bulk Flotation Tests CP101 to CP104

Test #	P ₁₀₀	Roughers (Cleaner Feed)				Cleaners			
	µm	Na ₂ SiO ₃ g/t	Calgon g/t	3900 g/t	3000 g/t	Na ₂ SiO ₃ g/t	Calgon g/t	3900 g/t	3000 g/t
CP101	53	250	50	1800	80	75	10	200	10
CP102	53	250	50	1800	80	75	15	200	10
CP103	53	250	50	1800	80	75	10	200	10
CP104	53	250	50	1800	80	75	15	200	10

Table 27: Bulk Concentrate Product Test Results of CP101 to CP104

Test #	Products	Wt	Assay, %					Distribution, %			
		%	Y ₂ O ₃	TREO, est	CaO	SiO ₂	Fe ₂ O ₃	Y ₂ O ₃	CaO	SiO ₂	Fe ₂ O ₃
CP101	2nd Cl Con 1-2	3.3	2.08	4.77	6.51	15.8	46.8	65.6	2.0	1.1	23.1
	Ro Conc 1-4	12.3	0.69	1.57	13.9	29.1	21.0	81.7	16.2	7.3	39.1
CP102	2nd Cl Con NonMags	2.7	2.58	5.90	8.01	20.9	31.4	65.5	2.1	1.2	24.2
	Ro Conc 1-4	13.6	0.63	1.45	13.5	32.1	17.8	77.1	17.6	9.1	37.2
CP103	2nd Cl Con NonMags	3.2	2.24	5.13	11.6	18.4	29.2	75.4	3.4	1.3	27.0
	Ro Conc 1-4	16.5	0.51	1.16	15.7	30.7	16.3	84.8	23.0	10.6	42.2
CP104	2nd Cl Con NonMags	2.9	3.31	7.59	9.44	24.4	26.8	61.2	2.6	1.46	11.6
	Ro Conc 1-4	17.5	0.79	1.80	14.6	32.2	16.9	87.5	24.1	11.6	43.9
2nd Cl Conc NonMags CP101 to CP104 Average		3.0	2.55	5.85	8.89	19.9	33.5	66.9	2.5	1.3	21.5

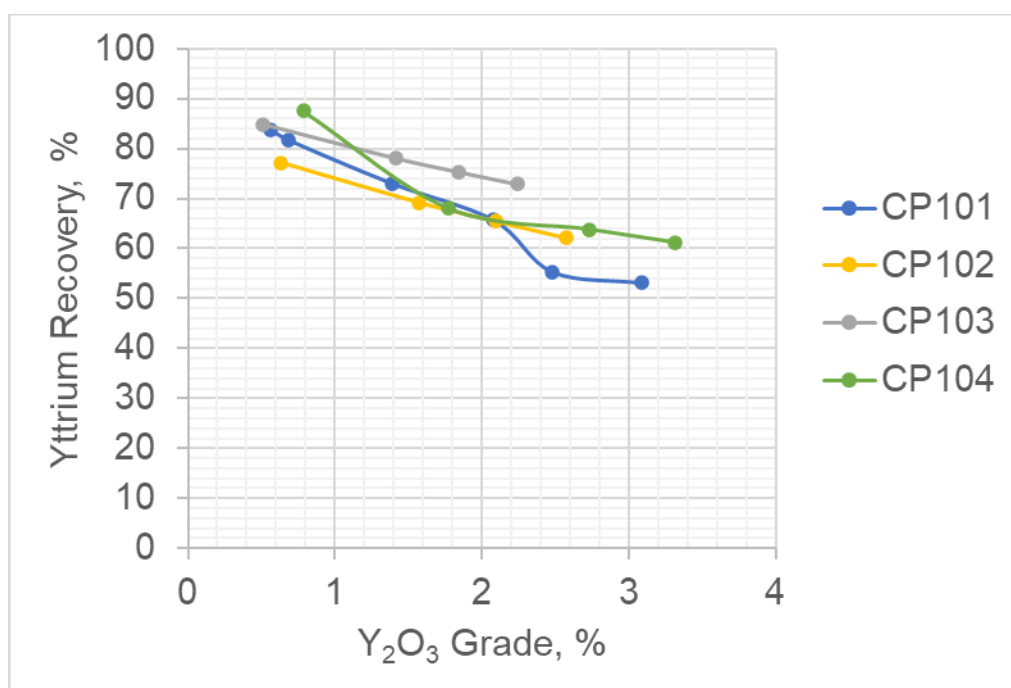


Figure 12: Flotation Results of CP101 to CP104

Table 28: Summary of Bulk Concentrate Production

Products	Wt, g	Assay, %							
		Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
CP-101 2nd Cl Con A NonMags@2A	170	3.09	7.08	7.28	16.9	35.5	10.8	1.37	3.40
CP-101 2nd Cl Conc B	91	1.12	2.56	8.37	20.6	41.1	6.08	1.97	4.24
CP-102 2nd Cl Con NonMags@2A	266	2.58	5.90	8.01	20.9	31.4	9.98	1.89	4.27
CP-103 2nd Cl Con NonMags@2A	327	2.24	5.13	11.6	18.4	29.2	9.24	1.69	3.75
CP-104 2nd Cl Con NonMags@2A	286	3.31	7.59	9.44	24.4	26.8	7.96	2.60	5.27
Combined - Exp	1140	2.63	6.01	9.32	20.4	31.0	9.07	1.94	4.24

Conclusions and Recommendations

The following conclusions may be drawn from the testwork performed on the fresh, lower grade samples taken from the Lofdal Heavy Rare Earth Project:

- Flotation performance with the run of mine sample (ROM) was reasonable, despite the lower head grade. A yttrium upgrade ratio of ~20 was achieved at ~70% yttrium recovery, in only ~3% mass pull.
- The Calgon dosage established in earlier testwork can likely be reduced, particularly if it is added to high intensity conditioning. However, this can result in lower concentrate grade owing to increased calcium extraction so it will have to be carefully optimized bearing in mind the effect on downstream operating costs.
- The collector Florrea 3900 still seems to be the best collector tested in this program to date, although the modified hydroxamate collector 8905Z performed very similarly. A reduced collector dosage of ~1600 g/t generated similar yttrium grade and recovery compared to tests at 2000 g/t. Further collector dosage reduction (i.e., to ~1000 g/t) is worth further investigation.
- It is unlikely that ore sorting or magnetic separation will prove to be viable options for pre-concentrating the ore prior to flotation.
- However, magnetic separation at low intensity (~2000 Gauss) was still beneficial for removing paramagnetic iron minerals from the flotation concentrate. It made little difference whether this was performed on the flotation feed or the flotation concentrate.
- A finer primary grind size of 100% passing 38 µm achieved the highest yttrium (and HREO) recovery. However, this was achieved at a slightly higher mass pull, which could negatively impact downstream leaching costs. This grind size was consistent with the maximum practical REE liberation based on the mineralogy investigation. Financial modelling including milling costs and downstream hydrometallurgy costs as well as tailings handling is needed to determine the optimum primary grind size.

The following recommendations are made for future testwork:

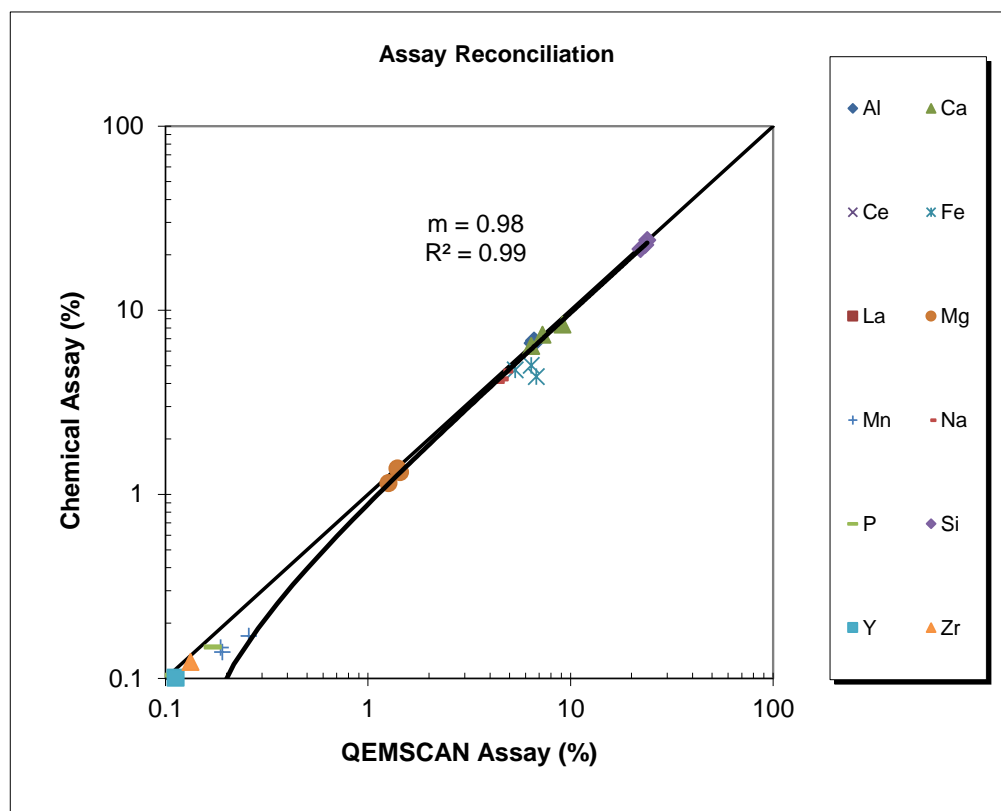
- Continue to evaluate alternative collectors and new depressants to improve grade and recovery and reduce operating costs as they become available.
- Perform locked cycle tests to evaluate the effect of recirculation streams on the flotation performance

- Test water from the mine site to evaluate its impact on flotation performance.
- Evaluate the developed flowsheet on further variability samples.
- Conduct pilot plant flotation tests to confirm flotation regime and produce adequate concentrate for downstream hydrometallurgical design and optimization.
- Solid liquid tests should be conducted on the new samples to provide data that can be used for thickener sizing and tailings pumping.
- Tailings tests for self-neutralizing capacities and geochemistry.

Appendix A – Head Mineralogy

Namibia
 CALR-18299-01/03
 MI5029-APR22 & MI5009-MAY22

Assay Reconciliation



Sample	ROM Head		LG XRF Comp
Fraction	+38 um	-38 um	As Rec'd
Al (TIMA)	6.59	6.68	6.49
Al (Chemical)	6.83	6.72	6.62
Ca (TIMA)	6.40	9.11	7.26
Ca (Chemical)	6.40	8.36	7.36
Ce (TIMA)	0.03	0.03	0.03
Ce (Chemical)	0.02	0.03	0.03
Fe (TIMA)	6.77	5.32	6.40
Fe (Chemical)	4.35	4.74	5.01
La (TIMA)	0.01	0.01	0.01
La (Chemical)	0.02	0.02	0.02
Mg (TIMA)	1.26	1.44	1.40
Mg (Chemical)	1.15	1.32	1.38
Mn (TIMA)	0.19	0.26	0.19
Mn (Chemical)	0.14	0.17	0.15
Na (TIMA)	4.70	4.27	4.47
Na (Chemical)	4.68	4.10	4.23
P (TIMA)	0.11	0.17	0.11
P (Chemical)	0.10	0.15	0.10
Si (TIMA)	23.9	22.2	23.4
Si (Chemical)	24.0	21.5	22.7
Y (TIMA)	0.08	0.11	0.08
Y (Chemical)	0.08	0.10	0.10
Zr (TIMA)	0.08	0.13	-
Zr (Chemical)	0.09	0.12	-

Namibia
 CALR-18299-01/03
 MI5029-APR22 & MI5009-MAY22

Modals

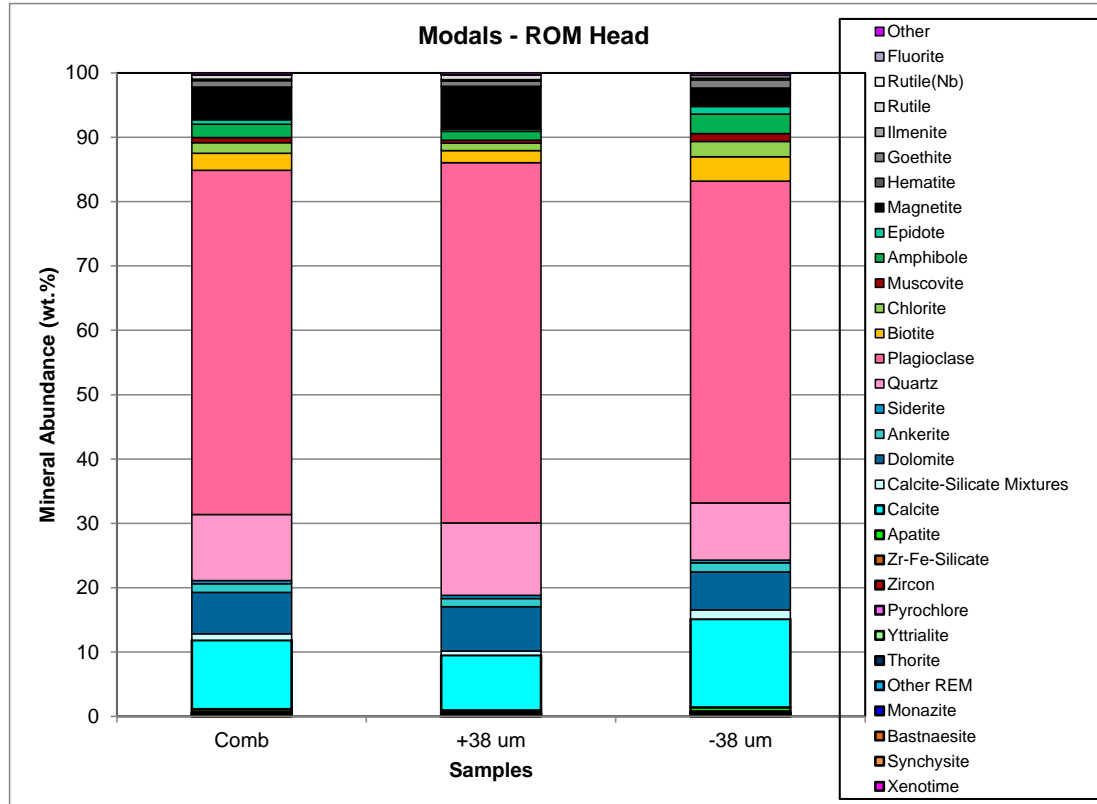
Survey		CALR-18299-01/03 / MI5029-APR22 & MI5009-MAY22			
Project		Namibia			
Sample		ROM Head			LG XRF Comp
Fraction		Comb	+38 um	-38 um	As Rec'd
Mass Size Distribution (%)		100.0	58.2	41.8	100.0
Mineral Mass (%)	Xenotime	0.26	0.22	0.32	0.23
	Synchysite	0.09	0.09	0.10	0.11
	Bastnaesite	0.00	0.00	0.00	0.00
	Monazite	0.03	0.04	0.03	0.02
	Other REM	0.00	0.00	0.01	0.00
	Thorite	0.00	0.00	0.00	0.00
	Yttrialite	0.02	0.02	0.01	0.01
	Pyrochlore	0.00	0.00	0.00	0.00
	Zircon	0.14	0.11	0.18	0.14
	Zr-Fe-Silicate	0.11	0.09	0.14	0.14
	Apatite	0.46	0.37	0.58	0.37
	Calcite	10.7	8.58	13.8	10.9
	Calcite-Silicate Mixtures	0.97	0.64	1.42	0.48
	Dolomite	6.47	6.88	5.90	7.21
	Ankerite	1.34	1.26	1.44	0.67
	Siderite	0.47	0.53	0.39	0.52
	Quartz	10.3	11.2	8.91	11.5
	Plagioclase	53.5	56.0	50.0	52.9
	Biotite	2.66	1.85	3.80	2.19
	Chlorite	1.66	1.15	2.38	1.38
	Muscovite	0.78	0.46	1.22	0.54
	Amphibole	2.08	1.41	3.01	1.83
	Epidote	0.67	0.28	1.21	0.40
	Magnetite	5.05	6.67	2.80	5.61
	Hematite	0.06	0.05	0.07	0.07
	Goethite	0.99	0.80	1.26	1.11
	Ilmenite	0.24	0.23	0.26	0.23
	Rutile	0.63	0.76	0.44	0.51
	Rutile(Nb)	0.01	0.01	0.01	0.01
	Fluorite	0.02	0.01	0.03	0.03
	Other	0.29	0.28	0.31	0.86
	Total	100.0	100.0	100.0	100.0

Modals Condensed

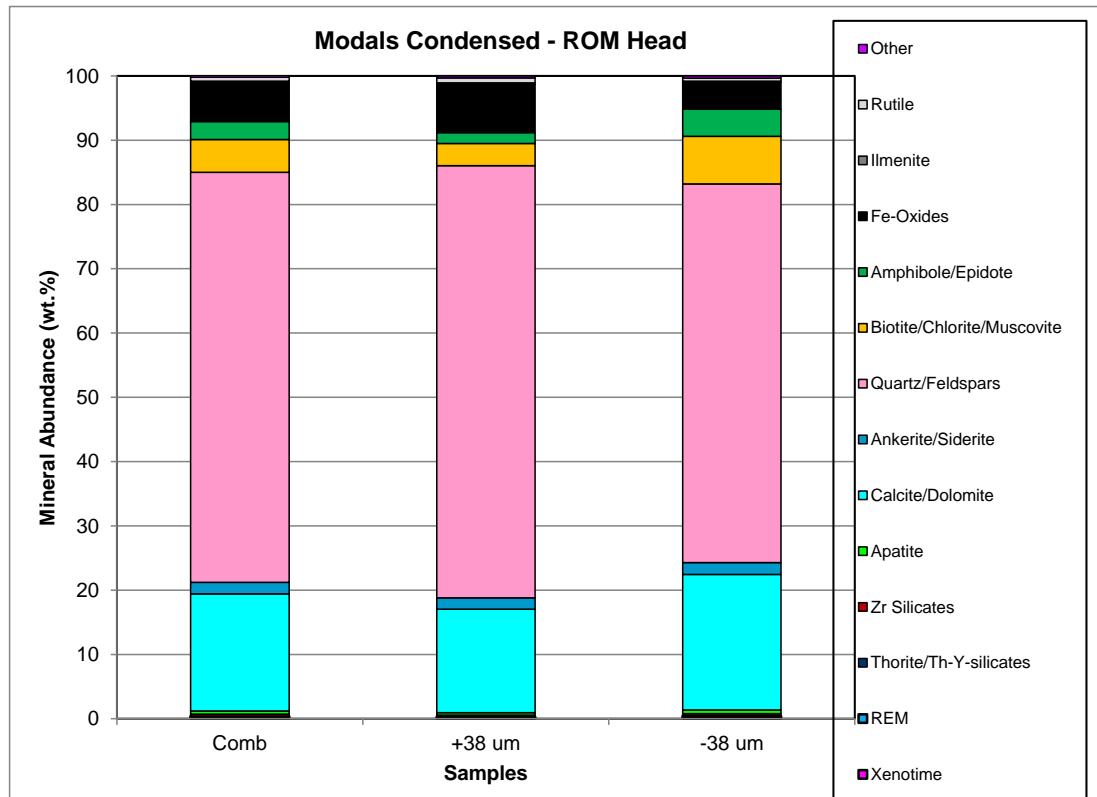
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Project		Namibia			
Sample		ROM Head			LG XRF Comp
Fraction		Comb	+38 um	-38 um	As Rec'd
Mass Size Distribution (%)		100.0	58.2	41.8	100.0
Mineral Mass (%)	Xenotime	0.30	0.22	0.32	0.20
	REM	0.10	0.12	0.13	0.10
	Thorite/Th-Y-silicates	0.00	0.02	0.02	0.00
	Zr Silicates	0.30	0.19	0.32	0.30
	Apatite	0.50	0.37	0.58	0.40
	Calcite/Dolomite	18.2	16.1	21.1	18.6
	Ankerite/Siderite	1.80	1.79	1.83	1.20
	Quartz/Feldspars	63.8	67.2	58.9	64.4
	Biotite/Chlorite/Muscovite	5.10	3.45	7.40	4.10
	Amphibole/Epidote	2.80	1.69	4.22	2.20
	Fe-Oxides	6.10	7.51	4.13	6.80
	Ilmenite	0.20	0.23	0.26	0.20
	Rutile	0.60	0.77	0.45	0.50
	Other	0.30	0.29	0.35	0.90
	Total	100.0	100.0	100.0	100.0

Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

Modal Chart

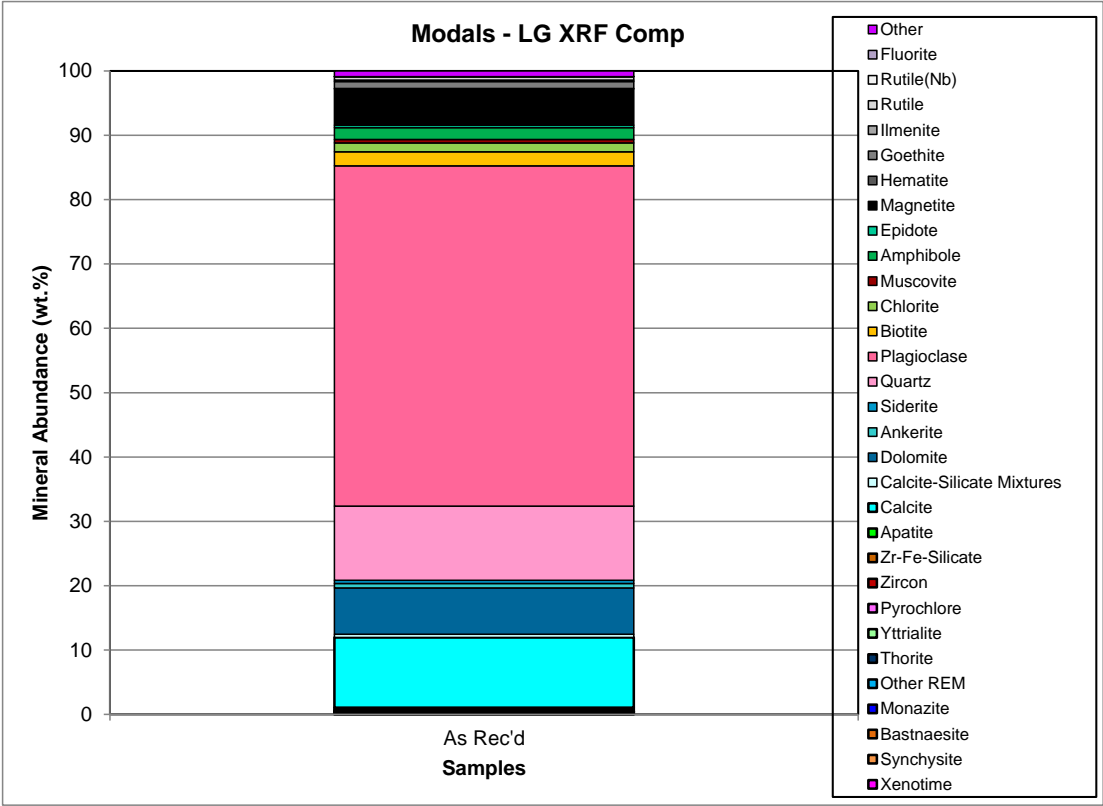


Modal Chart Condensed

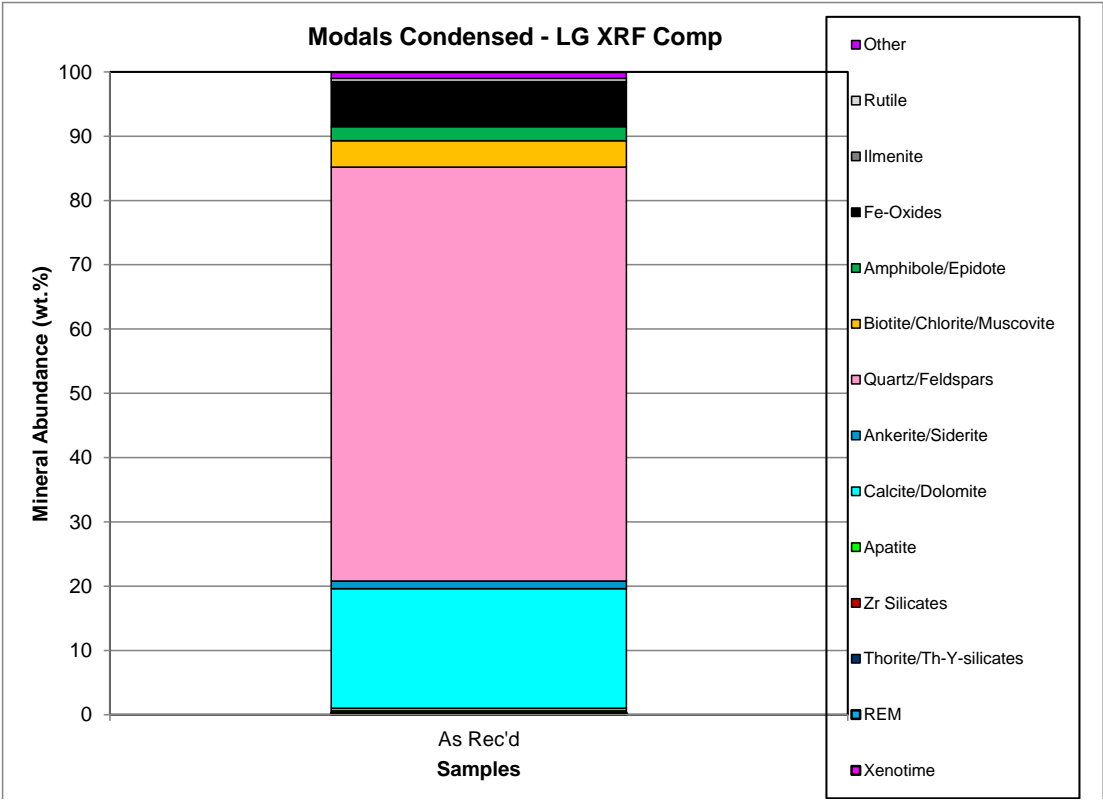


Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

Modal Chart

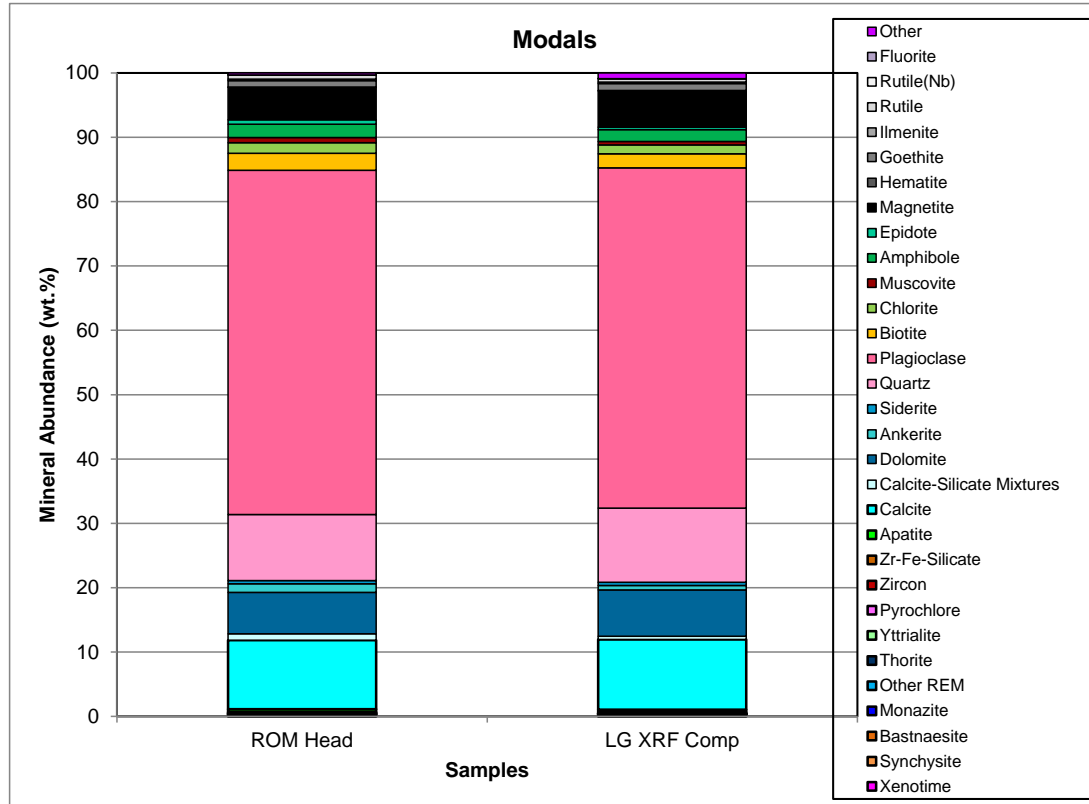


Modal Chart Condensed

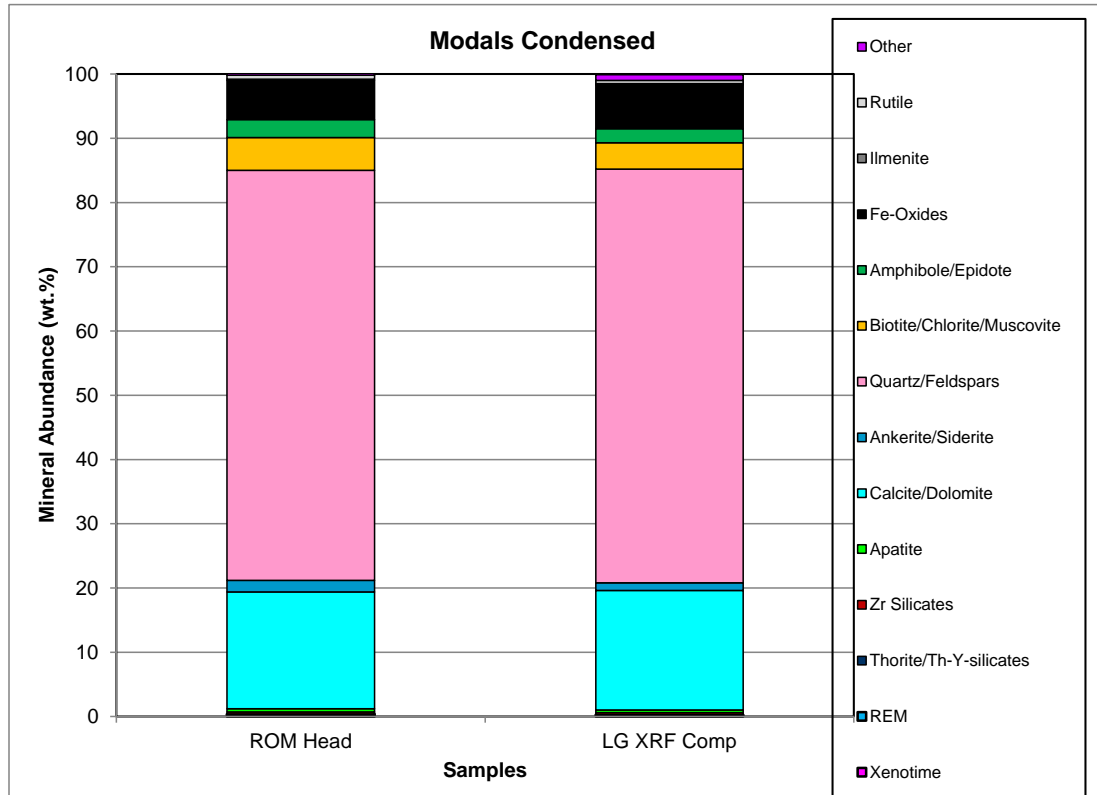


Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

Modal Chart

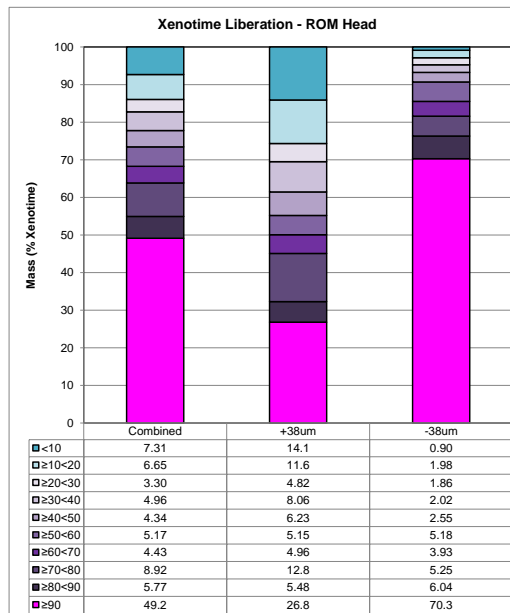
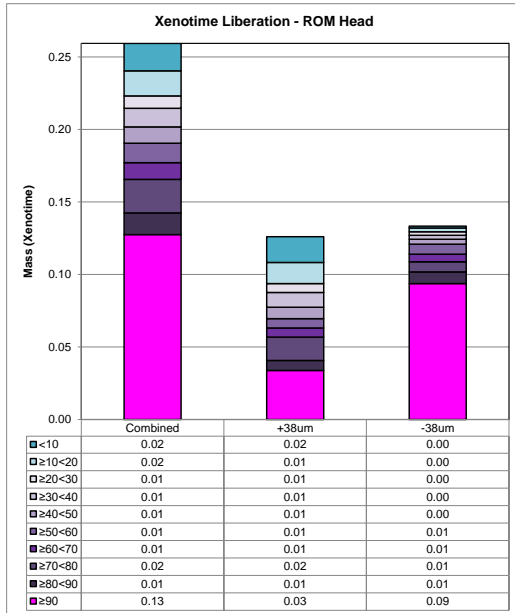


Modal Chart Condensed



Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

Xenotime Liberation

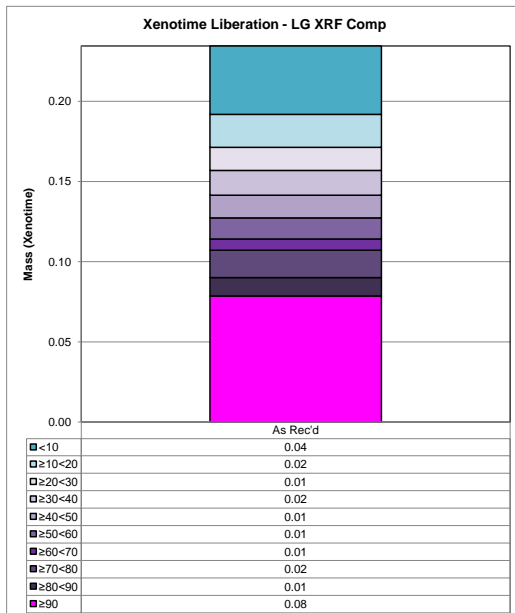


Absolute Mass of Xenotime Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	0.13	0.03	0.09
≥80<90	0.01	0.01	0.01
≥70<80	0.02	0.02	0.01
≥60<70	0.01	0.01	0.01
≥50<60	0.01	0.01	0.01
≥40<50	0.01	0.01	0.00
≥30<40	0.01	0.01	0.00
≥20<30	0.01	0.01	0.00
≥10<20	0.02	0.01	0.00
<10	0.02	0.02	0.00
Total	0.26	0.13	0.13

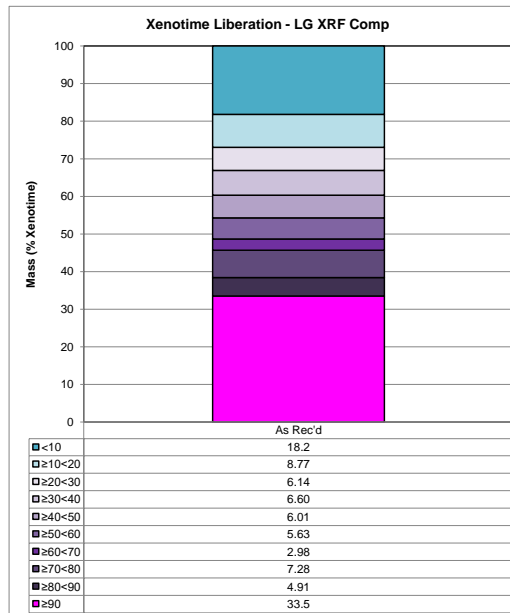
Normalized Mass of Xenotime Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	49.2	26.8	70.3
≥80<90	5.77	5.48	6.04
≥70<80	8.92	12.8	5.25
≥60<70	4.43	4.96	3.93
≥50<60	5.17	5.15	5.18
≥40<50	4.34	6.23	2.55
≥30<40	4.96	8.06	2.02
≥20<30	3.30	4.82	1.86
≥10<20	6.65	11.6	1.98
<10	7.31	14.1	0.90
Total	100.0	100.0	100.0



Absolute Mass of Xenotime Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	0.08
≥80<90	0.01
≥70<80	0.02
≥60<70	0.01
≥50<60	0.01
≥40<50	0.01
≥30<40	0.02
≥20<30	0.01
≥10<20	0.02
<10	0.04
Total	0.23

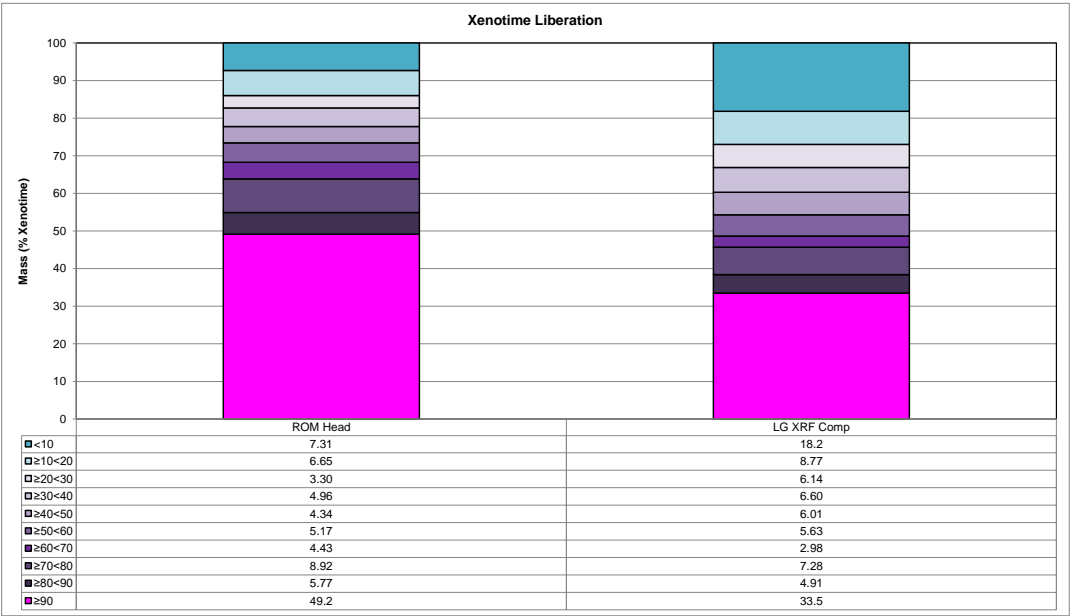


Normalized Mass of Xenotime Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	33.5
≥80<90	4.91
≥70<80	7.28
≥60<70	2.98
≥50<60	5.63
≥40<50	6.01
≥30<40	6.60
≥20<30	6.14
≥10<20	8.77
<10	18.2
Total	100.0

Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

Xenotime Liberation

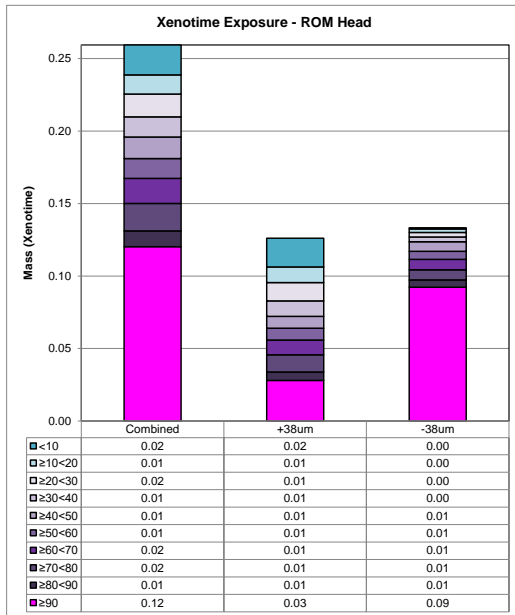


Normalized Mass of Xenotime Across Samples

Mineral Name	ROM Head	LG XRF Comp
≥90	49.2	33.5
≥80<90	5.77	4.91
≥70<80	8.92	7.28
≥60<70	4.43	2.98
≥50<60	5.17	5.63
≥40<50	4.34	6.01
≥30<40	4.96	6.60
≥20<30	3.30	6.14
≥10<20	6.65	8.77
<10	7.31	18.2
Total	100.0	100.0

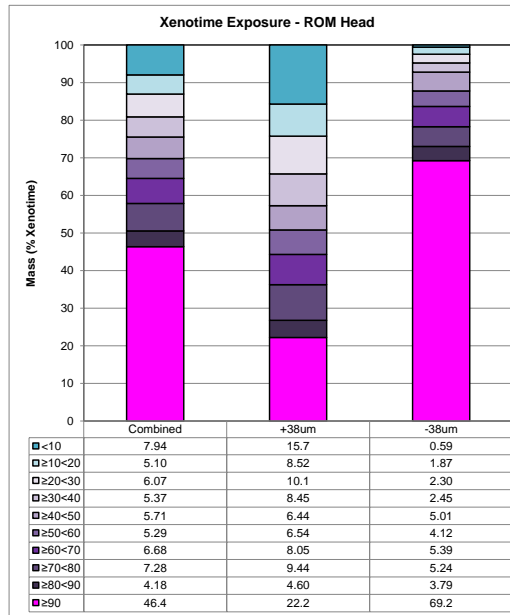
Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

Xenotime Exposure



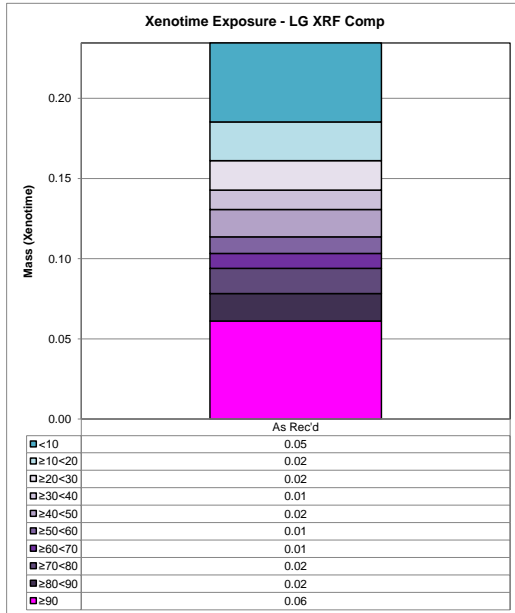
Absolute Mass of Xenotime Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	0.12	0.03	0.09
≥80<90	0.01	0.01	0.01
≥70<80	0.02	0.01	0.01
≥60<70	0.02	0.01	0.01
≥50<60	0.01	0.01	0.01
≥40<50	0.01	0.01	0.01
≥30<40	0.01	0.01	0.00
≥20<30	0.02	0.01	0.00
≥10<20	0.01	0.01	0.00
<10	0.02	0.02	0.00
Total	0.26	0.13	0.13



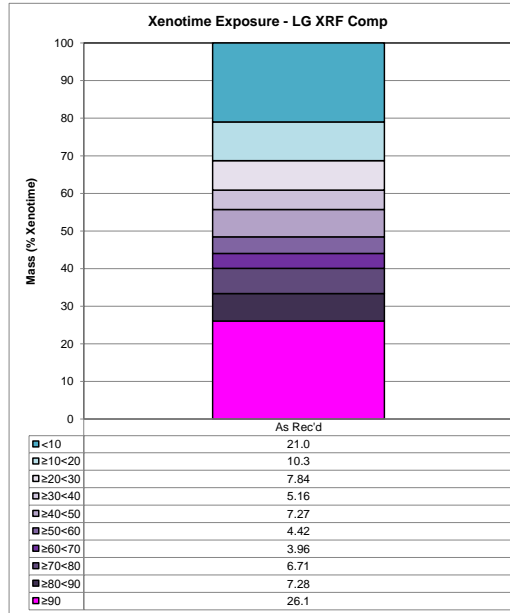
Normalized Mass of Xenotime Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	46.4	22.2	69.2
≥80<90	4.18	4.60	3.79
≥70<80	7.28	9.44	5.24
≥60<70	6.68	8.05	5.39
≥50<60	5.29	6.54	4.12
≥40<50	5.71	6.44	5.01
≥30<40	5.37	8.45	2.45
≥20<30	6.07	10.1	2.30
≥10<20	5.10	8.52	1.87
<10	7.94	15.7	0.59
Total	100.0	100.0	100.0



Absolute Mass of Xenotime Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	0.06
≥80<90	0.02
≥70<80	0.02
≥60<70	0.01
≥50<60	0.01
≥40<50	0.02
≥30<40	0.01
≥20<30	0.02
≥10<20	0.02
<10	0.05
Total	0.23

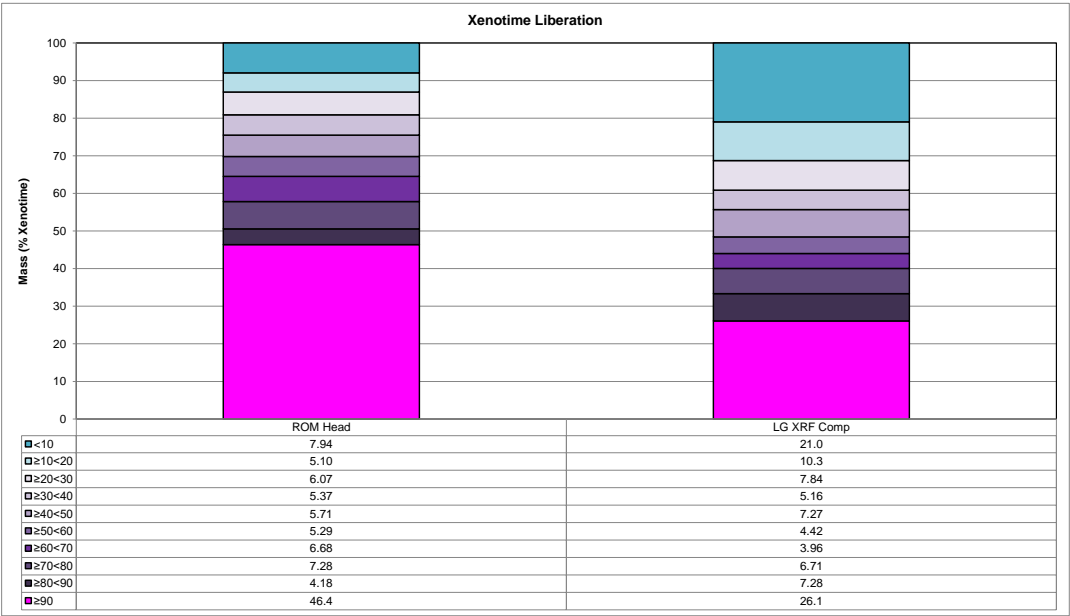


Normalized Mass of Xenotime Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	26.1
≥80<90	7.28
≥70<80	6.71
≥60<70	3.96
≥50<60	4.42
≥40<50	7.27
≥30<40	5.16
≥20<30	7.84
≥10<20	10.3
<10	21.0
Total	100.0

Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

Xenotime Liberation

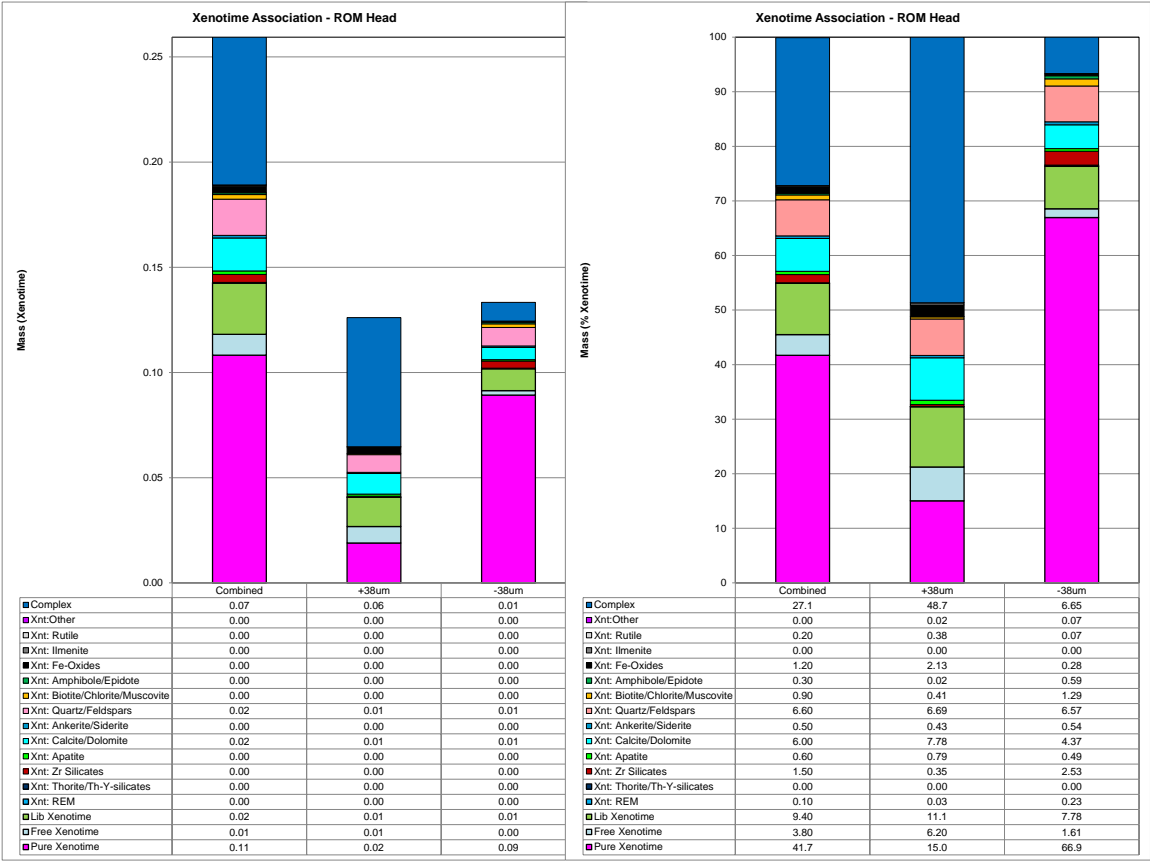


Normalized Mass of Xenotime Across Samples

Mineral Name	ROM Head	LG XRF Comp
≥90	46.4	26.1
≥80<90	4.18	7.28
≥70<80	7.28	6.71
≥60<70	6.68	3.96
≥50<60	5.29	4.42
≥40<50	5.71	7.27
≥30<40	5.37	5.16
≥20<30	6.07	7.84
≥10<20	5.10	10.3
<10	7.94	21.0
Total	100.0	100.0

Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

Xenotime Association



Absolute Mass of Xenotime Across Sample ROM Head

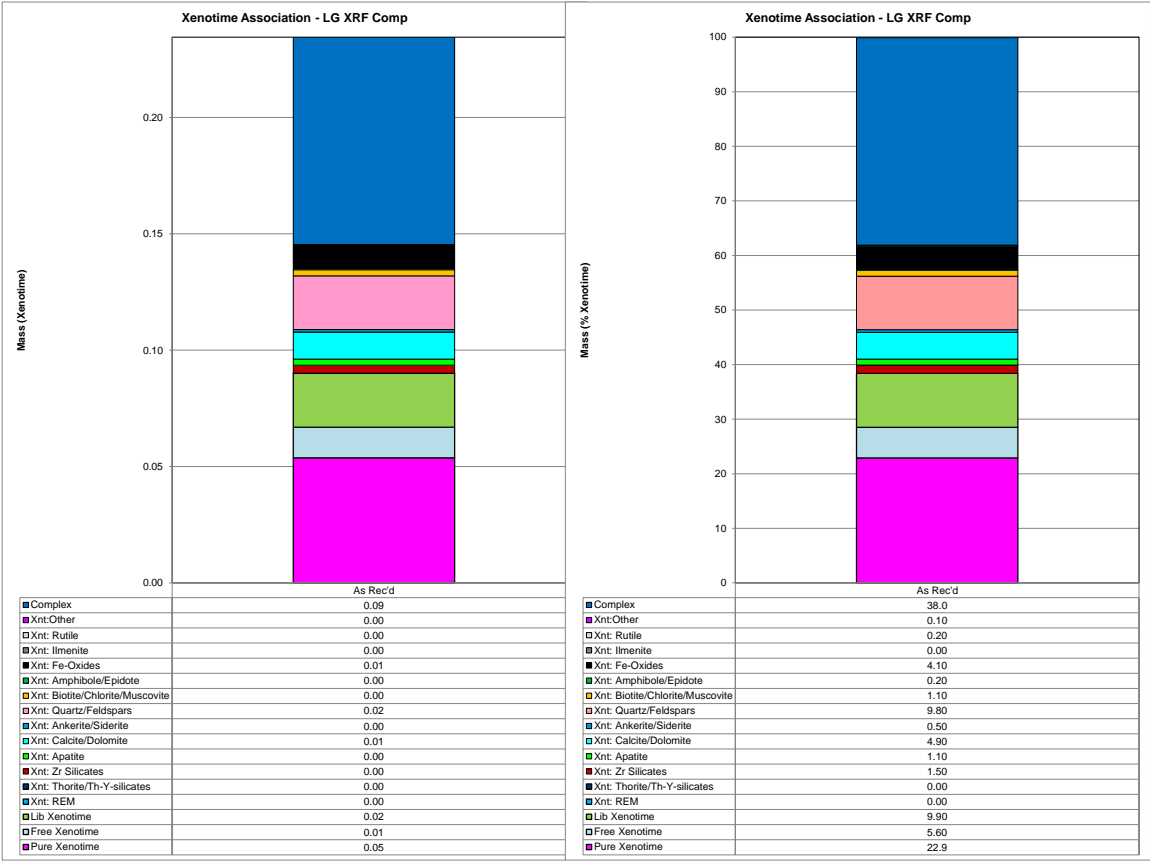
Mineral Name	Combined	+38um	-38um
Pure Xenotime	0.11	0.02	0.09
Free Xenotime	0.01	0.01	0.00
Lib Xenotime	0.02	0.01	0.01
Xnt: REM	0.00	0.00	0.00
Xnt: Thorite/Th-Y-silicates	0.00	0.00	0.00
Xnt: Zr Silicates	0.00	0.00	0.00
Xnt: Apatite	0.00	0.00	0.00
Xnt: Calcite/Dolomite	0.02	0.01	0.01
Xnt: Ankerite/Siderite	0.00	0.00	0.00
Xnt: Quartz/Feldspars	0.02	0.01	0.01
Xnt: Biotite/Chlorite/Muscovite	0.00	0.00	0.00
Xnt: Amphibole/Epidote	0.00	0.00	0.00
Xnt: Fe-Oxides	0.00	0.00	0.00
Xnt: Ilmenite	0.00	0.00	0.00
Xnt: Rutile	0.00	0.00	0.00
Xnt:Other	0.00	0.00	0.00
Complex	0.07	0.06	0.01
Total	0.26	0.13	0.13

