PRELIMINARY ECONOMIC ASSESSMENT OF NORRA KÄRR RARE EARTH DEPOSIT AND POTENTIAL BY-PRODUCTS, SWEDEN

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Prepared For

LEADING EDGE MATERIALS

Report Prepared by

srk consulting

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EXECUTIVE SUMMARY PRELIMINARY ECONOMIC ASSESSMENT OF NORRA KÄRR RARE EARTH DEPOSIT AND POTENTIAL BY-PRODUCTS, SWEDEN

1 INTRODUCTION

SRK Consulting (UK) Limited ("SRK") was requested by Leading Edge Materials Corporation ("LEM", hereinafter also referred to as the "Company"), to undertake an updated Preliminary Economic Assessment ("PEA") for the Norra Kärr rare earth elements ("REEs") deposit in Sweden ("Norra Kärr or the "Project"). This PEA represents an update to the previous technical study completed in 2015 and includes consideration of potential by-products of the REE processing route of nepheline syenite, niobium (Nb) and zirconium (Zr). The 2015 Prefeasibility Study ("PFS") was completed by GBM Minerals Engineering Consultants Ltd ("GBM"), Wardell Armstrong International ("WAI") and Golder Associates Ltd ("Golder") on behalf of Tasman Metals AB ("Tasman"), which is the 100%-owned subsidiary of LEM and owner of the Project. LEM's predecessor (Flinders Resources Limited) acquired Tasman Metals AB in 2016, thereafter changing the name to Leading Edge Materials Corporation and therefore all Tasman Metals AB sourced data is wholly owned by LEM and is stated as such in the report.

This PEA has been prepared in a Technical Report format in compliance with Canadian Securities Administrators' National Instrument 43-101 ("NI43-101") and Form 43-101F guidelines - Standards of Disclosure for Mineral Projects. In this report, SRK has considered all aspects of the Project including geology, mineral resources, mining, processing and environmental and social issues in accordance with the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Mineral Resource and Mineral Reserve definitions and in conformity with generally accepted CIM 'Exploration Best Practices' guidelines.

The Canadian Securities Administrators (CSA) published a staff notice in 2013 (Staff Notice 43-307 Mining Technical Reports – Preliminary Economic Assessments) clarifying its position regarding the use and disclosure of PEAs by issuers in the mining sector) as quoted below. This guidance however has not yet been published as an update to the NI 43-101 Standards of Disclosure:

"CSA broadened the definition of PEA in response to industry concerns that issuers needed to be able to take a step back and re-scope advanced stage projects based on new information or alternative production scenarios. In this context, the revised definition is based on the premise that the issuer is contemplating a significant change in the existing or proposed operation that is materially different from the previous mining study. In most cases, this will also involve considerably different economic parameters and capital investments. Examples of a significant change are a different scale of proposed operation (higher or lower throughput), a different scope of operation (higher or lower grade), the inclusion of other types of mineralisation (oxide vs.sulfide), the use of alternative mining methods (open pit vs. underground), or the use of alternative processing technology".

This statement provides the rationale for re-evaluation of the Norra Kärr project. Re-evaluation of the project is justified for the following reasons:

- Recognition of potentially economic commodities in the mineralization not evaluated in the 2015 report, namely nepheline syenite, niobium and zircon
- Recognition of the need to reduce the project footprint and assess alternatives to the previous large tailings facility at the mine site
- A need to minimize waste on the project and have greater utilization of the extracted materials
- The following elements are considered as REEs in the context of this report: lanthanum (La), cerium (Ce), praseodymium (Pr), neodymium (Nd), samarium (Sm), europium (Eu), gadolinium (Gd), terbium (Tb), dysprosium (Dy), holmium (Ho), erbium (Er), thulium (Tm), ytterbium (Yb), lutetium (Lu) and yttrium (Y). Of the REEs, La, Ce, Pr, Nd, Sm are considered light rare earth elements (“LREE”) and Eu, Gd, Tb, Dy, Ho, Er, Tm, Yb, Lu and Y are considered heavy rare earth elements (“HREE”).

1.1 Project Re-Evaluation

This report presents a re-evaluation of the Project to address issues raised after publication of the 2015 PFS report (Davidson et al 2015). The main issues being, the need to increase resource utilization/efficiency, improve environmental and social governance on the project and minimize local footprint by incorporating process improvements and changes in the mining and processing approach. The main changes are;