Normalized Mass of Xenotime Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
Pure Xenotime	41.7	15.0	66.9
Free Xenotime	3.80	6.20	1.61
Lib Xenotime	9.40	11.1	7.78
Xnt: REM	0.10	0.03	0.23
Xnt: Thorite/Th-Y-silicates	0.00	0.00	0.00
Xnt: Zr Silicates	1.50	0.35	2.53
Xnt: Apatite	0.60	0.79	0.49
Xnt: Calcite/Dolomite	6.00	7.78	4.37
Xnt: Ankerite/Siderite	0.50	0.43	0.54
Xnt: Quartz/Feldspars	6.60	6.69	6.57
Xnt: Biotite/Chlorite/Muscovite	0.90	0.41	1.29
Xnt: Amphibole/Epidote	0.30	0.02	0.59
Xnt: Fe-Oxides	1.20	2.13	0.28
Xnt: Ilmenite	0.00	0.00	0.00
Xnt: Rutile	0.20	0.38	0.07
Xnt:Other	0.00	0.02	0.07
Complex	27.1	48.7	6.65
Total	100.0	100.0	100.0
Liberated	54.9	32.3	76.3

Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

Xenotime Association



Absolute Mass of Xenotime Across Sample LG XRF Comp

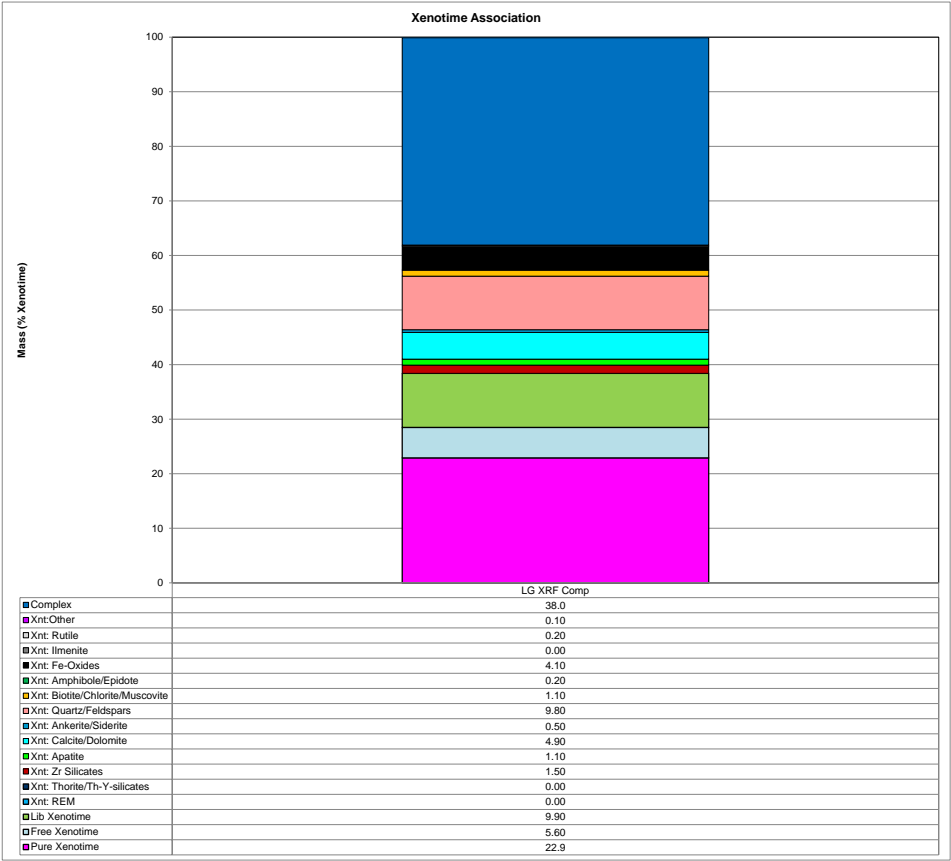
Mineral Name	As Rec'd
Pure Xenotime	0.05
Free Xenotime	0.01
Lib Xenotime	0.02
Xnt: REM	0.00
Xnt: Thorite/Th-Y-silicates	0.00
Xnt: Zr Silicates	0.00
Xnt: Apatite	0.00
Xnt: Calcite/Dolomite	0.01
Xnt: Ankerite/Siderite	0.00
Xnt: Quartz/Feldspars	0.02
Xnt: Biotite/Chlorite/Muscovite	0.00
Xnt: Amphibole/Epidote	0.00
Xnt: Fe-Oxides	0.01
Xnt: Ilmenite	0.00
Xnt: Rutile	0.00
Xnt:Other	0.00
Complex	0.09
Total	0.23

Normalized Mass of Xenotime Across Sample LG XRF Comp

Mineral Name	As Rec'd
Pure Xenotime	22.9
Free Xenotime	5.60
Lib Xenotime	9.90
Xnt: REM	0.00
Xnt: Thorite/Th-Y-silicates	0.00
Xnt: Zr Silicates	1.50
Xnt: Apatite	1.10
Xnt: Calcite/Dolomite	4.90
Xnt: Ankerite/Siderite	0.50
Xnt: Quartz/Feldspars	9.80
Xnt: Biotite/Chlorite/Muscovite	1.10
Xnt: Amphibole/Epidote	0.20
Xnt: Fe-Oxides	4.10
Xnt: Ilmenite	0.00
Xnt: Rutile	0.20
Xnt:Other	0.10
Complex	38.0
Total	100.0
Liberated	38.4

Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

Xenotime Association

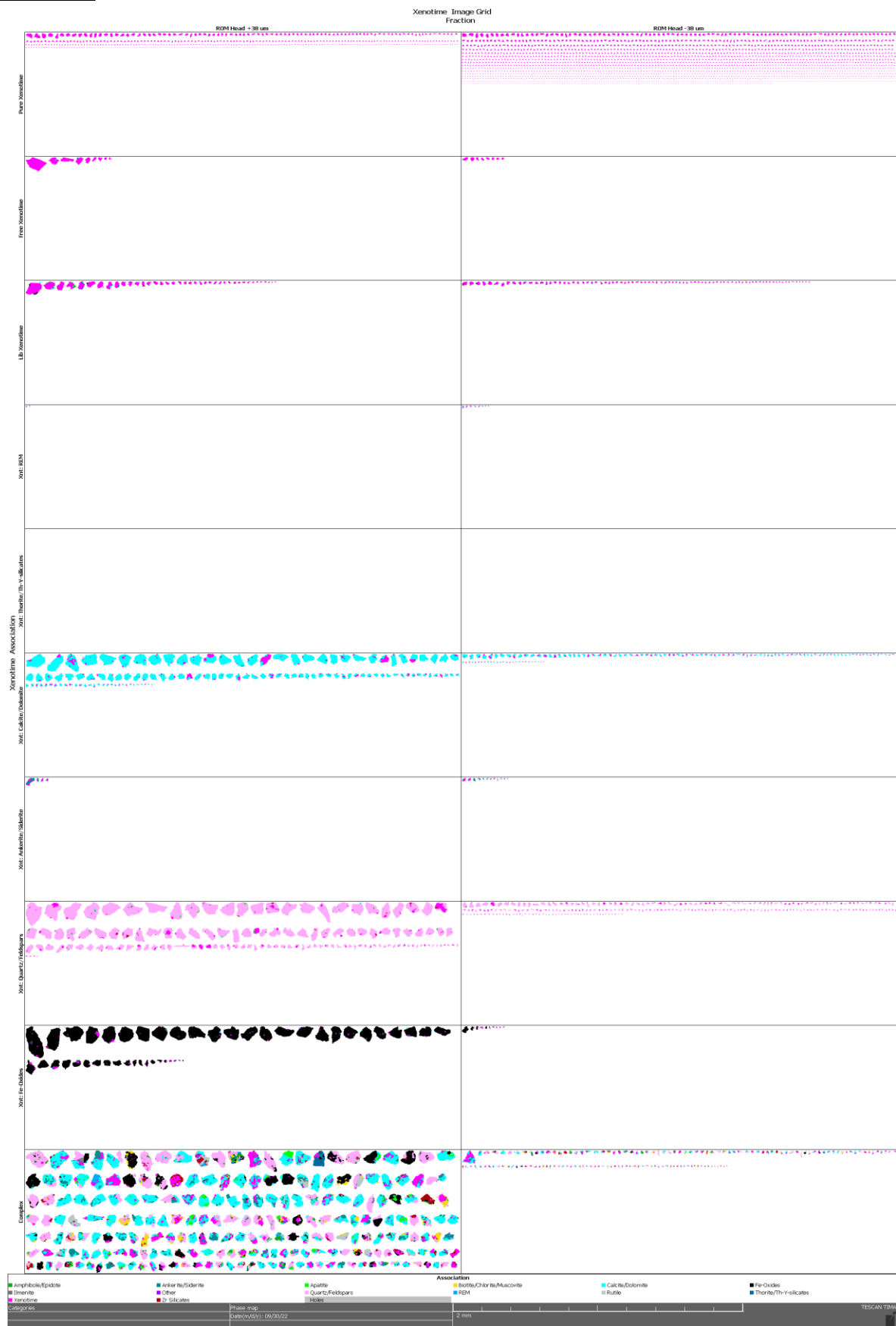


Normalized Mass of Xenotime Across Samples

Mineral Name	LG XRF Comp	ROM Head Comb	ROM +38 um	ROM -38 um
Pure Xenotime	22.9	41.7	15.0	66.9
Free Xenotime	5.60	3.80	6.20	1.61
Lib Xenotime	9.90	9.40	11.1	7.78
Xnt: REM	0.00	0.10	0.03	0.23
Xnt: Thorite/Th-Y-silicates	0.00	0.00	0.00	0.00
Xnt: Zr Silicates	1.50	1.50	0.35	2.53
Xnt: Apatite	1.10	0.60	0.79	0.49
Xnt: Calcite/Dolomite	4.90	6.00	7.78	4.37
Xnt: Ankerite/Siderite	0.50	0.50	0.43	0.54
Xnt: Quartz/Feldspars	9.80	6.60	6.69	6.57
Xnt: Biotite/Chlorite/Muscovite	1.10	0.90	0.41	1.29
Xnt: Amphibole/Epidote	0.20	0.30	0.02	0.59
Xnt: Fe-Oxides	4.10	1.20	2.13	0.28
Xnt: Ilmenite	0.00	0.00	0.00	0.00
Xnt: Rutile	0.20	0.20	0.38	0.07
Xnt: Other	0.10	0.00	0.02	0.07
Complex	38.0	27.1	48.7	6.6
Total	100.0	100.0	100.0	100.0
Liberated	38.4	54.9	32.3	76.3

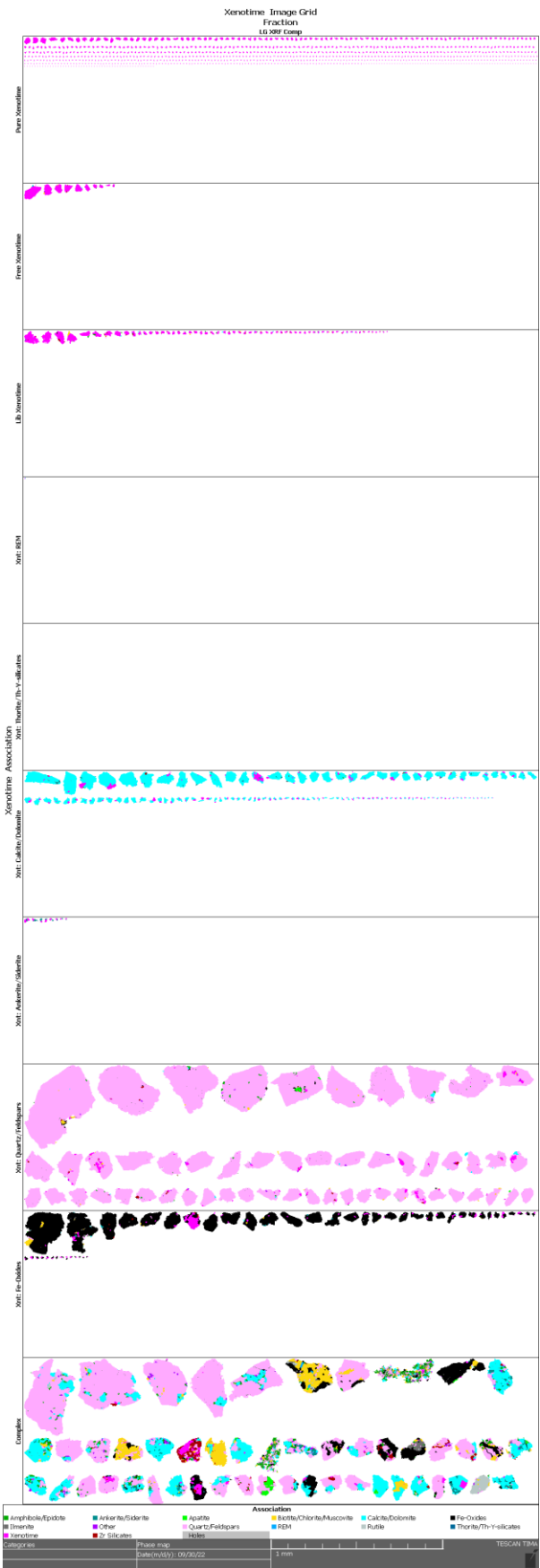
Mineral Name	LG XRF Comp	ROM Head Comb	ROM +38 um	ROM -38 um
Pure Xenotime	22.9	41.7	15.0	66.9
Free Xenotime	5.60	3.80	6.20	1.61
Lib Xenotime	9.90	9.40	11.1	7.78
Xnt: REM	0.00	0.10	0.03	0.23
Xnt: Zr Silicates	1.50	1.50	0.35	2.53
Xnt: Apatite	1.10	0.60	0.79	0.49
Xnt: Calcite/Dolomite	4.90	6.00	7.78	4.37
Xnt: Ankerite/Siderite	0.50	0.50	0.43	0.54
Xnt: Quartz/Feldspars	9.80	6.60	6.69	6.57
Xnt: Biotite/Chlorite/Muscovite	1.10	0.90	0.41	1.29
Xnt: Fe-Oxides	4.10	1.20	2.13	0.28
Xnt: Other	0.50	0.20	0.40	0.14
Complex	38.0	27.1	48.7	6.6
Total	100.0	100.0	100.0	100.0
Pure+Free + Liberated	38.4	54.9	32.3	76.3

Image Grid of Xenotime Association



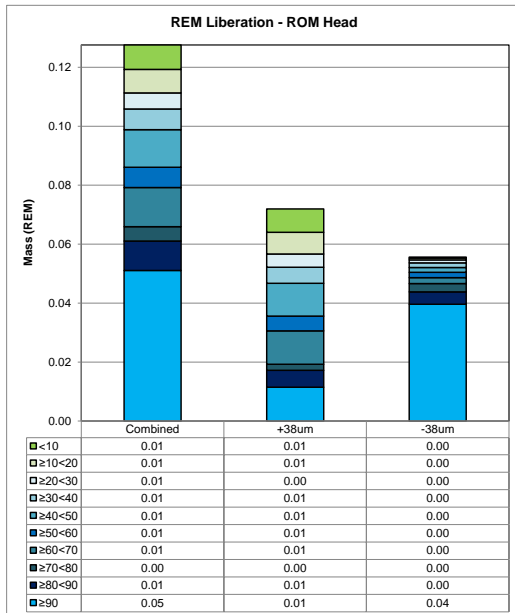
Nambia
CALR-18239-01/03
M65029-APR22 & M6509-MAY22

Image Grid of Xenotime Association



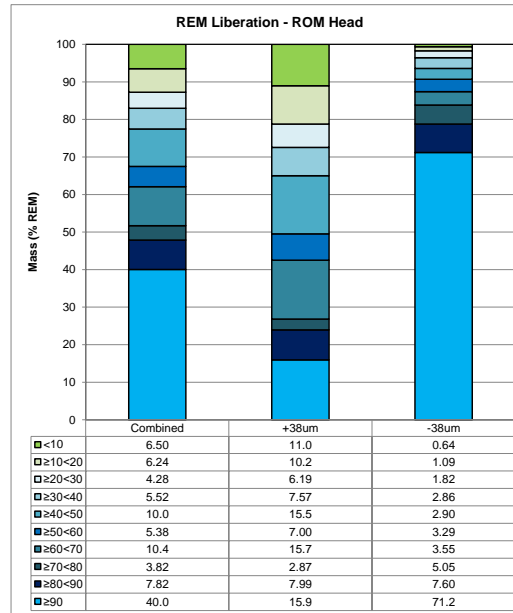
Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

REM Liberation



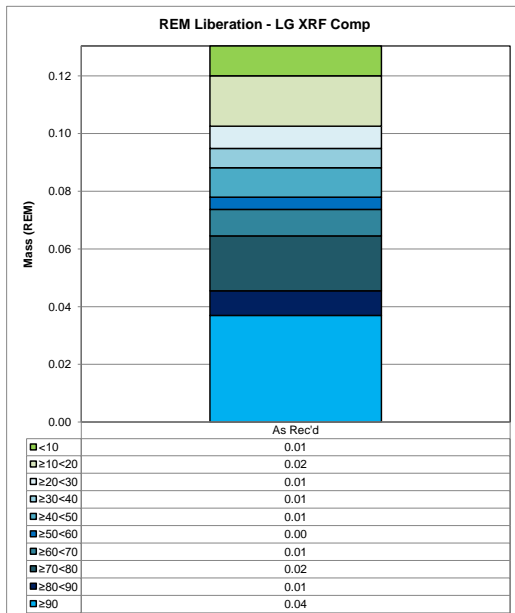
Absolute Mass of REM Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	0.05	0.01	0.04
≥80<90	0.01	0.01	0.00
≥70<80	0.00	0.00	0.00
≥60<70	0.01	0.01	0.00
≥50<60	0.01	0.01	0.00
≥40<50	0.01	0.01	0.00
≥30<40	0.01	0.01	0.00
≥20<30	0.01	0.00	0.00
≥10<20	0.01	0.01	0.00
<10	0.01	0.01	0.00
Total	0.13	0.07	0.06



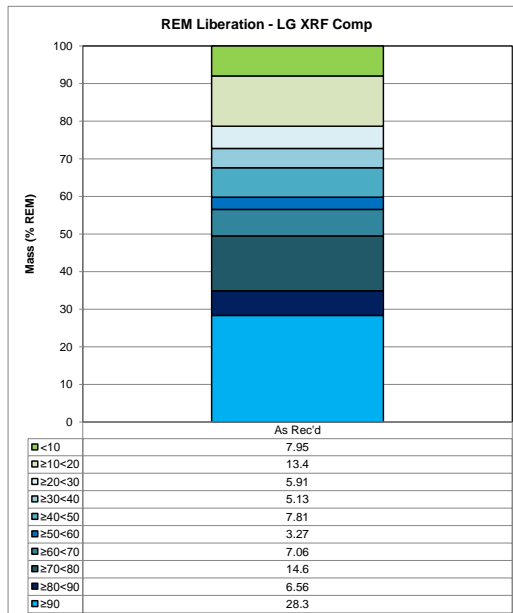
Normalized Mass of REM Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	40.0	15.9	71.2
≥80<90	7.82	7.99	7.60
≥70<80	3.82	2.87	5.05
≥60<70	10.4	15.7	3.55
≥50<60	5.38	7.00	3.29
≥40<50	10.0	15.5	2.90
≥30<40	5.52	7.57	2.86
≥20<30	4.28	6.19	1.82
≥10<20	6.24	10.2	1.09
<10	6.50	11.0	0.64
Total	100.0	100.0	100.0



Absolute Mass of REM Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	0.04
≥80<90	0.01
≥70<80	0.02
≥60<70	0.01
≥50<60	0.00
≥40<50	0.01
≥30<40	0.01
≥20<30	0.01
≥10<20	0.02
<10	0.01
Total	0.13

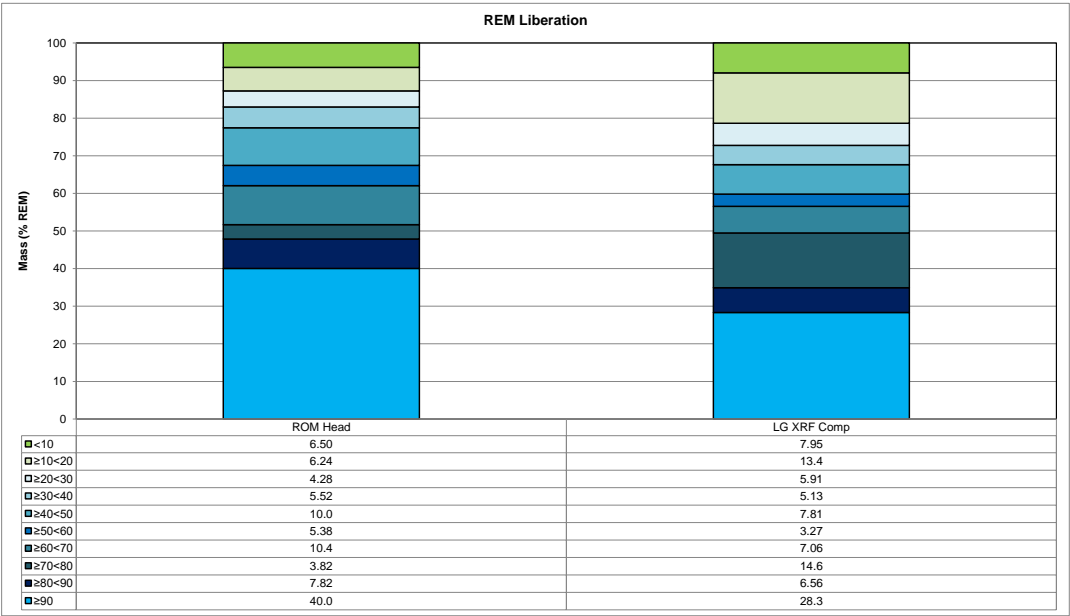


Normalized Mass of REM Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	28.3
≥80<90	6.56
≥70<80	14.6
≥60<70	7.06
≥50<60	3.27
≥40<50	7.81
≥30<40	5.13
≥20<30	5.91
≥10<20	13.4
<10	7.95
Total	100.0

Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

REM Liberation

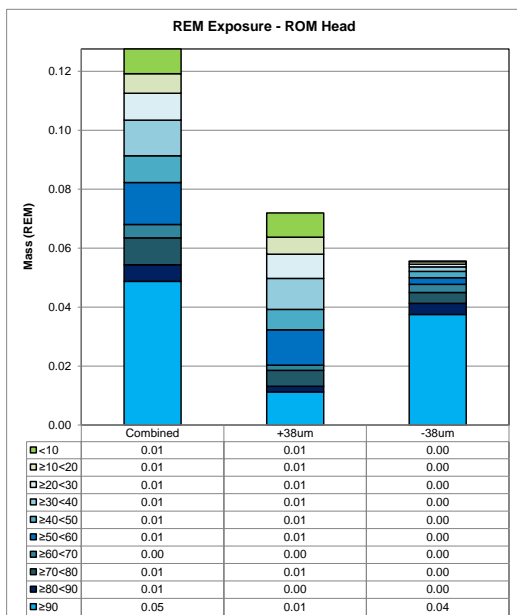


Normalized Mass of REM Across Samples

Mineral Name	ROM Head	LG XRF Comp
≥90	40.0	28.3
≥80<90	7.82	6.56
≥70<80	3.82	14.6
≥60<70	10.4	7.06
≥50<60	5.38	3.27
≥40<50	10.0	7.81
≥30<40	5.52	5.13
≥20<30	4.28	5.91
≥10<20	6.24	13.4
<10	6.50	7.95
Total	100.0	100.0

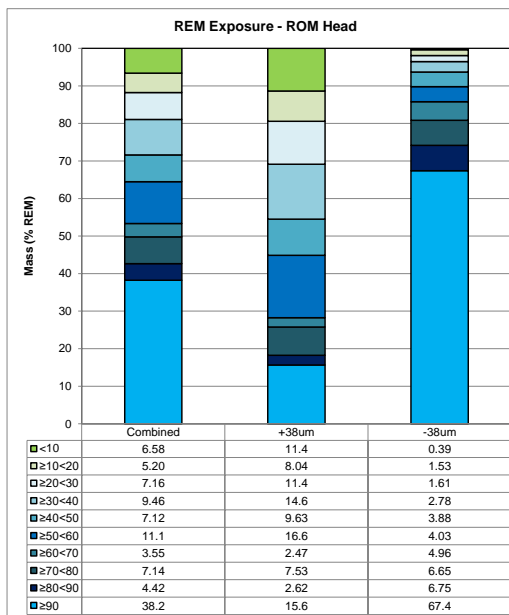
Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

REM Exposure



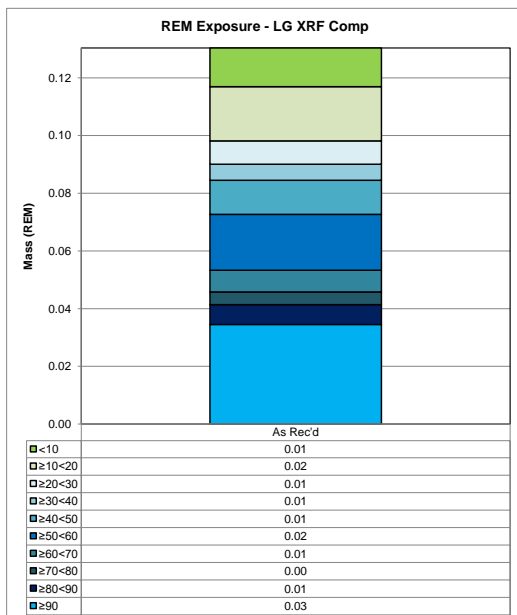
Absolute Mass of REM Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	0.05	0.01	0.04
≥80<90	0.01	0.00	0.00
≥70<80	0.01	0.01	0.00
≥60<70	0.00	0.00	0.00
≥50<60	0.01	0.01	0.00
≥40<50	0.01	0.01	0.00
≥30<40	0.01	0.01	0.00
≥20<30	0.01	0.01	0.00
≥10<20	0.01	0.01	0.00
<10	0.01	0.01	0.00
Total	0.13	0.07	0.06



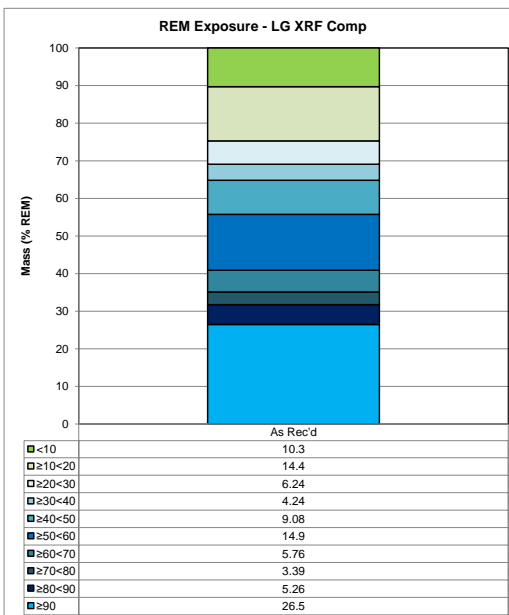
Normalized Mass of REM Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	38.2	15.6	67.4
≥80<90	4.42	2.62	6.75
≥70<80	7.14	7.53	6.65
≥60<70	3.55	2.47	4.96
≥50<60	11.1	16.6	4.03
≥40<50	7.12	9.63	3.88
≥30<40	9.46	14.6	2.78
≥20<30	7.16	11.4	1.61
≥10<20	5.20	8.04	1.53
<10	6.58	11.4	0.39
Total	100.0	100.0	100.0



Absolute Mass of REM Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	0.03
≥80<90	0.01
≥70<80	0.00
≥60<70	0.01
≥50<60	0.02
≥40<50	0.01
≥30<40	0.01
≥20<30	0.01
≥10<20	0.02
<10	0.01
Total	0.13

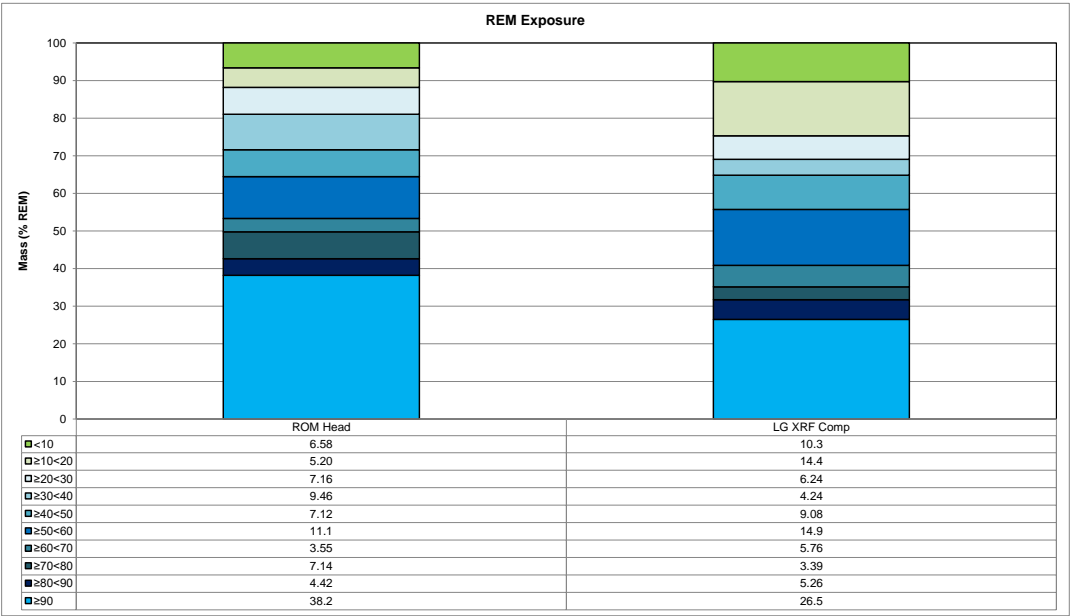


Normalized Mass of REM Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	26.5
≥80<90	5.26
≥70<80	3.39
≥60<70	5.76
≥50<60	14.9
≥40<50	9.08
≥30<40	4.24
≥20<30	6.24
≥10<20	14.4
<10	10.3
Total	100.0

Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

REM Exposure

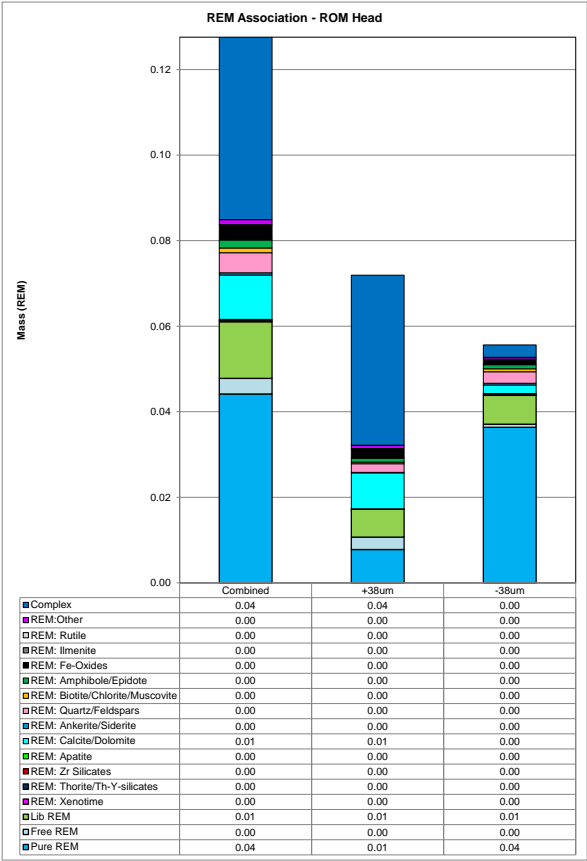


Normalized Mass of REM Across Samples

Mineral Name	ROM Head	LG XRF Comp
≥90	38.2	26.5
≥80<90	4.42	5.26
≥70<80	7.14	3.39
≥60<70	3.55	5.76
≥50<60	11.1	14.9
≥40<50	7.12	9.08
≥30<40	9.46	4.24
≥20<30	7.16	6.24
≥10<20	5.20	14.4
<10	6.58	10.3
Total	100.0	100.0

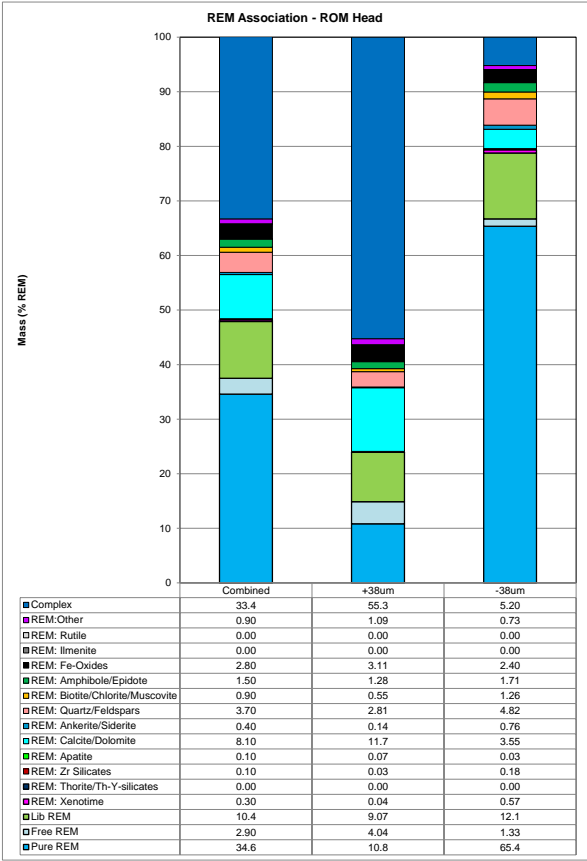
Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

REM Association



Absolute Mass of REM Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
Pure REM	0.04	0.01	0.04
Free REM	0.00	0.00	0.00
Lib REM	0.01	0.01	0.01
REM: Xenotime	0.00	0.00	0.00
REM: Thorite/Th-Y-silicates	0.00	0.00	0.00
REM: Zr Silicates	0.00	0.00	0.00
REM: Apatite	0.00	0.00	0.00
REM: Calcite/Dolomite	0.01	0.01	0.00
REM: Ankerite/Siderite	0.00	0.00	0.00
REM: Quartz/Feldspars	0.00	0.00	0.00
REM: Biotite/Chlorite/Muscovite	0.00	0.00	0.00
REM: Amphibole/Epidote	0.00	0.00	0.00
REM: Fe-Oxides	0.00	0.00	0.00
REM: Ilmenite	0.00	0.00	0.00
REM: Rutile	0.00	0.00	0.00
REM:Other	0.00	0.00	0.00
Complex	0.04	0.04	0.00
Total	0.13	0.07	0.06



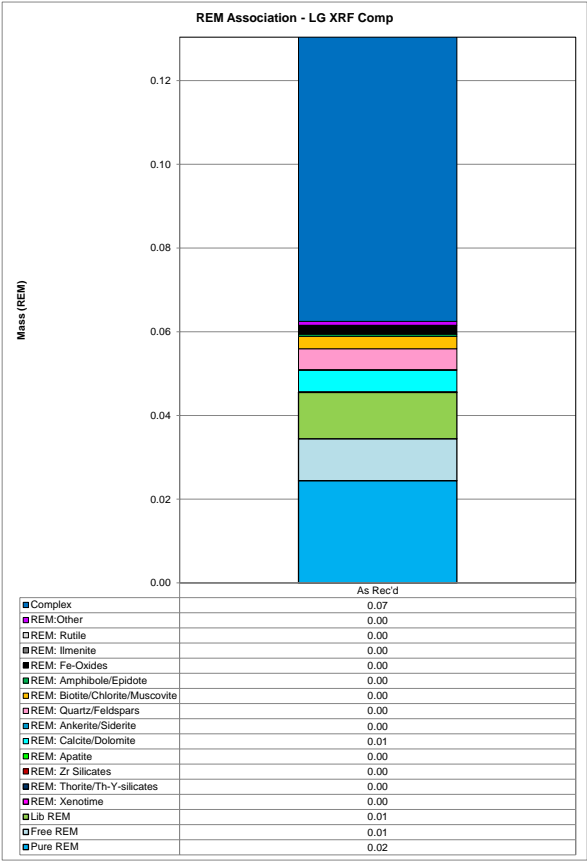
Normalized Mass of REM Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
Pure REM	34.6	10.8	65.4
Free REM	2.90	4.04	1.33
Lib REM	10.4	9.07	12.1
REM: Xenotime	0.30	0.04	0.57
REM: Thorite/Th-Y-silicates	0.00	0.00	0.00
REM: Zr Silicates	0.10	0.03	0.18
REM: Apatite	0.10	0.07	0.03
REM: Calcite/Dolomite	8.10	11.7	3.55
REM: Ankerite/Siderite	0.40	0.14	0.76
REM: Quartz/Feldspars	3.70	2.81	4.82
REM: Biotite/Chlorite/Muscovite	0.90	0.55	1.26
REM: Amphibole/Epidote	1.50	1.28	1.71
REM: Fe-Oxides	2.80	3.11	2.40
REM: Ilmenite	0.00	0.00	0.00
REM: Rutile	0.00	0.00	0.00
REM:Other	0.90	1.09	0.73
Complex	33.4	55.3	5.20
Total	100.0	100.0	100.0

Liberated 47.9 23.9 78.8

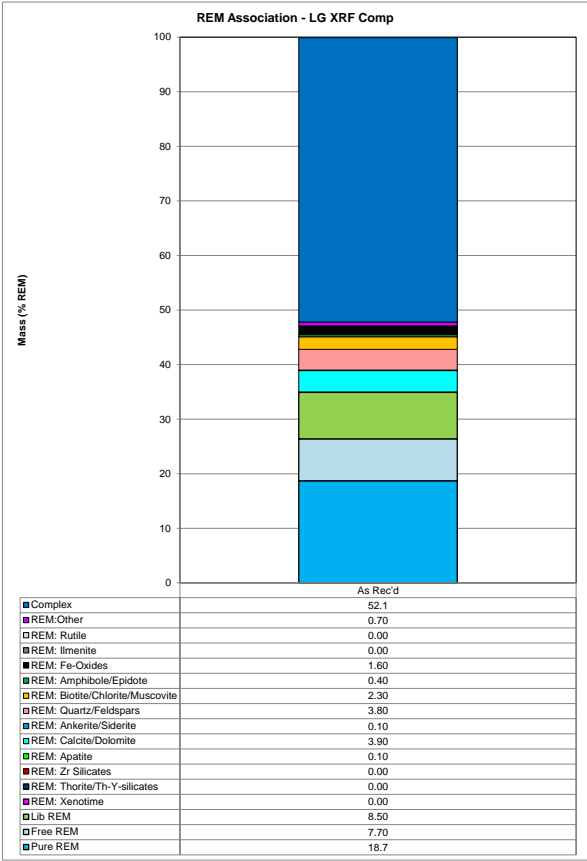
Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

REM Association



Absolute Mass of REM Across Sample LG XRF Comp

Mineral Name	As Rec'd
Pure REM	0.02
Free REM	0.01
Lib REM	0.01
REM: Xenotime	0.00
REM: Thorite/Th-Y-silicates	0.00
REM: Zr Silicates	0.00
REM: Apatite	0.00
REM: Calcite/Dolomite	0.01
REM: Ankerite/Siderite	0.00
REM: Quartz/Feldspars	0.00
REM: Biotite/Chlorite/Muscovite	0.00
REM: Amphibole/Epidote	0.00
REM: Fe-Oxides	0.00
REM: Ilmenite	0.00
REM: Rutile	0.00
REM:Other	0.00
Complex	0.07
Total	0.13



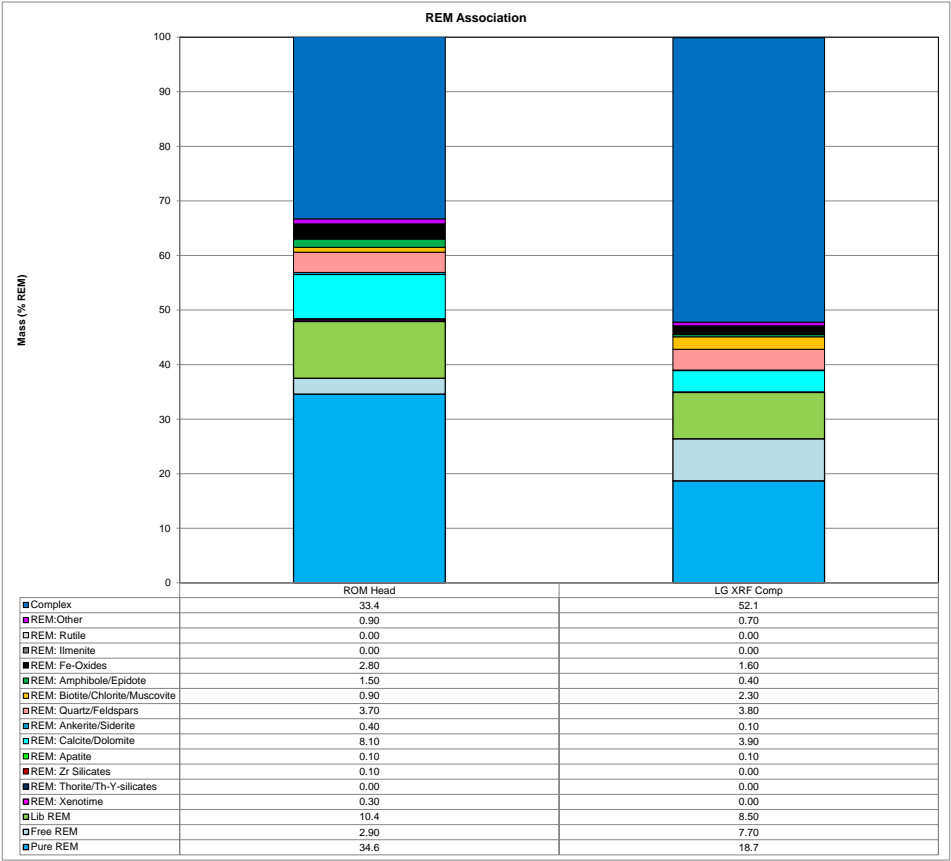
Normalized Mass of REM Across Sample LG XRF Comp

Mineral Name	As Rec'd
Pure REM	18.7
Free REM	7.70
Lib REM	8.50
REM: Xenotime	0.00
REM: Thorite/Th-Y-silicates	0.00
REM: Zr Silicates	0.00
REM: Apatite	0.10
REM: Calcite/Dolomite	3.90
REM: Ankerite/Siderite	0.10
REM: Quartz/Feldspars	3.80
REM: Biotite/Chlorite/Muscovite	2.30
REM: Amphibole/Epidote	0.40
REM: Fe-Oxides	1.60
REM: Ilmenite	0.00
REM: Rutile	0.00
REM:Other	0.70
Complex	52.1
Total	100.0

Liberated 34.9

Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

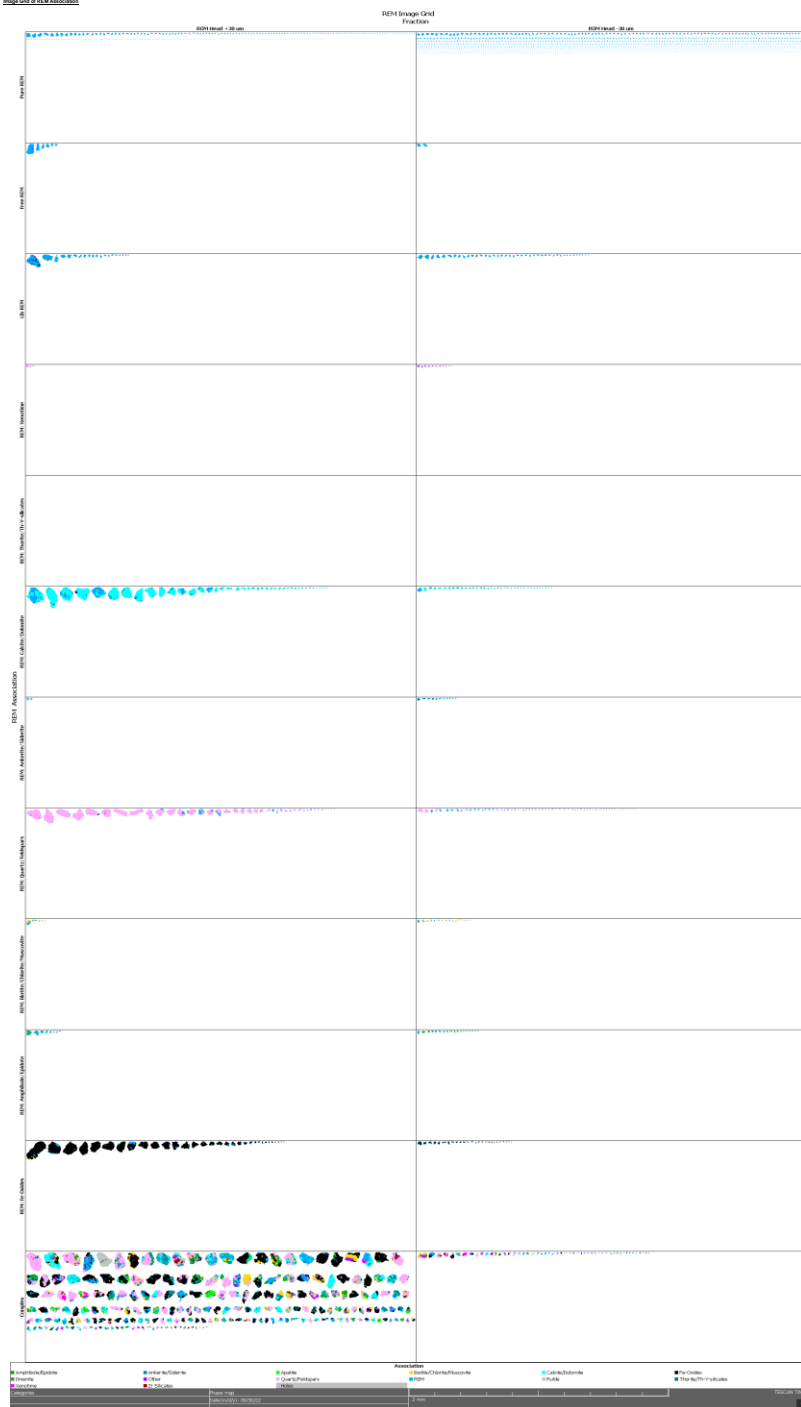
REM Association



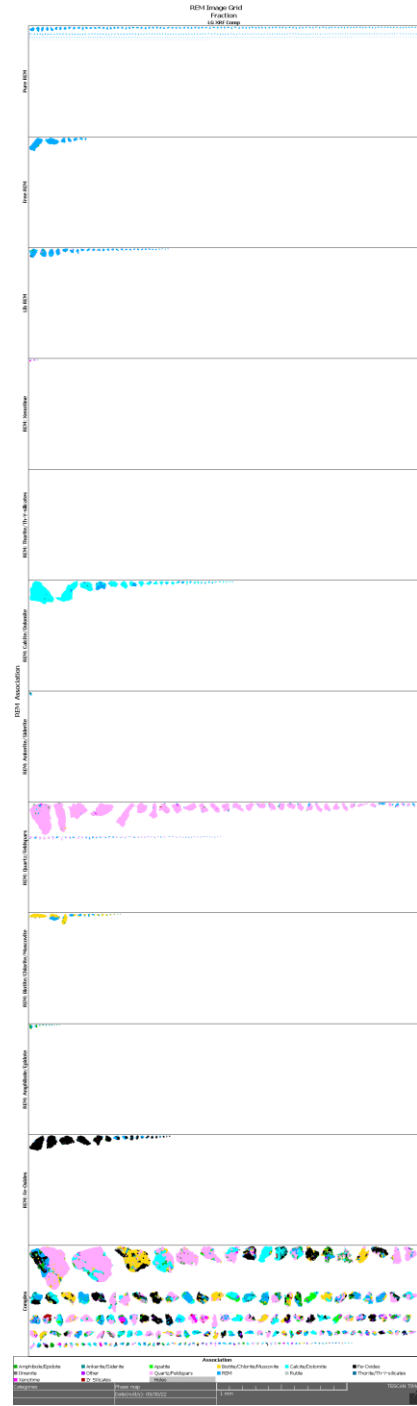
Normalized Mass of REM Across Samples

Mineral Name	ROM Head	LG XRF Comp
Pure REM	34.6	18.7
Free REM	2.90	7.70
Lib REM	10.4	8.50
REM: Xenotime	0.30	0.00
REM: Thorite/Th-Y-silicates	0.00	0.00
REM: Zr Silicates	0.10	0.00
REM: Apatite	0.10	0.10
REM: Calcite/Dolomite	8.10	3.90
REM: Ankerite/Siderite	0.40	0.10
REM: Quartz/Feldspars	3.70	3.80
REM: Biotite/Chlorite/Muscovite	0.90	2.30
REM: Amphibole/Epidote	1.50	0.40
REM: Fe-Oxides	2.80	1.60
REM: Ilmenite	0.00	0.00
REM: Rutile	0.00	0.00
REM: Other	0.90	0.70
Complex	33.4	52.1
Total	100.0	100.0
Liberated	47.9	34.9

Namibia
CALR-18299-01/03
ME029-APR02 & ME009-MAY22

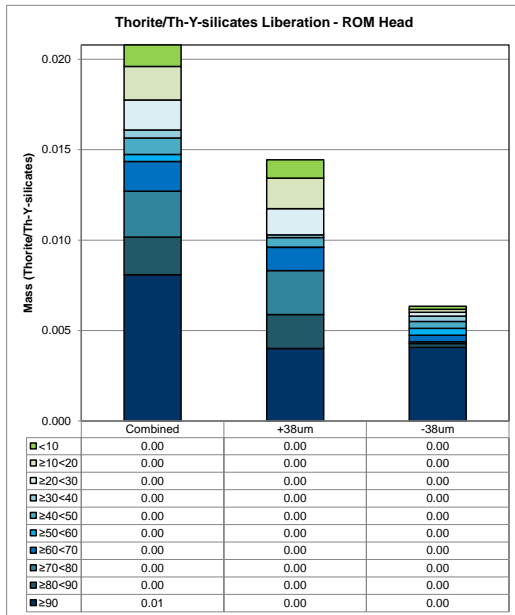


Barcode
CALR-1029-1-03
MSL2D-AP022 & MSL2D-AM022



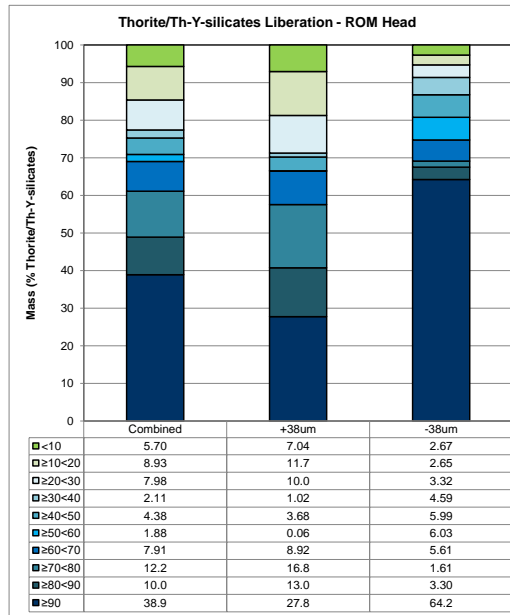
Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

Thorite/Th-Y-silicates Liberation



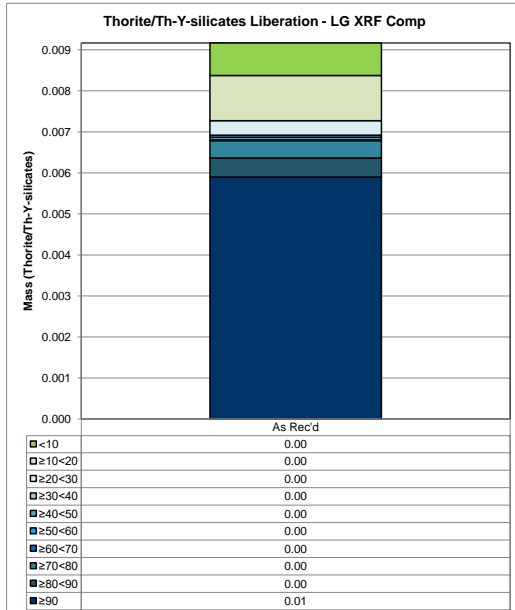
Absolute Mass of Thorite/Th-Y-silicates Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	0.01	0.00	0.00
≥80<90	0.00	0.00	0.00
≥70<80	0.00	0.00	0.00
≥60<70	0.00	0.00	0.00
≥50<60	0.00	0.00	0.00
≥40<50	0.00	0.00	0.00
≥30<40	0.00	0.00	0.00
≥20<30	0.00	0.00	0.00
≥10<20	0.00	0.00	0.00
<10	0.00	0.00	0.00
Total	0.02	0.01	0.01



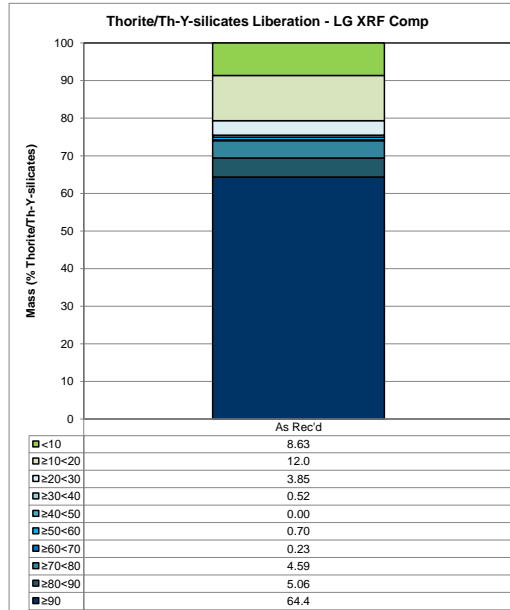
Normalized Mass of Thorite/Th-Y-silicates Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	38.9	27.8	64.2
≥80<90	10.0	13.0	3.30
≥70<80	12.2	16.8	1.61
≥60<70	7.91	8.92	5.61
≥50<60	1.88	0.06	6.03
≥40<50	4.38	3.68	5.99
≥30<40	2.11	1.02	4.59
≥20<30	7.98	10.0	3.32
≥10<20	8.93	11.7	2.65
<10	5.70	7.04	2.67
Total	100.0	100.0	100.0



Absolute Mass of Thorite/Th-Y-silicates Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	0.01
≥80<90	0.00
≥70<80	0.00
≥60<70	0.00
≥50<60	0.00
≥40<50	0.00
≥30<40	0.00
≥20<30	0.00
≥10<20	0.00
<10	0.00
Total	0.01

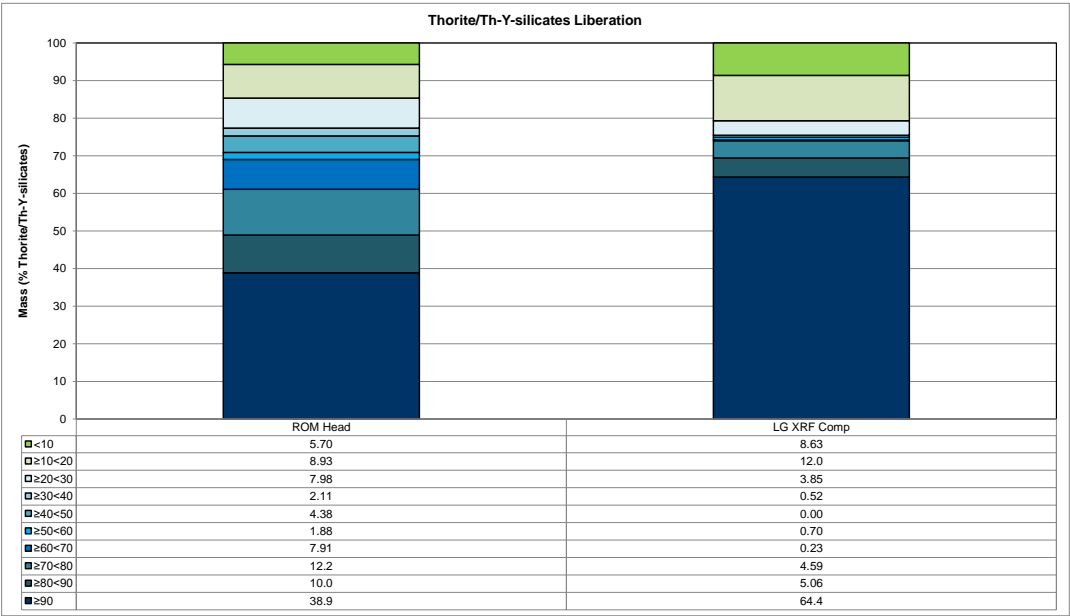


Normalized Mass of Thorite/Th-Y-silicates Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	64.4
≥80<90	5.06
≥70<80	4.59
≥60<70	0.23
≥50<60	0.70
≥40<50	0.00
≥30<40	0.00
≥20<30	0.00
≥10<20	0.00
<10	0.00
Total	100.0

Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

Thorite/Th-Y-silicates Liberation

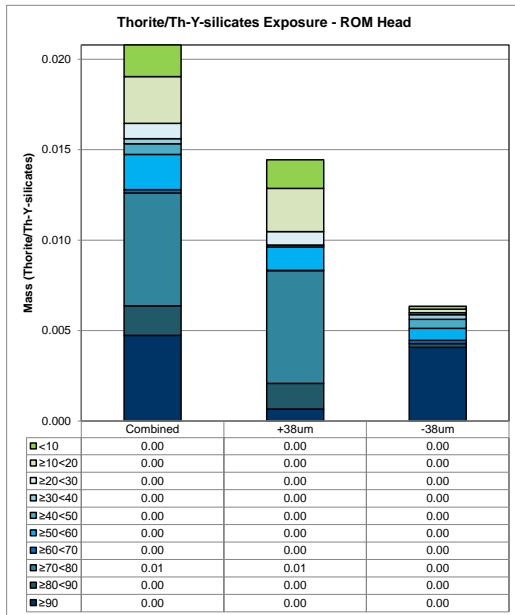


Normalized Mass of Thorite/Th-Y-silicates Across Samples

Mineral Name	ROM Head	LG XRF Comp
≥90	38.9	64.4
≥80<90	10.0	5.06
≥70<80	12.2	4.59
≥60<70	7.91	0.23
≥50<60	1.88	0.70
≥40<50	4.38	0.00
≥30<40	2.11	0.52
≥20<30	7.98	3.85
≥10<20	8.93	12.0
<10	5.70	8.63
Total	100.0	100.0

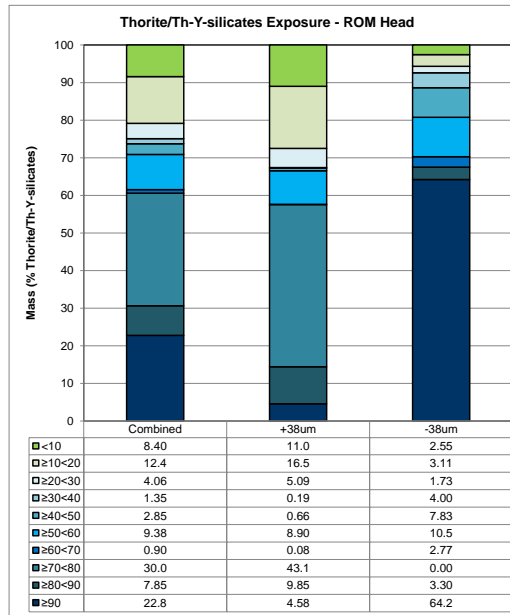
Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

Thorite/Th-Y-silicates Exposure



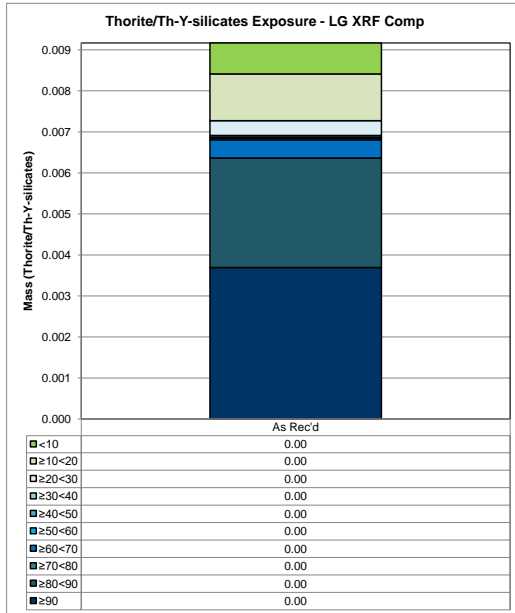
Absolute Mass of Thorite/Th-Y-silicates Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	0.00	0.00	0.00
≥80<90	0.00	0.00	0.00
≥70<80	0.01	0.01	0.00
≥60<70	0.00	0.00	0.00
≥50<60	0.00	0.00	0.00
≥40<50	0.00	0.00	0.00
≥30<40	0.00	0.00	0.00
≥20<30	0.00	0.00	0.00
≥10<20	0.00	0.00	0.00
<10	0.00	0.00	0.00
Total	0.02	0.01	0.01



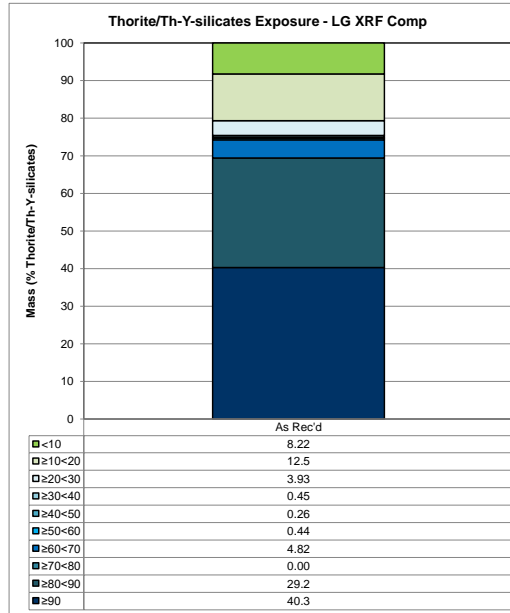
Normalized Mass of Thorite/Th-Y-silicates Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	22.8	4.58	64.2
≥80<90	7.85	9.85	3.30
≥70<80	30.0	43.1	0.00
≥60<70	0.90	0.08	2.77
≥50<60	9.38	8.90	10.5
≥40<50	2.85	0.66	7.83
≥30<40	1.35	0.19	4.00
≥20<30	4.06	5.09	1.73
≥10<20	12.4	16.5	3.11
<10	8.40	11.0	2.55
Total	100.0	100.0	100.0



Absolute Mass of Thorite/Th-Y-silicates Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	0.00
≥80<90	0.00
≥70<80	0.00
≥60<70	0.00
≥50<60	0.00
≥40<50	0.00
≥30<40	0.00
≥20<30	0.00
≥10<20	0.00
<10	0.00
Total	0.01

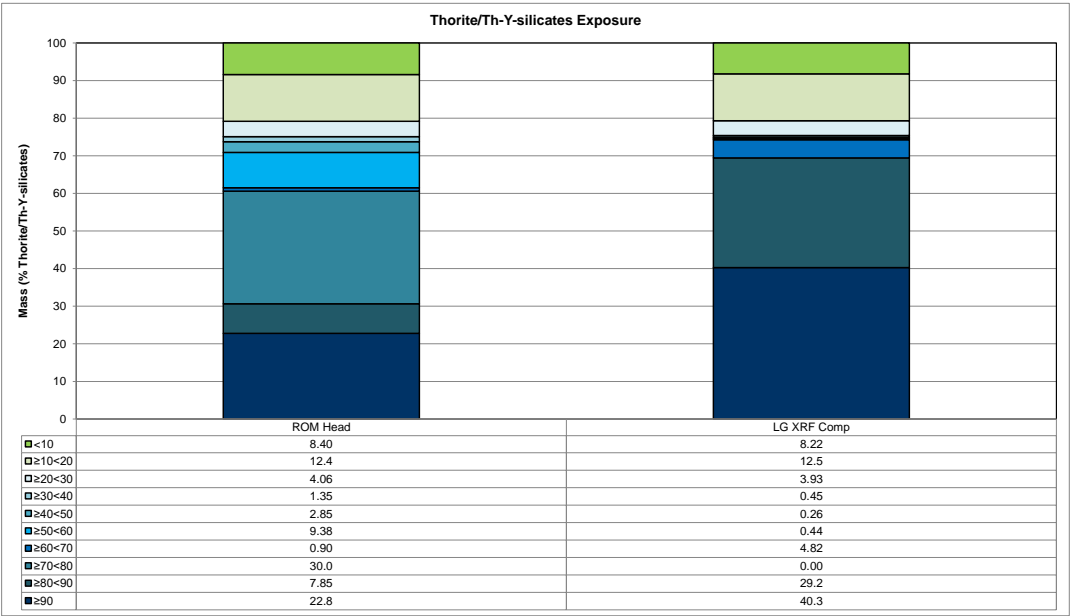


Normalized Mass of Thorite/Th-Y-silicates Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	40.3
≥80<90	29.2
≥70<80	0.00
≥60<70	4.82
≥50<60	0.44
≥40<50	0.26
≥30<40	0.45
≥20<30	3.93
≥10<20	12.5
<10	8.22
Total	100.0

Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

Thorite/Th-Y-silicates Exposure

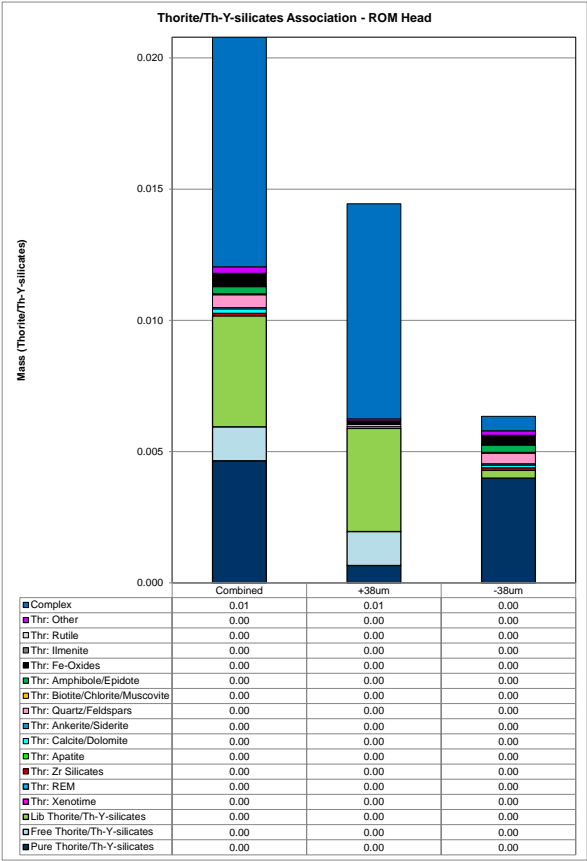


Normalized Mass of Thorite/Th-Y-silicates Across Samples

Mineral Name	ROM Head	LG XRF Comp
≥90	22.8	40.3
≥80<90	7.85	29.2
≥70<80	30.0	0.00
≥60<70	0.90	4.82
≥50<60	9.38	0.44
≥40<50	2.85	0.26
≥30<40	1.35	0.45
≥20<30	4.06	3.93
≥10<20	12.4	12.5
<10	8.40	8.22
Total	100.0	100.0

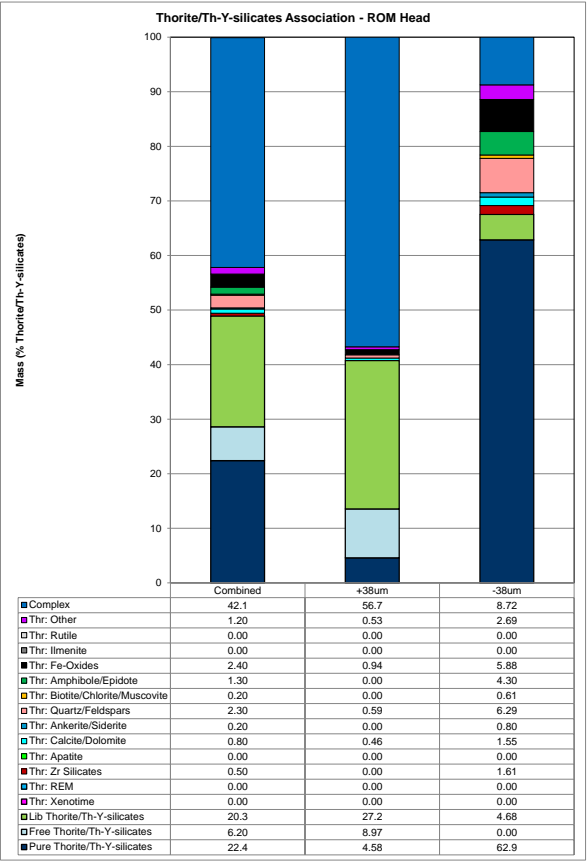
Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

Thorite/Th-Y-silicates Association



Absolute Mass of Thorite/Th-Y-silicates Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
Pure Thorite/Th-Y-silicates	0.00	0.00	0.00
Free Thorite/Th-Y-silicates	0.00	0.00	0.00
Lib Thorite/Th-Y-silicates	0.00	0.00	0.00
Thr: Xenotime	0.00	0.00	0.00
Thr: REM	0.00	0.00	0.00
Thr: Zr Silicates	0.00	0.00	0.00
Thr: Apatite	0.00	0.00	0.00
Thr: Calcite/Dolomite	0.00	0.00	0.00
Thr: Ankerite/Siderite	0.00	0.00	0.00
Thr: Quartz/Feldspars	0.00	0.00	0.00
Thr: Biotite/Chlorite/Muscovite	0.00	0.00	0.00
Thr: Amphibole/Epidote	0.00	0.00	0.00
Thr: Fe-Oxides	0.00	0.00	0.00
Thr: Ilmenite	0.00	0.00	0.00
Thr: Rutile	0.00	0.00	0.00
Thr: Other	0.00	0.00	0.00
Complex	0.01	0.01	0.00
Total	0.02	0.01	0.01



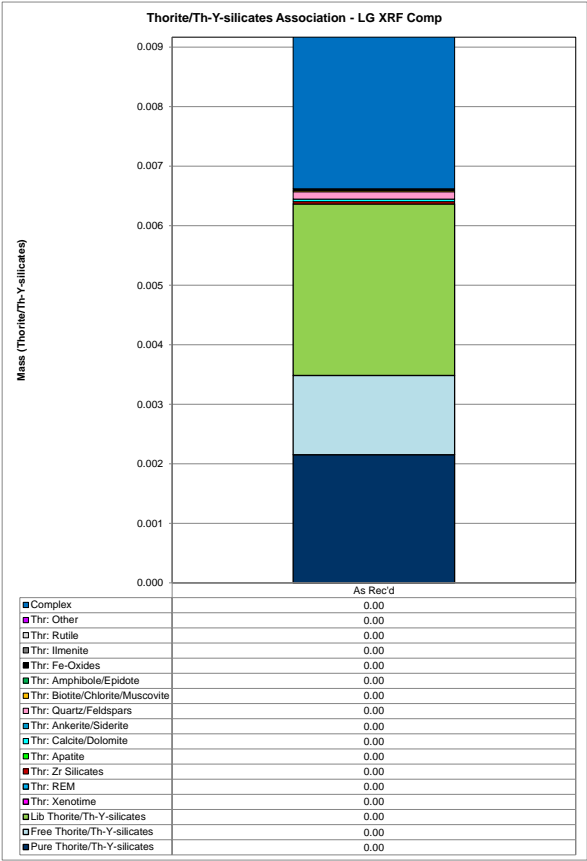
Normalized Mass of Thorite/Th-Y-silicates Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
Pure Thorite/Th-Y-silicates	22.4	4.58	62.9
Free Thorite/Th-Y-silicates	6.20	8.97	0.00
Lib Thorite/Th-Y-silicates	20.3	27.2	4.68
Thr: Xenotime	0.00	0.00	0.00
Thr: REM	0.00	0.00	0.00
Thr: Zr Silicates	0.50	0.00	1.61
Thr: Apatite	0.00	0.00	0.00
Thr: Calcite/Dolomite	0.80	0.46	1.55
Thr: Ankerite/Siderite	0.20	0.00	0.80
Thr: Quartz/Feldspars	2.30	0.59	6.29
Thr: Biotite/Chlorite/Muscovite	0.20	0.00	0.61
Thr: Amphibole/Epidote	1.30	0.00	4.30
Thr: Fe-Oxides	2.40	0.94	5.88
Thr: Ilmenite	0.00	0.00	0.00
Thr: Rutile	0.00	0.00	0.00
Thr: Other	1.20	0.53	2.69
Complex	42.1	56.7	8.72
Total	100.0	100.0	100.0

Liberated 48.9 40.7 67.5

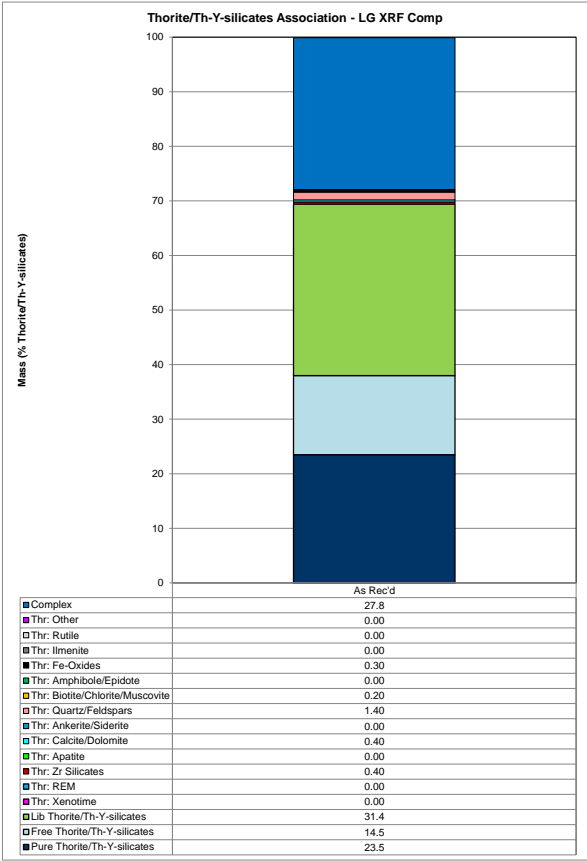
Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

Thorite/Th-Y-silicates Association



Absolute Mass of Thorite/Th-Y-silicates Across Sample LG XRF Comp

Mineral Name	As Rec'd
Pure Thorite/Th-Y-silicates	0.00
Free Thorite/Th-Y-silicates	0.00
Lib Thorite/Th-Y-silicates	0.00
Thr: Xenotime	0.00
Thr: REM	0.00
Thr: Zr Silicates	0.00
Thr: Apatite	0.00
Thr: Calcite/Dolomite	0.00
Thr: Ankerite/Siderite	0.00
Thr: Quartz/Feldspars	0.00
Thr: Biotite/Chlorite/Muscovite	0.00
Thr: Amphibole/Epidote	0.00
Thr: Fe-Oxides	0.00
Thr: Ilmenite	0.00
Thr: Rutile	0.00
Thr: Other	0.00
Complex	0.00
Total	0.01



Normalized Mass of Thorite/Th-Y-silicates Across Sample LG XRF Comp

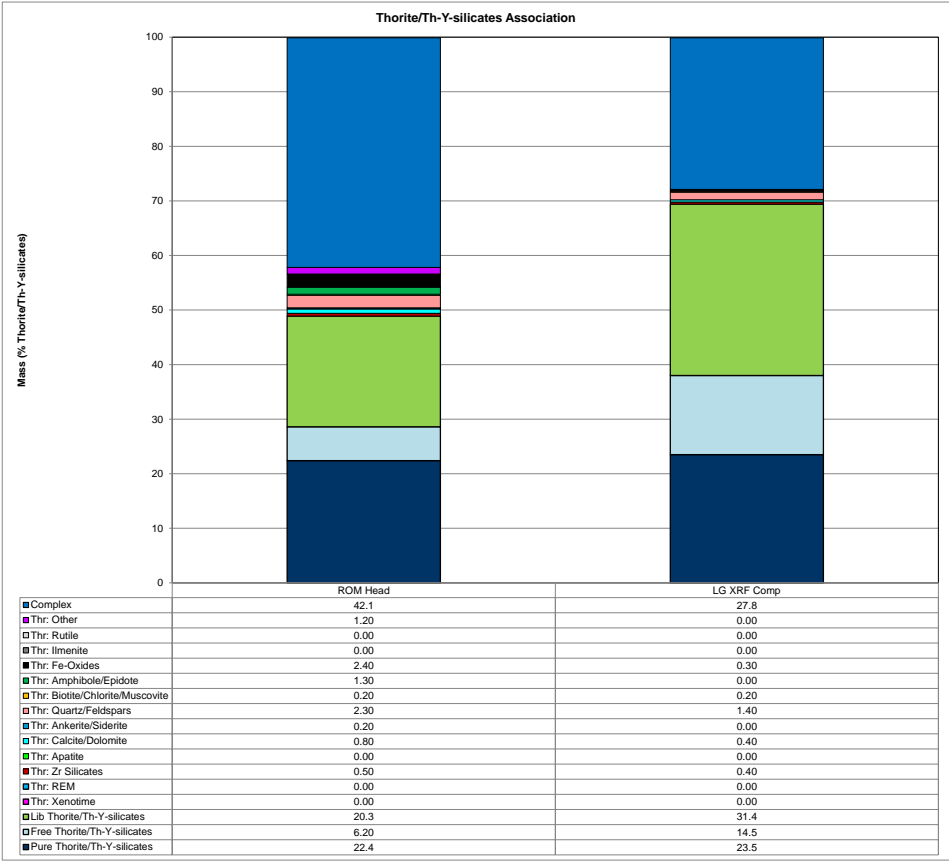
Mineral Name	As Rec'd
Pure Thorite/Th-Y-silicates	23.5
Free Thorite/Th-Y-silicates	14.5
Lib Thorite/Th-Y-silicates	31.4
Thr: Xenotime	0.00
Thr: REM	0.00
Thr: Zr Silicates	0.40
Thr: Apatite	0.00
Thr: Calcite/Dolomite	0.40
Thr: Ankerite/Siderite	0.00
Thr: Quartz/Feldspars	1.40
Thr: Biotite/Chlorite/Muscovite	0.20
Thr: Amphibole/Epidote	0.00
Thr: Fe-Oxides	0.30
Thr: Ilmenite	0.00
Thr: Rutile	0.00
Thr: Other	0.00
Complex	27.8
Total	100.0

Liberated

69.4

Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

Thorite/Th-Y-silicates Association

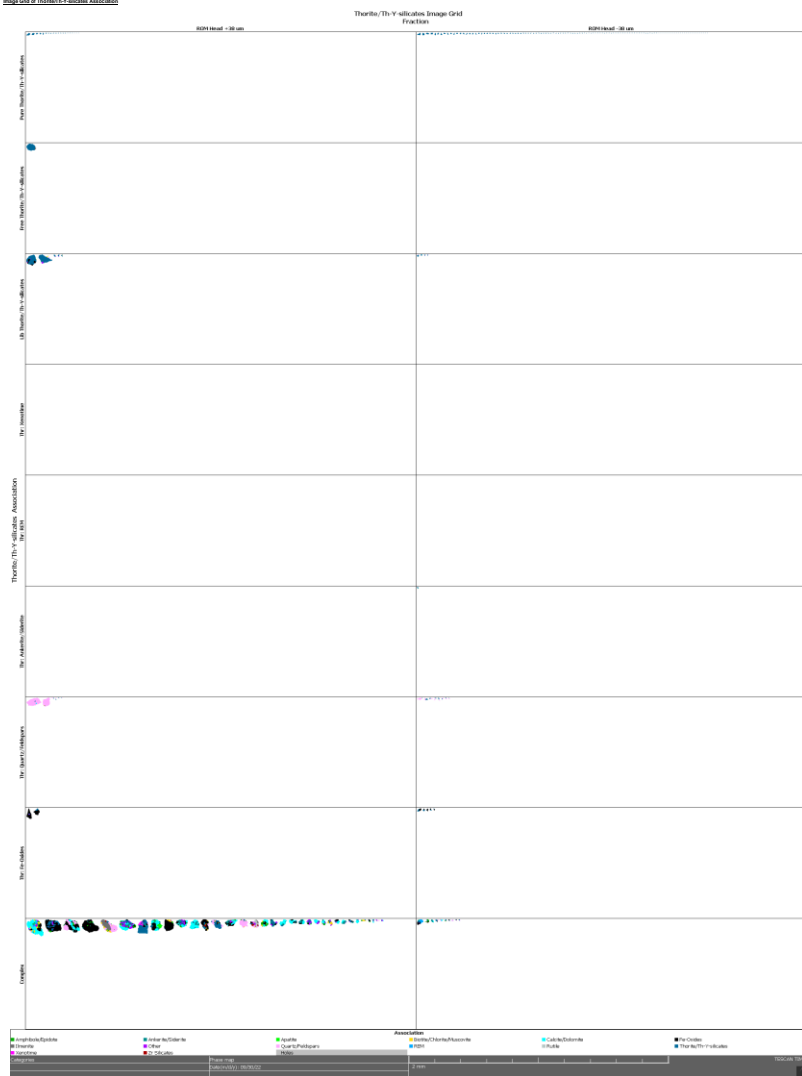


Normalized Mass of Thorite/Th-Y-silicates Across Samples

Mineral Name	ROM Head	LG XRF Comp
Pure Thorite/Th-Y-silicates	22.4	23.5
Free Thorite/Th-Y-silicates	6.20	14.5
Lib Thorite/Th-Y-silicates	20.3	31.4
Thr: Xenotime	0.00	0.00
Thr: REM	0.00	0.00
Thr: Zr Silicates	0.50	0.40
Thr: Apatite	0.00	0.00
Thr: Calcite/Dolomite	0.80	0.40
Thr: Ankerite/Siderite	0.20	0.00
Thr: Quartz/Feldspars	2.30	1.40
Thr: Biotite/Chlorite/Muscovite	0.20	0.20
Thr: Amphibole/Epidote	1.30	0.00
Thr: Fe-Oxides	2.40	0.30
Thr: Ilmenite	0.00	0.00
Thr: Rutile	0.00	0.00
Thr: Other	1.20	0.00
Complex	42.1	27.8
Total	100.0	100.0
Liberated	48.9	69.4

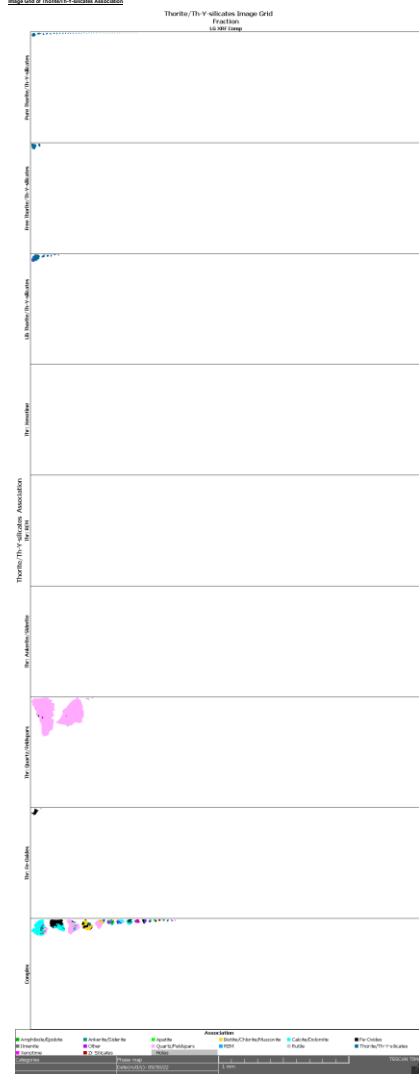
Namibia
CALR-18299-01/03
ME020-APR22 & ME009-MAY22

Image Grid of Thorite/Th-Y-silicates Association



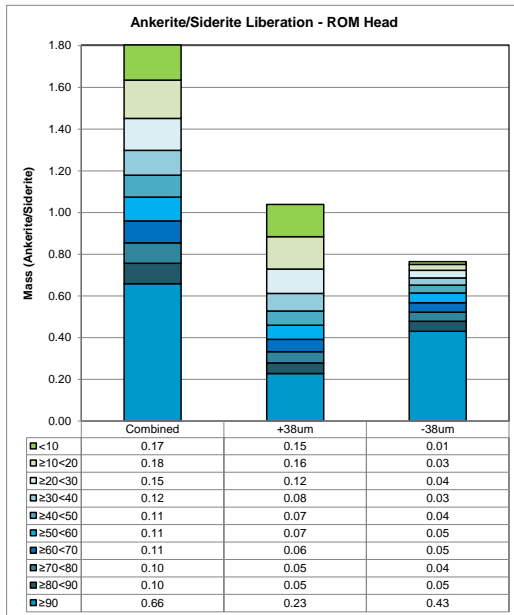
Namibia
CALR-18299-01/03
ME020-APR22 & ME009-MAY22

Image Grid of Thorite/Th-Y-silicates Association



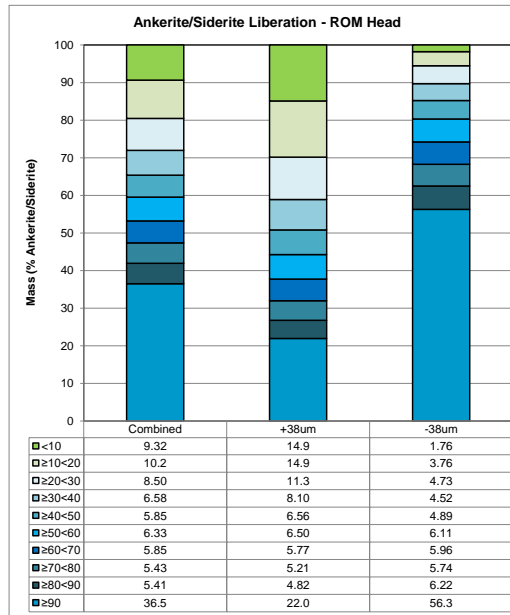
Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

Ankerite/Siderite Liberation



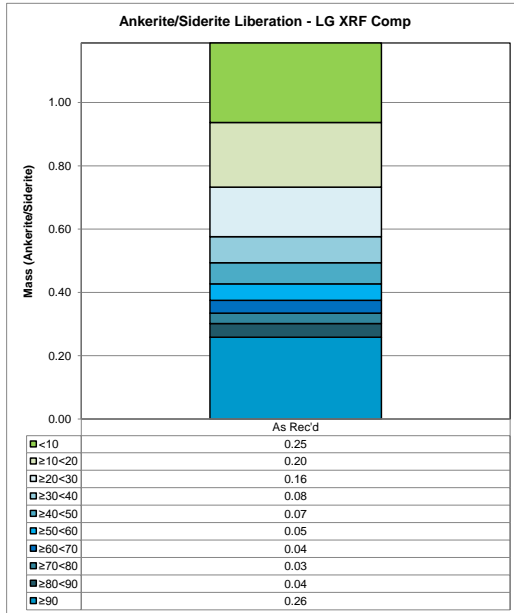
Absolute Mass of Ankerite/Siderite Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	0.66	0.23	0.43
≥80<90	0.10	0.05	0.05
≥70<80	0.10	0.05	0.04
≥60<70	0.11	0.06	0.05
≥50<60	0.11	0.07	0.05
≥40<50	0.11	0.07	0.04
≥30<40	0.12	0.08	0.03
≥20<30	0.15	0.12	0.04
≥10<20	0.18	0.16	0.03
<10	0.17	0.15	0.01
Total	1.80	1.04	0.76



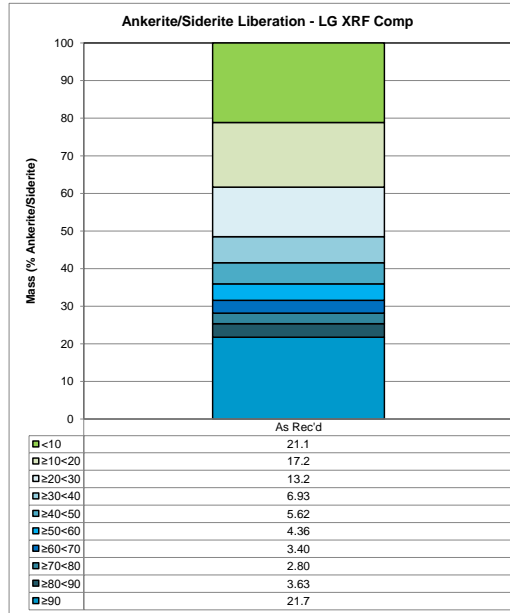
Normalized Mass of Ankerite/Siderite Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	36.5	22.0	56.3
≥80<90	5.41	4.82	6.22
≥70<80	5.43	5.21	5.74
≥60<70	5.85	5.77	5.96
≥50<60	6.33	6.50	6.11
≥40<50	5.85	6.56	4.89
≥30<40	6.58	8.10	4.52
≥20<30	8.50	11.3	4.73
≥10<20	10.2	14.9	3.76
<10	9.32	14.9	1.76
Total	100.0	100.0	100.0



Absolute Mass of Ankerite/Siderite Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	0.26
≥80<90	0.04
≥70<80	0.03
≥60<70	0.04
≥50<60	0.05
≥40<50	0.07
≥30<40	0.08
≥20<30	0.16
≥10<20	0.20
<10	0.25
Total	1.19

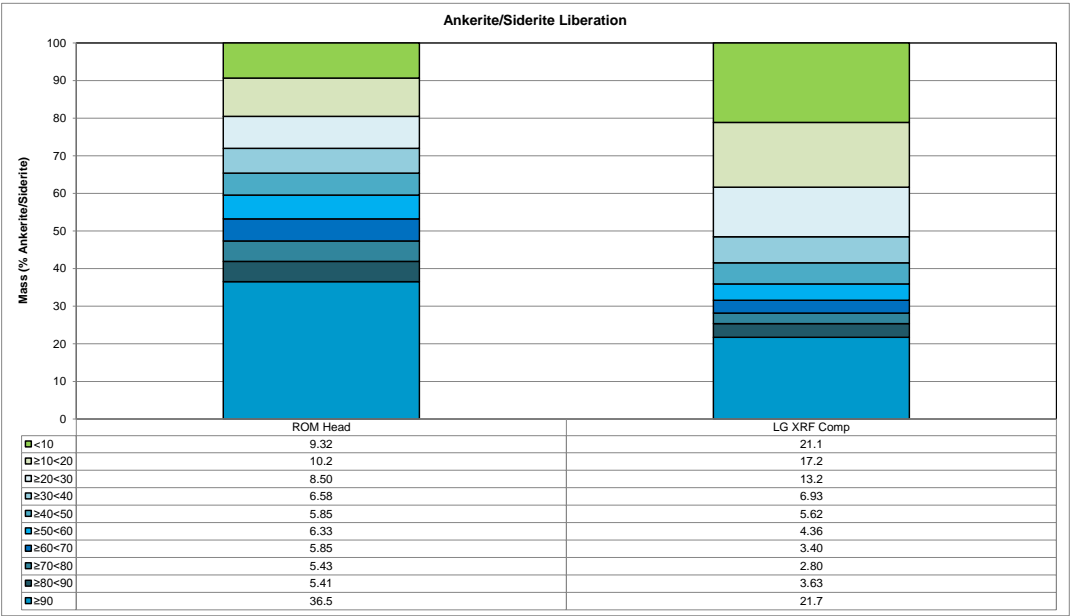


Normalized Mass of Ankerite/Siderite Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	21.7
≥80<90	3.63
≥70<80	2.80
≥60<70	3.40
≥50<60	4.36
≥40<50	5.62
≥30<40	6.93
≥20<30	13.2
≥10<20	17.2
<10	21.1
Total	100.0

Namibia
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MI5029-APR22 & MI5009-MAY22

Ankerite/Siderite Liberation

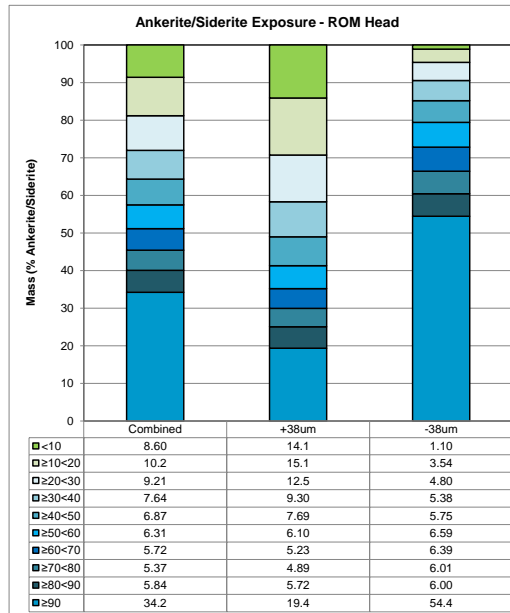
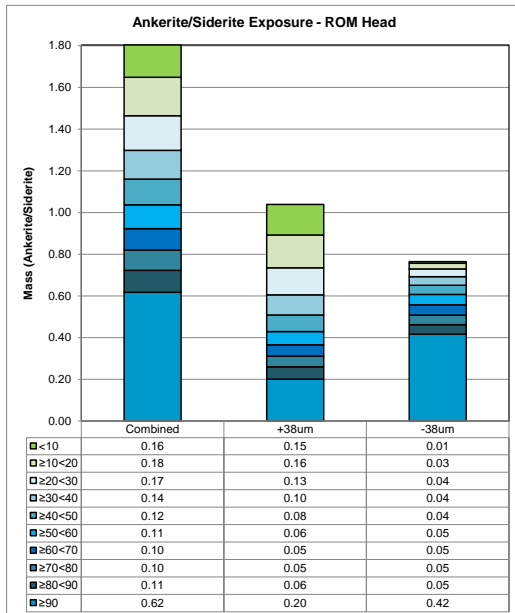


Normalized Mass of Ankerite/Siderite Across Samples

Mineral Name	ROM Head	LG XRF Comp
≥90	36.5	21.7
≥80<90	5.41	3.63
≥70<80	5.43	2.80
≥60<70	5.85	3.40
≥50<60	6.33	4.36
≥40<50	5.85	5.62
≥30<40	6.58	6.93
≥20<30	8.50	13.2
≥10<20	10.2	17.2
<10	9.32	21.1
Total	100.0	100.0

Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

Ankerite/Siderite Exposure

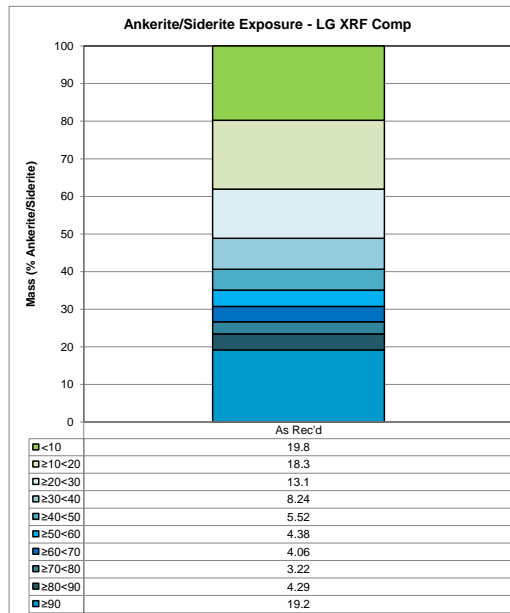
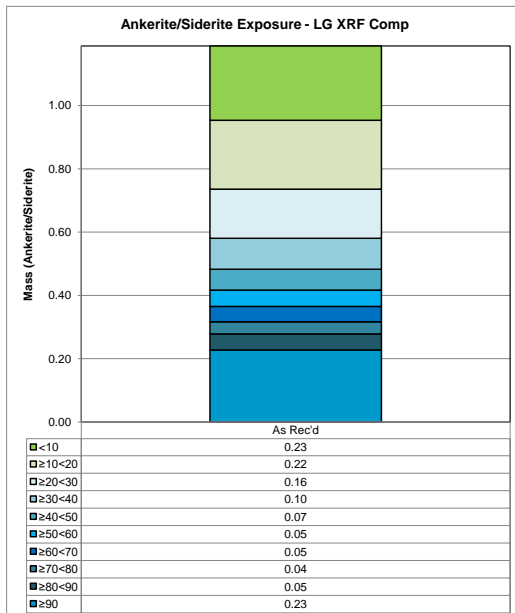


Absolute Mass of Ankerite/Siderite Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	0.62	0.20	0.42
≥80<90	0.11	0.06	0.05
≥70<80	0.10	0.05	0.05
≥60<70	0.10	0.05	0.05
≥50<60	0.11	0.06	0.05
≥40<50	0.12	0.08	0.04
≥30<40	0.14	0.10	0.04
≥20<30	0.17	0.13	0.04
≥10<20	0.18	0.16	0.03
<10	0.16	0.15	0.01
Total	1.80	1.04	0.76

Normalized Mass of Ankerite/Siderite Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	34.2	19.4	54.4
≥80<90	5.84	5.72	6.00
≥70<80	5.37	4.89	6.01
≥60<70	5.72	5.23	6.39
≥50<60	6.31	6.10	6.59
≥40<50	6.87	7.69	5.75
≥30<40	7.64	9.30	5.38
≥20<30	9.21	12.5	4.80
≥10<20	10.2	15.1	3.54
<10	8.60	14.1	1.10
Total	100.0	100.0	100.0



Absolute Mass of Ankerite/Siderite Across Sample LG XRF Comp

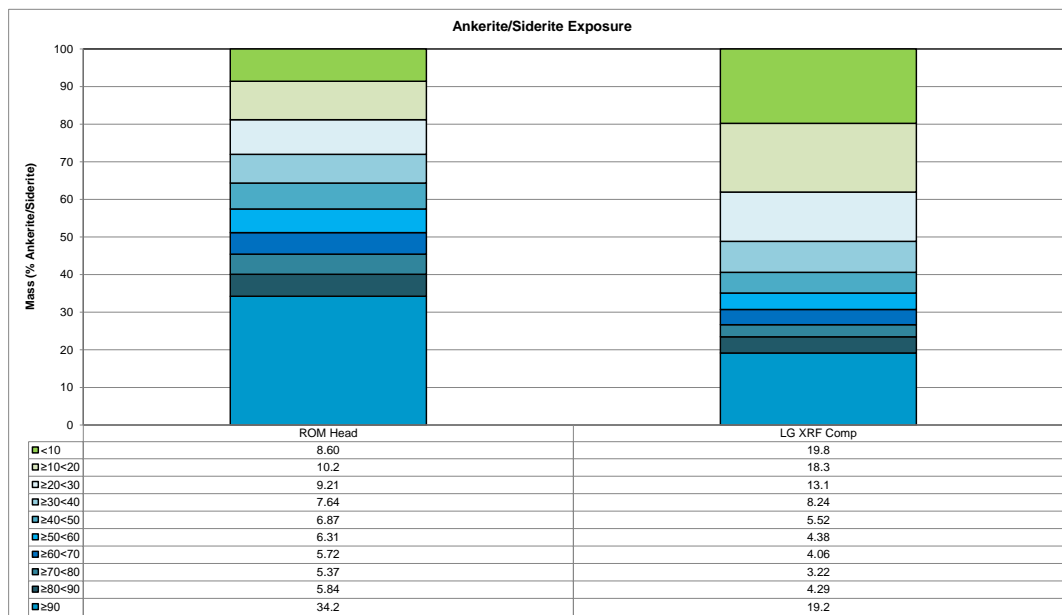
Mineral Name	As Rec'd
≥90	0.23
≥80<90	0.05
≥70<80	0.04
≥60<70	0.05
≥50<60	0.05
≥40<50	0.07
≥30<40	0.10
≥20<30	0.16
≥10<20	0.22
<10	0.23
Total	1.19

Normalized Mass of Ankerite/Siderite Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	19.2
≥80<90	4.29
≥70<80	3.22
≥60<70	4.06
≥50<60	4.38
≥40<50	5.52
≥30<40	8.24
≥20<30	13.1
≥10<20	18.3
<10	19.8
Total	100.0

Namibia
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 M15029-APR22 & M15009-MAY22

Ankerite/Siderite Exposure

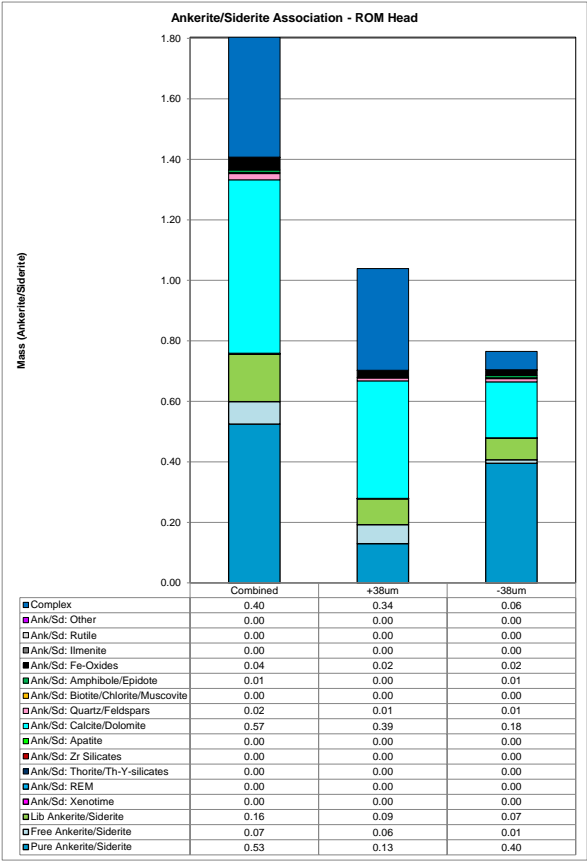


Normalized Mass of Ankerite/Siderite Across Samples

Mineral Name	ROM Head	LG XRF Comp
≥90	34.2	19.2
≥80<90	5.84	4.29
≥70<80	5.37	3.22
≥60<70	5.72	4.06
≥50<60	6.31	4.38
≥40<50	6.87	5.52
≥30<40	7.64	8.24
≥20<30	9.21	13.1
≥10<20	10.2	18.3
<10	8.60	19.8
Total	100.0	100.0

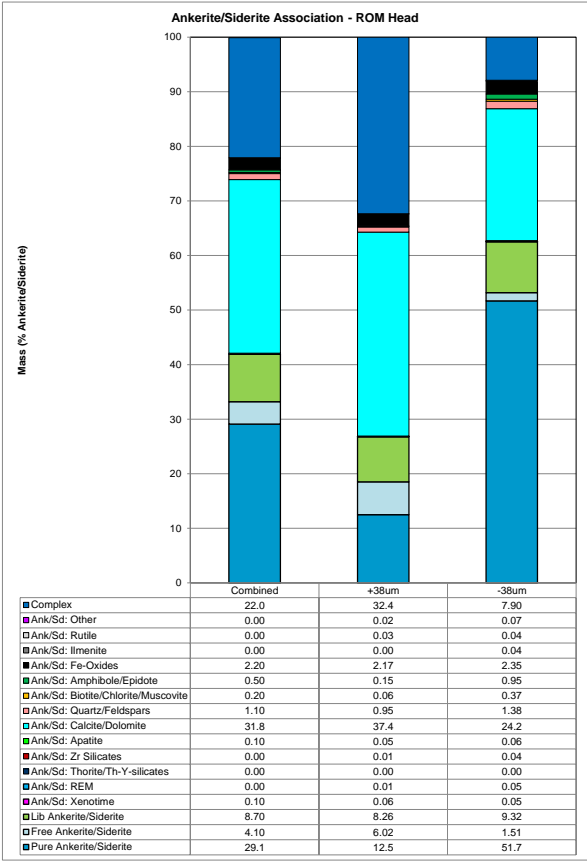
Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

Ankerite/Siderite Association



Absolute Mass of Ankerite/Siderite Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
Pure Ankerite/Siderite	0.53	0.13	0.40
Free Ankerite/Siderite	0.07	0.06	0.01
Lib Ankerite/Siderite	0.16	0.09	0.07
Ank/Sd: Xenotime	0.00	0.00	0.00
Ank/Sd: REM	0.00	0.00	0.00
Ank/Sd: Thorite/Th-Y-silicates	0.00	0.00	0.00
Ank/Sd: Zr Silicates	0.00	0.00	0.00
Ank/Sd: Apatite	0.00	0.00	0.00
Ank/Sd: Calcite/Dolomite	0.57	0.39	0.18
Ank/Sd: Quartz/Feldspars	0.02	0.01	0.01
Ank/Sd: Biotite/Chlorite/Muscovite	0.00	0.00	0.00
Ank/Sd: Amphibole/Epidote	0.01	0.00	0.01
Ank/Sd: Fe-Oxides	0.04	0.02	0.02
Ank/Sd: Ilmenite	0.00	0.00	0.00
Ank/Sd: Rutile	0.00	0.00	0.00
Ank/Sd: Other	0.00	0.00	0.00
Complex	0.40	0.34	0.06
Total	1.80	1.04	0.76

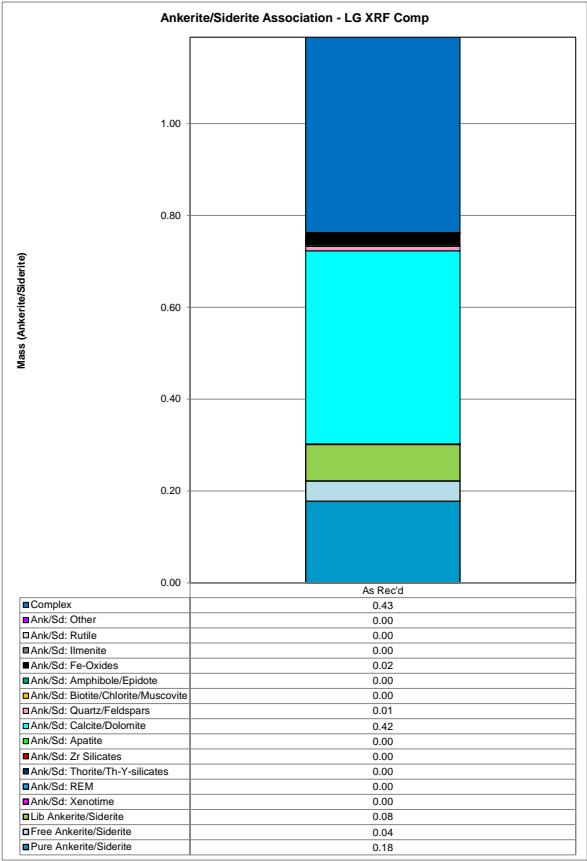


Normalized Mass of Ankerite/Siderite Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
Pure Ankerite/Siderite	29.1	12.5	51.7
Free Ankerite/Siderite	4.10	6.02	1.51
Lib Ankerite/Siderite	8.70	8.26	9.32
Ank/Sd: Xenotime	0.10	0.06	0.05
Ank/Sd: REM	0.00	0.01	0.05
Ank/Sd: Thorite/Th-Y-silicates	0.00	0.00	0.00
Ank/Sd: Zr Silicates	0.00	0.01	0.04
Ank/Sd: Apatite	0.10	0.05	0.06
Ank/Sd: Calcite/Dolomite	31.8	37.4	24.2
Ank/Sd: Quartz/Feldspars	1.10	0.95	1.38
Ank/Sd: Biotite/Chlorite/Muscovite	0.20	0.06	0.37
Ank/Sd: Amphibole/Epidote	0.50	0.15	0.95
Ank/Sd: Fe-Oxides	2.20	2.17	2.35
Ank/Sd: Ilmenite	0.00	0.00	0.04
Ank/Sd: Rutile	0.00	0.03	0.04
Ank/Sd: Other	0.00	0.02	0.07
Complex	22.0	32.4	7.90
Total	100.0	100.0	100.0
Liberated	41.9	26.8	62.5

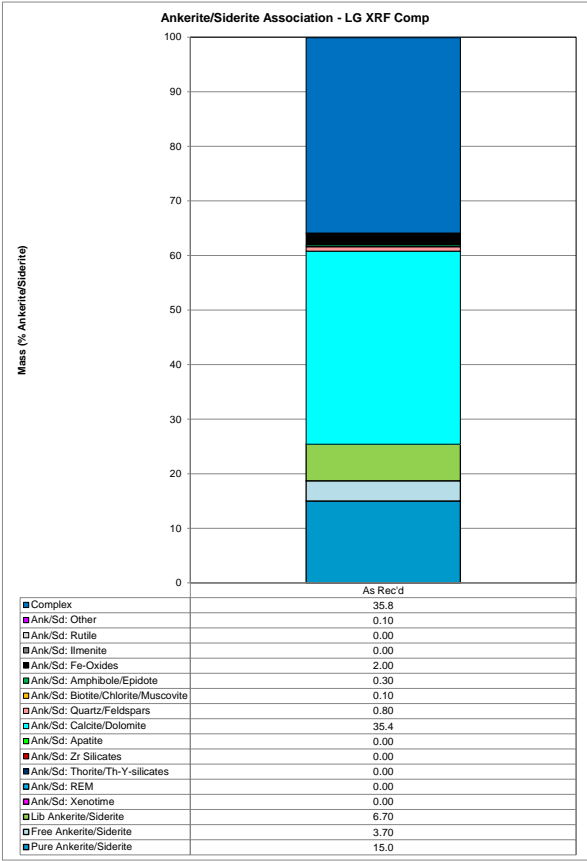
Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

Ankerite/Siderite Association



Absolute Mass of Ankerite/Siderite Across Sample LG XRF Comp

Mineral Name	As Rec'd
Pure Ankerite/Siderite	0.18
Free Ankerite/Siderite	0.04
Lib Ankerite/Siderite	0.08
Ank/Sd: Xenotime	0.00
Ank/Sd: REM	0.00
Ank/Sd: Thorite/Th-Y-silicates	0.00
Ank/Sd: Zr Silicates	0.00
Ank/Sd: Apatite	0.00
Ank/Sd: Calcite/Dolomite	0.42
Ank/Sd: Quartz/Feldspars	0.01
Ank/Sd: Biotite/Chlorite/Muscovite	0.00
Ank/Sd: Amphibole/Epidote	0.00
Ank/Sd: Fe-Oxides	0.02
Ank/Sd: Ilmenite	0.00
Ank/Sd: Rutile	0.00
Ank/Sd: Other	0.00
Complex	0.43
Total	1.19