- Mining and non-chemical processing at the Norra Kärr site will be limited to produce a multi-element mineral concentrate and a nepheline syenite by-product. The mineral concentrate will be shipped to a process facility conceptually proposed in the industrial centre of Luleå where production of REEs, niobium (“Nb”) and zirconium (“Zr”) products through leaching will occur. This results in a significant increase in resource utilization compared with the 2015 PFS where only REEs were considered to be produced.
- Smaller footprint for the mine and associated facilities resulting in an approximately 75-80% reduction in land area use at the Norra Kärr site compared with the 2015 PFS.
- All chemical processing will be undertaken at a conceptually chosen brownfield location in Luleå. With no chemicals required for processing at the Norra Kärr site this also results in a smaller plant footprint consisting of only crushing, magnetic separation and storage facilities for concentrates. In addition, water requirements and discharge volumes become significantly reduced compared to the 2015 PFS.
- Control of water on site and management to prevent impacts to catchment of Lake Vättern includes minimising the need to abstract water from the lake and aiming to have a zero-discharge circuit with utilization of all site contact water and minimising run-on by diversion ditches. Total on-site mine waste has been reduced from 42Mt to 16.9 Mt over the Life of Mine (9.4Mt waste rock and 7.5Mt magnetic separation waste) of which 21 % of mine waste rock will be backfilled into the pit. The reduced amount of waste enables a switch to dry tailings leaving a significantly smaller and more benign waste footprint during mine life and on closure at the Norra Kärr site relative to the 2015 PFS.

- At the off-site facility process waste will be separated into neutralized leach residue and gypsum to be placed in geomembrane lined impoundments. The gypsum also offers the future potential to be processed for the construction industry, even further increasing resource utilization, and minimizing the total amount of waste from the Project.

2 PROPERTY SUMMARY

The Project is located in southern Sweden some 15 km northeast of the town of Gränna and approximately 240 km south-east of Stockholm. The Project area is adjacent to the E4 highway that runs north-south through the entire country with excellent infrastructure including transport, power and water supply. The Project is situated within the outer zone of the Eastern Vattern Escarpment UNESCO Biosphere Reserve area, within 1 km of two EU-Natura2000 protected areas of Holkberg and Narbäck on the eastern shore of Lake Vättern– the second largest lake in Sweden and also a Natura2000 protected area (Figure ES 1).

An Exploration Permit covers the Norra Kärr deposit area that was first granted to Tasman Metals AB in 2009. This Exploration Permit was renewed on two prior occasions, and a request for a further five-year extension was submitted to the Mining Inspectorate in August 2019. The extension was granted in June 2020 with the Exploration Permit being valid until August 2024. In July 2020, the Exploration Permit was further extended to August 2025; however, this extension has been appealed by project objectors and the legal case is on-going at the time of writing. 7