Normalized Mass of Ankerite/Siderite Across Sample LG XRF Comp

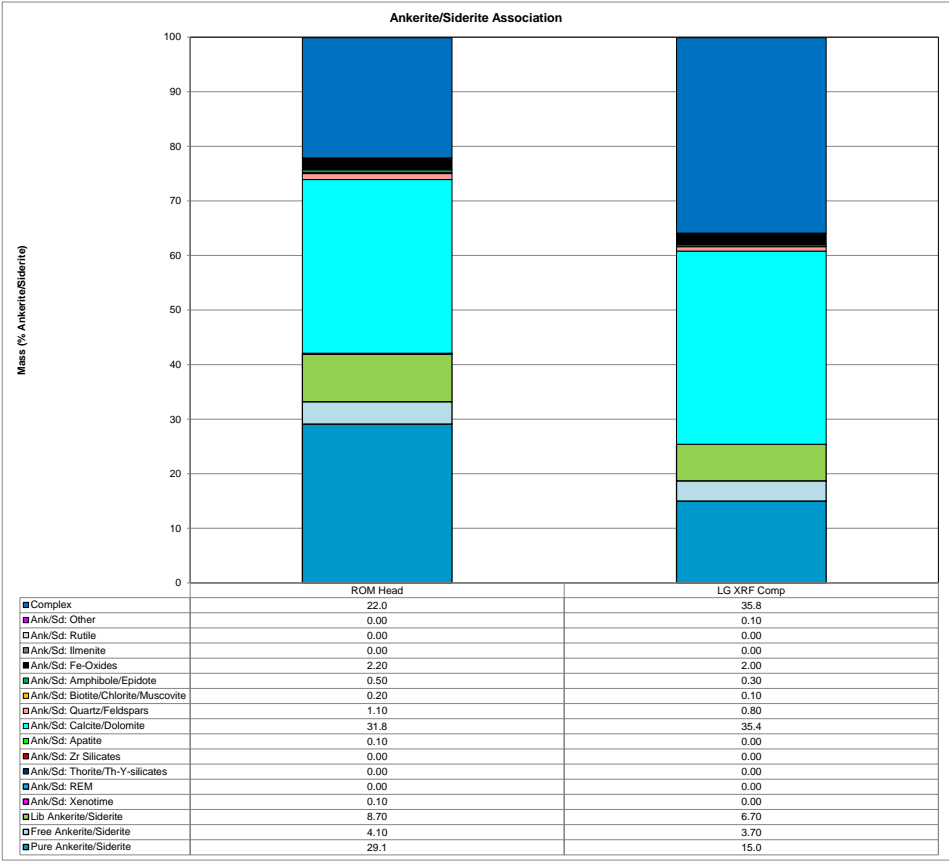
Mineral Name	As Rec'd
Pure Ankerite/Siderite	15.0
Free Ankerite/Siderite	3.70
Lib Ankerite/Siderite	6.70
Ank/Sd: Xenotime	0.00
Ank/Sd: REM	0.00
Ank/Sd: Thorite/Th-Y-silicates	0.00
Ank/Sd: Zr Silicates	0.00
Ank/Sd: Apatite	0.00
Ank/Sd: Calcite/Dolomite	35.4
Ank/Sd: Quartz/Feldspars	0.80
Ank/Sd: Biotite/Chlorite/Muscovite	0.10
Ank/Sd: Amphibole/Epidote	0.30
Ank/Sd: Fe-Oxides	2.00
Ank/Sd: Ilmenite	0.00
Ank/Sd: Rutile	0.00
Ank/Sd: Other	0.10
Complex	35.8
Total	100.0

Liberated

25.4

Namibia
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M15029-APR22 & M15009-MAY22

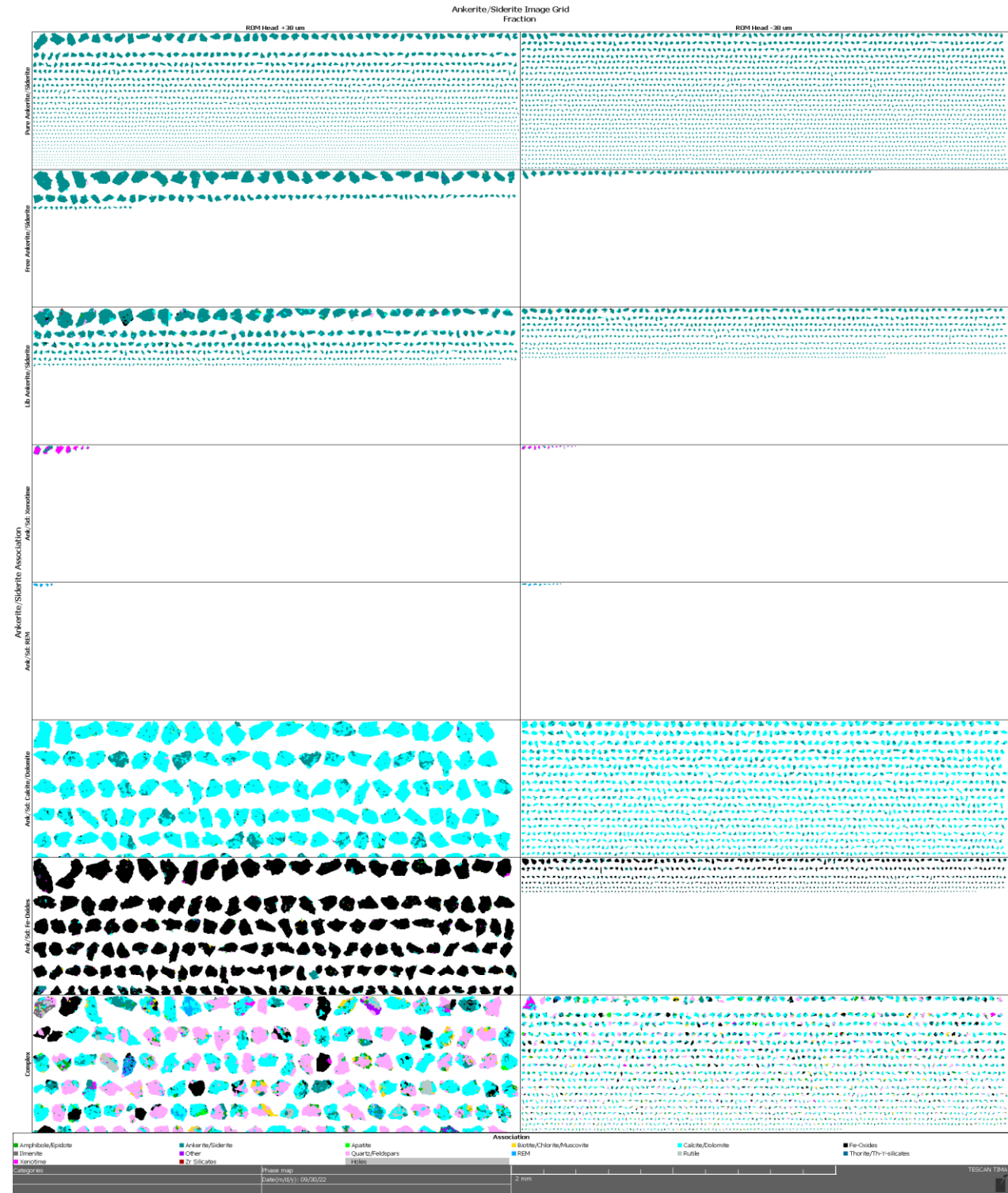
Ankerite/Siderite Association



Normalized Mass of Ankerite/Siderite Across Samples

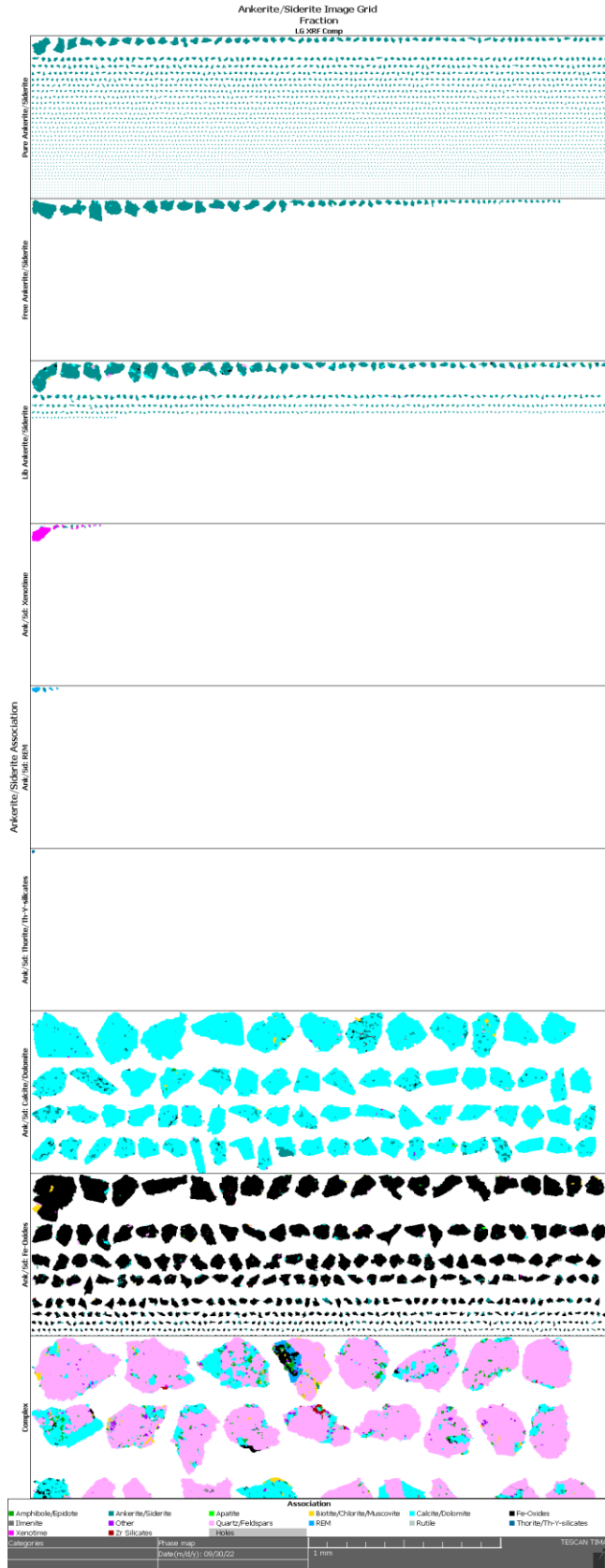
Mineral Name	ROM Head	LG XRF Comp
Pure Ankerite/Siderite	29.1	15.0
Free Ankerite/Siderite	4.10	3.70
Lib Ankerite/Siderite	8.70	6.70
Ank/Sd: Xenotime	0.10	0.00
Ank/Sd: REM	0.00	0.00
Ank/Sd: Thorite/Th-Y-silicates	0.00	0.00
Ank/Sd: Zr Silicates	0.00	0.00
Ank/Sd: Apatite	0.10	0.00
Ank/Sd: Calcite/Dolomite	31.8	35.4
Ank/Sd: Quartz/Feldspars	1.10	0.80
Ank/Sd: Biotite/Chlorite/Muscovite	0.20	0.10
Ank/Sd: Amphibole/Epidote	0.50	0.30
Ank/Sd: Fe-Oxides	2.20	2.00
Ank/Sd: Ilmenite	0.00	0.00
Ank/Sd: Rutile	0.00	0.00
Ank/Sd: Other	0.00	0.10
Complex	22.0	35.8
Total	100.0	100.0
Liberated	41.9	25.4

Namibia
CALR-18299-01/03
M5025-APR22 & M5009-MAY22
Image Grid of Ankerite/Siderite Association



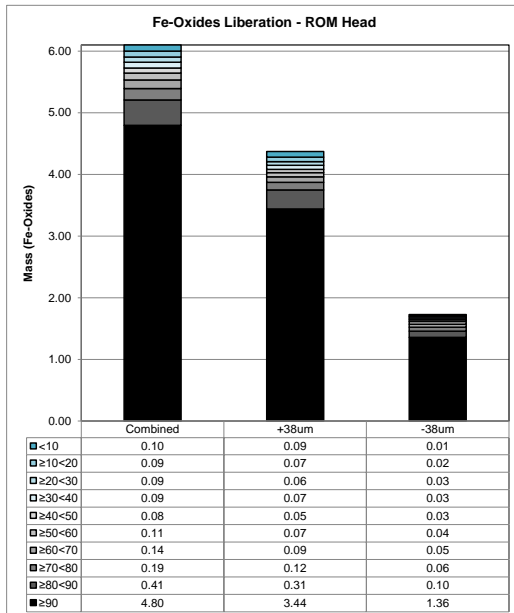
Namibia
CALR-18299-01/03
MS026-APR22 & MS009-MAY22

Image Grid of Ankerite/Siderite Association



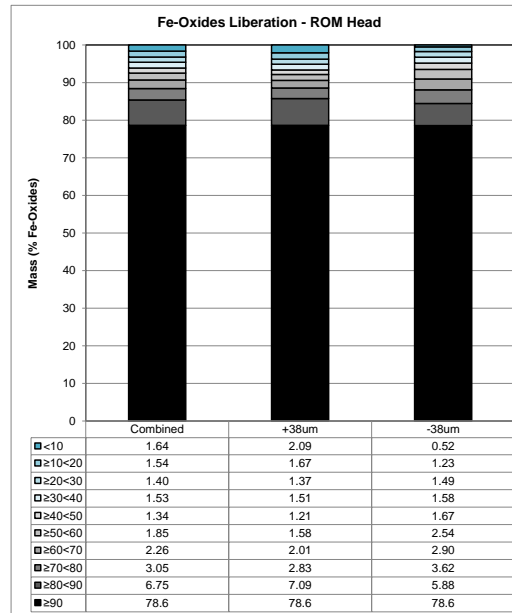
Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

Fe-Oxides Liberation



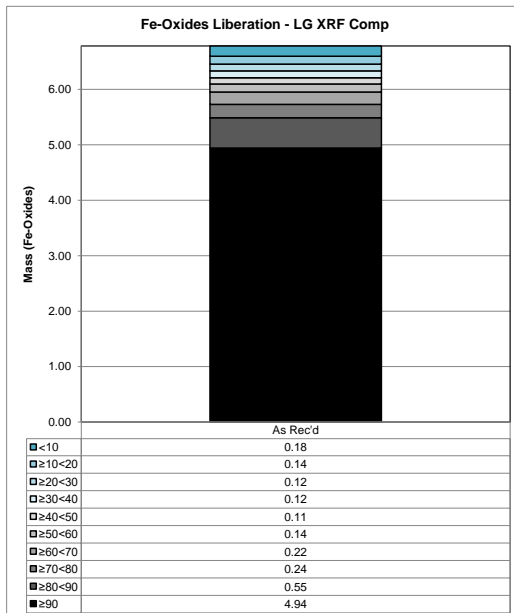
Absolute Mass of Fe-Oxides Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	4.80	3.44	1.36
≥80<90	0.41	0.31	0.10
≥70<80	0.19	0.12	0.06
≥60<70	0.14	0.09	0.05
≥50<60	0.11	0.07	0.04
≥40<50	0.08	0.05	0.03
≥30<40	0.09	0.07	0.03
≥20<30	0.09	0.06	0.03
≥10<20	0.09	0.07	0.02
<10	0.10	0.09	0.01
Total	6.10	4.37	1.73



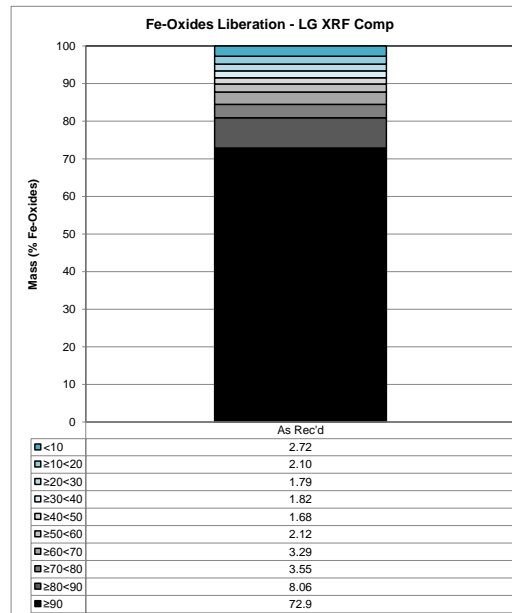
Normalized Mass of Fe-Oxides Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	78.6	78.6	78.6
≥80<90	6.75	7.09	5.88
≥70<80	3.05	2.83	3.62
≥60<70	2.26	2.01	2.90
≥50<60	1.85	1.58	2.54
≥40<50	1.34	1.21	1.67
≥30<40	1.53	1.51	1.58
≥20<30	1.40	1.37	1.49
≥10<20	1.54	1.67	1.23
<10	1.64	2.09	0.52
Total	100.0	100.0	100.0



Absolute Mass of Fe-Oxides Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	4.94
≥80<90	0.55
≥70<80	0.24
≥60<70	0.22
≥50<60	0.14
≥40<50	0.11
≥30<40	0.12
≥20<30	0.12
≥10<20	0.14
<10	0.18
Total	6.78

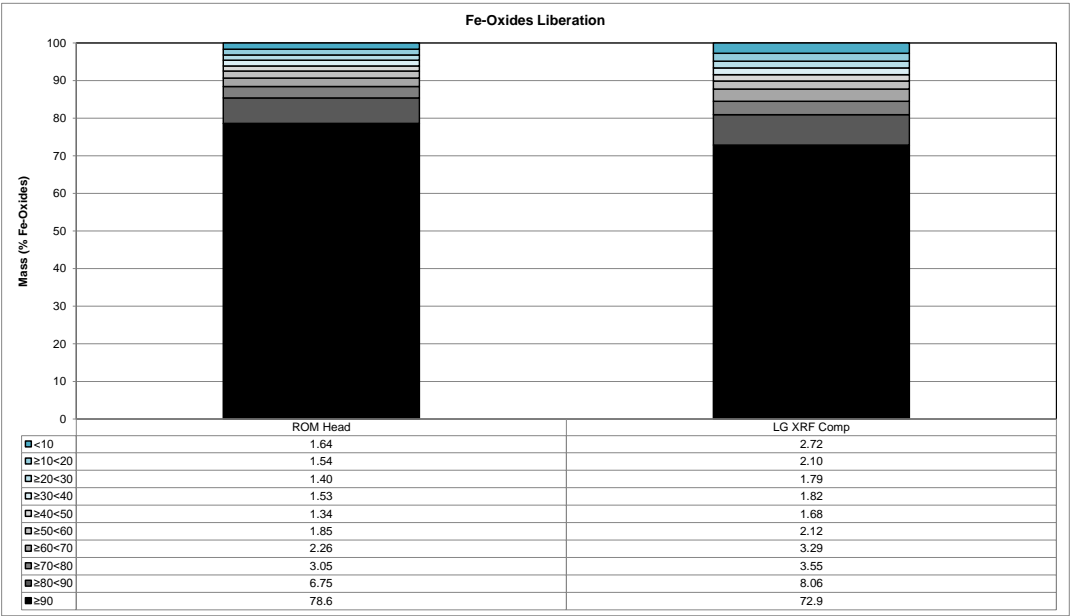


Normalized Mass of Fe-Oxides Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	72.9
≥80<90	8.06
≥70<80	3.55
≥60<70	3.29
≥50<60	2.12
≥40<50	1.68
≥30<40	1.82
≥20<30	1.79
≥10<20	2.10
<10	2.72
Total	100.0

Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

Fe-Oxides Liberation

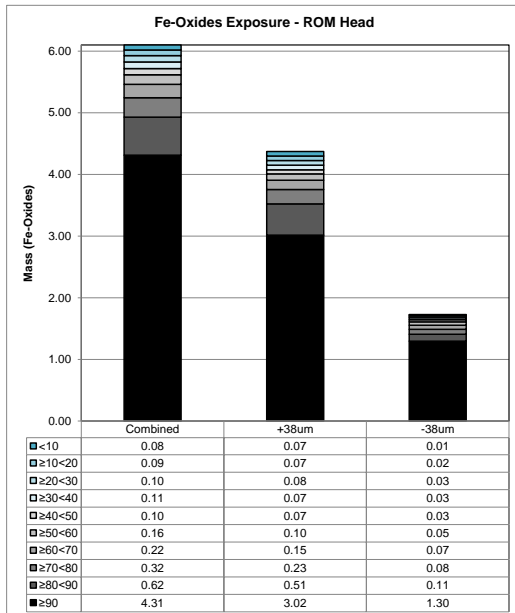


Normalized Mass of Fe-Oxides Across Samples

Mineral Name	ROM Head	LG XRF Comp
≥90	78.6	72.9
≥80<90	6.75	8.06
≥70<80	3.05	3.55
≥60<70	2.26	3.29
≥50<60	1.85	2.12
≥40<50	1.34	1.68
≥30<40	1.53	1.82
≥20<30	1.40	1.79
≥10<20	1.54	2.10
<10	1.64	2.72
Total	100.0	100.0

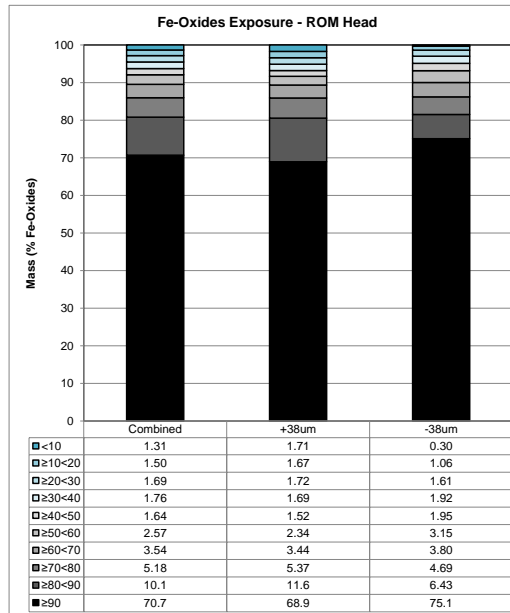
Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

Fe-Oxides Exposure



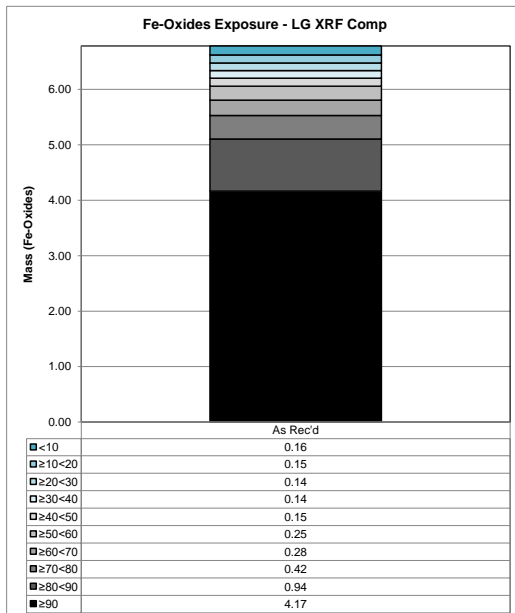
Absolute Mass of Fe-Oxides Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	4.31	3.02	1.30
≥80<90	0.62	0.51	0.11
≥70<80	0.32	0.23	0.08
≥60<70	0.22	0.15	0.07
≥50<60	0.16	0.10	0.05
≥40<50	0.10	0.07	0.03
≥30<40	0.11	0.07	0.03
≥20<30	0.10	0.08	0.03
≥10<20	0.09	0.07	0.02
<10	0.08	0.07	0.01
Total	6.10	4.37	1.73



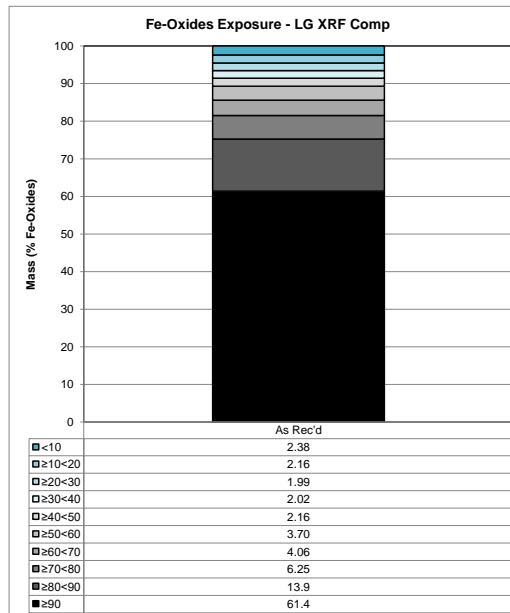
Normalized Mass of Fe-Oxides Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
≥90	70.7	68.9	75.1
≥80<90	10.1	11.6	6.43
≥70<80	5.18	5.37	4.69
≥60<70	3.54	3.44	3.80
≥50<60	2.57	2.34	3.15
≥40<50	1.64	1.52	1.95
≥30<40	1.76	1.69	1.92
≥20<30	1.69	1.72	1.61
≥10<20	1.50	1.67	1.06
<10	1.31	1.71	0.30
Total	100.0	100.0	100.0



Absolute Mass of Fe-Oxides Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	4.17
≥80<90	0.94
≥70<80	0.42
≥60<70	0.28
≥50<60	0.25
≥40<50	0.15
≥30<40	0.14
≥20<30	0.14
≥10<20	0.15
<10	0.16
Total	6.78

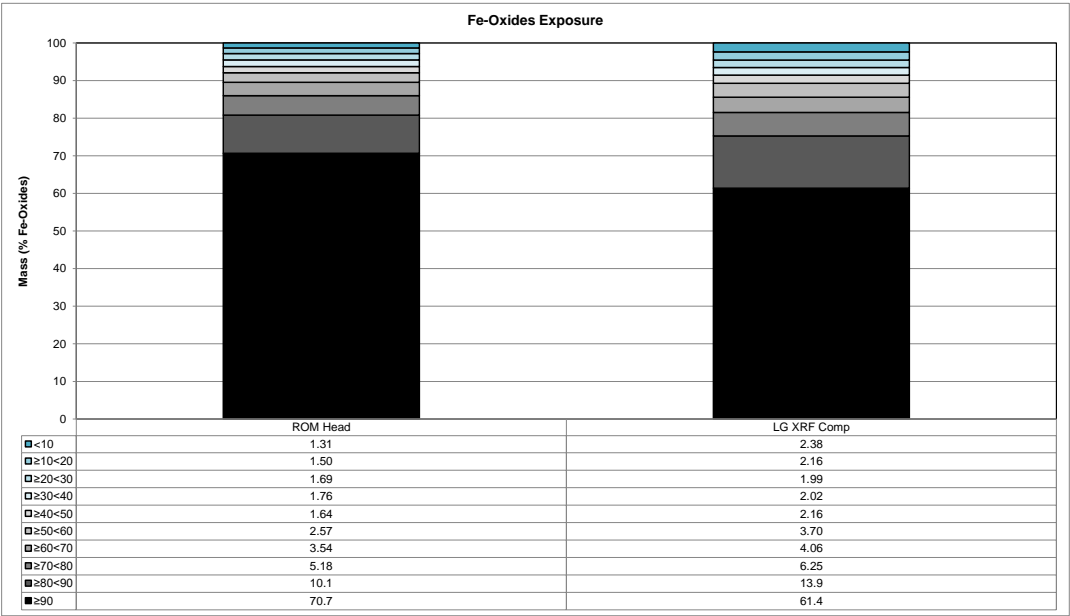


Normalized Mass of Fe-Oxides Across Sample LG XRF Comp

Mineral Name	As Rec'd
≥90	61.4
≥80<90	13.9
≥70<80	6.25
≥60<70	4.06
≥50<60	3.70
≥40<50	2.16
≥30<40	2.02
≥20<30	1.99
≥10<20	2.16
<10	2.38
Total	100.0

Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

Fe-Oxides Exposure

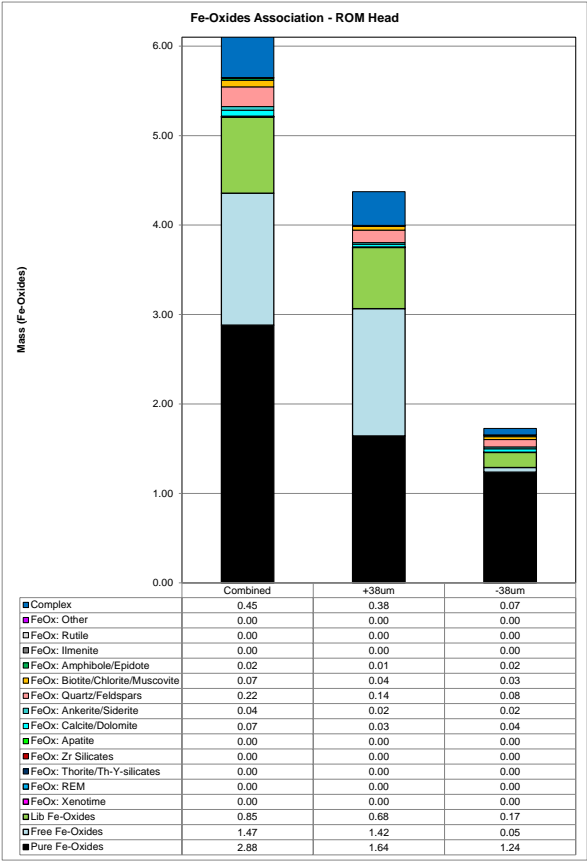


Normalized Mass of Fe-Oxides Across Samples

Mineral Name	ROM Head	LG XRF Comp
≥90	70.7	61.4
≥80<90	10.1	13.9
≥70<80	5.18	6.25
≥60<70	3.54	4.06
≥50<60	2.57	3.70
≥40<50	1.64	2.16
≥30<40	1.76	2.02
≥20<30	1.69	1.99
≥10<20	1.50	2.16
<10	1.31	2.38
Total	100.0	100.0

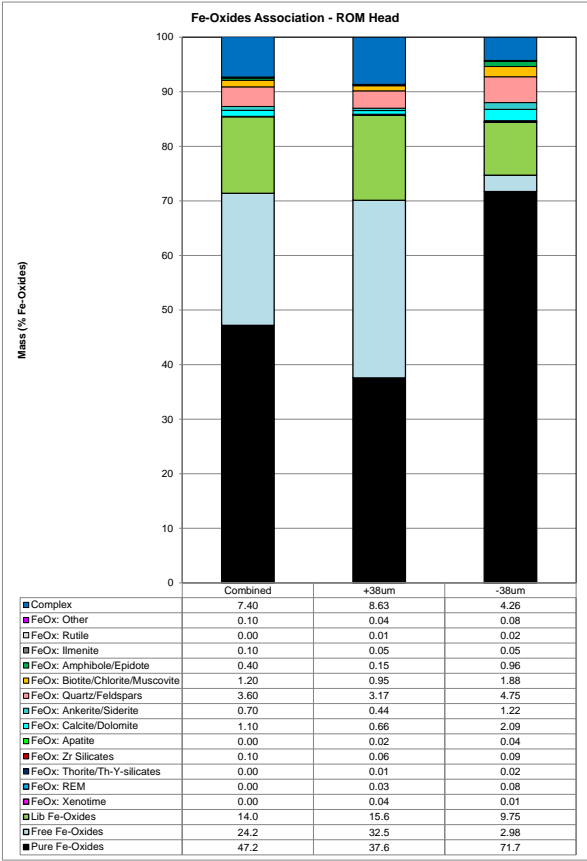
Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

Fe-Oxides Association



Absolute Mass of Fe-Oxides Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
Pure Fe-Oxides	2.88	1.64	1.24
Free Fe-Oxides	1.47	1.42	0.05
Lib Fe-Oxides	0.85	0.68	0.17
FeOx: Xenotime	0.00	0.00	0.00
FeOx: REM	0.00	0.00	0.00
FeOx: Thorite/Th-Y-silicates	0.00	0.00	0.00
FeOx: Zr Silicates	0.00	0.00	0.00
FeOx: Apatite	0.00	0.00	0.00
FeOx: Calcite/Dolomite	0.07	0.03	0.04
FeOx: Ankerite/Siderite	0.04	0.02	0.02
FeOx: Quartz/Feldspars	0.22	0.14	0.08
FeOx: Biotite/Chlorite/Muscovite	0.07	0.04	0.03
FeOx: Amphibole/Epidote	0.02	0.01	0.02
FeOx: Ilmenite	0.00	0.00	0.00
FeOx: Rutile	0.00	0.00	0.00
FeOx: Other	0.00	0.00	0.00
Complex	0.45	0.38	0.07
Total	6.10	4.37	1.73



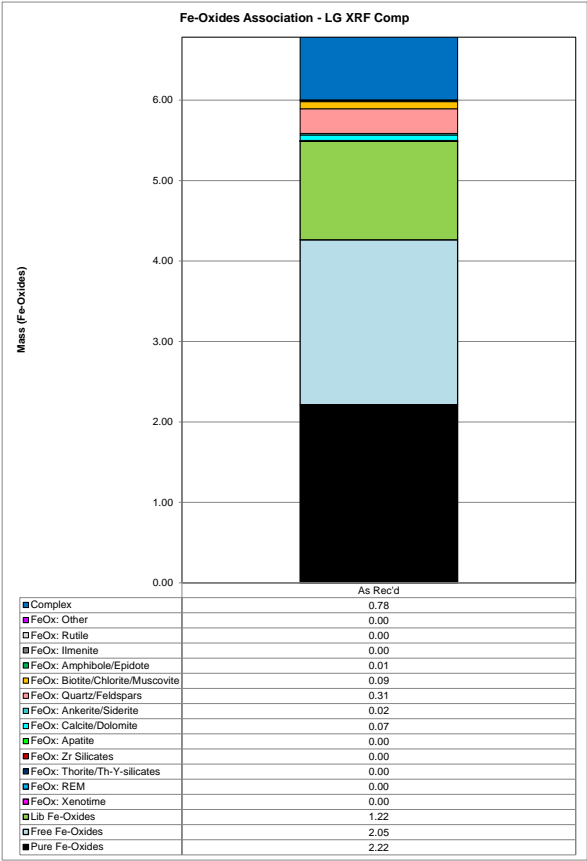
Normalized Mass of Fe-Oxides Across Sample ROM Head

Mineral Name	Combined	+38um	-38um
Pure Fe-Oxides	47.2	37.6	71.7
Free Fe-Oxides	24.2	32.5	2.98
Lib Fe-Oxides	14.0	15.6	9.75
FeOx: Xenotime	0.00	0.04	0.01
FeOx: REM	0.00	0.03	0.08
FeOx: Thorite/Th-Y-silicates	0.00	0.01	0.02
FeOx: Zr Silicates	0.10	0.06	0.09
FeOx: Apatite	0.00	0.02	0.04
FeOx: Calcite/Dolomite	1.10	0.66	2.09
FeOx: Ankerite/Siderite	0.70	0.44	1.22
FeOx: Quartz/Feldspars	3.60	3.17	4.75
FeOx: Biotite/Chlorite/Muscovite	1.20	0.95	1.88
FeOx: Amphibole/Epidote	0.40	0.15	0.96
FeOx: Ilmenite	0.10	0.05	0.05
FeOx: Rutile	0.00	0.01	0.02
FeOx: Other	0.10	0.04	0.08
Complex	7.40	8.63	4.26
Total	100.0	100.0	100.0

Liberated 85.4 85.7 84.5

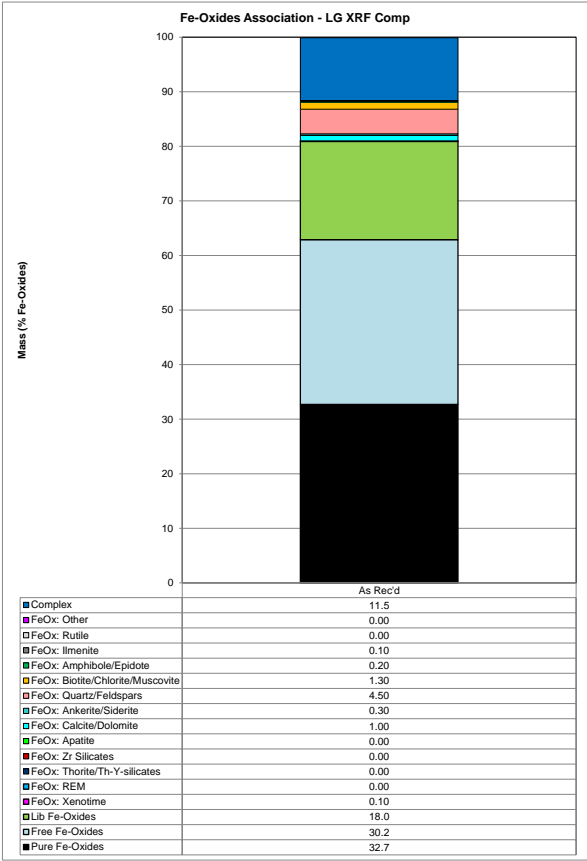
Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

Fe-Oxides Association



Absolute Mass of Fe-Oxides Across Sample LG XRF Comp

Mineral Name	As Rec'd
Pure Fe-Oxides	2.22
Free Fe-Oxides	2.05
Lib Fe-Oxides	1.22
FeOx: Xenotime	0.00
FeOx: REM	0.00
FeOx: Thorite/Th-Y-silicates	0.00
FeOx: Zr Silicates	0.00
FeOx: Apatite	0.00
FeOx: Calcite/Dolomite	0.07
FeOx: Ankerite/Siderite	0.02
FeOx: Quartz/Feldspars	0.31
FeOx: Biotite/Chlorite/Muscovite	0.09
FeOx: Amphibole/Epidote	0.01
FeOx: Ilmenite	0.00
FeOx: Rutile	0.00
FeOx: Other	0.00
Complex	0.78
Total	6.78



Normalized Mass of Fe-Oxides Across Sample LG XRF Comp

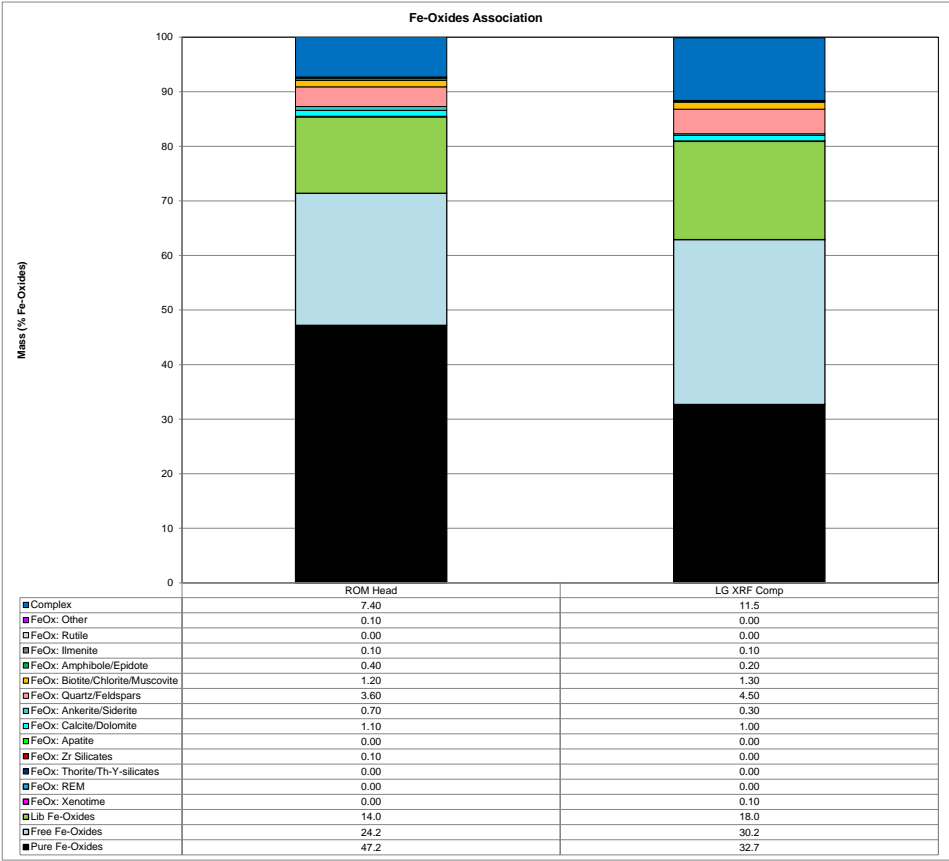
Mineral Name	As Rec'd
Pure Fe-Oxides	32.7
Free Fe-Oxides	30.2
Lib Fe-Oxides	18.0
FeOx: Xenotime	0.10
FeOx: REM	0.00
FeOx: Thorite/Th-Y-silicates	0.00
FeOx: Zr Silicates	0.00
FeOx: Apatite	0.00
FeOx: Calcite/Dolomite	1.00
FeOx: Ankerite/Siderite	0.30
FeOx: Quartz/Feldspars	4.50
FeOx: Biotite/Chlorite/Muscovite	1.30
FeOx: Amphibole/Epidote	0.20
FeOx: Ilmenite	0.10
FeOx: Rutile	0.00
FeOx: Other	0.00
Complex	11.5
Total	100.0

Liberated

80.9

Namibia
CALR-18299-01/03
M15029-APR22 & M15009-MAY22

Fe-Oxides Association



Normalized Mass of Fe-Oxides Across Samples

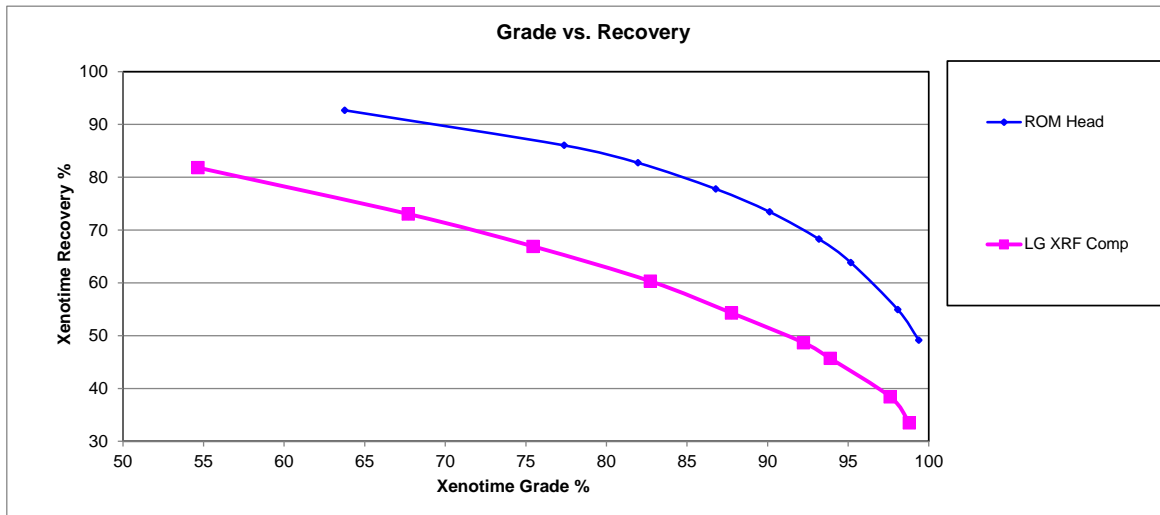
Mineral Name	ROM Head	LG XRF Comp
Pure Fe-Oxides	47.2	32.7
Free Fe-Oxides	24.2	30.2
Lib Fe-Oxides	14.0	18.0
FeOx: Xenotime	0.00	0.10
FeOx: REM	0.00	0.00
FeOx: Thorite/Th-Y-silicates	0.00	0.00
FeOx: Zr Silicates	0.10	0.00
FeOx: Apatite	0.00	0.00
FeOx: Calcite/Dolomite	1.10	1.00
FeOx: Ankerite/Siderite	0.70	0.30
FeOx: Quartz/Feldspars	3.60	4.50
FeOx: Biotite/Chlorite/Muscovite	1.20	1.30
FeOx: Amphibole/Epidote	0.40	0.20
FeOx: Ilmenite	0.10	0.10
FeOx: Rutile	0.00	0.00
FeOx: Other	0.10	0.00
Complex	7.40	11.5
Total	100.0	100.0

Liberated 85.4 80.9



Namibia
 CALR-18299-01/03
 MI5029-APR22 & MI5009-MAY22

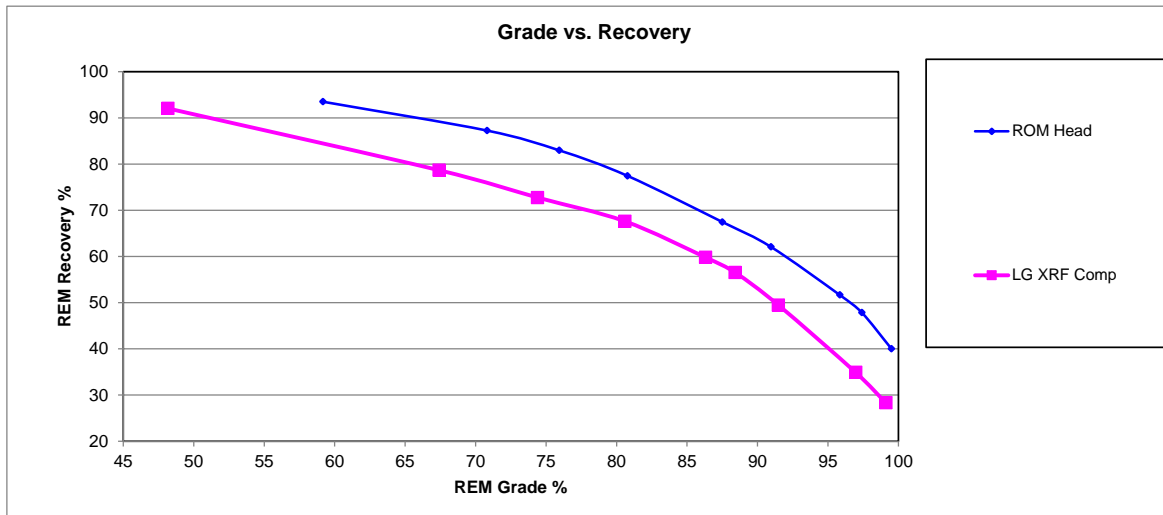
Xenotime Grade vs. Recovery:



Volume % of Xenotime/ Sample	ROM Head		LG XRF Comp	
	Grade	Recovery	Grade	Recovery
All particles	0.26	100.0	0.23	100.0
≥10	63.8	92.7	54.7	81.8
≥20	77.4	86.0	67.7	73.1
≥30	82.0	82.7	75.5	66.9
≥40	86.8	77.8	82.7	60.3
≥50	90.1	73.5	87.8	54.3
≥60	93.2	68.3	92.2	48.7
≥70	95.2	63.9	93.9	45.7
≥80	98.1	54.9	97.6	38.4
≥90	99.4	49.2	98.8	33.5

Namibia
 CALR-18299-01/03
 MI5029-APR22 & MI5009-MAY22

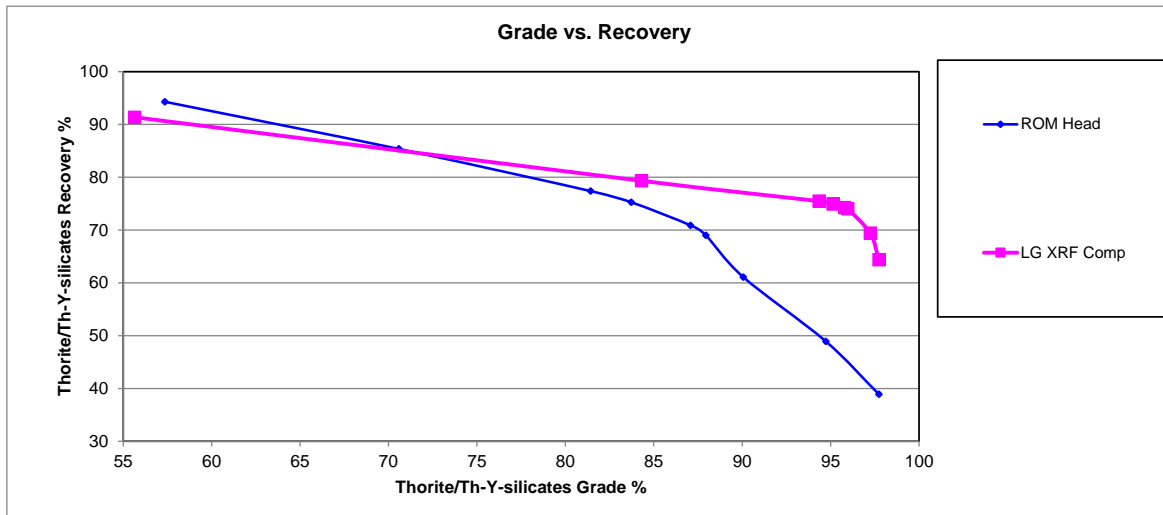
REM Grade vs. Recovery:



Volume % of REM / Sample	ROM Head		LG XRF Comp	
	Grade	Recovery	Grade	Recovery
All particles	0.13	100.0	0.13	100.0
≥10	59.2	93.5	48.2	92.1
≥20	70.8	87.3	67.4	78.7
≥30	75.9	83.0	74.4	72.8
≥40	80.8	77.5	80.6	67.6
≥50	87.5	67.5	86.3	59.8
≥60	91.0	62.1	88.4	56.6
≥70	95.8	51.7	91.5	49.5
≥80	97.4	47.9	97.0	34.9
≥90	99.5	40.0	99.1	28.4

Namibia
 CALR-18299-01/03
 MI5029-APR22 & MI5009-MAY22

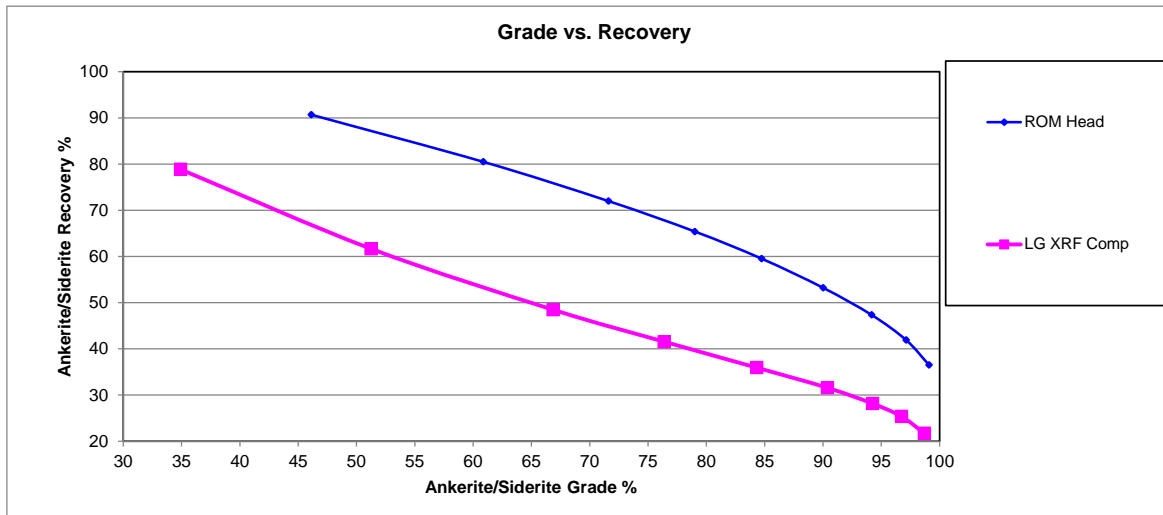
Thorite/Th-Y-silicates Grade vs. Recovery:



Volume % of Thorite/Th-Y- silicates / Sample	ROM Head		LG XRF Comp	
	Grade	Recovery	Grade	Recovery
All particles	0.02	100.0	0.01	100.0
≥10	57.4	94.3	55.6	91.4
≥20	70.6	85.4	84.3	79.3
≥30	81.4	77.4	94.3	75.5
≥40	83.7	75.3	95.2	75.0
≥50	87.1	70.9	95.2	75.0
≥60	87.9	69.0	95.8	74.3
≥70	90.1	61.1	96.0	74.0
≥80	94.7	48.9	97.2	69.4
≥90	97.7	38.9	97.7	64.4

Namibia
 CALR-18299-01/03
 MI5029-APR22 & MI5009-MAY22

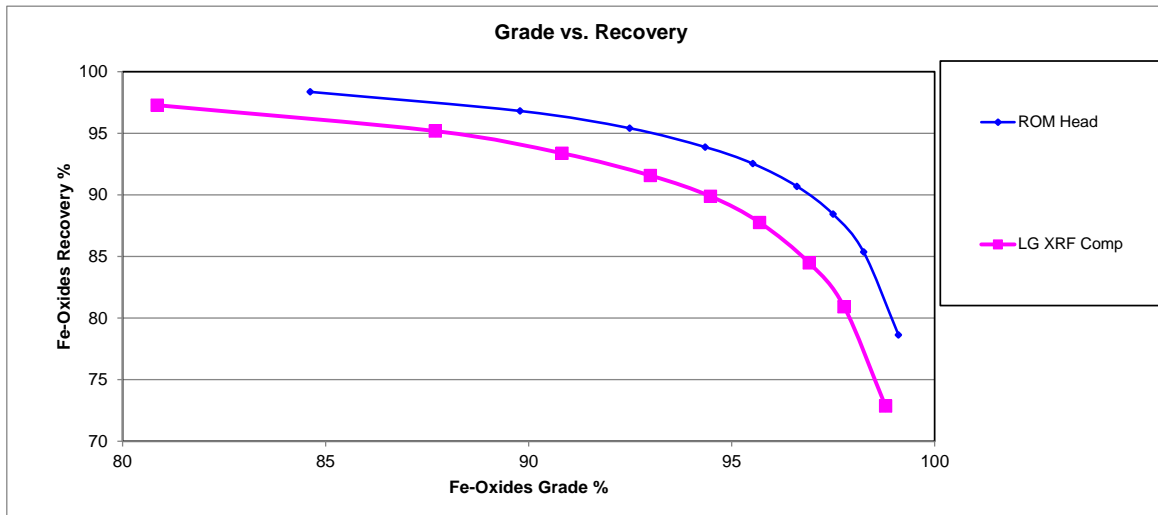
Ankerite/Siderite Grade vs. Recovery:



Volume % of Ankerite/Siderite / Sample	ROM Head		LG XRF Comp	
	Grade	Recovery	Grade	Recovery
All particles	1.80	100.0	1.19	100.0
≥10	46.1	90.7	34.9	78.9
≥20	60.9	80.5	51.2	61.7
≥30	71.6	72.0	66.8	48.5
≥40	79.0	65.4	76.4	41.6
≥50	84.7	59.6	84.3	35.9
≥60	90.0	53.2	90.4	31.6
≥70	94.2	47.4	94.2	28.2
≥80	97.1	41.9	96.7	25.4
≥90	99.1	36.5	98.7	21.7

Namibia
 CALR-18299-01/03
 MI5029-APR22 & MI5009-MAY22

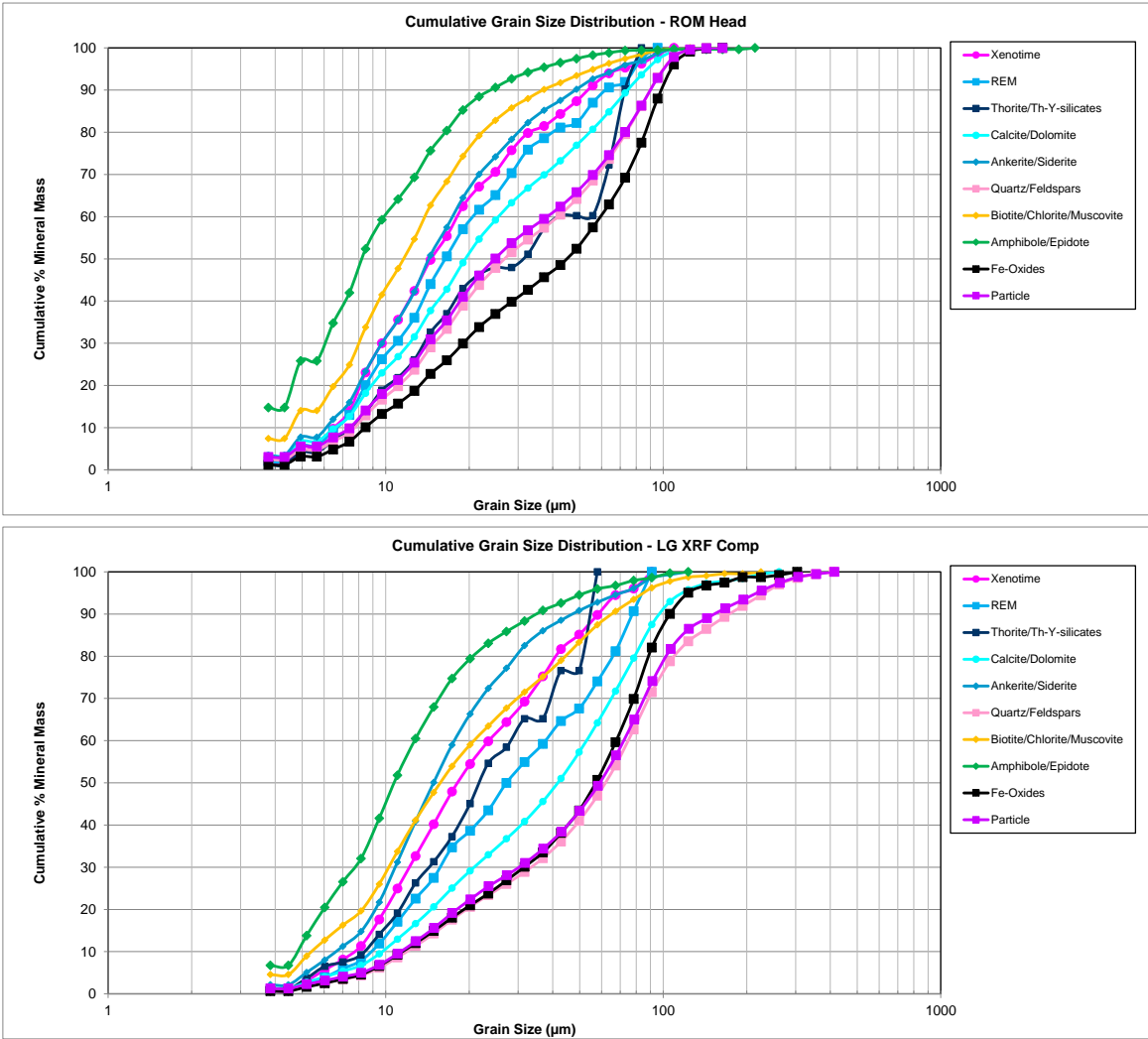
Fe-Oxides Grade vs. Recovery:



Volume % of Fe-Oxides / Sample	ROM Head		LG XRF Comp	
	Grade	Recovery	Grade	Recovery
All particles	6.10	100.0	6.78	100.0
≥10	84.6	98.4	80.9	97.3
≥20	89.8	96.8	87.7	95.2
≥30	92.5	95.4	90.8	93.4
≥40	94.4	93.9	93.0	91.6
≥50	95.5	92.5	94.5	89.9
≥60	96.6	90.7	95.7	87.8
≥70	97.5	88.4	96.9	84.5
≥80	98.3	85.4	97.8	80.9
≥90	99.1	78.6	98.8	72.9

Namibia
CALR-18299-01/03
MI5029-APR22 & MI5009-MAY22

Grain size distribution



Sample	Percentile (Mass % of phase) / Sample	Xenotime	REM	Thorite/ Th-Y- silicates	Calcite/ Dolomite	Ankerite/ Siderite	Quartz/ Feldspars	Biotite/ Chlorite/ Muscovite	Amphibole/ Epidote	Fe-Oxides	Particle
ROM Head	Median	14.8	16.3	29.5	19.5	14.4	26.7	11.3	8.31	45.1	24.7
	P80	34.2	40.1	70.9	54.4	30.0	73.6	22.5	16.3	86.4	72.8
LG XRF Comp	Median	18.0	27.5	20.6	41.8	14.8	62.0	15.9	10.7	57.3	59.1
	P80	42.0	64.4	55.6	78.9	29.4	109.4	44.5	20.7	89.1	102.2



Quantitative X-Ray Diffraction by Rietveld Refinement

Report Prepared for: Metallurgical Operations

Project Number/ LIMS No. 18299-03/MI5029-APR22

Sample Receipt: April 20, 2022

Sample Analysis: April 25, 2022

Reporting Date: Revised September 27, 2022

Instrument: BRUKER AXS D8 Advance Diffractometer

Test Conditions: Co radiation, 35 kV, 40 mA; Detector: LYNXEYE
Regular Scanning: Step: 0.02°, Step time: 0.75s, 2θ range: 6-80°

Interpretations : PDF2/PDF4 powder diffraction databases issued by the International Center for Diffraction Data (ICDD). DiffracPlus Eva and Topas software.

Detection Limit: 0.5-2%. Strongly dependent on crystallinity.

Contents:

- 1) Method Summary
- 2) Quantitative XRD Results
- 3) XRD Pattern(s)

Kim Gibbs, H.B.Sc., P.Geo.
Senior Mineralogist

Huiyun Zhou, Ph.D., P.Geo.
Senior Mineralogist

ACCREDITATION: SGS Natural Resources Lakefield is accredited to the requirements of ISO/IEC 17025 for specific tests as listed on our scope of accreditation, including geochemical, mineralogical and trade mineral tests. To view a list of the accredited methods, please visit the following website and search SGS Canada Inc. - Minerals: <https://www.scc.ca/en/search/palcan>.



Method Summary

The Rietveld Method of Mineral Identification by XRD (ME-LR-MIN-MET-MN-D05) method used by SGS Natural Resources is accredited to the requirements of ISO/IEC 17025.

Mineral Identification and Interpretation:

Mineral identification and interpretation involves matching the diffraction pattern of an unknown material to patterns of single-phase reference materials. The reference patterns are compiled by the Joint Committee on Powder Diffraction Standards - International Center for Diffraction Data (JCPDS-ICDD) database and released on software as Powder Diffraction Files (PDF).

Interpretations do not reflect the presence of non-crystalline and/or amorphous compounds, except when internal standards have been added by request. Mineral proportions may be strongly influenced by crystallinity, crystal structure and preferred orientations. Mineral or compound identification and quantitative analysis results should be accompanied by supporting chemical assay data or other additional tests.

Quantitative Rietveld Analysis:

Quantitative Rietveld Analysis is performed by using Topas 4.2 (Bruker AXS), a graphics based profile analysis program built around a non-linear least squares fitting system, to determine the amount of different phases present in a multicomponent sample. Whole pattern analyses are predicated by the fact that the X-ray diffraction pattern is a total sum of both instrumental and specimen factors. Unlike other peak intensity-based methods, the Rietveld method uses a least squares approach to refine a theoretical line profile until it matches the obtained experimental patterns.

Rietveld refinement is completed with a set of minerals specifically identified for the sample. Zero values indicate that the mineral was included in the refinement calculations, but the calculated concentration was less than 0.05wt%. Minerals not identified by the analyst are not included in refinement calculations for specific samples and are indicated with a dash.

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WARNING: The sample(s) to which the findings recorded herein (the "Findings") relate was(were) drawn and / or provided by the Client or by a third party acting at the Client's direction. The Findings constitute no warranty of the sample's representativeness of any goods and strictly relate to the sample(s). The Company accepts no liability with regard to the origin or source from which the sample(s) is/are said to be extracted.

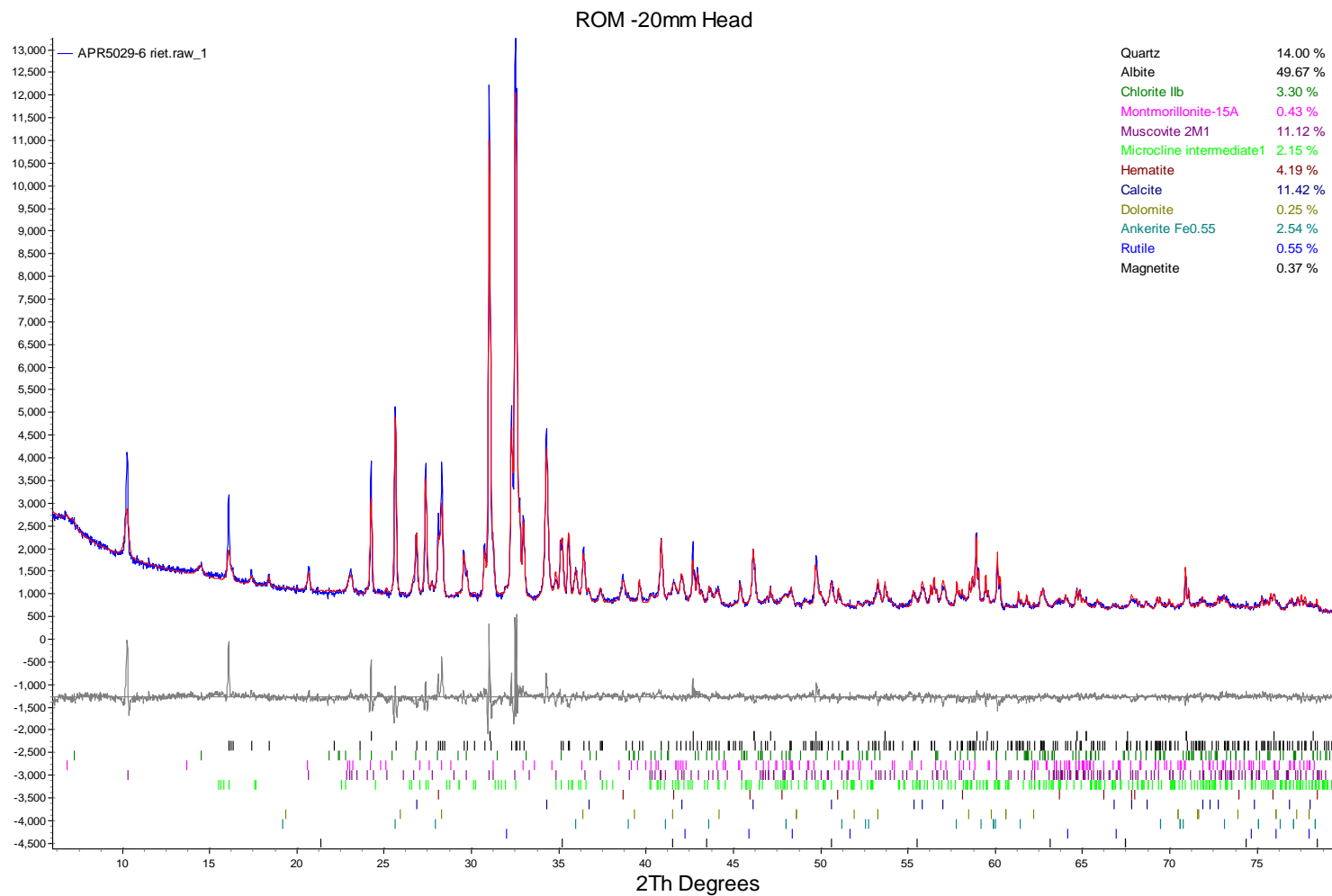
Summary of Rietveld Quantitative Analysis X-Ray Diffraction Results

Mineral/Compound	ROM -20mm Head APR5029-01 (wt %)
Quartz	14.0
Albite	49.7
Chlorite	3.3
Montmorillonite	0.4
Muscovite	11.1
Microcline	2.2
Hematite	4.2
Calcite	11.4
Dolomite	0.3
Ankerite	2.5
Rutile	0.6
Magnetite	0.4
TOTAL	100

The weight percent quantities indicated have been normalized to a sum of 100%.

The quantity of amorphous material has not been determined.

Mineral/Compound	Formula
Quartz	SiO ₂
Albite	NaAlSi ₃ O ₈
Chlorite	(Fe,(Mg,Mn) ₅ ,Al)(Si ₃ Al)O ₁₀ (OH) ₈
Montmorillonite	(Na,Ca) _{0.3} (Al,Mg) ₂ Si ₄ O ₁₀ (OH) ₂ ·10H ₂ O
Muscovite	KAl ₂ (AlSi ₃ O ₁₀)(OH) ₂
Microcline	KAlSi ₃ O ₈
Hematite	Fe ₂ O ₃
Calcite	CaCO ₃
Dolomite	CaMg(CO ₃) ₂
Ankerite	CaFe(CO ₃) ₂
Rutile	TiO ₂
Magnetite	Fe ₃ O ₄



Appendix B – Batch Flotation Testing

Test No.: F-101 Project No: 18299-03 Operator: Marteen Date: April 19,2022

Purpose: Based on old F58, test ROM sample

Sample: ROM

Feed: 2 kg of SG2 at -53 µm

Water DI Water Flot Fd K₈₀ = 42 µm

Notes:

Conditions:

Stage	Reagents Added, g/t								Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon		3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp °C	Pulp Density %
Strength	10%	5%		2%	100%			10%					
Pass through WHIMS at 2,000 Gauss (~2 Amps), the mags pass through the WHIMS as a cleaner													
Filter Non-Mags													
High Density Conditioning -50% @ 1000rpm													
Condition 1	250	200						100	3		7.5	49	50
Condition 2				1000	10				3		9.0	50	50
											8	50	
Rougher 1										3	8	53	35.0
Rougher 2				300	20				2	5	8.2	53	
Rougher 3				300	20				2	5	8.2	50	
Rougher 4				100	20				2	5	8.3	52	
Rougher 5				100	20				2	5	8.2	52	
Rougher 6 (Assay)					10				1	2	8.5	50	
1st Cleaner Stage: Combine Ro 1 to Ro 5 as 1st Cl feed													
Condition 3	100							75	1		8.6		
											9.5	51	
											9.5	52	
1st Cleaner A							12.5			3	9.5	52	-5
1st Cleaner B				25					1	2	9.3	52	
1st Cleaner C				35	10				1	1	9.1	49	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed													
Condition 4	25	20							1		9	50	
											3	9	50
2nd Cleaner A										3	9	50	
2nd Cleaner B				25		10			1	5	9	49	
Ro Total	375	220	0	1885	110	10		175	20	39	8.3	51.3	

Stage	Rougher	1st/2nd Clnr
Flotation Cell	4L (2kg flot cell)	4L (2kg flot cell)
Speed rpm	1600	1500

Lime: 4 in Ro Tail
4.5 in 1st Clnr Tail
4.59 in 2nd Clnr Tail

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
F1 2nd Cl Conc A	19	0.9	2.75	6.30	3.9	10.8	46.6	14.5	0.67	2.12	23.5	23.5	0.3	0.2	6.2	22.6	0.1	0.2
F1 2nd Cl Conc B	10	0.5	1.86	4.26	5.8	17.1	39.9	12.5	1.37	3.50	8.4	8.4	0.3	0.2	2.8	10.4	0.1	0.1
F1 2nd Cl Tail	26	1.2	0.58	1.33	13.7	29.0	23.3	2.8	3.22	7.82	6.8	6.8	1.7	0.7	4.3	5.9	0.7	0.8
F1 1st Cl Tail	161	7.8	0.23	0.53	10.9	41.9	12.2	1.11	4.95	11.8	16.8	16.8	8.2	6.7	14.0	14.8	6.7	7.2
F1 Ro Conc 6	22	1.1	0.21	0.48	9.3	41.5	15.5	2.6	4.83	10.9	2.1	2.1	1.0	0.9	2.4	4.7	0.9	0.9
F1 Ro Tail	1806	87.0	0.05	0.11	10.4	50.9	4.13	0.27	6.03	13.3	41.0	41.0	88.2	90.8	53.2	40.4	91.1	90.5
F1 Mag @ 2A	32	1.5	0.10	0.23	2.5	15.0	74.2	0.5	1.44	3.64	1.5	1.5	0.4	0.5	17.0	1.4	0.4	0.4
Head (Calc.)	2076	100.0	0.11	0.24	10.3	48.7	6.75	0.58	5.76	12.8	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

Cumulative Products

Products	Time,	Weight	Assay, %									Distribution, %							
			%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
2nd Cl Con 1	3	0.9	2.75	6.30	3.9	10.8	46.6	14.5	0.7	2.1	23.5	23.5	0.3	0.2	6.2	22.6	0.1	0.2	
2nd Cl Con 1-2	8	1.4	2.44	5.59	4.6	13.0	44.3	13.8	0.9	2.6	31.9	31.9	0.6	0.4	9.1	32.9	0.2	0.3	
1st Cl Con 1-3	6	2.6	1.56	3.58	8.9	20.5	34.4	8.6	2.0	5.1	38.7	38.7	2.3	1.1	13.4	38.8	0.9	1.0	
Ro Conc 1-5	23	10.4	0.57	1.30	10.4	36.5	17.8	3.0	4.2	10.1	55.5	55.5	10.5	7.8	27.4	53.6	7.6	8.2	
Ro Conc 1-6	25	11.4	0.53	1.22	10.3	37.0	17.6	3.0	4.3	10.2	57.6	57.6	11.5	8.7	29.8	58.3	8.5	9.1	
Flotation Feed		98.5	0.11	0.24	10.4	49.3	5.7	0.6	5.8	12.9	98.5	98.5	99.6	99.5	83.0	98.6	99.6	99.6	
Head (Calc.)		100.0	0.11	0.24	10.3	48.7	6.75	0.58	5.76	12.8	100	100	100	100	100	100	100	100	
Head (Direct)			0.10	0.22	9.9	49.7	6.44	0.58	5.92	12.9									

Test No.: F-102

Project No: 18299-03 Operator: Marteen Date: April 27,2022

Purpose: Based on F1, with half Calgon

Sample: ROM

Feed: 2 kg of SG2 at -53 µm

Water DI Water Flot Fd $K_{80} = 42 \mu\text{m}$

Notes:

Conditions:

Stage	Reagents Added, g/t								Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon		3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp	Pulp Density
Strength	10%	5%		2%	100%			10%				°C	%
Pass through WHIMS at 2,000 Gauss (~2 Amps), the mags pass through the WHIMS as a cleaner													
Filter Non-Mags													
High Density Conditioning ~50% @ 1000rpm													
Condition 1	250	100						100	3		7.5	51	50
Condition 2				1000	20				3		8.0	52	
Rougher 1						2.5				3	8.0	54	35.0
Rougher 2				300	20				2	5	8.1	52	
Rougher 3				300	20				2	5	8.2	52	
Rougher 4				200	20				2	5	8.2	51	
Rougher 5 (Assay)				500	20				1	2	8.3	52	
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed													
Condition 3	100							150	1		8.6		
1st Cleaner A				50			12.5		1	3	9.5	54	
1st Cleaner B				50		5			1	2	9.5	52	
1st Cleaner C				50	10	5			1	2	9.2	51	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed													
Condition 4	25	10							1		8.9	51	
2nd Cleaner A				25						3	8.9	52	
2nd Cleaner B				25		10			1	5	8.8	52	
Ro Total	375	110	0	2500	110	22.5		250	19	35	8.3	52.1	

Ro 3 and 4 lighter orangy bubbles

used forced air in cleaners

Comment:				Lime:	7.34 in Ro Tail 5.31 in 1st Clnr Tail 4.59 in 2nd Clnr Tail
Stage	Rougher	1st Clnr	2nd Clnr		
Flotation Cell	4L (2kg flot cell)	4L (2kg flot cell)	2L (1kg flot cell)		
Speed rpm	1600	1500	1300		

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
F2 2nd Cl Conc - A	21	1.0	3.65	8.36	5.1	13.1	39.5	14.5	0.80	2.43	35.6	35.6	0.5	0.3	5.7	24.8	0.1	0.2
F2 2nd Cl Conc - B	9	0.4	3.19	7.31	6.0	16.5	36.9	13.7	1.11	2.96	12.9	12.9	0.2	0.1	2.2	9.7	0.1	0.1
F2 2nd Cl Tail	23	1.1	1.35	3.09	13.6	24.1	27.8	4.8	2.42	5.90	14.6	14.6	1.5	0.5	4.5	9.0	0.5	0.5
F2 1st Cl Tail	173	8.3	0.19	0.44	11.4	42.6	12.0	1.01	5.05	11.7	15.5	15.5	9.2	7.2	14.5	14.4	7.2	7.5
F2 Ro Conc 5	44	2.1	0.11	0.25	15.9	37.1	9.4	0.6	4.35	10.2	2.3	2.3	3.2	1.6	2.9	2.2	1.6	1.7
F2 Ro Tail	1781	85.4	0.02	0.05	10.3	52.0	4.06	0.26	6.12	13.5	16.8	16.8	85.1	89.9	50.5	38.2	90.1	89.6
F2 Mags	36	1.7	0.13	0.30	1.8	13.5	77.9	0.5	1.35	3.01	2.2	2.2	0.3	0.5	19.7	1.6	0.4	0.4
Head (Calc.)	2086	100.0	0.10	0.23	10.3	49.4	6.86	0.58	5.80	12.9	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

Cumulative Products

Products	Time.	Weight		Assay, %							Distribution, %							
		%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
2nd Cl Con 1	3	1.0	3.65	8.36	5.1	13.1	39.5	14.5	0.8	2.4	35.6	35.6	0.5	0.3	5.7	24.8	0.1	0.2
2nd Cl Con 1-2	8	1.4	3.51	8.05	5.4	14.1	38.7	14.3	0.9	2.6	48.6	48.6	0.7	0.4	7.9	34.5	0.2	0.3
1st Cl Con 1-3	7	2.5	2.56	5.87	9.0	18.5	33.9	10.1	1.6	4.0	63.2	63.2	2.2	0.9	12.4	43.5	0.7	0.8
Ro Conc 1-4	18	10.8	0.74	1.70	10.8	37.0	17.1	3.1	4.2	9.9	78.7	78.7	11.3	8.1	26.9	57.9	7.9	8.3
Ro Conc 1-5	20	12.9	0.64	1.46	11.7	37.0	15.8	2.7	4.3	10.0	81.0	81.0	14.6	9.7	29.8	60.2	9.5	10.0
Flotation Feed		98.3	0.10	0.23	10.5	50.0	5.6	0.6	5.9	13.0	97.8	97.8	99.7	99.5	80.3	98.4	99.6	99.6
Head (Calc.)		100.0	0.10	0.23	10.3	49.4	6.86	0.58	5.80	12.9	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.9	49.7	6.44	0.58	5.92	12.9								

Test No.: F-103

Project No: 18299-03 Operator: Marteen Date: April 27,2022

Purpose: Based on F2, with half Calgon and try SM-15 as collector

Sample: ROM

Feed: 2 kg of SG2 at -53 µm

Water DI Water Flot Fd $K_{80} = 42 \mu\text{m}$

Notes:

Conditions:

Stage	Reagents Added, g/t								Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon		SM-15		Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp °C	Pulp Density %
Strength	10%	5%		100%				10%					
Pass through WHIMS at 2,000 Gauss (~2 Amps), the mags pass through the WHIMS as a cleaner													
Filter Non-Mags													
High Density Conditioning -50% @ 1000rpm													
Condition 1	250	100						100	3		9.0	53	50
Condition 2				300					3		8.5	53	
		100											
Rougher 1						5				3	8.5	53	35.0
Rougher 2				100		20			2	5	8.4	53	
Rougher 3				100					2	5	8.3	52	
Rougher 4				100		10			2	5	8.3	53	
Rougher 5 (Assay)				100		20			2	2	8.2	53	
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed													
Condition 3	100	50						75	1		8.6		
											9.5	53	
1st Cleaner A				25			12.5		1	3	9.5	52	~15
1st Cleaner B				25					1	2	9.3	53	
1st Cleaner C				25					1	2	9.1	52	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed													
Condition 4	25	25							1		8.9	51	
2nd Cleaner A				10		10				3	8.9	51	
2nd Cleaner B				0					1	5	8.9	50	
Ro Total	375	275	0	785	0	65		175	20	35	8.5	52.9	

Comment:

Stage	Rougher	1st Clnr	2nd Clnr
Flotation Cell	4L (2kg flot cell)	4L (2kg flot cell)	2L (1kg flot cell)
Speed rpm	1600	1500	1500

Lime: 3.91 in Ro Tail
4.54 in 1st Clnr Tail
in 2nd Clnr Tail

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
F3 2nd Cl Conc - A	143	6.9	0.31	0.71	42.0	2.6	6.2	0.4	0.22	0.68	21.4	21.4	28.4	0.4	6.5	4.9	0.3	0.4
F3 2nd Cl Conc - B	153	7.4	0.33	0.76	40.8	4.0	6.7	0.6	0.34	1.01	24.2	24.2	29.4	0.6	7.4	7.8	0.4	0.6
F3 2nd Cl Tail	155	7.5	0.32	0.73	31.6	15.9	8.7	1.2	1.55	4.06	23.9	23.9	23.1	2.4	9.8	16.1	2.0	2.4
F3 1st Cl Tail	231	11.2	0.11	0.25	7.3	51.1	8.2	0.78	5.74	14.4	12.2	12.2	7.9	11.5	13.8	15.1	11.0	12.5
F3 Ro Conc 5	56	2.7	0.14	0.32	8.8	47.6	9.9	1.7	5.16	12.7	3.8	3.8	2.3	2.6	4.1	7.9	2.4	2.7
F3 Ro Tail	1299	62.9	0.02	0.05	1.4	65.0	4.54	0.43	7.77	16.6	12.5	12.5	8.6	82.2	43.0	46.8	83.6	81.2
F3 Mags	28	1.4	0.15	0.34	2.0	14.5	75.4	0.6	1.44	3.24	2.0	2.0	0.3	0.4	15.4	1.5	0.3	0.3
Head (Calc.)	2065	100.0	0.10	0.23	10.3	49.8	6.65	0.58	5.85	12.9	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

Cumulative Products

Products	Time.	Weight		Assay, %								Distribution, %							
		%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	
2nd Cl Con 1	3	6.9	0.31	0.71	42.0	2.6	6.2	0.4	0.2	0.7	21.4	21.4	28.4	0.4	6.5	4.9	0.3	0.4	
2nd Cl Con 1-2	8	14.3	0.32	0.73	41.4	3.3	6.5	0.5	0.3	0.9	45.6	45.6	57.8	1.0	14.0	12.7	0.7	0.9	
1st Cl Con 1-3	7	21.8	0.32	0.73	38.0	7.7	7.2	0.8	0.7	2.0	69.5	69.5	80.9	3.4	23.8	28.8	2.7	3.3	
Ro Conc 1-4	18	33.0	0.25	0.57	27.6	22.4	7.6	0.8	2.4	6.2	81.7	81.7	88.8	14.8	37.6	43.9	13.6	15.8	
Ro Conc 1-5	20	35.7	0.24	0.55	26.2	24.3	7.7	0.8	2.6	6.7	85.5	85.5	91.2	17.4	41.6	51.8	16.0	18.5	
Flotation Feed		98.6	0.10	0.23	10.4	50.3	5.7	0.6	5.9	13.0	98.0	98.0	99.7	99.6	84.6	98.5	99.7	99.7	
Head (Calc.)		100.0	0.10	0.23	10.3	49.8	6.65	0.58	5.85	12.9	100	100	100	100	100	100	100	100	
Head (Direct)			0.10	0.22	9.9	49.7	6.44	0.58	5.92	12.9									

Test No.: F-104

Project No: 18299-03 Operator: Marteen Date: May 02,2022

Purpose: Based on F102, with higher collector dosage in Ro, **natural pH in CI**

Sample: ROM

Feed: 2 kg of SG3 at -75 µm

Water DI Water Flot Fd $K_{80} =$ 42 µm

Notes:

Conditions:

Stage	Reagents Added, g/t								Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon		3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp	Pulp Density
Strength	10%	5%		2%	100%			10%				°C	%
Pass through WHIMS at 2,000 Gauss (~2 Amps), the mags pass through the WHIMS as a cleaner													
Filter Non-Mags													
High Density Conditioning ~50% @ 1000rpm													
Condition 1	250	100						100	3		9.0	52	50
Condition 2				2000	20				5		9.0	52	
Rougher 1						2.5				3	8.5	53	35.0
Rougher 2				300	20				2	5	8.4	53	
Rougher 3				300	20				2	5	8.3	52	
Rougher 4				200	20				2	5	8.2	52	
Rougher 5 (Assay)				500	20				2	2	8.3	53	
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st CI feed													
Condition 3	50								1		natural	54	
1st Cleaner A				50			12.5		1	3	8.4	52	~7
1st Cleaner B				50		5			1	2	8.3	49	
1st Cleaner C				50	10	5			1	1	8.3	50	
2nd Cleaner Stage: Combine 1st CI A-C as 2nd CI feed													
Condition 4	25	10							1		natural	51	
2nd Cleaner A				25					1	3	8.2	52	
2nd Cleaner B				25		10			1	4	8.2	52	
Ro Total	325	110	0	3500	110	22.5		100	23	33	8.5	52.4	

thick sticky froth at beginning of float

Comment:

Stage	Rougher	1st Clnr	2nd Clnr
Flotation Cell	4L (2kg flot cell)	3L (750 g)	1L (500 g)??
Speed rpm	1600		

Lime: 7.34 in Ro Tail
4.15 in 1st Clnr Tail
3.87 in 2nd Clnr Tail**Metallurgical Balance**

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
F104 2nd CI Conc - A	21	1.1	3.57	8.18	5.9	13.8	39.3	12.6	0.91	2.48	36.4	36.4	0.6	0.3	6.4	23.6	0.2	0.2
F104 2nd CI Conc - B	11	0.5	2.82	6.46	7.6	16.9	35.8	11.3	1.26	3.08	14.6	14.6	0.4	0.2	3.0	10.7	0.1	0.1
F104 2nd CI Tail	31	1.6	0.78	1.79	17.2	24.7	21.7	3.0	2.74	6.23	11.7	11.7	2.7	0.8	5.2	8.3	0.7	0.8
F104 1st CI Tail	166	8.3	0.11	0.25	13.8	39.7	10.5	0.79	4.84	11.0	8.7	8.7	11.3	6.7	13.4	11.5	6.7	7.2
F104 Ro Conc 5	36	1.8	0.13	0.30	17.7	33.4	9.9	0.9	3.97	9.0	2.2	2.2	3.2	1.2	2.8	2.9	1.2	1.3
F104 Ro Tail	1696	85.3	0.03	0.07	9.7	52.4	4.07	0.28	6.39	13.5	24.4	24.4	81.6	90.4	53.0	41.8	90.8	90.1
F104 Mags @2A	27	1.4	0.15	0.34	1.9	14.0	77.1	0.5	1.39	3.12	2.0	2.0	0.3	0.4	16.2	1.3	0.3	0.3
Head (Calc.)	1989	100.0	0.11	0.24	10.2	49.4	6.55	0.57	6.00	12.8	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

Cumulative Products

Products	Time.	Weight		Assay, %								Distribution, %							
		%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	
2nd Cl Con 1	3	1.1	3.57	8.18	5.9	13.8	39.3	12.6	0.9	2.5	36.4	36.4	0.6	0.3	6.4	23.6	0.2	0.2	
2nd Cl Con 1-2	8	1.6	3.32	7.60	6.5	14.8	38.1	12.2	1.0	2.7	51.0	51.0	1.0	0.5	9.4	34.3	0.3	0.3	
1st Cl Con 1-3	7	3.2	2.06	4.72	11.8	19.7	30.0	7.6	1.9	4.4	62.7	62.7	3.7	1.3	14.6	42.6	1.0	1.1	
Ro Conc 1-4	18	11.5	0.65	1.49	13.2	34.2	15.9	2.7	4.0	9.2	71.4	71.4	15.0	8.0	28.0	54.1	7.7	8.3	
Ro Conc 1-5	20	13.3	0.58	1.33	13.8	34.1	15.1	2.4	4.0	9.2	73.7	73.7	18.2	9.2	30.8	57.0	8.9	9.6	
Flotation Feed		98.6	0.10	0.24	10.3	49.9	5.6	0.6	6.1	12.9	98.0	98.0	99.7	99.6	83.8	98.7	99.7	99.7	
Head (Calc.)		100.0	0.11	0.24	10.2	49.4	6.55	0.57	6.00	12.8	100	100	100	100	100	100	100	100	
Head (Direct)			0.10	0.22	9.9	49.7	6.44	0.58	5.92	12.9									

Test No.: F-105

Project No: 18299-03 Operator: Marteen Date: May 12,2022

target 2nd Cl Con A 40 g wet

Purpose: Based on F102, with half Calgon

Sample: ROM

Feed: 2 kg of SG2 at -53 µm

Water DI Water Flot Fd $K_{80} = 42 \mu\text{m}$

Notes:

Conditions:

Stage	Reagents Added, g/t								Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon		3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp	Pulp Density
Strength	10%	5%		2%	100%			10%				°C	%
Pass through WHIMS at 2,000 Gauss (~2 Amps), the mags pass through the WHIMS as a cleaner													
Filter Non-Mags													
High Density Conditioning -50% @ 1000rpm													
Condition 1	250	50						50	3		9.0	54	50
Condition 2				1000	20				3		8.0	53	
Rougher 1						2.5				3	8.0	54	35.0
Rougher 2				300	20				2	5	8.1	53	
Rougher 3				300	20				2	5	8	53	
Rougher 4				200	20				2	5	8.2	53	
Rougher 5 (Assay)				500	20				1	2	8	52	
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed													
Condition 3	50								1		natural	52	
1st Cleaner A				50			12.5		1	3	8.3	50	~5
1st Cleaner B				50		5			1	2	8.2	50	
1st Cleaner C				50	10	5			1	1	8.1	50	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed													
Condition 4	25								1		natural	53	
2nd Cleaner A				25						3	8.2	53	
2nd Cleaner B				25		10			1	4	8.3	53	
Ro Total	325	50	0	2500	110	22.5		50	19	33	8.2	53.1	

More foamy but lighter bubbles towards end. f

wet wts:
 2nd Cl Con A 81.4g
 2nd Cl Con B 41.8g
 2nd Cl Tail: 88.8g
 1st Cl Tail 442g
 Ro Con 5: 142g

Stage	Rougher	1st Clnr	2nd Clnr	Lime: 5.02 in Ro Tail 2.17 in 1st Clnr Tail 2.45 in 2nd Clnr Tail
Flotation Cell	4L (2kg flot cell)	3L (750 g)	1L (500 g)	
Speed rpm	1600	1350	1200	
same as F104				

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
F105 2nd Cl Conc A	62	3.0	2.13	4.88	17.4	10.8	27.1	7.4	0.84	2.12	65.9	65.9	5.1	0.7	13.4	39.0	0.4	0.5
F105 2nd Cl Conc B	23	1.1	0.94	2.15	20.3	14.0	22.1	4.8	1.29	2.85	10.7	10.7	2.2	0.3	4.0	9.3	0.2	0.2
F105 2nd Cl Tail	51	2.5	0.16	0.37	21.5	24.5	14.4	1.2	2.97	6.56	4.1	4.1	5.2	1.2	5.9	5.2	1.2	1.3
F105 1st Cl Tail	314	15.3	0.05	0.11	16.8	39.1	7.6	0.61	4.82	10.8	7.8	7.8	25.1	12.2	19.0	16.3	12.3	12.8
F105 Ro Conc 5	104	5.1	0.06	0.14	24.7	27.9	7.0	0.5	3.34	7.5	3.1	3.1	12.2	2.9	5.8	4.2	2.8	3.0
F105 Ro Tail	1480	72.2	<0.01	0.02	7.1	56.1	3.62	0.20	6.87	14.6	7.4	7.4	50.0	82.4	42.8	25.2	82.7	82.0
F105 Mags @2A	16	0.8	0.12	0.27	2.3	17.1	70.1	0.7	1.72	3.82	1.0	1.0	0.2	0.3	9.0	1.0	0.2	0.2
Head (Calc.)	2050	100.0	0.10	0.22	10.3	49.1	6.11	0.57	5.99	12.9	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

Cumulative Products

Products	Time	Weight		Assay, %								Distribution, %						
		%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
2nd Cl Con 1	3	3.0	2.13	4.88	17.4	10.8	27.1	7.4	0.8	2.1	65.9	65.9	5.1	0.7	13.4	39.0	0.4	0.5
2nd Cl Con 1-2	7	4.1	1.81	4.14	18.2	11.7	25.8	6.7	1.0	2.3	76.7	76.7	7.3	1.0	17.5	48.2	0.7	0.7
1st Cl Con 1-3	6	6.6	1.19	2.73	19.4	16.5	21.5	4.6	1.7	3.9	80.7	80.7	12.6	2.2	23.3	53.4	1.9	2.0
Ro Conc 1-4	18	21.9	0.40	0.90	17.6	32.3	11.8	1.8	3.9	8.7	88.6	88.6	37.6	14.4	42.4	69.7	14.2	14.9
Ro Conc 1-5	20	27.0	0.33	0.76	18.9	31.4	10.9	1.6	3.8	8.5	91.7	91.7	49.8	17.3	48.2	73.8	17.0	17.8
Flotation Feed		99.2	0.10	0.22	10.3	49.4	5.6	0.6	6.0	12.9	99.0	99.0	99.8	99.7	91.0	99.0	99.8	99.8
Head (Calc.)		100.0	0.10	0.22	10.3	49.1	6.11	0.57	5.99	12.9	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.9	49.7	6.44	0.58	5.92	12.9								

Test No.: F-106

Project No: 18299-03

Operator: Marteen

Date: May 12,2022

target 2nd Cl Con A

40 g wet

Purpose: Based on F102/104, with coarser size, ~75 um

Sample: ROM

Feed: 2 kg of SG3 at -75 µm

Water: DI Water

Flot Fd $K_{80} = 52 \mu\text{m}$

Notes:

Conditions:

Stage	Reagents Added, g/t								Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon		3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp °C	Pulp Density %
Strength	10%	5%		2%	100%			10%					
Pass through WHIMS at 2,000 Gauss (~2 Amps), the mags pass through the WHIMS as a cleaner													
Filter Non-Mags													
High Density Conditioning ~50% @ 1000rpm													
Condition 1	250	100						50	3		9.0	53	50
Condition 2				1000	20				3		8.0	53	
Rougher 1						2.5				3	8.0	54	35.0
Rougher 2				300	20				2	5	8.1	53	
Rougher 3				300	20				2	5	8.2	52	
Rougher 4				200	20				2	5	8.2	53	
Rougher 5 (Assay)				500	20				1	2	8.3	52	
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed													
Condition 3	50								1		natural	52	
1st Cleaner A				50			12.5		1	3	8.3	50	~5
1st Cleaner B				50		5			1	2	8.2	50	
1st Cleaner C				50	10	5			1+1	1	8.1	50	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed													
Condition 4	25								1		natural	53	
2nd Cleaner A				25						3	8.2	53	
2nd Cleaner B				25		10			1	4	8.3	53	
Ro Total	325	100	0	2500	110	22.5		50	18	33	8.3	52.9	

wet wts:
 2nd Cl Con A 56g
 2nd Cl Con 27g
 2nd Cl Tail: 65.9g
 1st Cl Tail 223g
 Ro Con 5: 102g

Comment:				Lime:	4.77 in Ro Tail 3.1 in 1st Clnr Tail 2.61 in 2nd Clnr Tail
Stage	Rougher	1st Clnr	2nd Clnr		
Flotation Cell	4L (2kg flot cell)	3L (750 g)	1L (500 g)		
Speed rpm	1600	1350	1200		
same as F104				same as F104	

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
F106 2nd Cl Conc A	39	1.9	2.79	6.39	6.6	20.8	33.6	10.5	1.70	3.80	52.0	52.0	1.3	0.8	9.1	35.3	0.6	0.6
F106 2nd Cl Conc B	12	0.6	1.30	2.98	8.6	33.3	23.9	6.1	3.37	6.51	7.4	7.4	0.5	0.4	2.0	6.3	0.3	0.3
F106 2nd Cl Tail	35	1.7	0.27	0.62	11.7	39.3	15.8	1.5	4.95	9.93	4.5	4.5	2.0	1.4	3.8	4.5	1.4	1.4
F106 1st Cl Tail	148	7.4	0.05	0.11	11.4	44.2	9.0	0.48	5.47	12.8	3.6	3.6	8.3	6.7	9.3	6.2	6.8	7.5
F106 Ro Conc 5	69	3.4	0.18	0.41	17.1	34.1	9.4	0.9	4.13	9.2	5.9	5.9	5.8	2.4	4.5	5.6	2.4	2.5
F106 Ro Tail	1650	82.5	0.03	0.07	10.0	51.8	4.00	0.28	6.34	13.4	23.8	23.8	81.5	87.6	46.2	40.1	87.9	87.1
F106 Mags @2A	48	2.4	0.12	0.27	2.4	14.6	74.9	0.5	1.42	3.53	2.8	2.8	0.6	0.7	25.0	2.0	0.6	0.7
Head (Calc.)	1999	100.0	0.10	0.24	10.1	48.8	7.14	0.58	5.95	12.7	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

Cumulative Products

Products	Time.	Weight		Assay, %								Distribution, %							
		%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	
2nd Cl Con 1	3	1.9	2.79	6.39	6.6	20.8	33.6	10.5	1.7	3.8	52.0	52.0	1.3	0.8	9.1	35.3	0.6	0.6	
2nd Cl Con 1-2	7	2.5	2.44	5.59	7.1	23.7	31.3	9.5	2.1	4.4	59.4	59.4	1.8	1.2	11.1	41.6	0.9	0.9	
1st Cl Con 1-3	6	4.3	1.56	3.56	9.0	30.1	25.0	6.2	3.3	6.7	63.9	63.9	3.8	2.6	14.9	46.1	2.3	2.2	
Ro Conc 1-4	18	11.7	0.60	1.38	10.5	39.0	14.9	2.6	4.7	10.6	67.5	67.5	12.1	9.3	24.3	52.3	9.1	9.7	
Ro Conc 1-5	20	15.1	0.51	1.16	12.0	37.9	13.6	2.2	4.5	10.3	73.4	73.4	17.9	11.7	28.8	57.9	11.5	12.2	
Flotation Feed		97.6	0.10	0.24	10.3	49.7	5.5	0.6	6.1	12.9	97.2	97.2	99.4	99.3	75.0	98.0	99.4	99.3	
Head (Calc.)		100.0	0.10	0.24	10.1	48.8	7.14	0.58	5.95	12.7	100	100	100	100	100	100	100	100	
Head (Direct)			0.10	0.22	9.9	49.7	6.44	0.58	5.92	12.9									

Test No.: F-107

Project No: 18299-03 Operator: Marteen Date: May 25,2022

target 2nd Cl Con A 40 g wet

Purpose: Based on F102/104, with SP sample

Sample: LG XRF SP

Feed: 2 kg of SG6 at -53 µm

Water DI Water

Flot Fd K₈₀ = 42 µm

Notes:

Conditions:

Stage	Reagents Added, g/t								Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon		3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp °C	Pulp Density %
Strength	10%	5%		2%	100%			10%					
Pass through WHIMS at 2,000 Gauss (~2 Amps), the mags pass through the WHIMS as a cleaner													
Filter Non-Mags													
High Density Conditioning ~50% @ 1000rpm													
Condition 1	250	100						100	3		9.0	53	50
Condition 2				1000	20				3		8.0	53	
Rougher 1						2.5				3	8.0	53	35.0
Rougher 2				300	20				2	5	8.1	53	
Rougher 3				300	20				2	5	8.1	53	
Rougher 4				200	20				2	5	8.1	52	
Rougher 5 (Assay)				500	20				1	2	8	53	
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed													
Condition 3	50								1		natural	54	
1st Cleaner A				50			12.5		1	3	8.3	52	~5
1st Cleaner B				50		5			1	2	8.3	52	
1st Cleaner C				50	10	5			1	1	8.3	50	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed													
Condition 4	25								1		natural	51	
2nd Cleaner A				25						3	8.4	53	
2nd Cleaner B				25		10			1	2.5	8.4	52	
Ro Total	325	100	0	2500	110	22.5		100	19	31.5	8.2	52.9	

Con A 69.9g
B-29g

Comment:			
Stage	Rougher	1st Clnr	2nd Clnr
Flotation Cell	4L (2kg flot cell)	3L (750 g)	1L (500 g)
Speed rpm	1600		

Lime: 8.7 in Ro Tail
4.06 in 1st Clnr Tail
2.67 in 2nd Clnr Tail

same as F104 same as F104

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
F107 2nd Cl Conc -A	47	2.4	2.34	5.10	8.4	19.6	35.5	8.3	1.58	3.66	53.5	53.5	2.0	1.0	12.3	36.1	0.7	0.7
F107 2nd Cl Conc -B	12	0.6	1.11	2.42	10.7	27.6	28.2	5.0	2.73	5.63	6.4	6.4	0.6	0.4	2.5	5.5	0.3	0.3
F107 2nd Cl Tail	25	1.3	0.32	0.70	15.5	31.0	19.1	1.5	3.70	8.13	3.9	3.9	1.9	0.8	3.5	3.5	0.8	0.8
F107 1st Cl Tail	149	7.7	0.11	0.24	14.5	38.3	11.0	0.67	4.64	10.9	8.0	8.0	10.7	6.2	12.2	9.3	6.3	6.8
F107 Ro Conc 5	37	1.9	0.15	0.33	13.9	37.6	11.6	1.0	4.44	10.0	2.7	2.7	2.5	1.5	3.2	3.3	1.5	1.5
F107 Ro Tail	1642	85.0	0.03	0.07	10.1	50.7	4.57	0.27	6.09	13.1	24.1	24.1	82.1	89.8	55.7	41.3	90.2	89.6
F107 Mags @2A	20	1.0	0.15	0.33	1.7	17.0	73.3	0.5	1.67	3.50	1.4	1.4	0.2	0.4	10.7	0.9	0.3	0.3
Head (Calc.)	1932	100.0	0.11	0.23	10.5	48.0	6.98	0.56	5.74	12.4	100	100	100	100	100	100	100	100
Head (Direct)			0.12	0.27	10.3	48.5	7.16	0.55	5.70	12.5								

Cumulative Products

Products	Time.	Weight %	Assay, %									Distribution, %							
			Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	
2nd Cl Con 1	3	2.4	2.34	5.10	8.4	19.6	35.5	8.3	1.6	3.7	53.5	53.5	2.0	1.0	12.3	36.1	0.7	0.7	
2nd Cl Con 1-2	5.5	3.0	2.09	4.56	8.9	21.2	34.0	7.6	1.8	4.1	59.9	59.9	2.6	1.3	14.8	41.7	1.0	1.0	
1st Cl Con 1-3	6	4.3	1.56	3.41	10.9	24.1	29.6	5.8	2.4	5.3	63.8	63.8	4.5	2.2	18.3	45.2	1.8	1.8	
Ro Conc 1-4	18	12.1	0.63	1.38	13.2	33.2	17.7	2.5	3.8	8.9	71.8	71.8	15.2	8.3	30.5	54.5	8.0	8.6	
Ro Conc 1-5	20	14.0	0.57	1.23	13.3	33.8	16.8	2.3	3.9	9.0	74.5	74.5	17.7	9.8	33.7	57.8	9.5	10.1	
Flotation Feed		99.0	0.11	0.23	10.6	48.3	6.3	0.6	5.8	12.5	98.6	98.6	99.8	99.6	89.3	99.1	99.7	99.7	
Head (Calc.)		100.0	0.11	0.23	10.5	48.0	6.98	0.56	5.74	12.4	100	100	100	100	100	100	100	100	
Head (Direct)			0.12	0.27	10.3	48.5	7.16	0.55	5.70	12.5									

Test No.: F-108

Project No: 18299-03 Operator: Marteen Date: May 25,2022

target 2nd Cl Con A

70 wet

Purpose: Based on F105, with SP sample

Sample: LG XRF SP

Feed: 2 kg of SG6 at -53 µm

Water DI Water Flot Fd K₈₀ = 42 µm

Notes:

Conditions:

Stage	Reagents Added, g/t							Time, min		Pulp			
	Sodium Silicate (N Type)	Calgon		3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp	Pulp Density
Strength	10%	5%		2%	100%			10%				°C	%
Pass through WHIMS at 2,000 Gauss (~2 Amps), the mags pass through the WHIMS as a cleaner													
Filter Non-Mags													
High Density Conditioning ~50% @ 1000rpm											7.5	53	50
Condition 1	250	50						50	3		9.0	53	50
Condition 2				1000	20				3		8.0	53	
Rougher 1						2.5				3	8.0	54	35.0
Rougher 2				300	20				2	5	8	53	
Rougher 3				300	20				2	5	8.1	53	
Rougher 4				200	20				2	5	8.1	53	
Rougher 5 (Assay)				500	20				1	2	8.2	52	
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed													
Condition 3	50								1		natural	54	
1st Cleaner A				50			12.5		1	3	8.2	52	~5
1st Cleaner B				50		5			1	2	8.2	52	
1st Cleaner C				50	10	5			1	1	8.2	52	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed													
Condition 4	25								1		natural	51	
2nd Cleaner A				25						3	8.2	51	
2nd Cleaner B				25		10			1	2.5	8.3	50	
Ro Total	325	50	0	2500	110	22.5		50	19	31.5	8.2	53.0	

Ro 5 con was very foamy
Looked like more mass2nd Cl con A 59g
B 29g

Comment:				Lime:	5.22 in Ro Tail 4.03 in 1st Clnr Tail 2.39 in 2nd Clnr Tail
Stage	Rougher	1st Clnr	2nd Clnr		
Flotation Cell	4L (2kg flot cell)	3L (750 g)	1L (500 g)		
Speed rpm	1600				
		same as F104	same as F104		

same as F104

same as F104

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
F108 2nd Cl Conc - A	41	2.2	2.45	5.61	13.4	12.2	33.8	7.3	0.87	2.37	49.8	49.8	2.8	0.5	9.4	29.2	0.3	0.4
F108 2nd Cl Conc - B	12	0.6	1.40	3.21	17.2	14.7	27.7	4.9	1.29	3.09	8.2	8.2	1.0	0.2	2.2	5.6	0.1	0.2
F108 2nd Cl Tail	33	1.7	0.40	0.92	21.1	21.4	18.9	1.5	2.38	5.70	6.6	6.6	3.6	0.8	4.3	4.8	0.7	0.8
F108 1st Cl Tail	174	9.2	0.13	0.30	20.1	30.0	11.0	0.77	3.50	8.2	11.3	11.3	17.9	5.7	13.1	13.1	5.7	6.1
F108 Ro Conc 5	164	8.7	0.09	0.21	25.2	25.5	8.5	0.7	2.93	6.6	7.4	7.4	21.2	4.6	9.6	10.7	4.5	4.6
F108 Ro Tail	1426	75.5	0.02	0.05	7.3	55.8	4.27	0.25	6.66	14.3	14.3	14.3	53.2	87.4	41.7	34.9	88.1	87.3
F108 Mags @2A	39	2.1	0.12	0.27	1.9	16.8	73.9	0.4	1.68	3.60	2.3	2.3	0.4	0.7	19.7	1.7	0.6	0.6
Head (Calc.)	1888	100.0	0.11	0.24	10.3	48.2	7.73	0.54	5.71	12.4	100	100	100	100	100	100	100	100
Head (Direct)			0.12	0.27	10.3	48.5	7.16	0.55	5.70	12.5								

Cumulative Products

Products	Time.	Weight	Assay, %									Distribution, %								
			%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	
2nd Cl Con 1	3	2.2	2.45	5.61	13.4	12.2	33.8	7.3	0.9	2.4	49.8	49.8	2.8	0.5	9.4	29.2	0.3	0.4		
2nd Cl Con 1-2	8	2.8	2.22	5.07	14.3	12.8	32.4	6.8	1.0	2.5	58.0	58.0	3.8	0.7	11.6	34.7	0.5	0.6		
1st Cl Con 1-3	6	4.5	1.51	3.47	16.9	16.1	27.2	4.7	1.5	3.8	64.6	64.6	7.4	1.5	15.9	39.6	1.2	1.4		
Ro Conc 1-4	18	13.7	0.58	1.34	19.0	25.4	16.3	2.1	2.8	6.7	76.0	76.0	25.3	7.3	29.0	52.7	6.8	7.5		
Ro Conc 1-5	20	22.4	0.39	0.90	21.4	25.5	13.3	1.5	2.9	6.7	83.4	83.4	46.5	11.8	38.6	63.4	11.3	12.1		
Flotation Feed		97.9	0.11	0.24	10.5	48.9	6.3	0.5	5.8	12.6	97.7	97.7	99.6	99.3	80.3	98.3	99.4	99.4		
Head (Calc.)		100.0	0.11	0.24	10.3	48.2	7.73	0.54	5.71	12.4	100	100	100	100	100	100	100	100		
Head (Direct)			0.12	0.27	10.3	48.5	7.16	0.55	5.70	12.5										

Test No.: F-109

Project No: 18299-03 Operator: Marteen Date: 06/06/2022

target 2nd Cl Con A

70 wet

Purpose: Based on F105, with HIC

Sample: ROM

Feed: 2 kg of SG5 at -53 µm

Water DI Water Flot Fd $K_{80} = 42 \mu\text{m}$

Notes:

Conditions:

Stage	Reagents Added, g/t								Time, min		Pulp		
	Sodium Silicate (N Tyne)	Calgon		3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp	Pulp Density
Strength	10%	5%		2%	100%			10%				°C	%
Pass through WHIMS at 2,000 Gauss (~2 Amps), the mags pass through the WHIMS as a cleaner													
Filter Non-Mags													
High Density & Intensity Conditioning ~50% @ 1800rpm													
Condition 1	250	50						100	5		7.5	54	50
Condition 2				1000	20				10		9.0	53	50
											8.1	52	
Rougher 1						2.5				3	8.1	53	35.0
Rougher 2				300	20				2	5	8.1	53	
Rougher 3				300	20				2	5	8.2	52	
Rougher 4				200	20				2	5	8.2	52	
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed													
Condition 3	50	10							1		natural	54	
												53	
1st Cleaner A				50			12.5		1	3	8.3	53	~5
1st Cleaner B				50		5			1	2	8.3	52	
1st Cleaner C				50	10	5			1	1	8.3	50	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed													
Condition 4	25								1		natural	51	
2nd Cleaner A				25						3	8.3	53	
2nd Cleaner B				25		10			1	4	8.34	53	
Ro Total	325	60	0	2000	90	22.5		100	27	31	8.3	52.5	

Comment:

Stage	Rougher	1st Clnr	2nd Clnr
Flotation Cell	4L (2kg flot cell)	3L (750 g)	1L (500 g)
Speed rpm	1600		

Lime:(g)

4.13 in Ro Tail

4.4 in 1st Clnr Tail

3.14 in 2nd Clnr Tail

same as F104

same as F104

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
F109 2nd Cl Conc A	43	2.0	2.97	6.81	9.3	13.7	35.8	11.3	0.98	2.66	56.2	56.2	1.8	0.6	11.0	40.2	0.3	0.4
F109 2nd Cl Conc B	18	0.9	1.63	3.72	12.9	19.0	30.1	7.6	1.65	3.77	13.3	13.3	1.1	0.3	4.0	11.7	0.2	0.3
F109 2nd Cl Tail	30	1.4	0.29	0.67	17.4	28.1	18.4	1.8	3.28	7.33	3.9	3.9	2.4	0.8	4.0	4.6	0.8	0.8
F109 1st Cl Tail	194	9.2	0.08	0.17	18.5	34.5	8.8	0.45	4.25	9.7	6.6	6.6	16.7	6.6	12.3	7.3	6.6	7.0
F109 Ro Tail	1784	84.8	0.02	0.05	9.4	52.1	4.0	0.2	6.46	13.7	17.6	17.6	77.6	91.1	51.9	34.3	91.5	91.0
F109 Mags @2A	33	1.6	0.16	0.37	2.1	18.5	69.2	0.7	2.05	4.1	2.4	2.4	0.3	0.6	16.7	1.8	0.5	0.5
Head (Calc.)	2102	100	0.11	0.24	10.2	48.5	6.58	0.57	5.99	12.8	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

Cumulative Products

Products	Time,	Weight %	Assay, %								Distribution, %							
			Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
2nd Cl Con 1	3	2.0	2.97	6.81	9.3	13.7	35.8	11.3	1.0	2.7	56.2	56.2	1.8	0.6	11.0	40.2	0.3	0.4
2nd Cl Con 1-2	8	2.9	2.57	5.87	10.4	15.3	34.1	10.2	1.2	3.0	69.5	69.5	2.9	0.9	15.0	51.9	0.6	0.7
1st Cl Con 1-3	6	4.3	1.81	4.15	12.7	19.5	28.9	7.4	1.9	4.4	73.4	73.4	5.4	1.7	19.0	56.5	1.4	1.5
Ro Conc 1-4	18	13.6	0.63	1.44	16.6	29.7	15.2	2.7	3.5	8.0	80.0	80.0	22.0	8.3	31.4	63.8	7.9	8.5
Flotation Feed		98.4	0.11	0.24	10.4	49.0	5.6	0.6	6.1	12.9	98	98	100	99	83	98	99	99
Head (Calc.)		100.0	0.11	0.24	10.2	48.5	6.58	0.57	5.99	12.8	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.9	49.7	6.44	0.58	5.92	12.9								

Test No.: F-110

Project No: 18299-03 Operator: Marteen Date: 06-20-2022

target 2nd Cl Conc A

70 wet

Purpose: Based on F109, with HIC, with new collector 8905Z

Sample: ROM

Feed: 2 kg of SG5 at -53 µm, WHIMS @2A performed

Water: DI Water Flot Fd K₈₀ = 42 µm

Notes: prepare 8905Z similar to SHA (dissolve in hot water)

1. Reagent Preparation: make 1% solution, must heat up and keep heated to keep dissolved in solution, approx 60C (used sigma)

Conditions:

Stage	Reagents Added, g/t								Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon		8905Z	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp	Pulp Density
Strength	10%	5%		1%	100%			10%				°C	%
High Density & Intensity Conditioning -50% @ 1800rpm													
Condition 1	250	50						100	5		7.5	54	50
Condition 2				1000	20				5		9.0	53	50
											8.1	52	
Rougher 1						2.5				3	8.1	53	35.0
Rougher 2				300	20				2	5	8.1	53	
Rougher 3				300	20				2	5	8.2	52	
Rougher 4				200	20				2	5	8.2	52	
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed													
Condition 3	50	10							1		-8	54	
												53	
1st Cleaner A				50			12.5		1	3	8.3	53	-5
1st Cleaner B				50		5			1	2	8.3	52	
1st Cleaner C				50	10	5			1	1	8.3	50	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed													
Condition 4	25								1		-8	51	
2nd Cleaner A				25						3	8.3	53	
2nd Cleaner B				25		10			1	4	8.34	53	
Ro Total	325	60	0	2000	90	22.5		100	22	31	8.3	52.5	

keep pH -8

how's the froth like

Comment:

Stage	Rougher	1st Clnr	2nd Clnr
Flotation Cell	4L (2kg flot cell)	3L (750 g)	1L (500 g)
Speed rpm	1600		

 Lime:(g) 4.13 in Ro Tail
 4.4 in 1st Clnr Tail
 3.14 in 2nd Clnr Tail
Metallurgical Balance

Products	Weight		Assay, %									Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	
F110 2nd Cl Conc - A	66	3.1	2.11	4.83	9.2	24.5	28.1	9.3	2.28	4.97	67.2	67.2	2.8	1.6	13.3	51.0	1.2	1.2	
F110 2nd Cl Conc - B	13	0.6	1.19	2.73	10.4	31.1	23.7	5.8	3.24	6.63	7.7	7.7	0.6	0.4	2.3	6.5	0.4	0.3	
F110 2nd Cl Tail	59	2.8	0.20	0.47	14.6	38.0	12.5	1.1	4.57	10.10	5.9	5.9	4.0	2.2	5.3	5.6	2.2	2.2	
F110 1st Cl Tail	152	7.2	0.08	0.17	15.3	39.4	8.5	0.41	4.60	11.6	5.6	5.6	10.8	5.9	9.3	5.2	5.7	6.5	
F110 Ro Tail	1781	84.6	0.01	0.03	9.9	51.3	4.1	0.2	6.13	13.5	11.0	11.0	81.4	89.4	53.1	29.8	89.9	89.2	
F110 Mags @2A	33	1.6	0.16	0.37	2.1	18.5	69.2	0.7	2.05	4.1	2.6	2.6	0.3	0.6	16.7	1.8	0.6	0.5	
Head (Calc.)	2104	100	0.10	0.22	10.2	48.6	6.59	0.57	5.77	12.8	100	100	100	100	100	100	100	100	
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9									

Cumulative Products

Products	Time,	Weight %	Assay, %									Distribution, %						
			Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
2nd Cl Conc 1	3	3.1	2.11	4.83	9.2	24.5	28.1	9.3	2.3	5.0	67.2	67.2	2.8	1.6	13.3	51.0	1.2	1.2
2nd Cl Conc 1-2	7	3.8	1.95	4.47	9.4	25.6	27.4	8.7	2.4	5.2	74.9	74.9	3.4	2.0	15.6	57.5	1.6	1.5
1st Cl Conc 1-3	6	6.6	1.20	2.75	11.6	30.9	21.0	5.4	3.4	7.3	80.8	80.8	7.5	4.2	20.9	63.1	3.8	3.8
Ro Conc 1-4	18	13.8	0.61	1.40	13.6	35.4	14.5	2.8	4.0	9.6	86.4	86.4	18.2	10.0	30.3	68.3	9.6	10.3
Flotation Feed		98.4	0.10	0.22	10.4	49.1	5.6	0.6	5.8	12.9	97	97	100	99	83	98	99	99
Head (Calc.)		100.0	0.10	0.22	10.2	48.6	6.59	0.57	5.77	12.8	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.9	49.7	6.44	0.58	5.92	12.9								

Test No.: F-111

Project No: 18299-03 Operator: Marteen Date: 06-21-2022

target 2nd Cl Con A

70 wet

Purpose: Based on F109, with HIC, -half dosage of collectors (1000 g/t)

Sample: ROM

Feed: 2 kg of SG9 at -53 µm, WHIMS performed

Water DI Water Flot Fd $K_{80} = 42 \mu\text{m}$

Notes:

Conditions:

Stage	Reagents Added, g/t								Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon		3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp	Pulp Density
Strength	10%	5%		2%	100%			10%				°C	%
High Density & Intensity Conditioning ~50% @ 1800rpm													
Condition 1	250	50						100	5		7.5	54	50
Condition 2				800	20				5		9.0	53	50
											8.1	52	
Rougher 1						2.5				3	8.1	53	35.0
Rougher 2				100	20				2	5	8.1	53	
Rougher 3				100	20				2	5	8.2	52	
Rougher 4				0	20				2	5	8.2	52	
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed													
Condition 3	50	10							1		natural	54	
												53	
1st Cleaner A				50			12.5		1	3	8.3	53	~5
1st Cleaner B				50		5			1	2	8.3	52	
1st Cleaner C				50	10	5			1	1	8.3	50	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed													
Condition 4	25								1		natural	51	
2nd Cleaner A				25						3	8.3	53	
2nd Cleaner B				25		10			1	4	8.34	53	
Ro Total	325	60	0	1200	90	22.5		100	22	31	8.3	52.5	

Comment:

Stage	Rougher	1st Clnr	2nd Clnr
Flotation Cell	4L (2kg flot cell)	3L (750 g)	1L (500 g)
Speed rpm	1600		

same as F104

same as F104

 Lime:(g)
 4.13 in Ro Tail
 4.4 in 1st Clnr Tail
 3.14 in 2nd Clnr Tail
Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
F111 2nd Cl Conc A	28	1.4	3.52	8.06	5.4	14.5	39.3	12.5	1.10	2.81	45.8	46.0	0.7	0.4	8.3	29.5	0.3	0.3
F111 2nd Cl Conc B	13	0.6	1.94	4.45	7.0	26.8	30.4	8.2	2.64	5.32	11.7	11.7	0.4	0.4	3.0	8.9	0.3	0.3
F111 2nd Cl Tail	21	1.0	0.44	1.02	12.6	33.4	20.1	2.3	4.05	8.90	4.2	4.2	1.2	0.7	3.1	3.9	0.7	0.7
F111 1st Cl Tail	152	7.4	0.09	0.20	11.9	43.1	9.3	0.55	5.30	12.6	6.2	6.2	8.6	6.5	10.4	6.9	6.5	7.3
F111 Ro Tail	1821	88.6	0.04	0.08	10.2	50.5	4.6	0.3	6.25	13.2	30.8	30.9	88.8	91.8	61.8	49.9	92.1	91.3
SG-9 Mag @ 2A	22	1.1	0.14	0.23	1.15	10.3	83.9	0.4	1.01	2.2	1.4	1.0	0.1	0.2	13.5	0.8	0.2	0.2
Head (Calc.)	2056	100	0.11	0.24	10.2	48.7	6.55	0.59	6.01	12.8	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

Cumulative Products

Products	Time,	Weight	Assay, %									Distribution, %								
			%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	
2nd Cl Con 1	3	1.4	3.52	8.06	5.4	14.5	39.3	12.5	1.1	2.8	45.8	46.0	0.7	0.4	8.3	29.5	0.3	0.3		
2nd Cl Con 1-2	8	2.0	3.02	6.92	5.9	18.4	36.5	11.1	1.6	3.6	57.5	57.7	1.2	0.8	11.2	38.4	0.5	0.6		
1st Cl Con 1-3	6	3.0	2.17	4.97	8.1	23.3	31.1	8.2	2.4	5.4	61.7	61.9	2.4	1.4	14.3	42.3	1.2	1.3		
Ro Conc 1-4	18	10.4	0.69	1.59	10.8	37.4	15.6	2.8	4.5	10.5	67.9	68.1	11.0	8.0	24.7	49.3	7.7	8.5		
Flotation Feed		98.9	0.11	0.24	10.3	49.1	5.7	0.6	6.1	12.9	99	99	100	100	87	99	100	100		
Head (Calc.)		100.0	0.11	0.24	10.2	48.7	6.55	0.59	6.01	12.8	100	100	100	100	100	100	100	100		
Head (Direct)			0.10	0.22	9.9	49.7	6.44	0.58	5.92	12.9										

Test No.: F-112

Project No: 18299-03 Operator: Marteen Date: 11/07/2022

target 2nd Cl Con A

50 wet

Purpose: Based on F109, with HIC, ~65% less collectors (700 g/t)

Sample: ROM

Feed: 2 kg of SG9 at -53 µm, WHIMS performed

Water DI Water Flot Fd K₈₀ = 42 µm

Notes:

Conditions:

Stage	Reagents Added, g/t								Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon		3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp °C	Pulp Density %
Strength	10%	5%		2%	100%			10%					
High Density & Intensity Conditioning ~50% @ 1800rpm											7.5	53	50
Condition 1	250	50						100	5		9.0	53	50
Condition 2				500	40				5		8.0	53	
Rougher 1						2.5				3	8.0	52	35.0
Rougher 2				0	20	2.5			2	5	7.9	50	
Rougher 3				0	20				2	5	8	50	
Rougher 4				0	20				2	5	8	50	
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed													
Condition 3	50	10							1		natural		
											8	53	
1st Cleaner A				50			5.5		1	3	8	53	~5
1st Cleaner B				50					1	2	7.9	53	
1st Cleaner C				50	10	5			1	1	7.9	52	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed													
Condition 4	25								1		natural		
											8.2	52	
2nd Cleaner A				25						3	8.1	52	
2nd Cleaner B				25		10			1	4	8	50	
Ro Total	325	60	0	700	110	20		100	22	31	8.2	51.3	

Comment:

Stage	Rougher	1st Clnr	2nd Clnr
Flotation Cell	4L (2kg flot cell)	3L (750 g)	1L (500 g)
Speed rpm	1600		

Lime:(g) 5.31 in Ro Tail
3.17 in 1st Clnr Tail
4.16 in 2nd Clnr Tail

same as F104 same as F104

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
F112 2nd Cl Conc A	11	0.6	2.34	5.35	5.3	13.0	41.0	12.4	0.92	2.60	12.6	12.6	0.3	0.1	3.5	12.4	0.1	0.1
F112 2nd Cl Conc B	3	0.1	1.00*	2.29							1.3	1.3	0.0	0.0	0.0	0.0	0.0	0.0
F112 2nd Cl Tail	19	0.9	0.65	1.48	15.3	30.7	19.8	2.9	3.44	8.13	5.9	5.9	1.4	0.6	2.9	4.9	0.5	0.6
F112 1st Cl Tail	110	5.3	0.42	0.96	10.9	41.1	13.5	1.23	4.79	11.5	21.7	21.8	5.7	4.4	11.1	11.9	4.3	4.8
F112 Ro Tail	1894	92.0	0.06	0.15	10.2	51.0	4.8	0.4	6.07	13.1	57.0	57.3	92.5	94.6	68.8	70.0	94.9	94.3
SG-9 Mag @ 2A	22	1.1	0.14	0.23	1.15	10.3	83.9	0.4	1.01	2.2	1.4	1.0	0.1	0.2	13.7	0.8	0.2	0.2
Head (Calc.)	2059	100	0.10	0.23	10.1	49.6	6.46	0.55	5.89	12.8	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

*NSS, estimate

Cumulative Products

Products	Time,	Weight %	Assay, %								Distribution, %							
			Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
2nd Cl Con 1	3	0.6	2.34	5.35	5.3	13.0	41.0	12.4	0.9	2.6	12.6	12.6	0.3	0.1	3.5	12.4	0.1	0.1
2nd Cl Con 1-2	8	0.7	2.07	4.75	4.2	10.4	32.9	10.0	0.7	2.1	13.9	14.0	0.3	0.1	3.5	12.4	0.1	0.1
1st Cl Con 1-3	6	1.6	1.25	2.87	10.6	22.1	25.4	5.9	2.3	5.6	19.8	19.9	1.7	0.7	6.4	17.3	0.6	0.7
Ro Conc 1-4	18	7.0	0.61	1.41	10.8	36.7	16.3	2.3	4.2	10.1	41.5	41.7	7.4	5.1	17.5	29.2	5.0	5.5
Flotation Feed		98.9	0.10	0.23	10.2	50.0	5.6	0.6	5.9	12.9	99	99	100	100	86	99	100	100
Head (Calc.)		100.0	0.10	0.23	10.1	49.6	6.46	0.55	5.89	12.8	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.9	49.7	6.44	0.58	5.92	12.9								

Test No.: F-113

Project No: 18299-03 Operator: Marteen Date: 11/07/2022

target 2nd Cl Con A

70 wet

Purpose: Based on F109, with HIC, new collector TJ-B3

Sample: ROM

Feed: 2 kg of SG9 at -53 µm, WHIMS performed

Water DI Water Flot Fd $K_{80} = 42 \mu\text{m}$

Notes:

Conditions:

Stage	Reagents Added, g/t							Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon	TJ-B3	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp	Pulp Density
Strength	10%	5%	2%	100%			10%				°C	%
High Density & Intensity Conditioning ~50% @ 1800rpm										7.5	54	50
Condition 1	250	50					100	5		9.0	53	50
Condition 2			1000	20				5		8.1	52	
Rougher 1					2.5				3	8.1	53	35.0
Rougher 2			300	20				2	5	8.1	53	
Rougher 3			300	20				2	5	8.2	52	
Rougher 4			200	20				2	5	8.2	52	
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed												
Condition 3	50	10						1		natural	54	
											53	
1st Cleaner A			50			12.5		1	3	8.3	53	~5
1st Cleaner B			50		5			1	2	8.3	52	
1st Cleaner C			50	10	5			1	1	8.3	50	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed												
Condition 4	25							1		natural	51	
2nd Cleaner A			25						3	8.3	53	
2nd Cleaner B			25		10			1	4	8.34	53	
Ro Total	325	60	0	2000	90	22.5	100	22	31	8.3	52.5	

Comment:

Stage	Rougher	1st Clnr	2nd Clnr
Flotation Cell	4L (2kg flot cell)	3L (750 g)	1L (500 g)
Speed rpm	1600		

Lime:(g)
5.02 in Ro Tail
5.55 in 1st Clnr Tail
4.82 in 2nd Clnr Tail

same as F104

same as F104

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
F113 2nd Cl Conc A	40	1.9	1.65	3.78	10.5	20.5	33.0	9.8	1.73	4.03	29.6	29.7	2.0	0.8	9.8	33.2	0.6	0.6
F113 2nd Cl Conc B	14	0.7	1.52	3.49	10.7	24.3	28.9	8.6	2.21	4.90	9.7	9.7	0.7	0.3	3.0	10.3	0.3	0.3
F113 2nd Cl Tail	54	2.6	0.71	1.63	18.6	28.9	16.0	2.5	3.20	7.29	17.1	17.2	4.8	1.5	6.4	11.2	1.4	1.5
F113 1st Cl Tail	208	10.1	0.11	0.26	20.8	33.7	7.7	0.46	3.78	9.9	10.6	10.7	20.7	6.9	11.8	8.1	6.6	7.9
F113 Tail	1721	83.6	0.04	0.09	8.7	53.5	4.4	0.3	6.31	13.7	31.7	31.8	71.7	90.2	55.6	36.4	90.9	89.6
SG-9 Mag @ 2A	22	1.1	0.14	0.23	1.15	10.3	83.9	0.4	1.01	2.2	1.4	1.0	0.1	0.2	13.4	0.8	0.2	0.2
Head (Calc.)	2058	100	0.11	0.25	10.2	49.6	6.57	0.57	5.80	12.8	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

Cumulative Products

Products	Time,	Weight %	Assay, %								Distribution, %							
			Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
2nd Cl Con 1	3	1.9	1.65	3.78	10.5	20.5	33.0	9.8	1.7	4.0	29.6	29.7	2.0	0.8	9.8	33.2	0.6	0.6
2nd Cl Con 1-2	8	2.6	1.62	3.70	10.6	21.5	31.9	9.5	1.9	4.3	39.3	39.4	2.7	1.1	12.8	43.4	0.8	0.9
1st Cl Con 1-3	6	5.3	1.17	2.67	14.6	25.2	24.0	6.0	2.5	5.8	56.4	56.6	7.5	2.7	19.2	54.7	2.3	2.4
Ro Conc 1-4	18	15.4	0.47	1.09	18.7	30.8	13.3	2.3	3.4	8.5	67.0	67.3	28.2	9.5	31.0	62.8	8.9	10.2
Flotation Feed		98.9	0.11	0.25	10.3	50.0	5.7	0.6	5.9	12.9	99	99	100	100	87	99	100	100
Head (Calc.)		100.0	0.11	0.25	10.2	49.6	6.57	0.57	5.80	12.8	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.9	49.7	6.44	0.58	5.92	12.9								

Test No.: F-114

Project No: 18299-03 Operator: Marteen Date: 03/08/2022

target 2nd Cl Con A

70 wet

Purpose: Based on F109, with ~20% less collector

Sample: ROM

Feed: 2 kg of SG9 at -53 µm, WHIMS performed

Water DI Water Flot Fd $K_{80} = 42 \mu\text{m}$

Notes:

Conditions:

Stage	Reagents Added, g/t								Time, min		Pulp			
	Sodium Silicate (N Type)	Calgon		3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp	Pulp Density	
Strength	10%	5%		2%	100%			10%				°C	%	
High Density & Intensity Conditioning ~50% @ 1800rpm												7.5	52	50
Condition 1	250	50						100	5		9.0	53	50	
Condition 2				1000	20				5		8.0	53		
Rougher 1						2.5				3	8.1	53	35.0	
Rougher 2				200	20				2	5	8.1	52		
Rougher 3				100	20				2	5	8.2	53		
Rougher 4				100	20				2	5	8.2	53		
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed														
Condition 3	50	10							1		natural	54		
												53		
1st Cleaner A				50			12.5		1	3	8.3	53	~5	
1st Cleaner B				50		5			1	2	8.4	52		
1st Cleaner C				50	10	5			1	1	8.3	53		
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed														
Condition 4	25								1		natural	51		
											8.5			
2nd Cleaner A				25						3	8.4	53		
2nd Cleaner B				25		10			1	4	8.3	51		
Ro Total	325	60	0	1600	90	22.5		100	22	31	8.3	52.8		

Comment:

Stage	Rougher	1st Clnr	2nd Clnr
Flotation Cell	4L (2kg flot cell)	3L (750 g)	1L (500 g)
Speed rpm	1600		

Lime:(g)
 5.85 in Ro Tail
 4.22 in 1st Clnr Tail
 4.37 in 2nd Clnr Tail

same as F104

same as F104

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
F114 2nd Cl Conc - A	45	2.2	2.96	6.78	7.5	16.1	37.0	11.1	1.22	3.04	63.0	63.3	1.6	0.7	12.6	43.5	0.5	0.5
F114 2nd Cl Conc - B	9	0.4	1.37	3.14	10.7	25.7	28.5	6.4	2.43	5.07	5.7	5.7	0.5	0.2	1.9	4.9	0.2	0.2
F114 2nd Cl Tail	30	1.5	0.18	0.41	15.4	33.8	16.3	1.3	4.01	9.06	2.5	2.5	2.2	1.0	3.7	3.4	1.0	1.0
F114 1st Cl Tail	171	8.3	0.05	0.12	12.8	42.8	8.3	0.40	5.11	12.2	4.1	4.1	10.5	7.2	10.6	5.9	7.3	8.0
F114 Ro Tail	1776	86.5	0.03	0.06	10.0	51.7	4.4	0.3	6.15	13.3	23.3	23.4	85.1	90.6	57.7	41.4	90.9	90.1
SG-9 Mag @ 2A	22	1.1	0.14	0.23	1.2	10.3	83.9	0.4	1.01	2.2	1.4	1.0	0.1	0.2	13.6	0.8	0.2	0.2
Head (Calc.)	2053	100	0.10	0.24	10.2	49.4	6.52	0.56	5.85	12.8	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

Cumulative Products

Products	Time,	Weight %	Assay, %								Distribution, %							
			Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
2nd Cl Con 1	3	2.2	2.96	6.78	7.5	16.1	37.0	11.1	1.2	3.0	63.0	63.3	1.6	0.7	12.6	43.5	0.5	0.5
2nd Cl Con 1-2	8	2.6	2.70	6.18	8.0	17.7	35.6	10.3	1.4	3.4	68.7	69.0	2.1	0.9	14.5	48.4	0.6	0.7
1st Cl Con 1-3	6	4.1	1.80	4.13	10.6	23.4	28.7	7.1	2.3	5.4	71.2	71.5	4.3	1.9	18.1	51.9	1.6	1.7
Ro Conc 1-4	18	12.4	0.63	1.44	12.1	36.4	15.0	2.6	4.2	10.0	75.3	75.6	14.8	9.2	28.7	57.8	8.9	9.7
Flotation Feed		98.9	0.10	0.24	10.3	49.8	5.7	0.6	5.9	12.9	99	99	100	100	86	99	100	100
Head (Calc.)		100.0	0.10	0.24	10.2	49.4	6.52	0.56	5.85	12.8	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.9	49.7	6.44	0.58	5.92	12.9								

Test No.: F-115

Project No: 18299-03 Operator: Marteen Date: 03/08/2022

target 2nd Cl Con A

70 wet

Purpose: Based on F109, with finer grind size

Sample: ROM

Feed: 2 kg of SG12 at -38 µm

Water DI Water Flot Fd $K_{80} =$ µm

Notes:

Conditions:

Stage	Reagents Added, g/t								Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon		3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp °C	Pulp Density %
Strength	10%	5%		2%	100%			10%					
Pass through WHIMS at 2,000 Gauss (~2 Amps), the mags pass through the WHIMS as a cleaner													
Filter Non-Mags													
High Density & Intensity Conditioning -50% @ 1800rpm													
Condition 1	250	50						100	5		7.5	52	50
Condition 2				1000	20				5		9.0	51	50
											8.1	50	
Rougher 1						2.5				3	8.1	51	35.0
Rougher 2				300	20				2	5	8.1	51	
Rougher 3				300	20				2	5	8.1	53	
Rougher 4				200	20				2	5	8.2	52	
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed													
Condition 3	50	10							1		natural	52	
												50	
1st Cleaner A				50			12.5		1	3	8.3	53	~5
1st Cleaner B				50		5			1	2	8.3	52	
1st Cleaner C				50	10	5			1	1	8.3	51	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed													
Condition 4	25						5		1		natural	53	
2nd Cleaner A				25						3	8.4	53	
2nd Cleaner B				25		10			1	4	8.3	53	
Ro Total	325	60	0	2000	90	22.5		100	22	31	8.3	51.3	

Comment:

Stage	Rougher	1st Clnr	2nd Clnr
Flotation Cell	4L (2kg flot cell)	3L (750 g)	1L (500 g)
Speed rpm	1600		

Lime:(g)

6.98 in Ro Tail

10.43 in 1st Clnr Tail

4.69 in 2nd Clnr Tail

same as F104

same as F104

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
F115 2nd Cl Conc - A	72	3.4	2.34	5.35	10.3	15.9	34.4	9.3	1.27	3.10	74.6	74.6	3.3	1.1	17.9	55.1	0.7	0.8
F115 2nd Cl Conc - B	12	0.5	0.74	1.69	13.3	24.0	27.3	3.9	2.51	5.22	3.8	3.8	0.7	0.3	2.3	3.7	0.2	0.2
F115 2nd Cl Tail	33	1.5	0.09	0.20	17.7	32.7	14.1	0.7	3.86	8.93	1.3	1.3	2.6	1.0	3.3	1.9	1.0	1.1
F115 1st Cl Tail	263	12.4	0.04	0.09	15.3	41.8	7.2	0.33	5.04	11.6	4.4	4.4	18.0	10.6	13.7	7.1	10.8	11.2
F115 Ro Tail	1724	81.2	0.02	0.04	9.8	52.2	4.2	0.2	6.20	13.6	14.6	14.6	75.2	86.7	52.1	31.1	86.9	86.4
F115 Mags	21	1.0	0.15	0.35	2.1	16.2	72.1	0.7	1.60	3.6	1.4	1.4	0.2	0.3	10.7	1.1	0.3	0.3
Head (Calc.)	2124	100	0.11	0.24	10.5	48.9	6.52	0.57	5.79	12.8	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

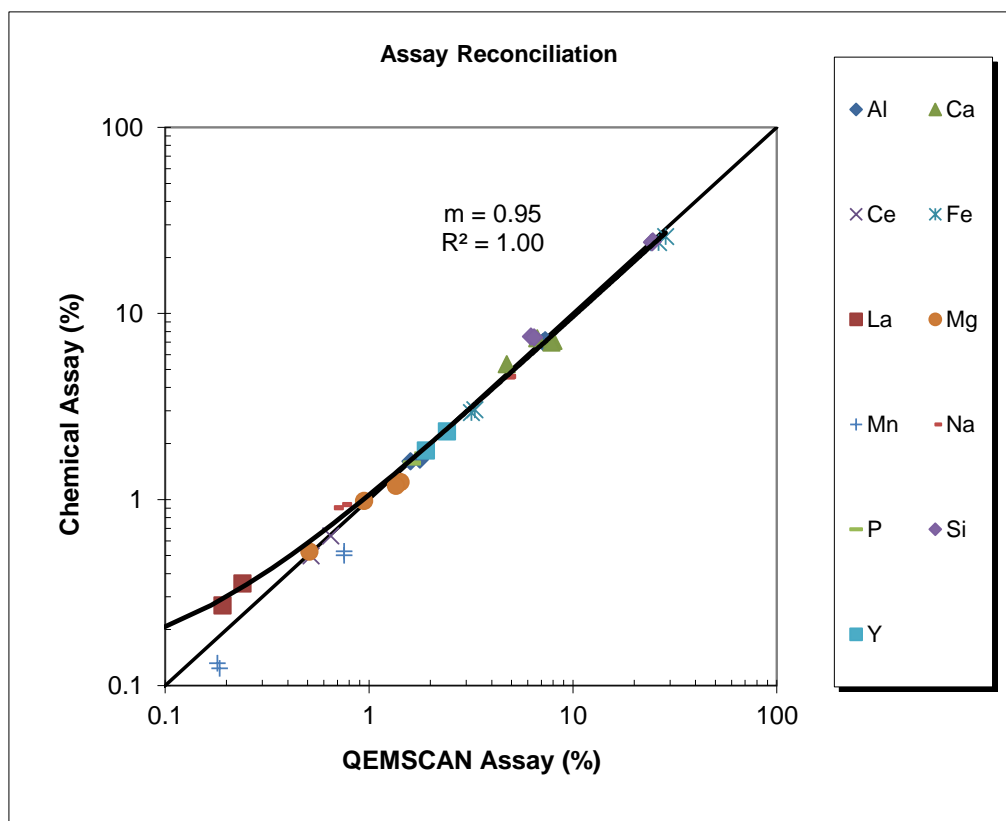
Cumulative Products

Products	Time,	Weight %	Assay, %								Distribution, %							
			Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
2nd Cl Con 1	3	3.4	2.34	5.35	10.3	15.9	34.4	9.3	1.3	3.1	74.6	74.6	3.3	1.1	17.9	55.1	0.7	0.8
2nd Cl Con 1-2	8	3.9	2.12	4.85	10.7	17.0	33.4	8.6	1.4	3.4	78.3	78.3	4.0	1.4	20.1	58.8	1.0	1.0
1st Cl Con 1-3	6	5.5	1.54	3.53	12.7	21.4	28.0	6.4	2.1	5.0	79.6	79.6	6.6	2.4	23.5	60.7	2.0	2.1
Ro Conc 1-4	18	17.9	0.50	1.14	14.5	35.6	13.6	2.2	4.1	9.6	84.1	84.1	24.6	13.0	37.1	67.8	12.8	13.4
Flotation Feed		99.0	0.11	0.24	10.6	49.2	5.9	0.6	5.8	12.9	99	99	100	100	89	99	100	100
Head (Calc.)		100.0	0.11	0.24	10.5	48.9	6.52	0.57	5.79	12.8	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.9	49.7	6.44	0.58	5.92	12.9								

Appendix C – Process Mineralogy

Namibia Critical Metals
 CALR-18299-03
 MI5067-SEP22

Assay Reconciliation



Sample	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
Al (TIMA)	7.15	1.60	7.32	1.77
Al (Chemical)	7.04	1.61	7.20	1.64
Ca (TIMA)	8.02	4.73	7.84	6.67
Ca (Chemical)	7.15	5.35	6.98	7.36
Ce (TIMA)	0.01	0.65	0.01	0.52
Ce (Chemical)	0.01	0.64	0.01	0.50
Fe (TIMA)	3.30	28.5	3.18	26.3
Fe (Chemical)	3.04	25.9	2.93	24.1
La (TIMA)	0.00	0.24	0.00	0.19
La (Chemical)	0.01	0.35	0.01	0.27
Mg (TIMA)	1.43	0.51	1.35	0.95
Mg (Chemical)	1.24	0.53	1.19	0.98
Mn (TIMA)	0.18	0.75	0.19	0.75
Mn (Chemical)	0.13	0.53	0.12	0.50
Na (TIMA)	4.71	0.68	4.74	0.75
Na (Chemical)	4.56	0.91	4.60	0.94
P (TIMA)	0.05	1.62	0.04	1.84
P (Chemical)	0.05	1.57	0.03	1.76
Si (TIMA)	24.4	6.22	24.7	6.46
Si (Chemical)	24.2	7.53	24.4	7.43
Y (TIMA)	0.02	2.41	0.02	1.90
Y (Chemical)	0.02	2.33	0.02	1.84

Namibia Critical Metals
 CALR-18299-03
 MI5067-SEP22

Modals

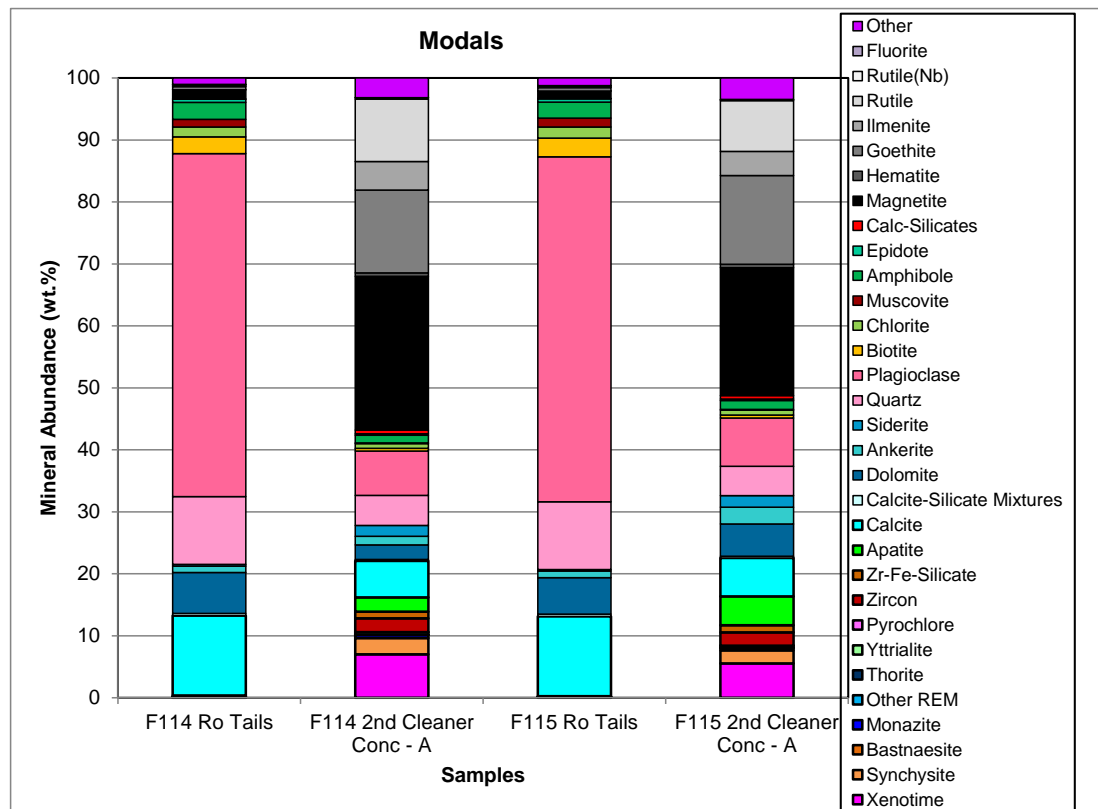
Survey		CALR-18299-03 / MI5067-SEP22			
Project		Namibia Critical Metals			
Sample		F114 Ro Tails	F114 2nd Cleaner Conc A	F115 Ro Tails	F115 2nd Cleaner Conc A
Mineral Mass (%)	Xenotime	0.06	7.00	0.05	5.54
	Synchysite	0.02	2.65	0.02	2.13
	Bastnaesite	0.00	0.00	0.00	0.00
	Monazite	0.01	0.49	0.00	0.38
	Other REM	0.00	0.01	0.00	0.01
	Thorite	0.00	0.01	0.00	0.01
	Yttrialite	0.01	0.44	0.00	0.32
	Pyrochlore	0.00	0.00	0.00	0.00
	Zircon	0.05	2.21	0.04	2.15
	Zr-Fe-Silicate	0.03	1.07	0.03	1.15
	Apatite	0.21	2.33	0.12	4.67
	Calcite	12.9	5.88	12.9	6.21
	Calcite-Silicate Mixtures	0.37	0.25	0.36	0.28
	Dolomite	6.57	2.32	5.88	5.18
	Ankerite	1.04	1.41	1.05	2.72
	Siderite	0.27	1.73	0.25	1.85
	Quartz	11.0	4.87	10.9	4.74
	Plagioclase	55.3	7.13	55.6	7.81
	Biotite	2.71	0.45	3.01	0.45
	Chlorite	1.59	0.75	1.79	0.81
	Muscovite	1.24	0.12	1.43	0.13
	Amphibole	2.72	1.24	2.59	1.43
	Epidote	0.53	0.24	0.57	0.25
	Calc-Silicates	0.01	0.61	0.01	0.54
	Magnetite	1.51	24.8	1.22	20.6
	Hematite	0.00	0.59	0.00	0.55
	Goethite	0.52	13.3	0.56	14.3
	Ilmenite	0.11	4.63	0.08	3.90
	Rutile	0.20	10.1	0.17	8.19
	Rutile(Nb)	0.00	0.22	0.00	0.19
	Fluorite	0.04	0.02	0.04	0.03
	Other	1.02	3.17	1.24	3.45
Total		100.0	100.0	100.0	100.0

Modals Condensed

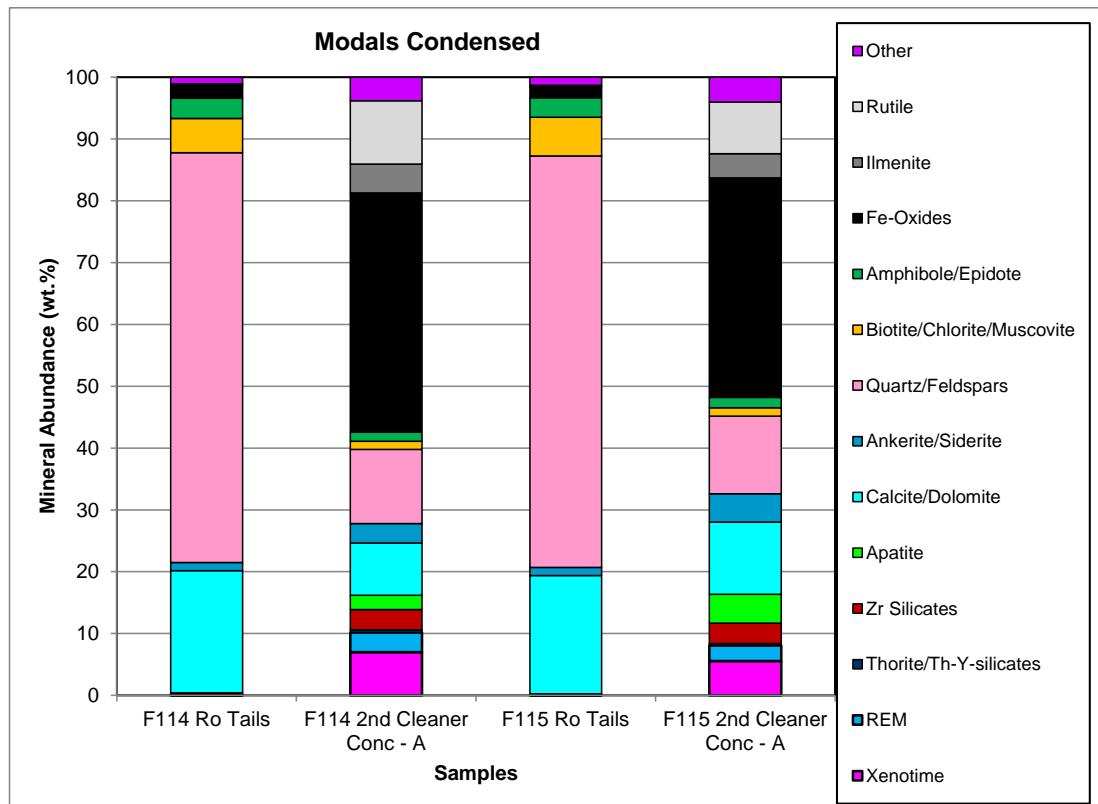
Survey		CALR-18299-03 / MI5067-SEP22			
Project		Namibia Critical Metals			
Sample		F114 Ro Tails	F114 2nd Cleaner Conc A	F115 Ro Tails	F115 2nd Cleaner Conc A
Mineral Mass (%)	Xenotime	0.06	7.00	0.05	5.54
	REM	0.03	3.15	0.03	2.53
	Thorite/Th-Y-silicates	0.01	0.44	0.00	0.33
	Zr Silicates	0.08	3.28	0.06	3.30
	Apatite	0.21	2.33	0.12	4.67
	Calcite/Dolomite	19.8	8.45	19.1	11.7
	Ankerite/Siderite	1.30	3.15	1.30	4.58
	Quartz/Feldspars	66.3	12.0	66.6	12.5
	Biotite/Chlorite/Muscovite	5.55	1.32	6.23	1.38
	Amphibole/Epidote	3.25	1.48	3.16	1.68
	Fe-Oxides	2.04	38.7	1.79	35.5
	Ilmenite	0.11	4.63	0.08	3.90
	Rutile	0.21	10.3	0.18	8.37
	Other	1.07	3.79	1.29	4.02
	Total	100.0	100.0	100.0	100.0

Namibia Critical Metals
CALR-18299-03
MI5067-SEP22

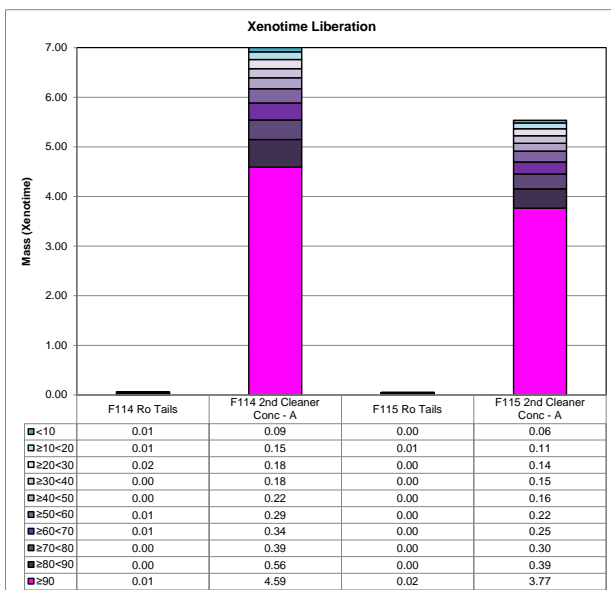
Modal Chart



Modal Chart Condensed



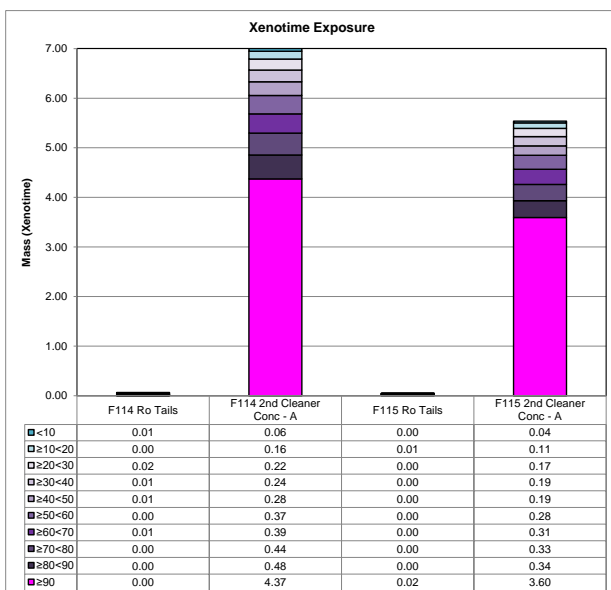
Xenotime Liberation



Absolute Mass of Xenotime Across Samples

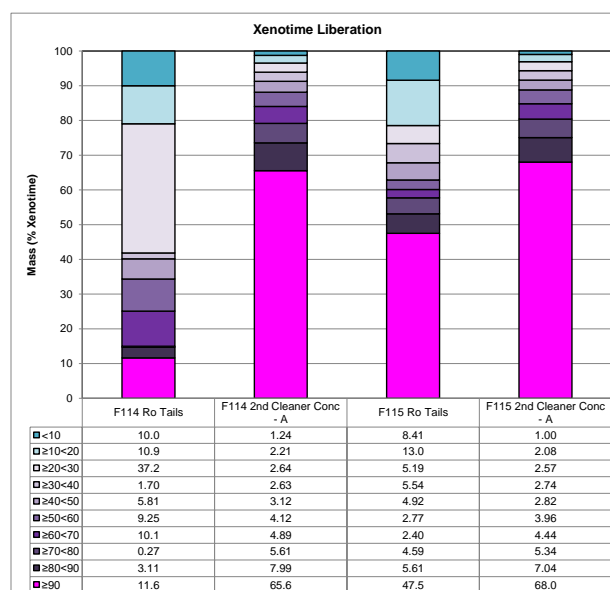
Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	0.01	4.59	0.02	3.77
≥80<90	0.00	0.56	0.00	0.39
≥70<80	0.00	0.39	0.00	0.30
≥60<70	0.01	0.34	0.00	0.25
≥50<60	0.01	0.29	0.00	0.22
≥40<50	0.00	0.22	0.00	0.16
≥30<40	0.00	0.18	0.00	0.15
≥20<30	0.02	0.18	0.00	0.14
≥10<20	0.01	0.15	0.01	0.11
<10	0.01	0.09	0.00	0.06
Total	0.06	7.00	0.05	5.54

Xenotime Exposure



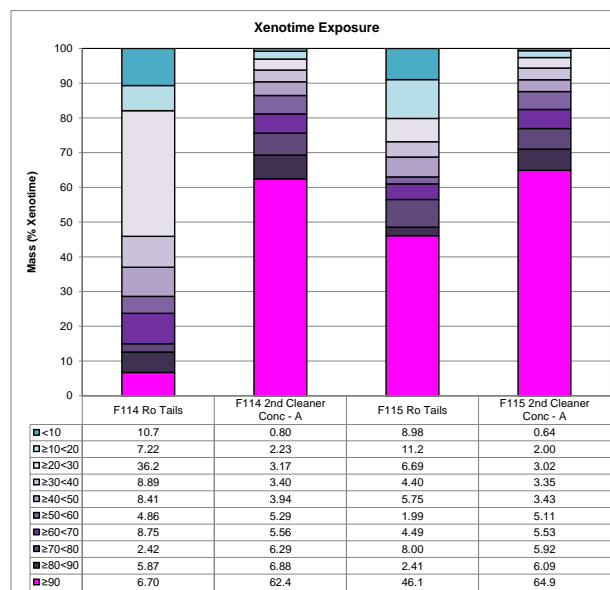
Absolute Mass of Xenotime Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	0.00	4.37	0.02	3.60
≥80<90	0.00	0.48	0.00	0.34
≥70<80	0.00	0.44	0.00	0.33
≥60<70	0.01	0.39	0.00	0.31
≥50<60	0.00	0.37	0.00	0.28
≥40<50	0.01	0.28	0.00	0.19
≥30<40	0.01	0.24	0.00	0.19
≥20<30	0.02	0.22	0.00	0.17
≥10<20	0.00	0.16	0.01	0.11
<10	0.01	0.06	0.00	0.04
Total	0.06	7.00	0.05	5.54



Normalized Mass of Xenotime Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	11.6	65.6	47.5	68.0
≥80<90	3.11	7.99	5.61	7.04
≥70<80	0.27	5.61	4.59	5.34
≥60<70	10.1	4.89	2.40	4.44
≥50<60	9.25	4.12	2.77	3.96
≥40<50	5.81	3.12	4.92	2.82
≥30<40	1.70	2.63	5.54	2.74
≥20<30	37.2	2.64	5.19	2.57
≥10<20	10.9	2.21	13.0	2.08
<10	10.0	1.24	8.41	1.00
Total	100.0	100.0	100.0	100.0

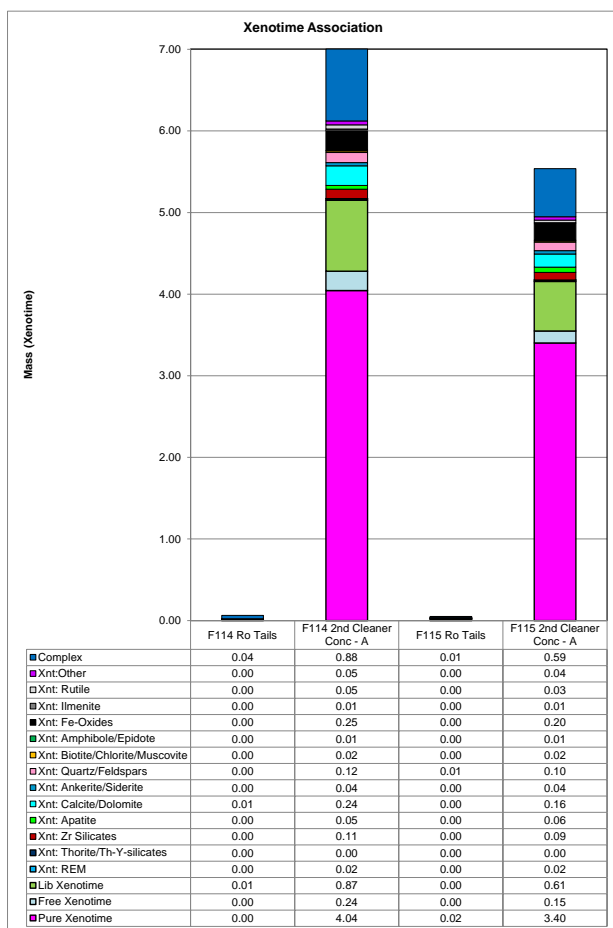


Normalized Mass of Xenotime Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	6.70	62.4	46.1	64.9
≥80<90	5.87	6.88	2.41	6.09
≥70<80	2.42	6.29	8.00	5.92
≥60<70	8.75	5.56	4.49	5.53
≥50<60	4.86	5.29	1.99	5.11
≥40<50	8.41	3.94	5.75	3.43
≥30<40	8.89	3.40	4.40	3.35
≥20<30	36.2	3.17	6.69	3.02
≥10<20	7.22	2.23	11.2	2.00
<10	10.7	0.80	8.98	0.64
Total	100.0	100.0	100.0	100.0

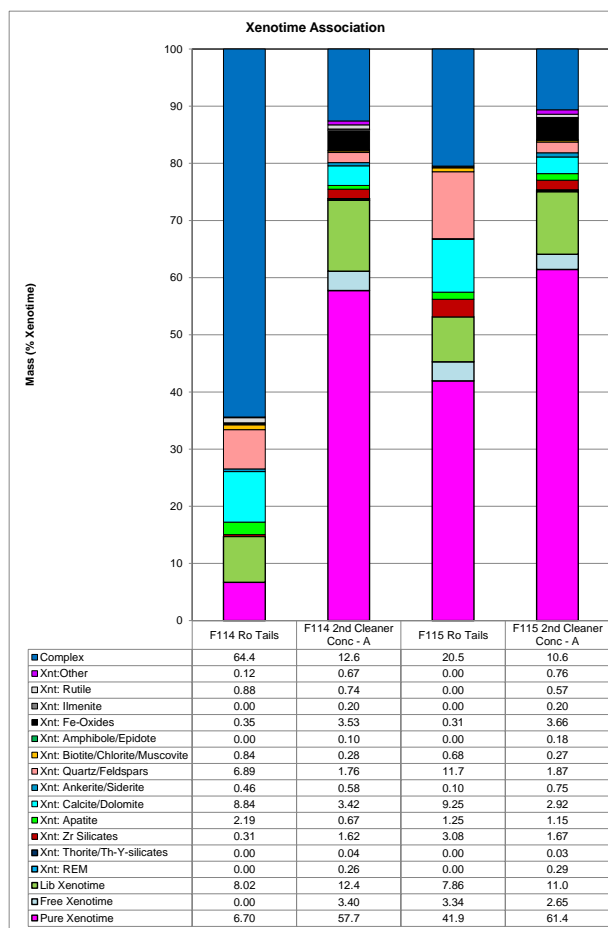
Namibia Critical Metals
CALR-18299-03
MI5067-SEP22

Xenotime Association



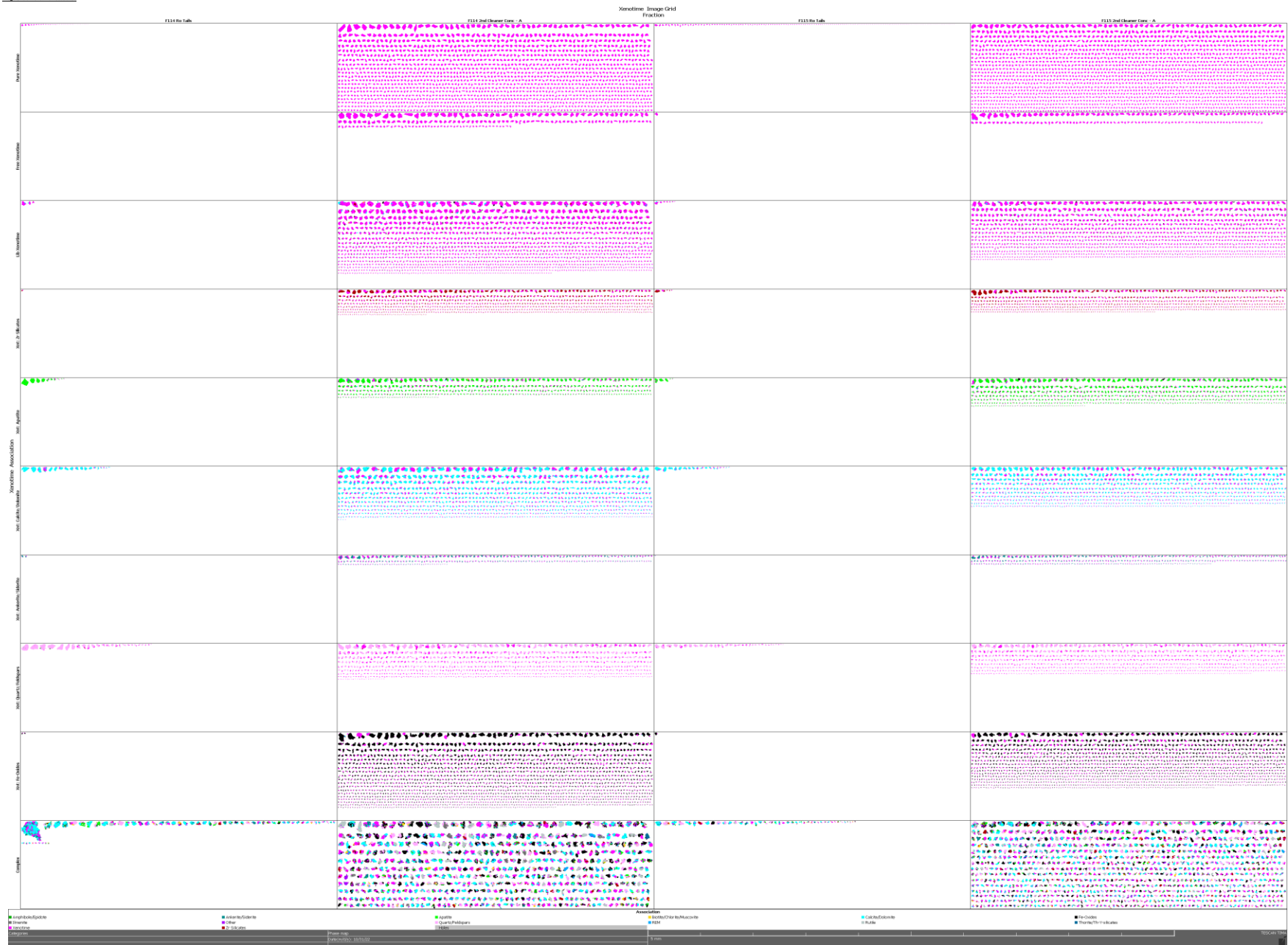
Absolute Mass of Xenotime Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
Pure Xenotime	0.00	4.04	0.02	3.40
Free Xenotime	0.00	0.24	0.00	0.15
Lib Xenotime	0.01	0.87	0.00	0.61
Xnt: REM	0.00	0.02	0.00	0.02
Xnt: Thorite/Th-Y-silicates	0.00	0.00	0.00	0.00
Xnt: Zr Silicates	0.00	0.11	0.00	0.09
Xnt: Apatite	0.00	0.05	0.00	0.06
Xnt: Calcite/Dolomite	0.01	0.24	0.00	0.16
Xnt: Ankerite/Siderite	0.00	0.04	0.00	0.04
Xnt: Quartz/Feldspars	0.00	0.12	0.01	0.10
Xnt: Biotite/Chlorite/Muscovite	0.00	0.02	0.00	0.02
Xnt: Amphibole/Epidote	0.00	0.01	0.00	0.01
Xnt: Fe-Oxides	0.00	0.25	0.00	0.20
Xnt: Ilmenite	0.00	0.01	0.00	0.01
Xnt: Rutile	0.00	0.05	0.00	0.03
Xnt:Other	0.00	0.05	0.00	0.04
Complex	0.04	0.88	0.01	0.59
Total	0.06	7.00	0.05	5.54



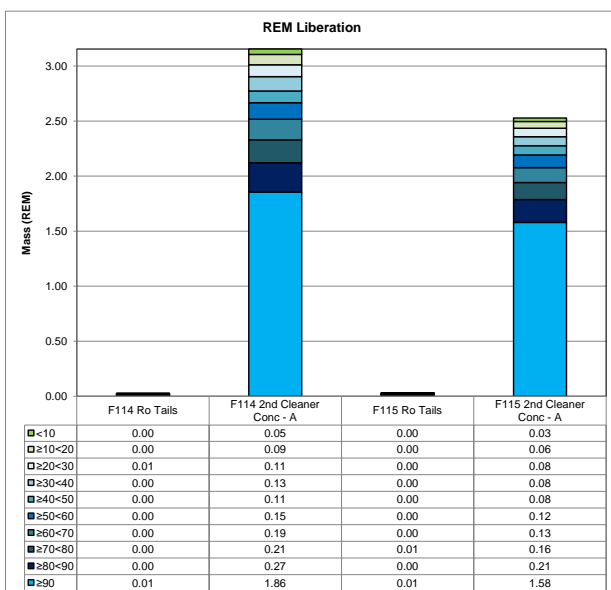
Normalized Mass of Xenotime Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
Pure Xenotime	6.70	57.7	41.9	61.4
Free Xenotime	0.00	3.40	3.34	2.65
Lib Xenotime	8.02	12.4	7.86	11.0
Xnt: REM	0.00	0.26	0.00	0.29
Xnt: Thorite/Th-Y-silicates	0.00	0.04	0.00	0.03
Xnt: Zr Silicates	0.31	1.62	3.08	1.67
Xnt: Apatite	2.19	0.67	1.25	1.15
Xnt: Calcite/Dolomite	8.84	3.42	9.25	2.92
Xnt: Ankerite/Siderite	0.46	0.58	0.10	0.75
Xnt: Quartz/Feldspars	6.89	1.76	11.7	1.87
Xnt: Biotite/Chlorite/Muscovite	0.84	0.28	0.68	0.27
Xnt: Amphibole/Epidote	0.00	0.10	0.00	0.18
Xnt: Fe-Oxides	0.35	3.53	0.31	3.66
Xnt: Ilmenite	0.00	0.20	0.00	0.20
Xnt: Rutile	0.88	0.74	0.00	0.57
Xnt:Other	0.12	0.67	0.00	0.76
Complex	64.4	12.6	20.5	10.6
Total	100.0	100.0	100.0	100.0
Liberated	14.7	73.5	53.1	75.1



Namibia Critical Metals
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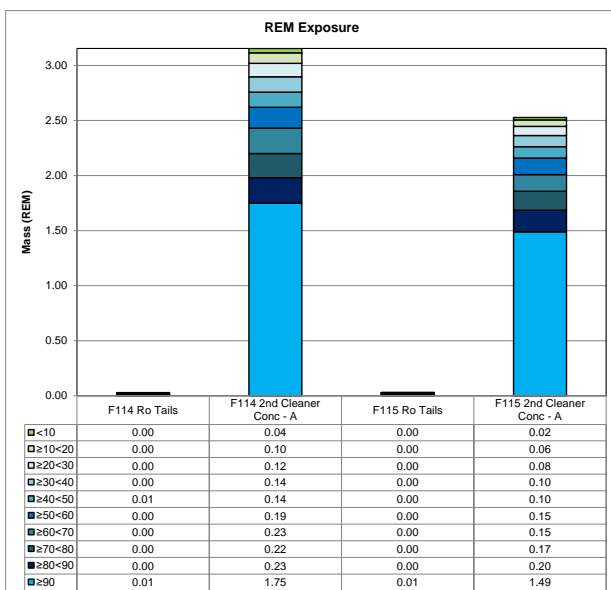
REM Liberation



Absolute Mass of REM Across Samples

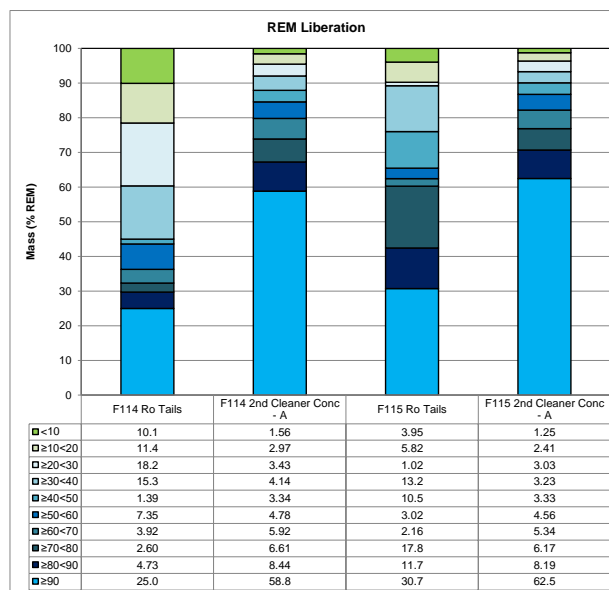
Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	0.01	1.86	0.01	1.58
≥80<90	0.00	0.27	0.00	0.21
≥70<80	0.00	0.21	0.01	0.16
≥60<70	0.00	0.19	0.00	0.13
≥50<60	0.00	0.15	0.00	0.12
≥40<50	0.00	0.11	0.00	0.08
≥30<40	0.00	0.13	0.00	0.08
≥20<30	0.01	0.11	0.00	0.08
≥10<20	0.00	0.09	0.00	0.06
<10	0.00	0.05	0.00	0.03
Total	0.03	3.15	0.03	2.53

REM Exposure



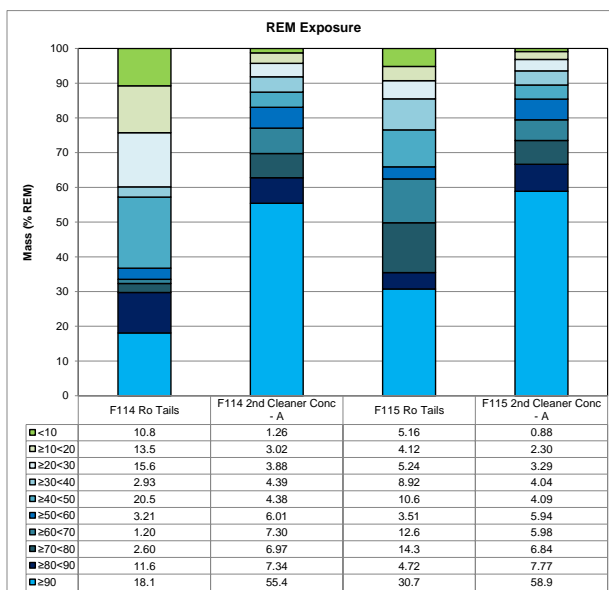
Absolute Mass of REM Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	0.01	1.75	0.01	1.49
≥80<90	0.00	0.23	0.00	0.20
≥70<80	0.00	0.22	0.00	0.17
≥60<70	0.00	0.23	0.00	0.15
≥50<60	0.00	0.19	0.00	0.15
≥40<50	0.01	0.14	0.00	0.10
≥30<40	0.00	0.14	0.00	0.10
≥20<30	0.00	0.12	0.00	0.08
≥10<20	0.00	0.10	0.00	0.06
<10	0.00	0.04	0.00	0.02
Total	0.03	3.15	0.03	2.53



Normalized Mass of REM Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	25.0	58.8	30.7	62.5
≥80<90	4.73	8.44	11.7	8.19
≥70<80	2.60	6.61	17.8	6.17
≥60<70	3.92	5.92	2.16	5.34
≥50<60	7.35	4.78	3.02	4.56
≥40<50	1.39	3.34	10.5	3.33
≥30<40	15.3	4.14	13.2	3.23
≥20<30	18.2	3.43	1.02	3.03
≥10<20	11.4	2.97	5.82	2.41
<10	10.1	1.56	3.95	1.25
Total	100.0	100.0	100.0	100.0

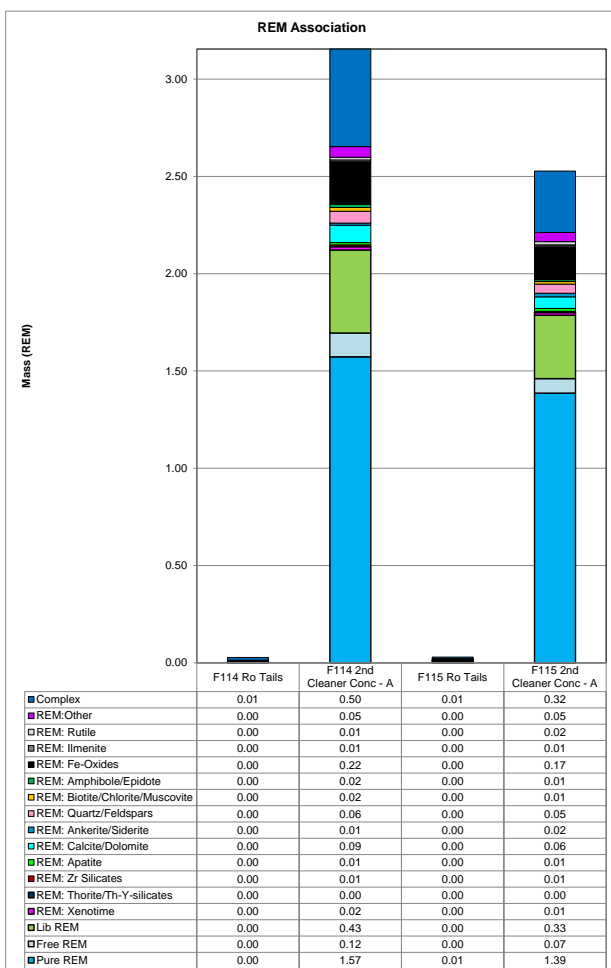


Normalized Mass of REM Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	18.1	55.4	30.7	58.9
≥80<90	11.6	7.34	4.72	7.77
≥70<80	2.60	6.97	14.3	6.84
≥60<70	1.20	7.30	12.6	5.98
≥50<60	3.21	6.01	3.51	5.94
≥40<50	20.5	4.38	10.6	4.09
≥30<40	2.93	4.39	8.92	4.04
≥20<30	15.6	3.88	5.24	3.29
≥10<20	13.5	3.02	4.12	2.30
<10	10.8	1.26	5.16	0.88
Total	100.0	100.0	100.0	100.0

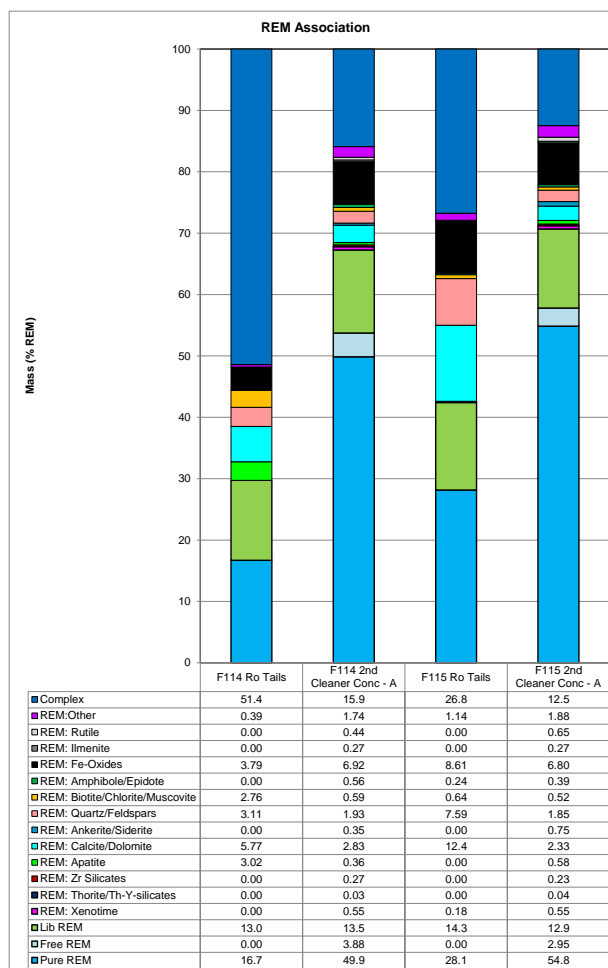
Namibia Critical Metals
CALR-18299-03
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REM Association



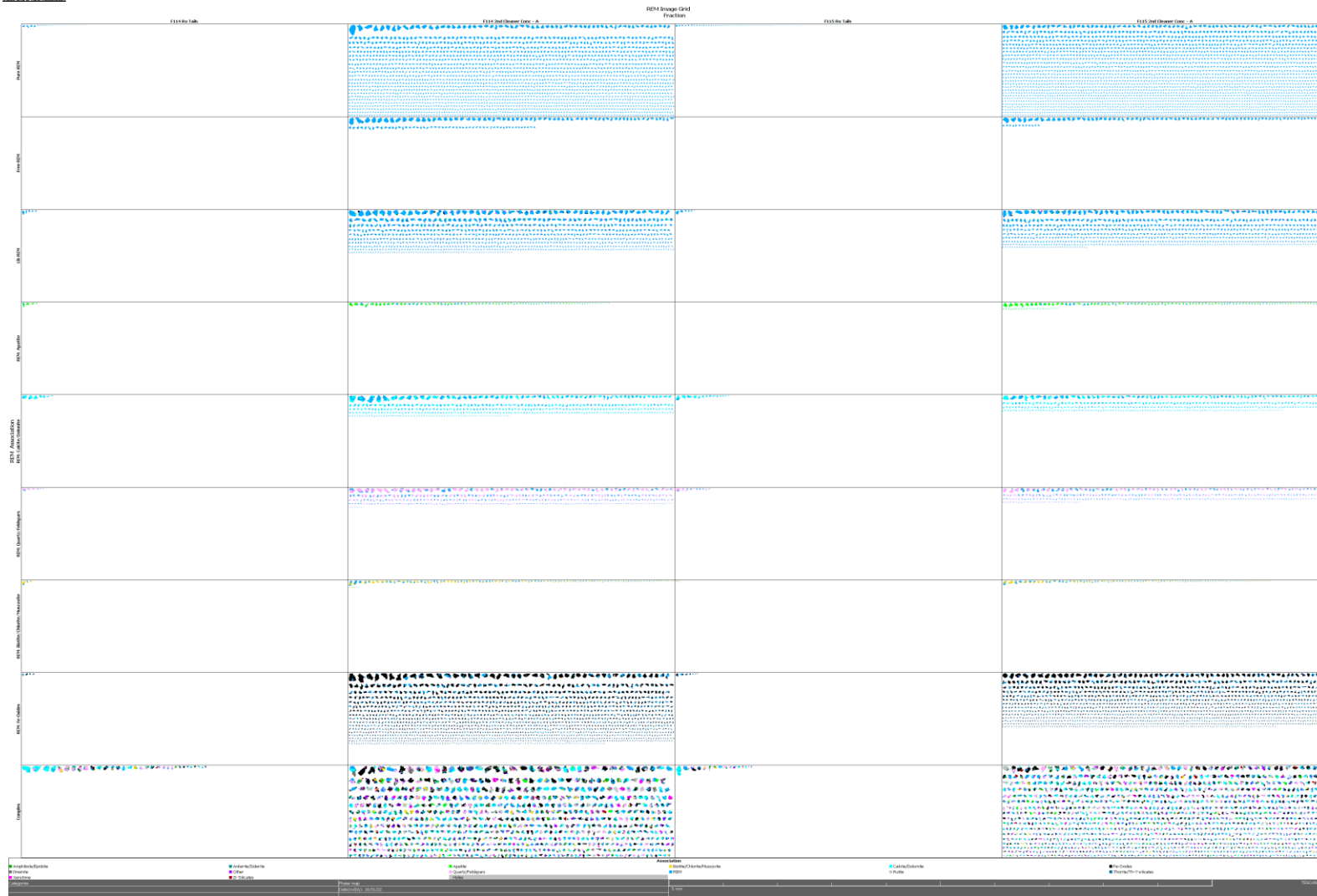
Absolute Mass of REM Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
Pure REM	0.00	1.57	0.01	1.39
Free REM	0.00	0.12	0.00	0.07
Lib REM	0.00	0.43	0.00	0.33
REM: Xenotime	0.00	0.02	0.00	0.01
REM: Thorite/Th-Y-silicates	0.00	0.00	0.00	0.00
REM: Zr Silicates	0.00	0.01	0.00	0.01
REM: Apatite	0.00	0.01	0.00	0.01
REM: Calcite/Dolomite	0.00	0.09	0.00	0.06
REM: Ankerite/Siderite	0.00	0.01	0.00	0.02
REM: Quartz/Feldspars	0.00	0.06	0.00	0.05
REM: Biotite/Chlorite/Muscovite	0.00	0.02	0.00	0.01
REM: Amphibole/Epidote	0.00	0.02	0.00	0.01
REM: Fe-Oxides	0.00	0.22	0.00	0.17
REM: Ilmenite	0.00	0.01	0.00	0.01
REM: Rutile	0.00	0.01	0.00	0.02
REM: Other	0.00	0.05	0.00	0.05
Complex	0.01	0.50	0.01	0.32
Total	0.03	3.15	0.03	2.53

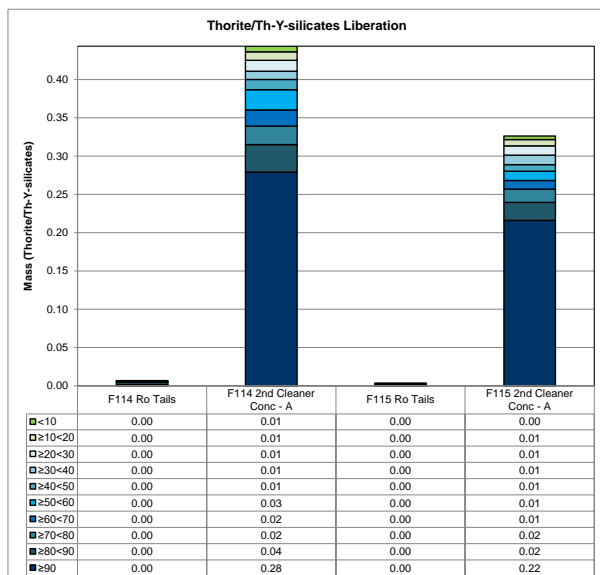


Normalized Mass of REM Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
Pure REM	16.7	49.9	28.1	54.8
Free REM	0.00	3.88	0.00	2.95
Lib REM	13.0	13.5	14.3	12.9
REM: Xenotime	0.00	0.55	0.18	0.55
REM: Thorite/Th-Y-silicates	0.00	0.03	0.00	0.04
REM: Zr Silicates	0.00	0.27	0.00	0.23
REM: Apatite	3.02	0.36	0.00	0.58
REM: Calcite/Dolomite	5.77	2.83	12.4	2.33
REM: Ankerite/Siderite	0.00	0.35	0.00	0.75
REM: Quartz/Feldspars	3.11	1.93	7.59	1.85
REM: Biotite/Chlorite/Muscovite	2.76	0.59	0.64	0.52
REM: Amphibole/Epidote	0.00	0.56	0.24	0.39
REM: Fe-Oxides	3.79	6.92	8.61	6.80
REM: Ilmenite	0.00	0.27	0.00	0.27
REM: Rutile	0.00	0.44	0.00	0.65
REM: Other	0.39	1.74	1.14	1.88
Complex	51.4	15.9	26.8	12.5
Total	100.0	100.0	100.0	100.0
Liberated	29.7	67.3	42.4	70.7



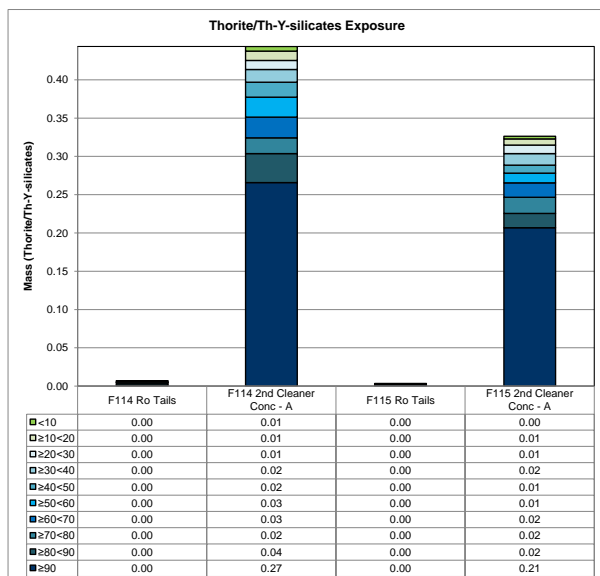
Thorite/Th-Y-silicates Liberation



Absolute Mass of Thorite/Th-Y-silicates Across Samples

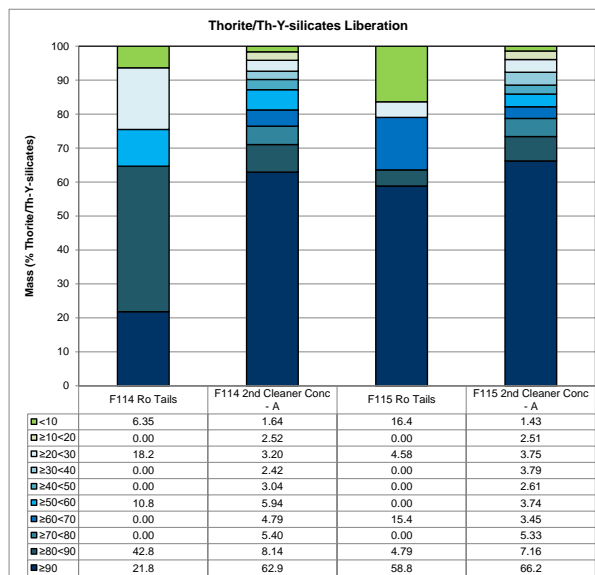
Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	0.00	0.28	0.00	0.22
≥80<90	0.00	0.04	0.00	0.02
≥70<80	0.00	0.02	0.00	0.02
≥60<70	0.00	0.02	0.00	0.01
≥50<60	0.00	0.03	0.00	0.01
≥40<50	0.00	0.01	0.00	0.01
≥30<40	0.00	0.01	0.00	0.01
≥20<30	0.00	0.01	0.00	0.01
≥10<20	0.00	0.01	0.00	0.01
<10	0.00	0.01	0.00	0.00
Total	0.01	0.44	0.00	0.33

Thorite/Th-Y-silicates Exposure



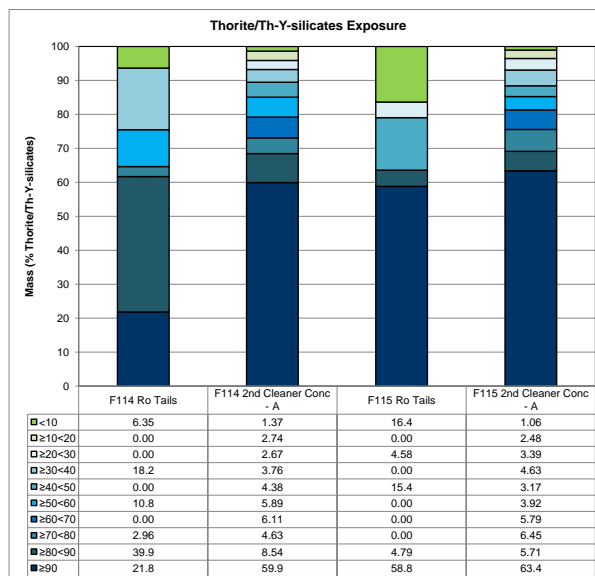
Absolute Mass of Thorite/Th-Y-silicates Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	0.00	0.27	0.00	0.21
≥80<90	0.00	0.04	0.00	0.02
≥70<80	0.00	0.02	0.00	0.02
≥60<70	0.00	0.03	0.00	0.02
≥50<60	0.00	0.03	0.00	0.01
≥40<50	0.00	0.02	0.00	0.01
≥30<40	0.00	0.02	0.00	0.02
≥20<30	0.00	0.01	0.00	0.01
≥10<20	0.00	0.01	0.00	0.01
<10	0.00	0.01	0.00	0.00
Total	0.01	0.44	0.00	0.33



Normalized Mass of Thorite/Th-Y-silicates Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	21.8	62.9	58.8	66.2
≥80<90	42.8	8.14	4.79	7.16
≥70<80	0.00	5.40	0.00	5.33
≥60<70	0.00	4.79	15.4	3.45
≥50<60	10.8	5.94	0.00	3.74
≥40<50	0.00	3.04	0.00	2.61
≥30<40	0.00	2.42	0.00	3.79
≥20<30	18.2	3.20	4.58	3.75
≥10<20	0.00	2.52	0.00	2.51
<10	6.35	1.64	16.4	1.43
Total	100.0	100.0	100.0	100.0

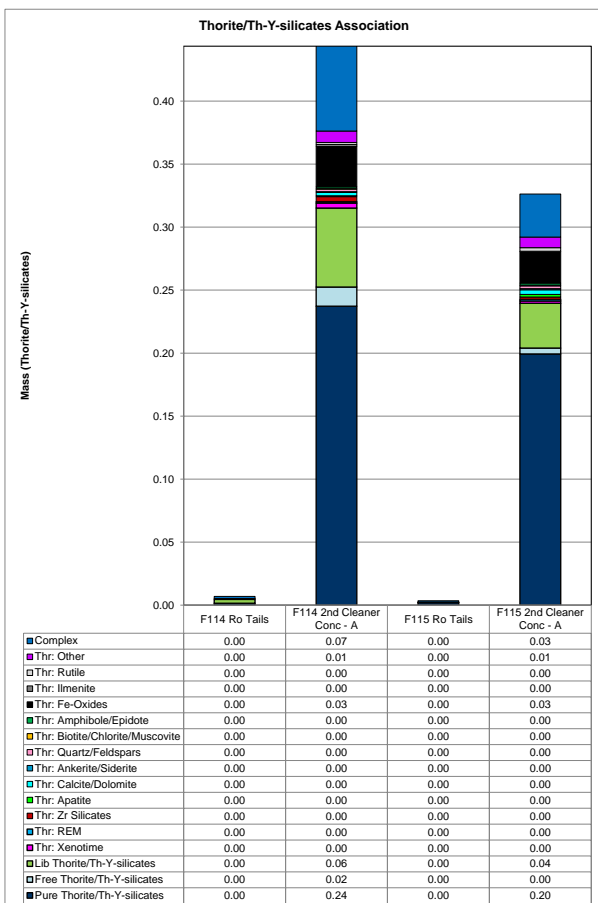


Normalized Mass of Thorite/Th-Y-silicates Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	21.8	59.9	58.8	63.4
≥80<90	39.9	8.54	4.79	5.71
≥70<80	2.96	4.63	0.00	6.45
≥60<70	0.00	6.11	0.00	5.79
≥50<60	10.8	5.89	0.00	3.92
≥40<50	0.00	4.38	15.4	3.17
≥30<40	18.2	3.76	0.00	4.63
≥20<30	0.00	2.67	4.58	3.39
≥10<20	0.00	2.74	0.00	2.48
<10	6.35	1.37	16.4	1.06
Total	100.0	100.0	100.0	100.0

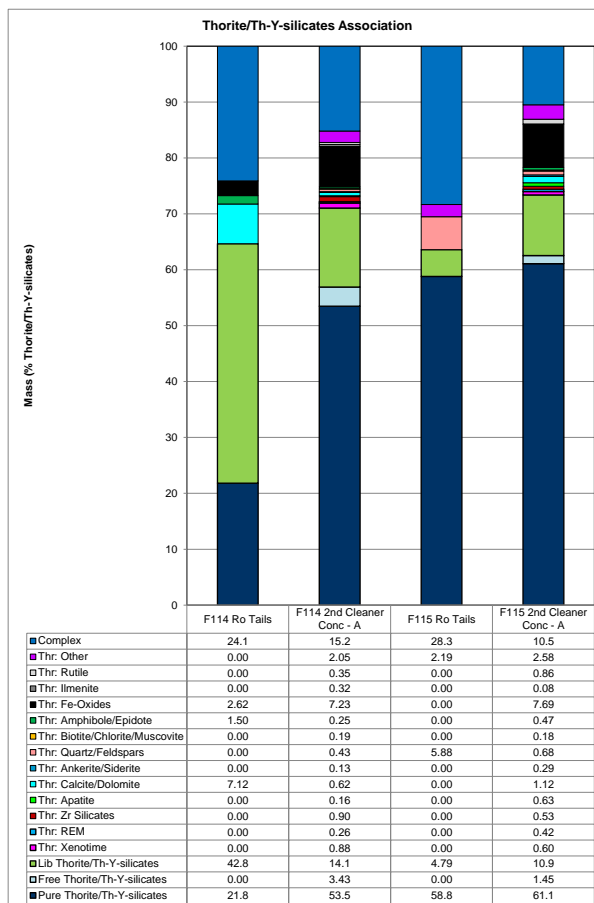
Namibia Critical Metals
CALR-18299-03
MI5067-SEP22

Thorite/Th-Y-silicates Association



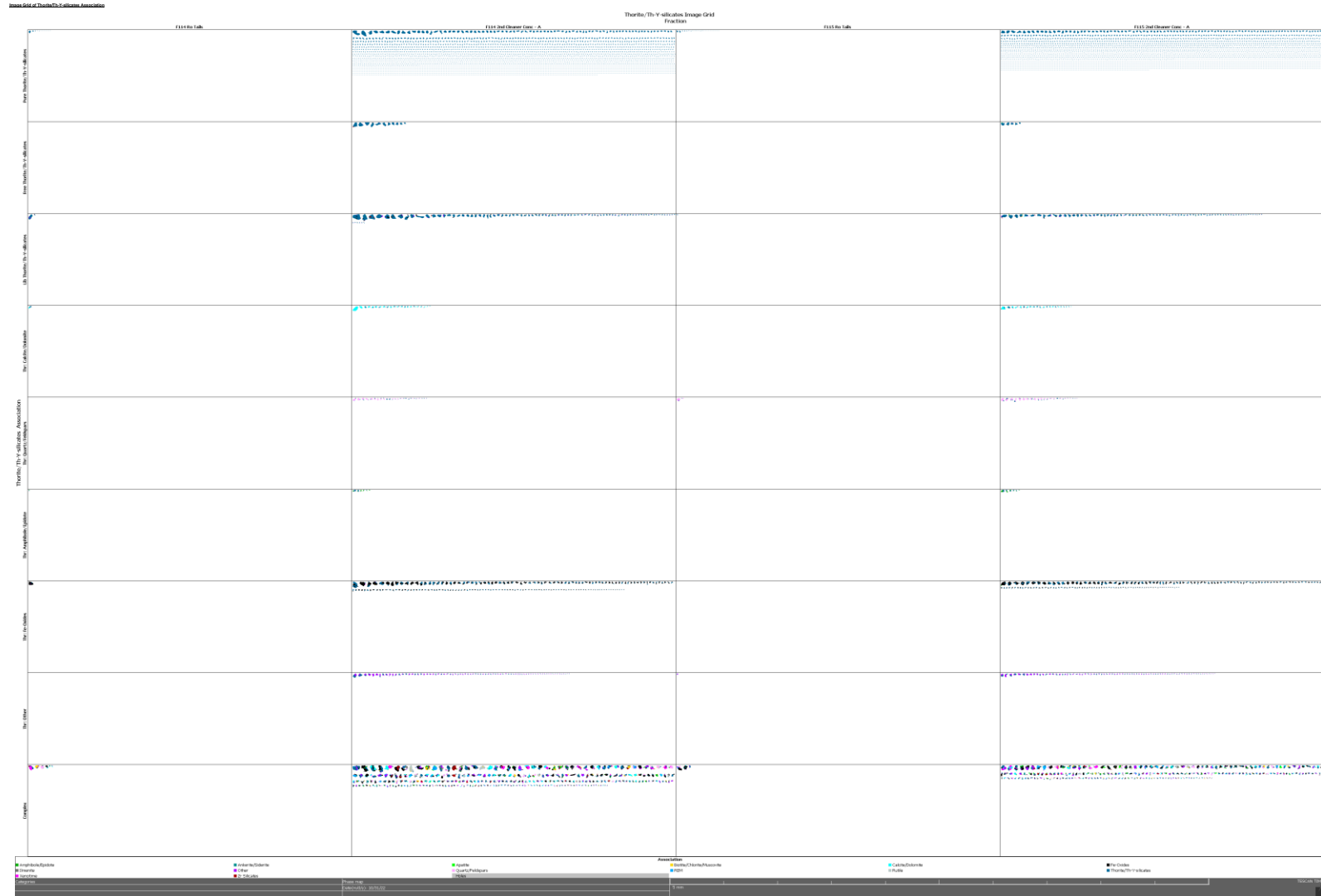
Absolute Mass of Thorite/Th-Y-silicates Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
Pure Thorite/Th-Y-silicates	0.00	0.24	0.00	0.20
Free Thorite/Th-Y-silicates	0.00	0.02	0.00	0.00
Lib Thorite/Th-Y-silicates	0.00	0.06	0.00	0.04
Thr: Xenotime	0.00	0.00	0.00	0.00
Thr: REM	0.00	0.00	0.00	0.00
Thr: Zr Silicates	0.00	0.00	0.00	0.00
Thr: Apatite	0.00	0.00	0.00	0.00
Thr: Calcite/Dolomite	0.00	0.00	0.00	0.00
Thr: Ankerite/Siderite	0.00	0.00	0.00	0.00
Thr: Quartz/Feldspars	0.00	0.00	0.00	0.00
Thr: Biotite/Chlorite/Muscovite	0.00	0.00	0.00	0.00
Thr: Amphibole/Epidote	0.00	0.00	0.00	0.00
Thr: Fe-Oxides	0.00	0.03	0.00	0.03
Thr: Ilmenite	0.00	0.00	0.00	0.00
Thr: Rutile	0.00	0.00	0.00	0.00
Thr: Other	0.00	0.01	0.00	0.01
Complex	0.00	0.07	0.00	0.03
Total	0.01	0.44	0.00	0.33

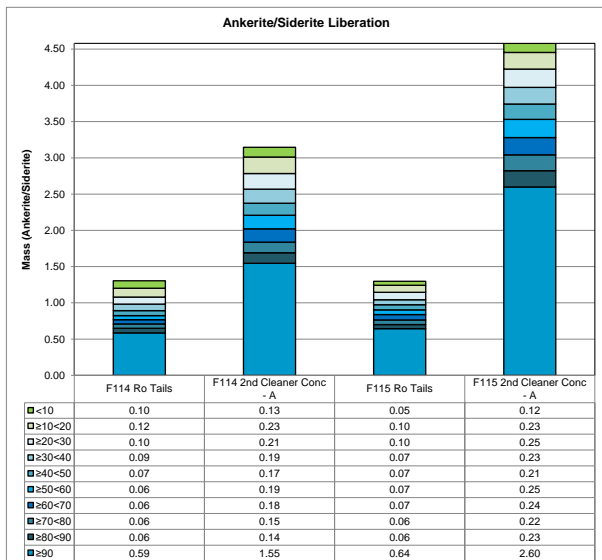


Normalized Mass of Thorite/Th-Y-silicates Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
Pure Thorite/Th-Y-silicates	21.8	53.5	58.8	61.1
Free Thorite/Th-Y-silicates	0.00	3.43	0.00	1.45
Lib Thorite/Th-Y-silicates	42.8	14.1	4.79	10.9
Thr: Xenotime	0.00	0.88	0.00	0.60
Thr: REM	0.00	0.26	0.00	0.42
Thr: Zr Silicates	0.00	0.90	0.00	0.53
Thr: Apatite	0.00	0.16	0.00	0.63
Thr: Calcite/Dolomite	7.12	0.62	0.00	1.12
Thr: Ankerite/Siderite	0.00	0.13	0.00	0.29
Thr: Quartz/Feldspars	0.00	0.43	5.88	0.68
Thr: Biotite/Chlorite/Muscovite	0.00	0.19	0.00	0.18
Thr: Amphibole/Epidote	1.50	0.25	0.00	0.47
Thr: Fe-Oxides	2.62	7.23	0.00	7.69
Thr: Ilmenite	0.00	0.32	0.00	0.08
Thr: Rutile	0.00	0.35	0.00	0.86
Thr: Other	0.00	2.05	2.19	2.58
Complex	24.1	15.2	28.3	10.5
Total	100.0	100.0	100.0	100.0
Liberated	64.7	71.1	63.6	73.4



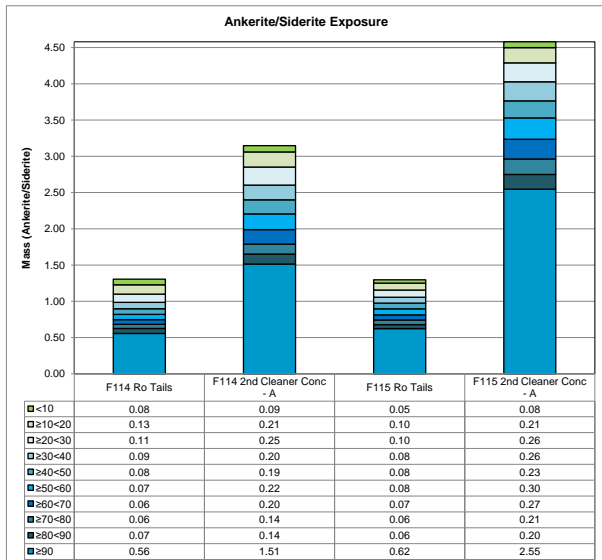
Ankerite/Siderite Liberation



Absolute Mass of Ankerite/Siderite Across Samples

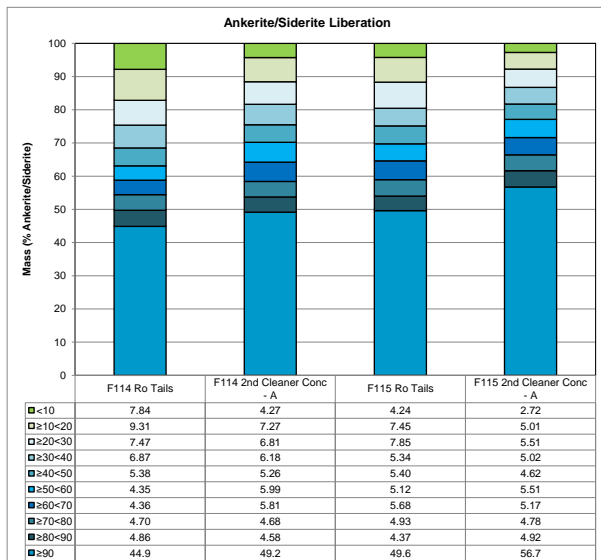
Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	0.59	1.55	0.64	2.60
≥80<90	0.06	0.14	0.06	0.23
≥70<80	0.06	0.15	0.06	0.22
≥60<70	0.06	0.18	0.07	0.24
≥50<60	0.06	0.19	0.07	0.25
≥40<50	0.07	0.17	0.07	0.21
≥30<40	0.09	0.19	0.07	0.23
≥20<30	0.10	0.21	0.10	0.25
≥10<20	0.12	0.23	0.10	0.23
<10	0.10	0.13	0.05	0.12
Total	1.30	3.15	1.30	4.58

Ankerite/Siderite Exposure



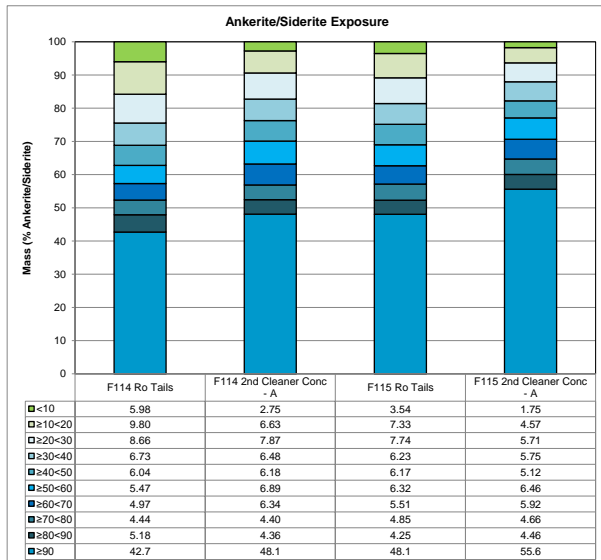
Absolute Mass of Ankerite/Siderite Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	0.56	1.51	0.62	2.55
≥80<90	0.07	0.14	0.06	0.20
≥70<80	0.06	0.14	0.06	0.21
≥60<70	0.06	0.20	0.07	0.27
≥50<60	0.07	0.22	0.08	0.30
≥40<50	0.08	0.19	0.08	0.23
≥30<40	0.09	0.20	0.08	0.26
≥20<30	0.11	0.25	0.10	0.26
≥10<20	0.13	0.21	0.10	0.21
<10	0.08	0.09	0.05	0.08
Total	1.30	3.15	1.30	4.58



Normalized Mass of Ankerite/Siderite Across Samples

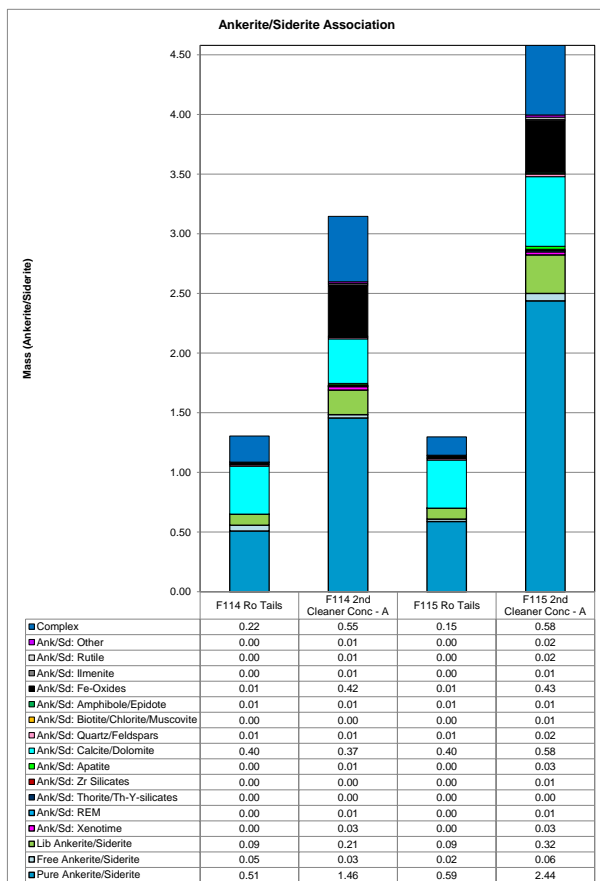
Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	44.9	49.2	49.6	56.7
≥80<90	4.86	4.58	4.37	4.92
≥70<80	4.70	4.68	4.93	4.78
≥60<70	4.36	5.81	5.68	5.17
≥50<60	4.35	5.99	5.12	5.51
≥40<50	5.38	5.26	5.40	4.62
≥30<40	6.87	6.18	5.34	5.02
≥20<30	7.47	6.81	7.85	5.51
≥10<20	9.31	7.27	7.45	5.01
<10	7.84	4.27	4.24	2.72
Total	100.0	100.0	100.0	100.0



Normalized Mass of Ankerite/Siderite Across Samples

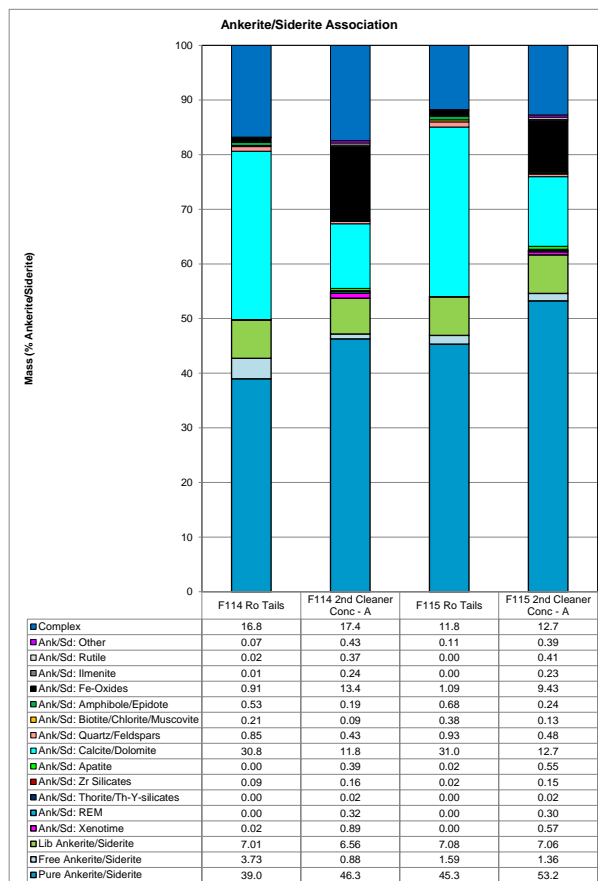
Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	42.7	48.1	48.1	55.6
≥80<90	5.18	4.36	4.25	4.46
≥70<80	4.44	4.40	4.85	4.66
≥60<70	4.97	6.34	5.51	5.92
≥50<60	5.47	6.89	6.32	6.46
≥40<50	6.04	6.18	6.17	5.12
≥30<40	6.73	6.48	6.23	5.75
≥20<30	8.66	7.87	7.74	5.71
≥10<20	9.80	6.63	7.33	4.57
<10	5.98	2.75	3.54	1.75
Total	100.0	100.0	100.0	100.0

Ankerite/Siderite Association



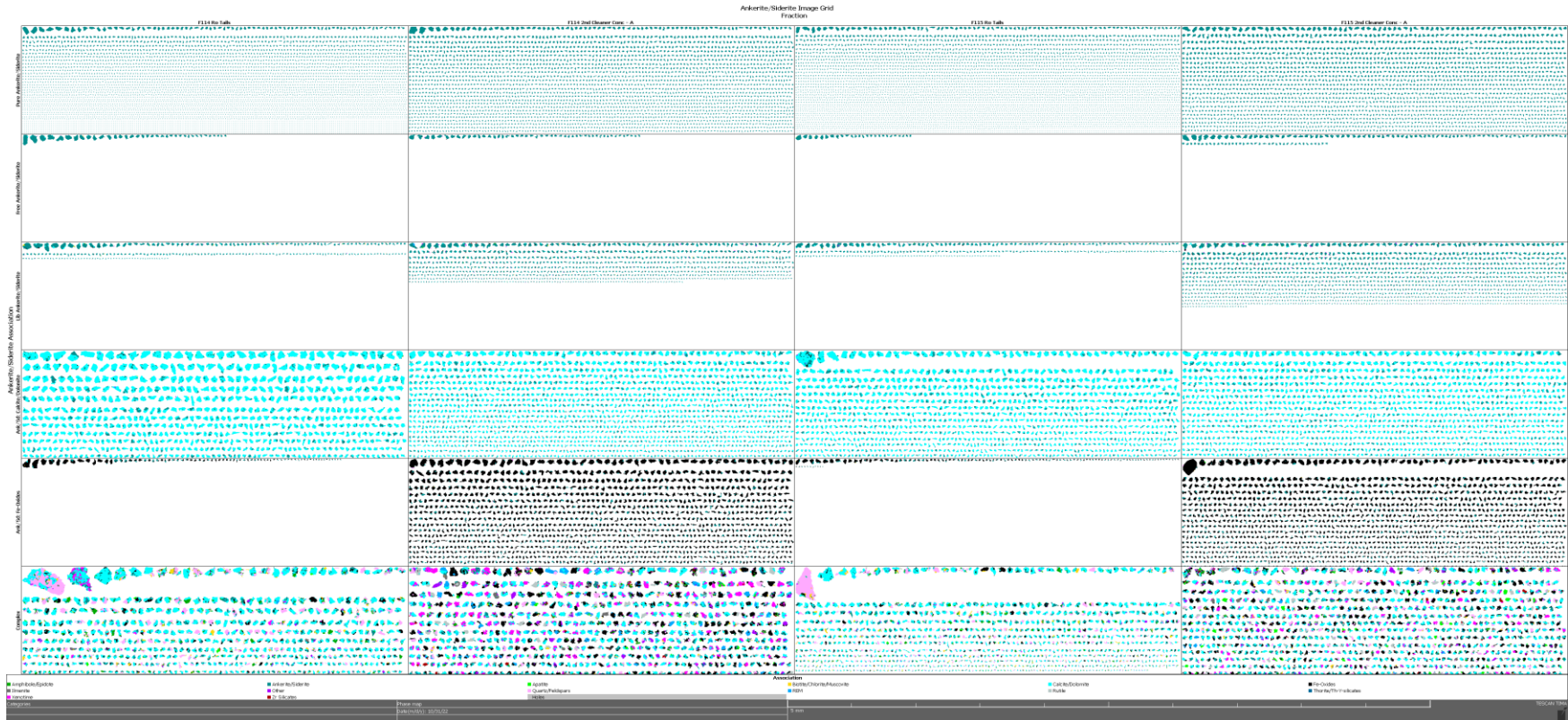
Absolute Mass of Ankerite/Siderite Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
Pure Ankerite/Siderite	0.51	1.46	0.59	2.44
Free Ankerite/Siderite	0.05	0.03	0.02	0.06
Lib Ankerite/Siderite	0.09	0.21	0.09	0.32
Ank/Sd: Xenotime	0.00	0.03	0.00	0.03
Ank/Sd: REM	0.00	0.01	0.00	0.01
Ank/Sd: Thorite/Th-Y-silicates	0.00	0.00	0.00	0.00
Ank/Sd: Zr Silicates	0.00	0.00	0.00	0.01
Ank/Sd: Apatite	0.00	0.01	0.00	0.03
Ank/Sd: Calcite/Dolomite	0.40	0.37	0.40	0.58
Ank/Sd: Quartz/Feldspars	0.01	0.01	0.01	0.02
Ank/Sd: Biotite/Chlorite/Muscovite	0.00	0.00	0.00	0.01
Ank/Sd: Amphibole/Epidote	0.01	0.01	0.01	0.01
Ank/Sd: Fe-Oxides	0.01	0.42	0.01	0.43
Ank/Sd: Ilmenite	0.00	0.01	0.00	0.01
Ank/Sd: Rutile	0.00	0.01	0.00	0.02
Ank/Sd: Other	0.00	0.01	0.00	0.02
Complex	0.22	0.55	0.15	0.58
Total	1.30	3.15	1.30	4.58

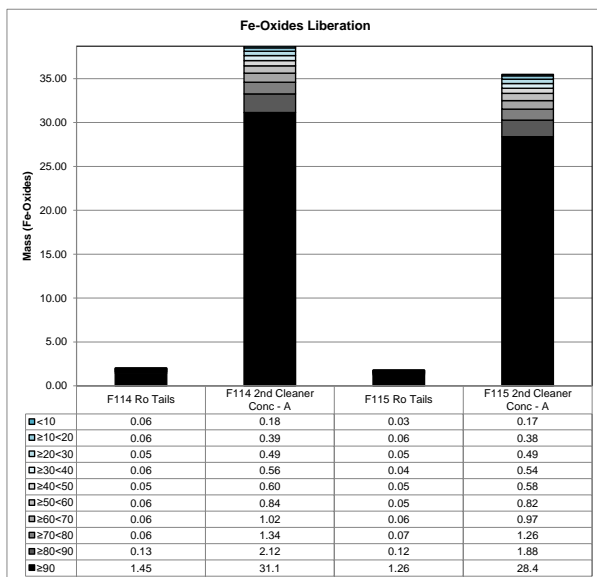


Normalized Mass of Ankerite/Siderite Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
Pure Ankerite/Siderite	39.0	46.3	45.3	53.2
Free Ankerite/Siderite	3.73	0.88	1.59	1.36
Lib Ankerite/Siderite	7.01	6.56	7.08	7.06
Ank/Sd: Xenotime	0.02	0.89	0.00	0.57
Ank/Sd: REM	0.00	0.32	0.00	0.30
Ank/Sd: Thorite/Th-Y-silicates	0.00	0.02	0.00	0.02
Ank/Sd: Zr Silicates	0.09	0.16	0.02	0.15
Ank/Sd: Apatite	0.00	0.39	0.02	0.55
Ank/Sd: Calcite/Dolomite	30.8	11.8	31.0	12.7
Ank/Sd: Quartz/Feldspars	0.85	0.43	0.93	0.48
Ank/Sd: Biotite/Chlorite/Muscovite	0.21	0.09	0.38	0.13
Ank/Sd: Amphibole/Epidote	0.53	0.19	0.68	0.24
Ank/Sd: Fe-Oxides	0.91	13.4	1.09	9.43
Ank/Sd: Ilmenite	0.01	0.24	0.00	0.23
Ank/Sd: Rutile	0.02	0.37	0.00	0.41
Ank/Sd: Other	0.07	0.43	0.11	0.39
Complex	16.8	17.4	11.8	12.7
Total	100.0	100.0	100.0	100.0
Liberated	49.7	53.7	54.0	61.7



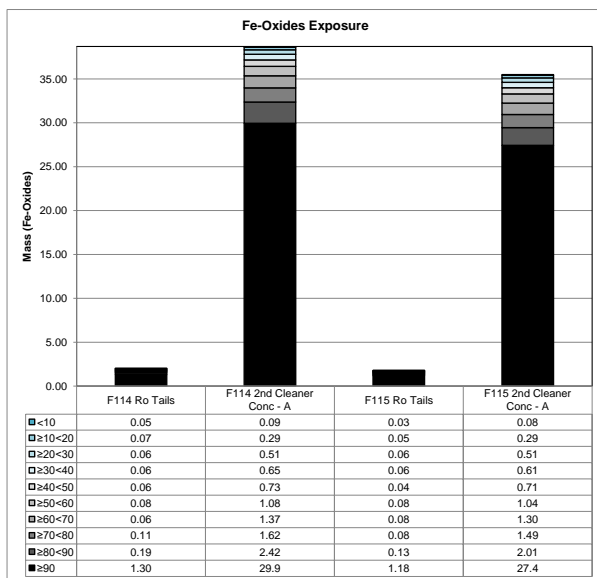
Fe-Oxides Liberation



Absolute Mass of Fe-Oxides Across Samples

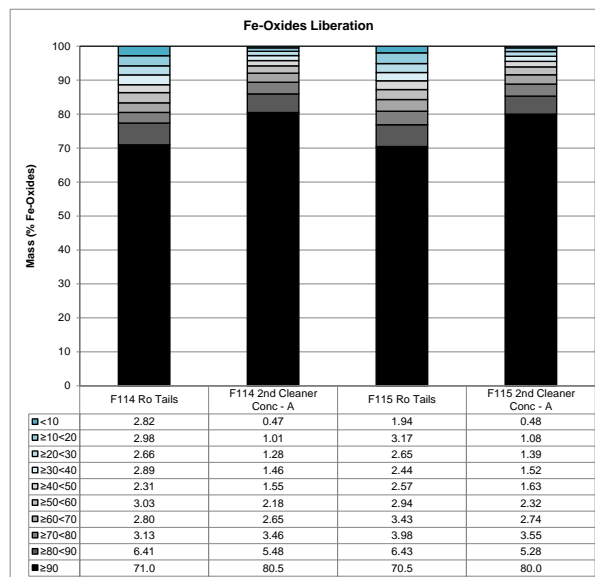
Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	1.45	31.1	1.26	28.4
≥80<90	0.13	2.12	0.12	1.88
≥70<80	0.06	1.34	0.07	1.26
≥60<70	0.06	1.02	0.06	0.97
≥50<60	0.06	0.84	0.05	0.82
≥40<50	0.05	0.60	0.05	0.58
≥30<40	0.06	0.56	0.04	0.54
≥20<30	0.05	0.49	0.05	0.49
≥10<20	0.06	0.39	0.06	0.38
<10	0.06	0.18	0.03	0.17
Total	2.04	38.7	1.79	35.5

Fe-Oxides Exposure



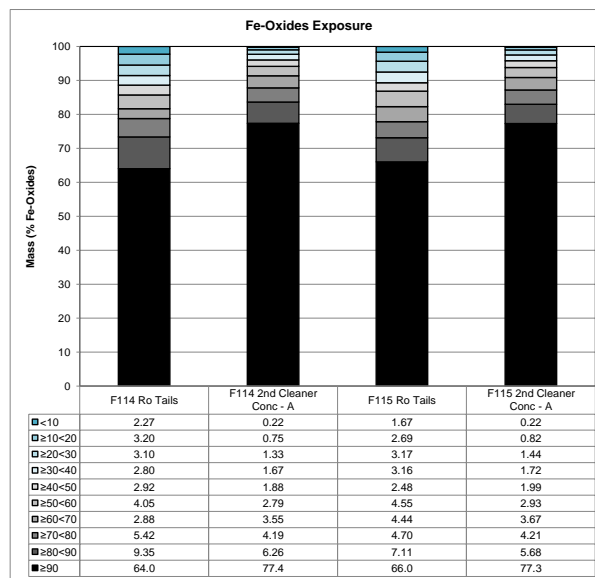
Absolute Mass of Fe-Oxides Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	1.30	29.9	1.18	27.4
≥80<90	0.19	2.42	0.13	2.01
≥70<80	0.11	1.62	0.08	1.49
≥60<70	0.06	1.37	0.08	1.30
≥50<60	0.08	1.08	0.08	1.04
≥40<50	0.06	0.73	0.04	0.71
≥30<40	0.06	0.65	0.06	0.61
≥20<30	0.06	0.51	0.06	0.51
≥10<20	0.07	0.29	0.05	0.29
<10	0.05	0.09	0.03	0.08
Total	2.04	38.7	1.79	35.5



Normalized Mass of Fe-Oxides Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	71.0	80.5	70.5	80.0
≥80<90	6.41	5.48	6.43	5.28
≥70<80	3.13	3.46	3.98	3.55
≥60<70	2.80	2.65	3.43	2.74
≥50<60	3.03	2.18	2.94	2.32
≥40<50	2.31	1.55	2.57	1.63
≥30<40	2.89	1.46	2.44	1.52
≥20<30	2.66	1.28	2.65	1.39
≥10<20	2.98	1.01	3.17	1.08
<10	2.82	0.47	1.94	0.48
Total	100.0	100.0	100.0	100.0

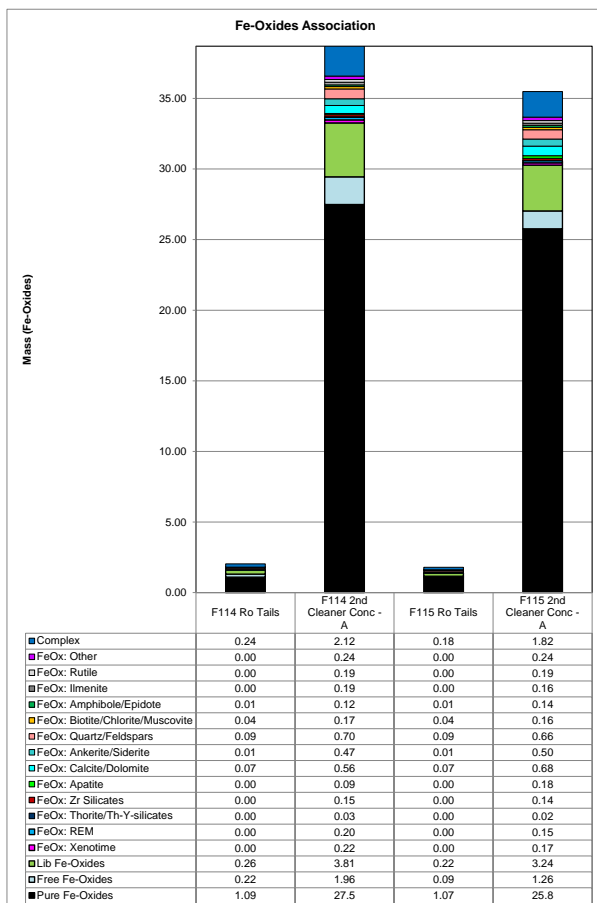


Normalized Mass of Fe-Oxides Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
≥90	64.0	77.4	66.0	77.3
≥80<90	9.35	6.26	7.11	5.68
≥70<80	5.42	4.19	4.70	4.21
≥60<70	2.88	3.55	4.44	3.67
≥50<60	4.05	2.79	4.55	2.93
≥40<50	2.92	1.88	2.48	1.99
≥30<40	2.80	1.67	3.16	1.72
≥20<30	3.10	1.33	3.17	1.44
≥10<20	3.20	0.75	2.69	0.82
<10	2.27	0.22	1.67	0.22
Total	100.0	100.0	100.0	100.0

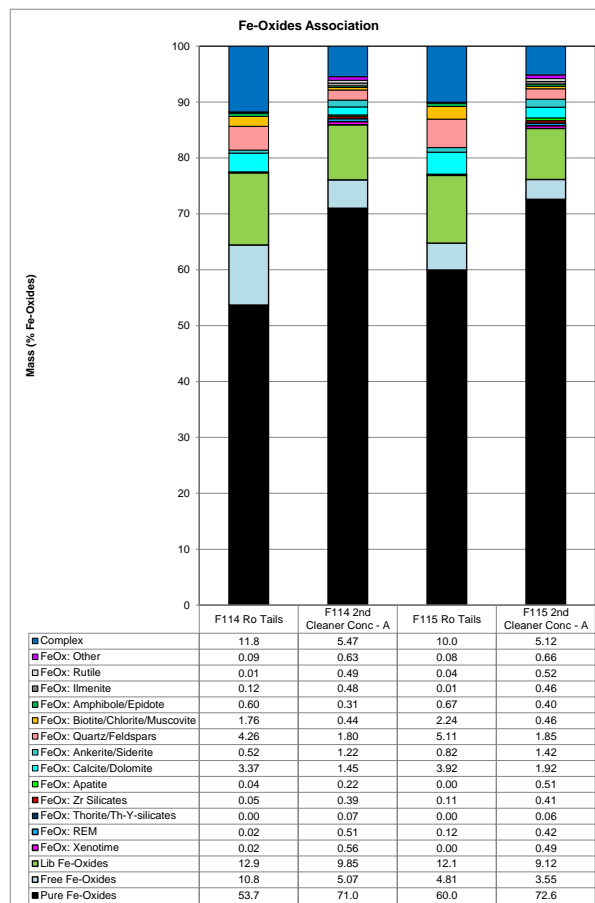
Namibia Critical Metals
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Fe-Oxides Association



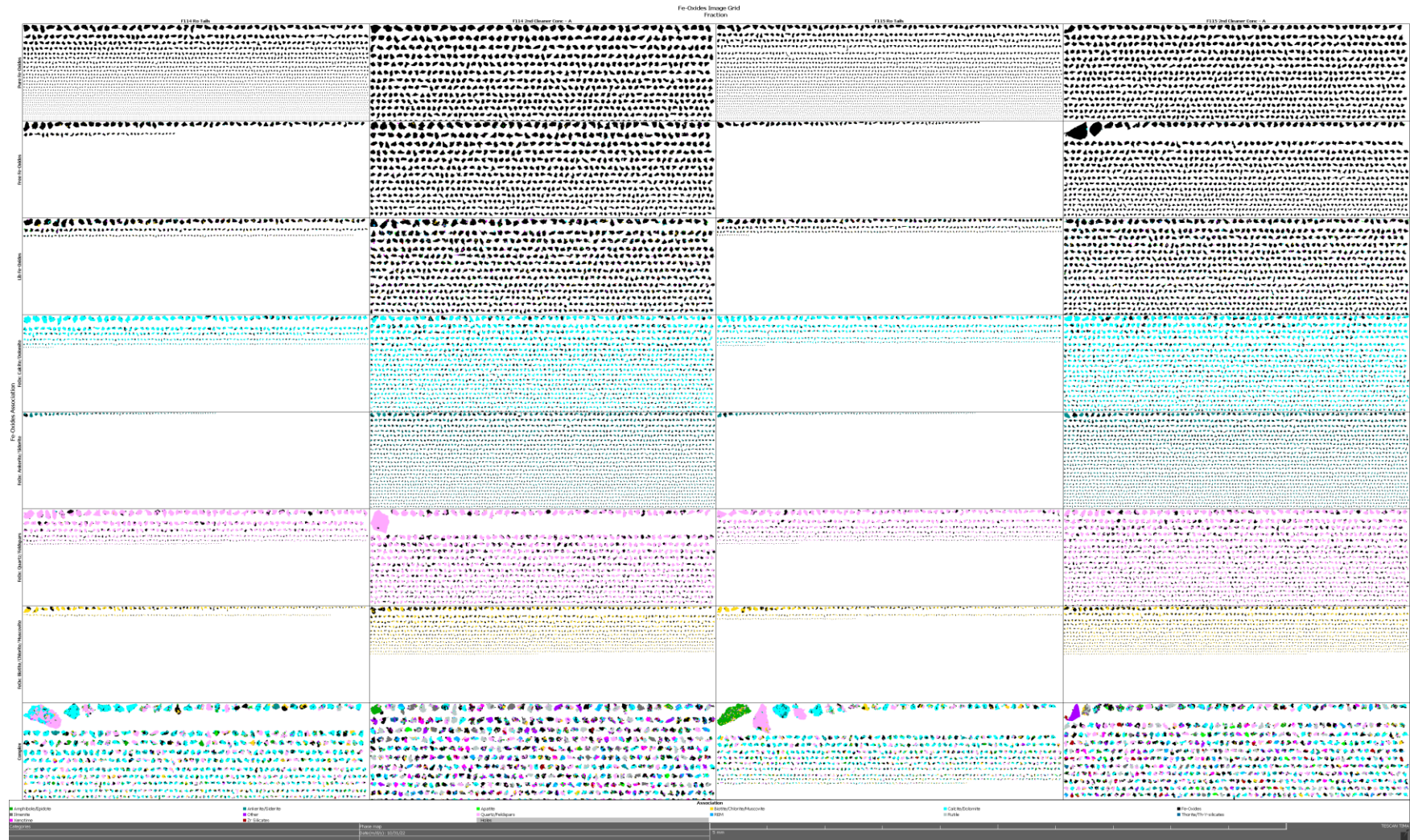
Absolute Mass of Fe-Oxides Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
Pure Fe-Oxides	1.09	27.5	1.07	25.8
Free Fe-Oxides	0.22	1.96	0.09	1.26
Lib Fe-Oxides	0.26	3.81	0.22	3.24
FeOx: Xenotime	0.00	0.22	0.00	0.17
FeOx: REM	0.00	0.20	0.00	0.15
FeOx: Thorite/Th-Y-silicates	0.00	0.03	0.00	0.02
FeOx: Zr Silicates	0.00	0.15	0.00	0.14
FeOx: Apatite	0.00	0.09	0.00	0.18
FeOx: Calcite/Dolomite	0.07	0.56	0.07	0.68
FeOx: Ankerite/Siderite	0.01	0.47	0.01	0.50
FeOx: Quartz/Feldspars	0.09	0.70	0.09	0.66
FeOx: Biotite/Chlorite/Muscovite	0.04	0.17	0.04	0.16
FeOx: Amphibole/Epilote	0.01	0.12	0.01	0.14
FeOx: Ilmenite	0.00	0.19	0.00	0.16
FeOx: Rutile	0.00	0.19	0.00	0.19
FeOx: Other	0.00	0.24	0.00	0.24
Complex	0.24	2.12	0.18	1.82
Total	2.04	38.7	1.79	35.5



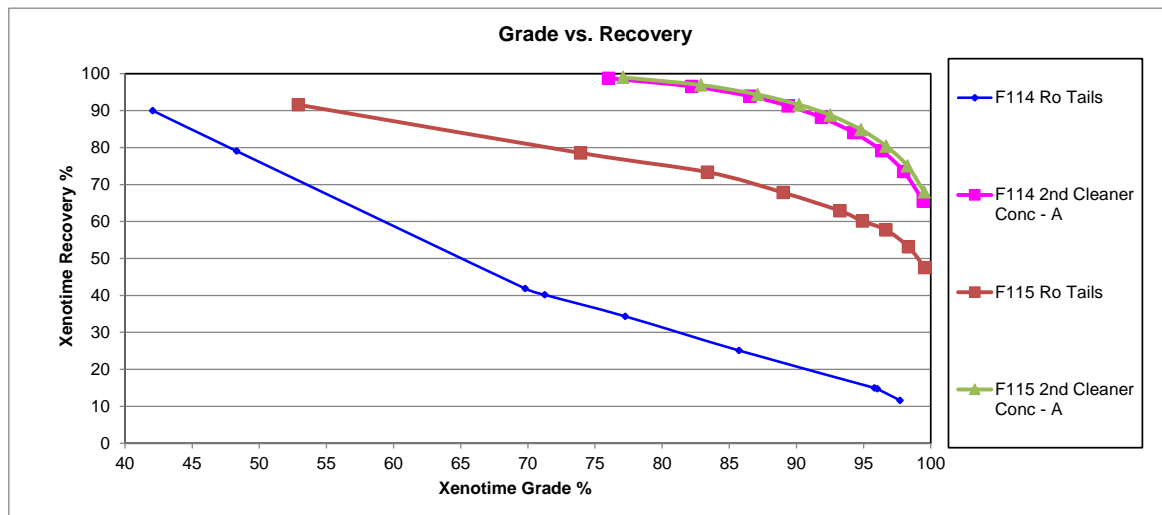
Normalized Mass of Fe-Oxides Across Samples

Mineral Name	F114 Ro Tails	F114 2nd Cleaner Conc - A	F115 Ro Tails	F115 2nd Cleaner Conc - A
Pure Fe-Oxides	53.7	71.0	60.0	72.6
Free Fe-Oxides	10.8	5.07	4.81	3.55
Lib Fe-Oxides	12.9	9.85	12.1	9.12
FeOx: Xenotime	0.02	0.56	0.00	0.49
FeOx: REM	0.02	0.51	0.12	0.42
FeOx: Thorite/Th-Y-silicates	0.00	0.07	0.00	0.06
FeOx: Zr Silicates	0.05	0.39	0.11	0.41
FeOx: Apatite	0.04	0.22	0.00	0.51
FeOx: Calcite/Dolomite	3.37	1.45	3.92	1.92
FeOx: Ankerite/Siderite	0.52	1.22	0.82	1.42
FeOx: Quartz/Feldspars	4.26	1.80	5.11	1.85
FeOx: Biotite/Chlorite/Muscovite	1.76	0.44	2.24	0.46
FeOx: Amphibole/Epilote	0.60	0.31	0.67	0.40
FeOx: Ilmenite	0.12	0.48	0.01	0.46
FeOx: Rutile	0.01	0.49	0.04	0.52
FeOx: Other	0.09	0.63	0.08	0.66
Complex	11.8	5.47	10.0	5.12
Total	100.0	100.0	100.0	100.0
Liberated	77.4	85.9	76.9	85.3



Namibia Critical Metals
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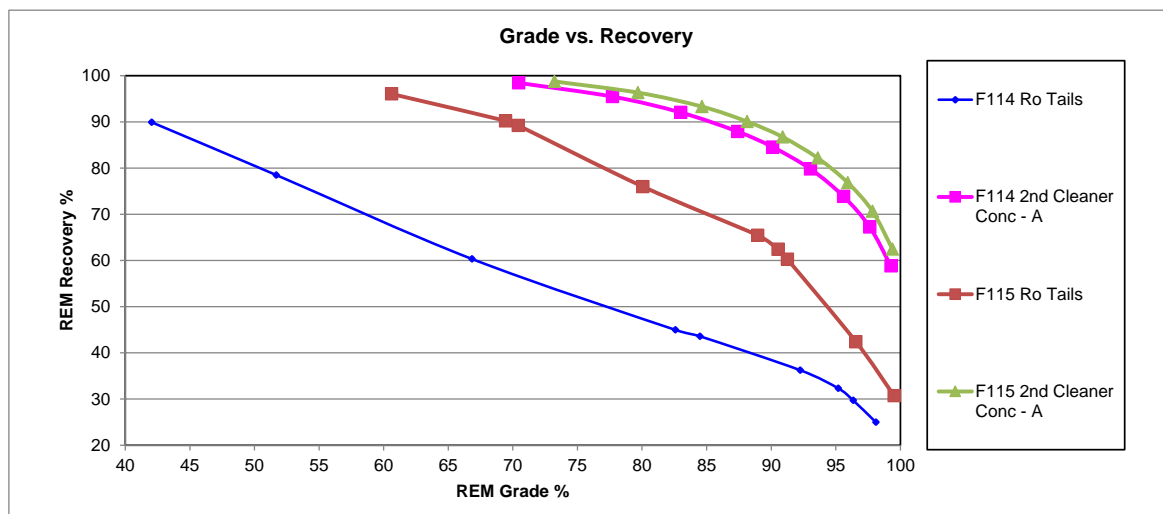
Xenotime Grade vs. Recovery:



Volume % of Xenotime/ Sample	F114 Ro Tails		F114 2nd Cleaner Conc - A		F115 Ro Tails		F115 2nd Cleaner Conc - A	
	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery
All particles	0.06	100.0	7.00	100.0	0.05	100.0	5.54	100.0
≥10	42.1	90.0	76.0	98.8	52.9	91.6	77.1	99.0
≥20	48.3	79.1	82.2	96.6	73.9	78.5	82.9	96.9
≥30	69.8	41.9	86.5	93.9	83.4	73.4	87.1	94.4
≥40	71.3	40.2	89.4	91.3	89.0	67.8	90.2	91.6
≥50	77.2	34.4	91.9	88.2	93.2	62.9	92.5	88.8
≥60	85.7	25.1	94.3	84.0	94.9	60.1	94.8	84.8
≥70	95.8	15.0	96.4	79.2	96.7	57.7	96.7	80.4
≥80	96.0	14.7	98.0	73.6	98.3	53.1	98.3	75.1
≥90	97.7	11.6	99.4	65.6	99.6	47.5	99.5	68.0

Namibia Critical Metals
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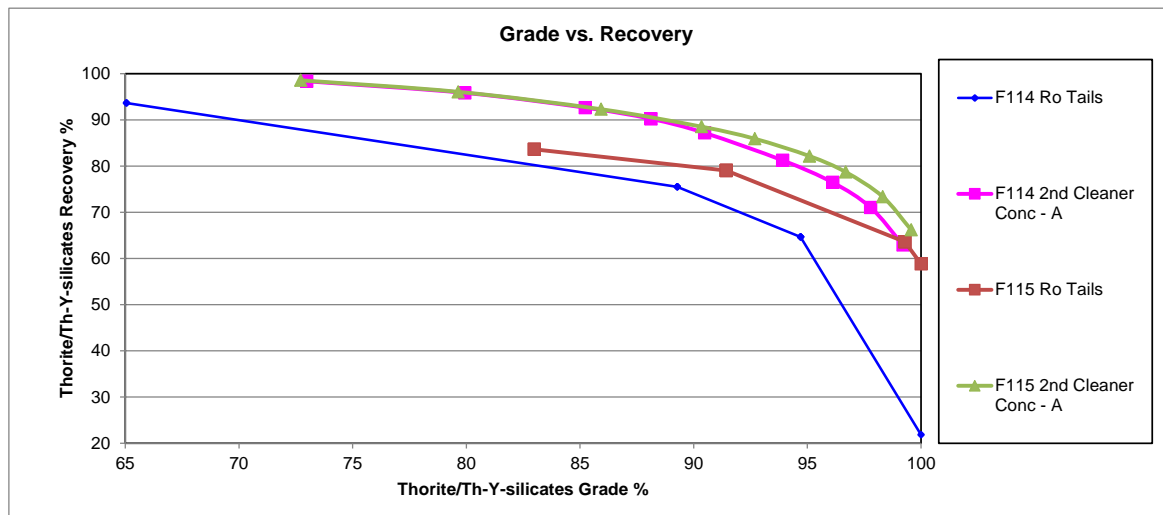
REM Grade vs. Recovery:



Volume % of REM / Sample	F114 Ro Tails		F114 2nd Cleaner Conc - A		F115 Ro Tails		F115 2nd Cleaner Conc - A	
	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery
All particles	0.03	100.0	3.15	100.0	0.03	100.0	2.53	100.0
≥10	42.0	89.9	70.4	98.4	60.6	96.1	73.2	98.8
≥20	51.7	78.5	77.7	95.5	69.4	90.2	79.7	96.3
≥30	66.8	60.3	83.0	92.0	70.4	89.2	84.6	93.3
≥40	82.6	45.0	87.4	87.9	80.1	76.0	88.1	90.1
≥50	84.5	43.6	90.1	84.6	89.0	65.4	90.9	86.7
≥60	92.2	36.3	93.0	79.8	90.5	62.4	93.6	82.2
≥70	95.2	32.3	95.6	73.9	91.3	60.3	95.9	76.8
≥80	96.3	29.7	97.6	67.3	96.6	42.4	97.8	70.7
≥90	98.1	25.0	99.3	58.8	99.5	30.7	99.4	62.5

Namibia Critical Metals
 CALR-18299-03
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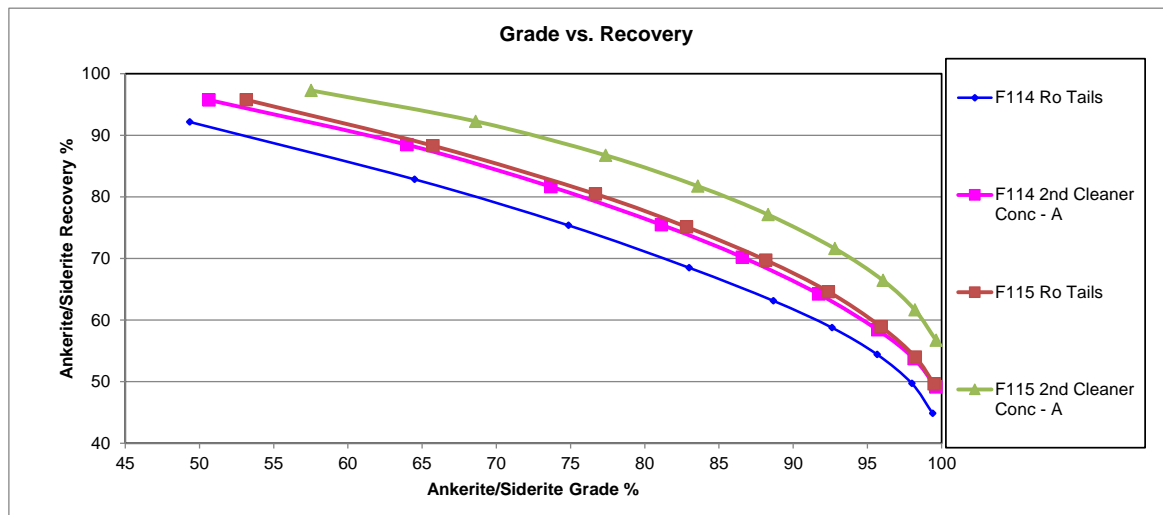
Thorite/Th-Y-silicates Grade vs. Recovery:



Volume % of Thorite/Th-Y- silicates / Sample	F114 Ro Tails		F114 2nd Cleaner Conc - A		F115 Ro Tails		F115 2nd Cleaner Conc - A	
	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery
All particles	0.01	100.0	0.44	100.0	0.00	100.0	0.33	100.0
≥10	65.1	93.7	73.0	98.4	83.0	83.6	72.7	98.6
≥20	65.1	93.7	79.9	95.8	83.0	83.6	79.6	96.1
≥30	89.3	75.5	85.2	92.6	91.4	79.0	85.9	92.3
≥40	89.3	75.5	88.1	90.2	91.4	79.0	90.3	88.5
≥50	89.3	75.5	90.5	87.2	91.4	79.0	92.7	85.9
≥60	94.7	64.7	93.9	81.2	91.4	79.0	95.1	82.2
≥70	94.7	64.7	96.1	76.5	99.3	63.6	96.7	78.7
≥80	94.7	64.7	97.8	71.1	99.3	63.6	98.3	73.4
≥90	100.0	21.8	99.2	62.9	100.0	58.8	99.6	66.2

Namibia Critical Metals
CALR-18299-03
MI5067-SEP22

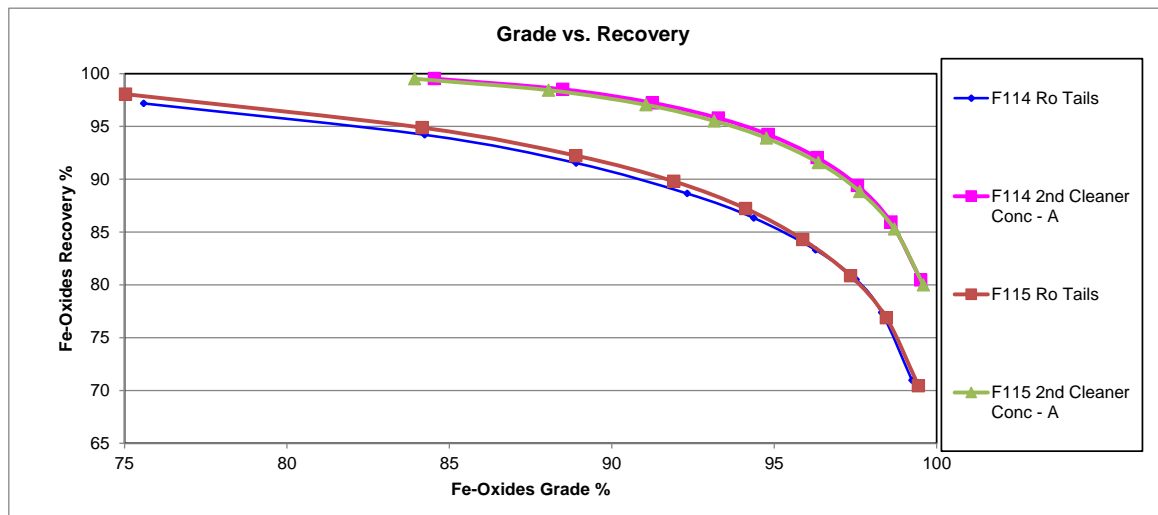
Ankerite/Siderite Grade vs. Recovery:



Volume % of Ankerite/Siderite / Sample	F114 Ro Tails		F114 2nd Cleaner Conc - A		F115 Ro Tails		F115 2nd Cleaner Conc - A	
	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery
All particles	1.30	100.0	3.15	100.0	1.30	100.0	4.58	100.0
≥10	49.3	92.2	50.6	95.7	53.1	95.8	57.5	97.3
≥20	64.5	82.9	64.0	88.5	65.7	88.3	68.6	92.3
≥30	74.9	75.4	73.7	81.7	76.7	80.5	77.4	86.8
≥40	83.0	68.5	81.1	75.5	82.8	75.1	83.6	81.7
≥50	88.7	63.1	86.6	70.2	88.2	69.7	88.3	77.1
≥60	92.6	58.8	91.7	64.2	92.4	64.6	92.8	71.6
≥70	95.7	54.4	95.7	58.4	95.9	58.9	96.1	66.4
≥80	98.0	49.7	98.1	53.7	98.2	54.0	98.2	61.7
≥90	99.4	44.9	99.6	49.2	99.5	49.6	99.6	56.7

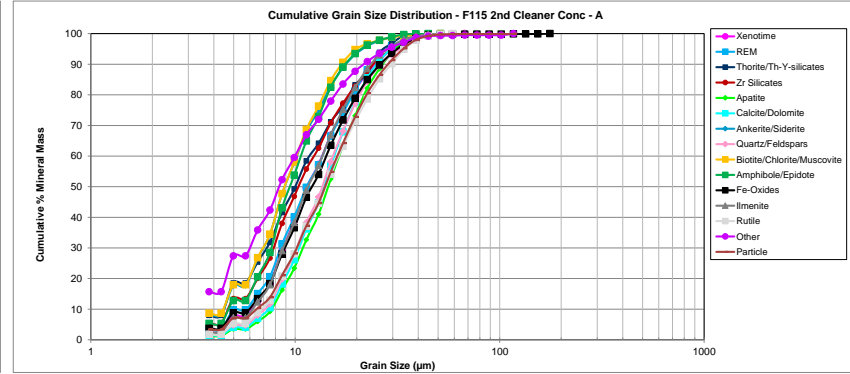
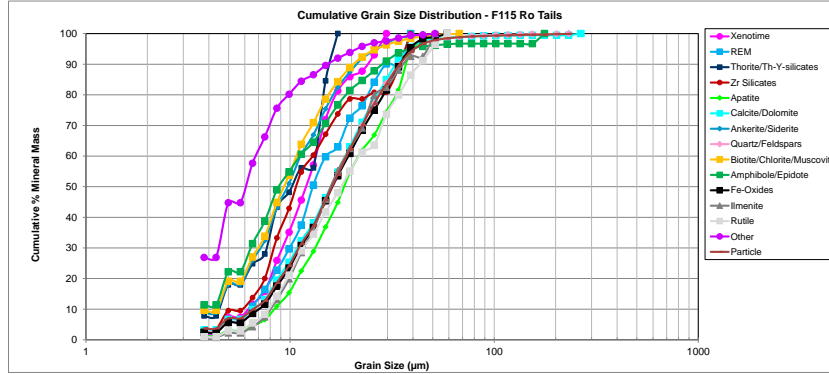
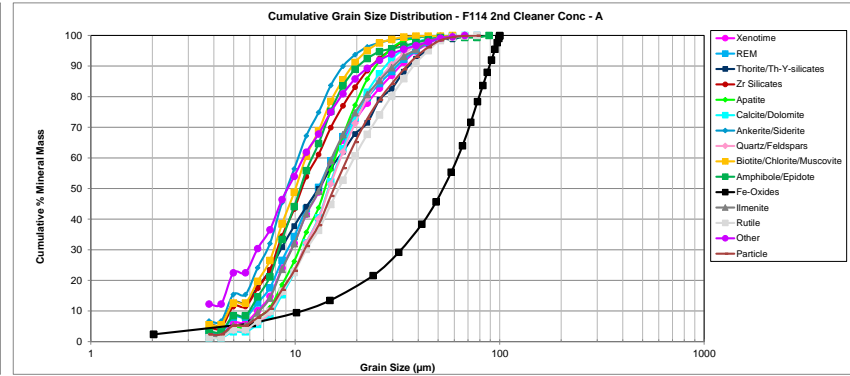
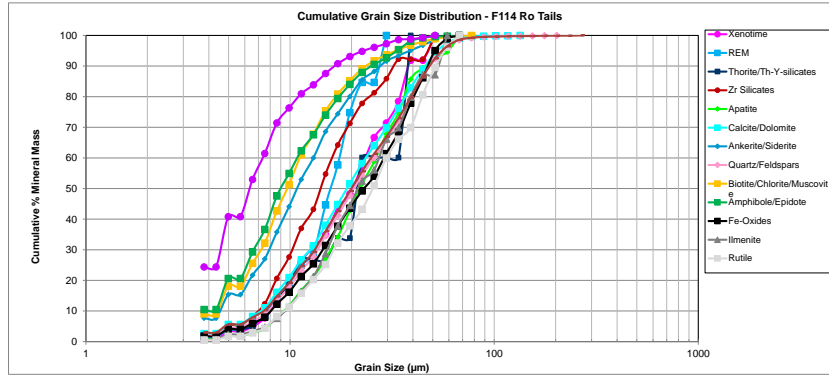
Namibia Critical Metals
CALR-18299-03
MI5067-SEP22

Fe-Oxides Grade vs. Recovery:



Volume % of Fe-Oxides / Sample	F114 Ro Tails		F114 2nd Cleaner Conc - A		F115 Ro Tails		F115 2nd Cleaner Conc - A	
	Grade	Recovery	Grade	Recovery	Grade	Recovery	Grade	Recovery
All particles	2.04	100.0	38.7	100.0	1.79	100.0	35.5	100.0
≥10	75.6	97.2	84.5	99.5	75.0	98.1	83.9	99.5
≥20	84.2	94.2	88.5	98.5	84.2	94.9	88.1	98.4
≥30	88.9	91.5	91.3	97.2	88.9	92.2	91.1	97.1
≥40	92.3	88.7	93.3	95.8	91.9	89.8	93.2	95.5
≥50	94.4	86.3	94.8	94.2	94.1	87.2	94.8	93.9
≥60	96.3	83.3	96.3	92.1	95.9	84.3	96.4	91.6
≥70	97.5	80.5	97.6	89.4	97.4	80.9	97.6	88.8
≥80	98.3	77.4	98.6	86.0	98.5	76.9	98.7	85.3
≥90	99.2	71.0	99.5	80.5	99.4	70.5	99.6	80.0

Grain size distribution



Sample	Percentile (Mass % of phase) / Sample	Xenotime	REM	Thorianite/Th-Y-silicates	Zr Silicates	Apatite	Calcite/Dolomite	Ankerite/Siderite	Quartz/Feldspars	Biotite/Chlorite/Muscovite	Amphibole/Epidote	Fe-Oxides	Ilmenite	Rutile	Other	Particle
F114 Ro Tails	Median	20.6	15.2	20.3	14.0	22.5	18.9	10.7	20.6	9.60	8.98	23.3	21.2	25.2	5.88	20.1
	P80	35.0	20.3	35.2	24.9	37.3	36.7	19.5	39.5	16.6	17.3	40.6	40.1	44.0	11.2	39.2
F114 2nd Cleaner Conc - A	Median	13.1	12.7	13.1	10.7	14.0	14.4	8.98	14.8	10.2	10.7	14.0	13.1	16.3	8.99	15.6
	P80	24.0	23.0	26.5	18.3	20.6	22.0	14.0	22.8	15.6	15.9	23.5	22.2	29.4	16.6	26.5
F115 Ro Tails	Median	12.2	12.7	10.7	10.7	18.9	15.9	9.60	16.3	9.59	8.98	16.3	15.9	17.6	5.88	15.9
	P80	17.0	22.8	14.8	23.5	32.4	26.9	16.3	27.5	15.6	18.9	28.8	29.0	34.2	9.60	27.6
F115 2nd Cleaner Conc - A	Median	11.7	11.7	10.2	10.2	14.4	14.0	8.98	13.6	8.98	9.59	12.2	11.7	14.0	8.30	14.0
	P80	18.9	19.2	18.3	18.3	21.7	20.4	14.0	20.6	14.0	14.4	20.1	18.9	23.3	15.6	22.5

Appendix D – Bulk Flotation Testing

Test No.: CP-101

Project No: 18299-03 Operator: Marteer Date: 05/30/2022

Purpose: Based on F102/105, first 10kg scale up test

Sample: ROM

Feed: 10 kg of at -53 µm SG7

Water DI Water Ro Tails K₉₀ = µm

Notes: RPM- 60%

Reagent Preparation for 3900: make **2% solution** with 3900 :NaOH (**10g 3900, 0.5g NaOH,500ml water**)(add NaOH to water first, then add in 3900)

Conditions:

Stage	Reagents Added, g/t							Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon	3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp °C	Pulp Density %
Strength	10%	5%	2%	100%			10%					
High Density Conditioning ~50% @ 1000rpm										7.5	50	50
Condition 1	250	50					80	3		9.0	50	50
Condition 2			1000	20				3		8.0	50	
Rougher 1									5	8.0	52	30.0
Rougher 2			300	20		1		2	5	8.1	51	
Rougher 3			300	20		1		2	7	8.2	51	
Rougher 4			200	20		2.5		2	7	8.1	51	
Rougher 5 (Assay separately)			500	20		5		2	5	8.1	52	
Ro Tails subsample for assay. Bulk Ro tails filter, oven dry, submit to bucking to weigh and bag.												
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed												
Condition 3	50	10						1		natural	50	
1st Cleaner A			50			1		1	3	8.3	50	-12
1st Cleaner B			50		5	1		1	3	8.2	50	
1st Cleaner C			50	10	5	2.5		1	3	8.2	50	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed										natural		
2nd Cleaner A	25							1				
2nd Cleaner B			25					1	3	8.2	52	
2nd Cleaner C			25		10	3.5		1	5	8.2	50	
Pass through 2nd Cl A concentrate through WHIMS at 2,000 Gauss (~2 Amps), the mags pass through the WHIMS as a cleaner												
Ro Total	325	60	2500	110	20	17.5	80	21	46	8.2	51.0	

Comment:

Stage	Rougher	1st Cleaner	2nd Cleaner
Flotation Cell	28L (10kg flot cell)	8L (4kg flot cell)	4L (2kg flot cell)
Speed rpm	60%	1200	

Lime Additions:

Ro Tail: 33.58g Sub : 8.84g

1st Clnr Tail: 11.05g

2nd Clnr Tail: 6.27g

Ro Con 5: 7.05g

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
CP-101 2nd Cl Conc A Non-Mag	170	1.8	3.09	7.08	7.3	16.9	35.5	10.8	1.37	3.4	53.1	53.1	1.2	0.6	9.5	33.6	0.4	0.5
CP-101 2nd Cl Conc A Mag	51	0.5	0.43	0.98	0.6	3.29	94.8	1.03	0.17	0.7	2.2	2.2	0.0	0.0	7.6	1.0	0.0	0.0
CP-101 2nd Cl Conc B	91	0.9	1.12	2.56	8.4	20.6	41.1	6.08	1.97	4.2	10.3	10.3	0.8	0.4	5.9	10.1	0.3	0.3
CP-101 2nd Cl Tail	209	2.2	0.35	0.80	14.3	31.2	19.0	2.21	3.71	8.3	7.4	7.4	3.0	1.4	6.3	8.4	1.4	1.4
CP-101 1st Cl Tail	658	6.9	0.13	0.30	17.3	34.7	9.4	0.54	4.25	9.5	8.7	8.7	11.3	4.9	9.7	6.5	4.9	5.1
CP-101 Ro Conc 5	307	3.2	0.07	0.16	17.1	36.6	9.3	0.59	4.39	9.9	2.2	2.2	5.2	2.4	4.5	3.3	2.4	2.5
CP-101 Ro Tail	8104	84.5	0.02	0.05	9.8	52.1	4.40	0.25	6.33	13.6	16.1	16.1	78.6	90.3	56.4	37.1	90.6	90.2
Head (Calc.)	9590	100.0	0.10	0.24	10.5	48.8	6.59	0.57	5.91	12.7	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

Cumulative Products

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
2nd Cl Con 1 NonMags	170	1.8	3.09	7.1	7.3	16.9	35.5	10.8	1.4	3.4	53.1	53.1	1.2	0.6	9.5	33.6	0.4	0.5
2nd Cl Con 1	221	2.3	2.48	5.7	5.7	13.8	49.1	8.6	1.1	2.8	55.3	55.3	1.3	0.6	17.1	34.5	0.4	0.5
2nd Cl Con 1-2	312	3.3	2.08	4.8	6.5	15.8	46.8	7.8	1.3	3.2	65.6	65.6	2.0	1.1	23.1	44.7	0.7	0.8
1st Cl Con 1-3	521	5.4	1.39	3.18	9.6	22.0	35.6	5.6	2.3	5.2	73.0	73.0	5.0	2.4	29.3	53.1	2.1	2.2
Ro Con 1-4	1179	12.3	0.69	1.57	13.9	29.1	21.0	2.8	3.4	7.6	81.7	81.7	16.2	7.3	39.1	59.6	7.1	7.3
Ro Con 1-5	1486	15.5	0.56	1.28	14.6	30.6	18.6	2.3	3.6	8.1	83.9	83.9	21.4	9.7	43.6	62.9	9.4	9.8
Ro Tail	8104	84.5	0.02	0.05	9.8	32.5	2.9	0.6	6.3	13.6	16.1	16.1	78.6	56.4	37.1	90.6	90.6	90.2
2nd Cl Conc A - nonmags& Conc B	261	2.7	2.40	5.5	7.7	18.2	37.5	9.2	1.6	3.7	63.4	63.4	2.0	1.0	15.5	43.7	0.7	0.8

Test No.: CP-102

Project No: 18299-03 Operator: Marteer Date: 06/08/2022

Purpose: Based on CP-101

Target weight

Sample: ROM

2nd cl Conc 250 g, dry

Feed: 10 kg of at -53 µm SG8

320 g, wet

Water DI Water Ro Tails K₈₀ = N/A µm

Notes: RPM- 60%

Subsample the tails, submit the subsample for assay and S/A. The remaining tails filter, weigh, and bag.

Reagent Preparation for 3900: make 2% solution with 3900 :NaOH (10g 3900, 0.5g NaOH,500ml water)(add NaOH to water first, then add in 3900)

Conditions:

Stage	Reagents Added, g/t							Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon	3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp °C	Pulp Density %
Strength	10%	5%	2%	100%			10%					
High Density Conditioning ~50% @ 1000rpm												
Condition 1	250	50						3		7.5	50	50
Condition 2			1000	20				3		9.0	50	50
										8.0	51	
Rougher 1									5	8.0	53	30.0
Rougher 2			300	20		1		2	5	8	52	
Rougher 3			300	20		1		2	7	8.1	51	
Rougher 4			200	20		2.5		2	7	8.1	52	
Ro Tails subsample for assay. Bulk Ro tails filter, oven dry, submit to bucking to weigh and bag.												
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed												
Condition 3	50	10						1		natural	50	
1st Cleaner A			50			1		1	3	8.2	50	~12
1st Cleaner B			50		5	1		1	3	8.2	50	
1st Cleaner C			50	10	5	2.5		1	3	8.1	50	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed												
	25	5						1		natural		
2nd Cleaner A			25				5	1	3	8.4	51	
2nd Cleaner B			25		10	3.5		1	5	8.4	50	
Pass through 2nd Cl A&B concentrate through WHIMS at 2,000 Gauss (~2 Amps), the mags pass through the WHIMS as a cleaner												
Ro Total	325	65	2000	90	20	12.5	5	19	41	8.2	51.5	

Comment:

Stage	Rougher	1st Cleaner	2nd Cleaner
Flotation Cell	28L (10kg flot cell)	8L (4kg flot cell)	4L (2kg flot cell)
Speed rpm	60%	1200	

Lime Additions:

Ro Tails: 28.26g
 Ro Tail Sub: 6.12g Flocc:
 1st Clnr Tail: 6.54g 0
 2nd Clnr Tail: 3.91g 0

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
CP-102 2nd Cl Conc Non-Mag	266	2.7	2.58	5.90	8.0	20.9	31.4	10.0	1.89	4.3	62.1	62.1	2.0	1.2	13.0	46.1	0.9	0.9
CP-102 2nd Cl Conc Mag	80	0.8	0.47	1.08	0.9	3.45	91.1	1.25	0.21	0.8	3.4	3.4	0.1	0.1	11.3	1.7	0.0	0.0
CP-102 2nd Cl Tail	141	1.4	0.29	0.67	15.0	34.2	13.7	1.00	4.16	9.7	3.7	3.7	2.0	1.0	3.0	2.5	1.0	1.1
CP-102 1st Cl Tail	858	8.7	0.10	0.23	16.2	37.9	7.5	0.44	4.49	10.6	7.9	7.9	13.4	6.8	10.0	6.6	6.8	7.2
CP-102 Ro Tail	8558	86.4	0.03	0.07	10.0	50.7	4.7	0.29	6.01	13.4	22.9	22.9	82.4	90.9	62.8	43.2	91.2	90.8
Head (Calc.)	9903	100.0	0.11	0.26	10.5	48.2	6.50	0.58	5.69	12.8	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

Cumulative Products

updated with ICP on Ro tails

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
2nd Cl Con NonMags	266	2.7	2.58	5.9	8.0	20.9	31.4	10.0	1.9	4.3	62.1	62.1	2.0	1.2	13.0	46.1	0.9	0.9
2nd Cl Con	345	3.5	2.09	4.8	6.4	16.9	45.2	8.0	1.5	3.5	65.5	65.5	2.1	1.2	24.2	47.8	0.9	0.9
1st Cl Con 1-3	487	4.9	1.57	3.6	8.9	21.9	36.0	5.9	2.3	5.3	69.2	69.2	4.2	2.2	27.2	50.3	2.0	2.0
Ro Con 1-4	1345	13.6	0.63	1.45	13.5	32.1	17.8	2.4	3.7	8.7	77.1	77.1	17.6	9.1	37.2	56.8	8.8	9.2
Ro Tail	8558	86.4	0.03	0.07	10.0	50.7	4.7	0.3	6.0	13.4	22.9	22.9	82.4	90.9	62.8	43.2	91.2	90.8

Test No.: CP-103

Project No: 18299-03 Operator: Marteer Date: 06/08/2022

Purpose: Based on CP-101

Target weight

Sample: ROM

2nd cl Conc 250 g, dry

Feed: 10 kg of at -53 µm SG10

302 g, wet

Water DI Water Ro Tails $K_{80} = 42 \mu\text{m}$

Notes: RPM- 60%

Subsample the tails, submit the subsample for assay and S/A. The remaining tails filter, weigh, and bag.

Reagent Preparation for 3900: make 2% solution with 3900 :NaOH (10g 3900, 0.5g NaOH,500ml water)(add NaOH to water first, then add in 3900)

Conditions:

Stage	Reagents Added, g/t							Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon	3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp °C	Pulp Density %
Strength	10%	5%	2%	100%			10%					
High Density Conditioning ~50% @ 1000rpm												
Condition 1	250	50						3		7.5	50	50
Condition 2			1000	20				3		9.0	50	50
										8.0	51	
Rougher 1									5	8.0	52	30.0
Rougher 2			300	20		1		2	5	8	52	
Rougher 3			300	20		1		2	7	8.1	51	
Rougher 4			200	20		2.5		2	7	8.1	50	
Ro Tails subsample for assay. Bulk Ro tails filter, oven dry, submit to bucking to weigh and bag.												
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed												
Condition 3	50	10						1		natural	50	
1st Cleaner A			50			1		1	3	8.2	50	~12
1st Cleaner B			50		5	1		1	3	8.2	52	
1st Cleaner C			50	10	5	2.5		1	3	8.3	51	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed												
	25	0						1		natural		
2nd Cleaner A			25				5	1	3	8.3	52	
2nd Cleaner B			25		10	3.5		1	5	8.3	52	
Pass through 2nd Cl A&B concentrate through WHIMS at 2,000 Gauss (~2 Amps), the mags pass through the WHIMS as a cleaner												
Ro Total	325	60	2000	90	20	12.5	5	19	41	8.2	51.0	

Comment:

Stage	Rougher	1st Cleaner	2nd Cleaner
Flotation Cell	28L (10kg flot cell)	8L (4kg flot cell)	4L (2kg flot cell)
Speed rpm	60%	1200	

Lime Additions:

Ro Tails: 36.39g
 Ro Tail Sub: 9.81g Floc:
 1st Clnr Tail: 14.82g 0
 2nd Clnr Tail: 6g 0

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
CP-103 2nd Cl Conc Non Mag	327	3.2	2.24	5.13	11.6	18.4	29.2	9.2	1.69	3.8	72.9	72.9	3.3	1.2	14.7	52.3	0.9	1.0
CP-103 2nd Cl Conc Mag	85	0.8	0.29	0.66	0.7	2.91	93.6	0.86	0.19	0.6	2.4	2.4	0.1	0.1	12.2	1.3	0.0	0.0
CP-103 2nd Cl Tail	143	1.4	0.19	0.44	16.1	32.9	14.0	1.23	4.17	9.2	2.7	2.7	2.0	1.0	3.1	3.1	1.0	1.0
CP-103 1st Cl Tail	1126	11.1	0.06	0.14	18.0	36.1	7.0	0.41	4.43	10.1	6.7	6.7	17.6	8.4	12.2	8.0	8.4	8.9
CP-103 Ro Tail	8507	83.5	0.02	0.04	10.4	51.0	4.4	0.24	6.29	13.4	15.2	15.2	77.0	89.4	57.8	35.4	89.7	89.1
Head (Calc.)	10188	100.0	0.10	0.23	11.3	47.7	6.36	0.57	5.86	12.6	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

Cumulative Products

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
2nd Cl Con NonMags	327	3.2	2.24	5.1	11.6	18.4	29.2	9.2	1.7	3.8	72.9	72.9	3.3	1.2	14.7	52.3	0.9	1.0
2nd Cl Con	412	4.0	1.84	4.2	9.4	15.2	42.4	7.5	1.4	3.1	75.4	75.4	3.4	1.3	27.0	53.6	1.0	1.0
1st Cl Con 1-3	555	5.4	1.41	3.2	11.1	19.8	35.1	5.9	2.1	4.7	78.1	78.1	5.4	2.3	30.1	56.7	2.0	2.0
Ro Con 1-4	1682	16.5	0.51	1.16	15.7	30.7	16.3	2.2	3.7	8.3	84.8	84.8	23.0	10.6	42.2	64.6	10.3	10.9

Test No.: CP-104

Project No: 18299-03 Operator: Marteer Date: 06/16/2022

Purpose: Based on CP-102

Target weight

Sample: ROM

2nd cl Conc 250 g, dry

Feed: 10 kg of at -53 µm SG8

320 g, wet

Water DI Water Ro Tails K₈₀ = 43 µm

Notes: RPM- 60%

Subsample the tails, submit the subsample for assay and S/A. The remaining tails filter, weigh, and bag.

Reagent Preparation for 3900: make 2% solution with 3900 :NaOH (10g 3900, 0.5g NaOH,500ml water)(add NaOH to water first, then add in 3900)

Conditions:

Stage	Reagents Added, g/t							Time, min		Pulp		
	Sodium Silicate (N Type)	Calgon	3900	3000	Pine Oil	Defoamer	NaOH	Cond.	Froth	pH	Temp °C	Pulp Density %
Strength	10%	5%	2%	100%			10%					
High Density Conditioning ~50% @ 1000rpm												
Condition 1	250	50						3		7.5	50	50
Condition 2			1000	20				3		9.0	50	50
										8.0	51	
Rougher 1									5	8.0	53	30.0
Rougher 2			300	20		1		2	5	8	52	
Rougher 3			300	20		1		2	7	8.1	51	
Rougher 4			200	20		2.5		2	7	8.1	52	
Ro Tails subsample for assay. Bulk Ro tails filter, oven dry, submit to bucking to weigh and bag.												
1st Cleaner Stage: Combine Ro 1 to Ro 4 as 1st Cl feed												
Condition 3	50	10						1		natural	50	
1st Cleaner A			50			1		1	3	8.2	50	~12
1st Cleaner B			50		5	1		1	3	8.2	50	
1st Cleaner C			50	10	5	2.5		1	3	8.1	50	
2nd Cleaner Stage: Combine 1st Cl A-C as 2nd Cl feed												
	25	5						1		natural		
2nd Cleaner A			25				5	1	3	8.4	51	
2nd Cleaner B			25		10	3.5		1	5	8.4	50	
Pass through 2nd Cl A&B concentrate through WHIMS at 2,000 Gauss (~2 Amps), the mags pass through the WHIMS as a cleaner												
Ro Total	325	65	2000	90	20	12.5	5	19	41	8.2	51.5	

Comment:

Stage	Rougher	1st Cleaner	2nd Cleaner
Flotation Cell	28L (10kg flot cell)	8L (4kg flot cell)	4L (2kg flot cell)
Speed rpm	60%	1200	

Lime Additions:

Ro Tails: 28.26g
 Ro Tail Sub: 6.12g Floc:
 1st Clnr Tail: 6.54g 0
 2nd Clnr Tail: 3.91g 0

Metallurgical Balance

Products	Weight		Assay, %								Distribution, %							
	g	%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
CP-104 2nd Cl Conc Non Mag	286	2.9	3.31	7.59	9.4	24.4	26.8	8.0	2.60	5.3	61.2	61.2	2.6	1.5	11.6	40.8	1.3	1.2
CP-104 2nd Cl Conc Mag	76	0.8	0.53	1.22	0.6	3.14	93.2	0.78	0.20	0.7	2.6	2.6	0.0	0.0	10.6	1.1	0.0	0.0
CP-104 2nd Cl Tail	231	2.3	0.28	0.64	16.9	33.1	12.4	0.91	4.12	9.4	4.2	4.2	3.8	1.6	4.3	3.8	1.6	1.7
CP-104 1st Cl Tail	1131	11.5	0.27	0.61	16.3	35.9	10.2	0.88	4.43	10.0	19.5	19.5	17.7	8.5	17.4	17.8	8.6	8.9
CP-104 Ro Tail	8145	82.5	0.02	0.05	9.7	51.7	4.6	0.25	6.35	13.6	12.5	12.5	75.9	88.4	56.1	36.5	88.5	88.1
Head (Calc.)	9869	100.0	0.16	0.36	10.5	48.3	6.71	0.57	5.92	12.7	100	100	100	100	100	100	100	100
Head (Direct)			0.10	0.22	9.89	49.7	6.44	0.58	5.92	12.9								

Cumulative Products

updated with ICP on Ro tails

Products	Weight		Assay, %								Distribution, %							
		%	Y ₂ O ₃	TREO (est)	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃	Y ₂ O ₃	TREO	CaO	SiO ₂	Fe ₂ O ₃	TiO ₂	Na ₂ O	Al ₂ O ₃
2nd Cl Con NonMags		2.9	3.31	7.6	9.4	24.4	26.8	8.0	2.6	5.3	61.2	61.2	2.6	1.5	11.6	40.8	1.3	1.2
2nd Cl Con		3.7	2.73	6.3	7.6	20.0	40.7	6.5	2.1	4.3	63.8	63.8	2.6	1.5	22.2	41.9	1.3	1.2
1st Cl Con 1-3		6.0	1.78	4.1	11.2	25.1	29.6	4.3	2.9	6.3	68.0	68.0	6.4	3.1	26.5	45.6	2.9	3.0
Ro Con 1-4		17.5	0.79	1.80	14.6	32.2	16.9	2.1	3.9	8.7	87.5	87.5	24.1	11.6	43.9	63.5	11.5	11.9