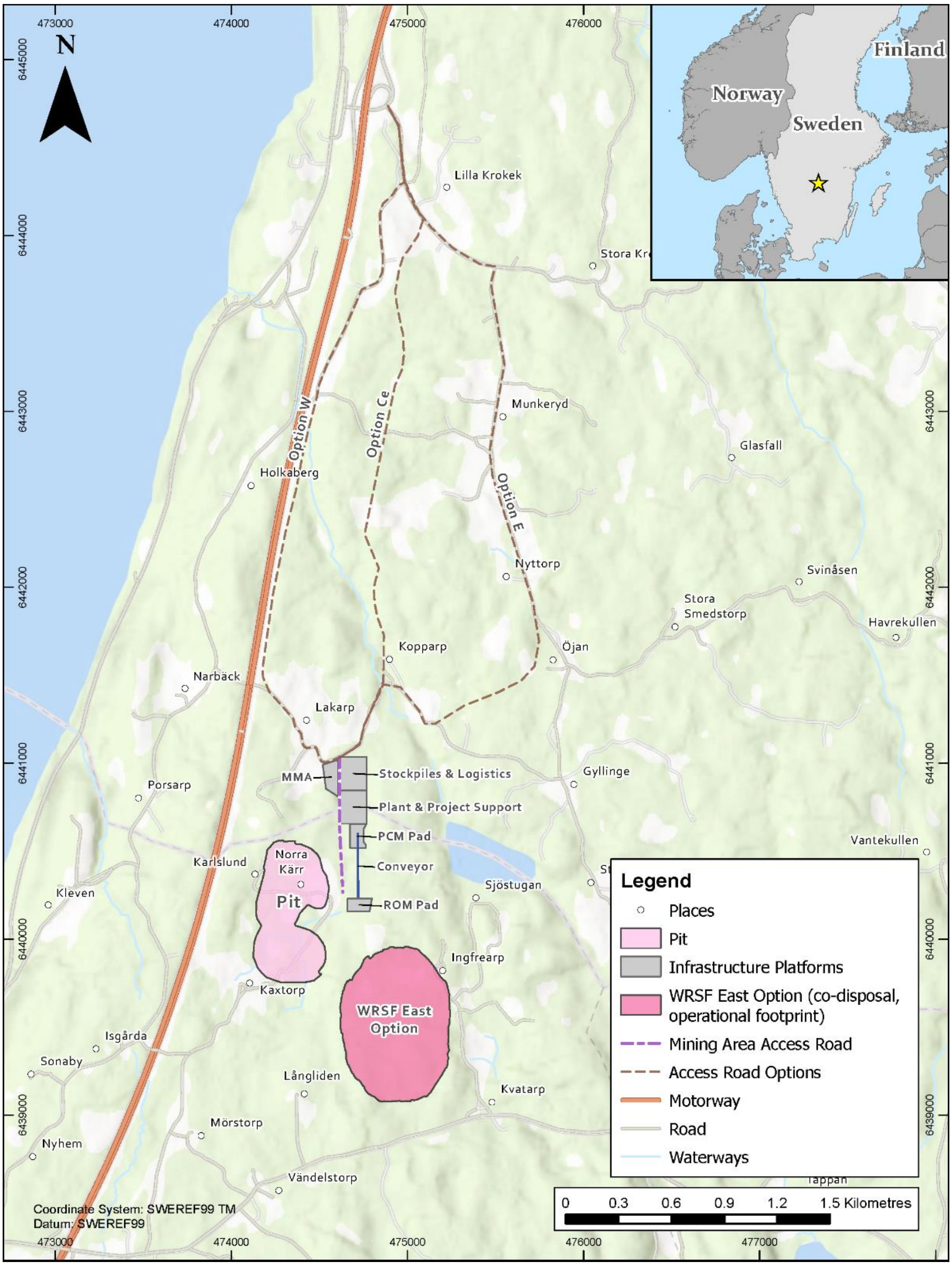

Figure ES 1: Proposed site layout and location. Insert show's location relative to the region.

The Project was explored extensively throughout the 20th century since the Swedish Geological Survey ("SGU") discovered the Norra Kärr alkaline igneous complex in 1906. Boliden AB explored the area extensively in the 1940s and again in the 1970s for zirconium and rare earth element potential. The Company commenced drilling in December 2009 and as of the end of 2012 had completed a total of 119 drillholes totalling 20,420 m. Drilling was carried out on east-west section lines with drilling inclined to the east at approximately 50°. Section lines are spaced approximately 50 m apart with drillholes spaced along the section lines at 60 to 80 m. No drilling has been undertaken on the deposit since 2012 except for 10 dedicated geotechnical and hydrogeological drillholes completed in 2014 (no samples were assayed).

3 GEOLOGY

The REE and Zr-bearing Norra Kärr deposit is hosted by an alkaline igneous intrusion dominated by various sub-categories of nepheline syenite. The intrusion is termed "agpaitic grennaite" (named after the nearby town of Gränna), which is described as a peralkaline nepheline syenite containing complex and exotic silicate minerals such as eudialyte and rinkite (containing zirconium, titanium, rare earth elements). It is part of the Precambrian age (~1 Ga) Trans Scandinavian Igneous Belt ("TIB"), which comprises a giant elongated array of batholiths extending 1,400 km across Scandinavia from southeast Sweden to northwest Norway.

The REE and Zr mineralisation is hosted by the complex -silicate mineral eudialyte. There are minor amounts of secondary REE-bearing britholite and trace mosandrite. The eudialyte at Norra Kärr is relatively rich in REE compared to most other similar deposits and also contains a very high proportion of HREO.

4 MINERAL RESOURCE ESTIMATE

The deposit model described herein was completed by WAI in 2015 as part of the PFS. SRK has used this model to produce an updated Mineral Resource estimate ("MRE") recognising additional product potential and using up-to-date pit optimization parameters. As part of this process, SRK analysed the potential qualities of nepheline syenite in addition to assigning value to other by-products. For each block, a net recoverable value (USD /t) has been calculated based on SRK's technical-economic model; this has been used as a cut-off grade for the purpose of reporting the MRE.

SRK did not change the block model generated by WAI, the model has been reviewed and adopted by SRK for the purpose of this PEA.

WAI utilised the drilling and historic mapping and trenching data to generate a 3D geological model based on lithological logging codes. A block model was generated and grades of each separate REE elements, zirconium, niobium, uranium and thorium were estimated into lithological domains separately. Variography was completed on the domained drillhole data and ordinary Kriging interpolation methodology was chosen for the grade estimate. The grades were validated using check estimates along with visual and statistical methods.Nepheline Syenite grades have been assigned by SRK according to mineralogical and metallurgical observations.

Quality control data is available for TREO grades only and no mining has been completed with which to verify the model through reconciliation. Density values were assigned to the model as an average per lithological unit to report tonnages.

Table ES 1 presents the updated Mineral Resource statement which has been prepared by SRK in accordance with the terminology, definitions and guidelines given in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014). Reasonable prospects for eventual economic extraction have been demonstrated according to the findings of this PEA which have been used to constrain the MRE to an updated pit shell and a cut-off grade of USD150/t. The statement has an effective date as of August 18 2021.In terms of the TREO composition the split is 48% LREO and 52% HREO (Table ES 2).

Table ES 1: Norra Kärr Mineral Resource Statement (SRK, 18 August 2021)*

Mineral Resource Classification	Tonnes (Mt)	TREO (%)	ZrO_2 (%)	Nb_2O_5 (%)	Nepheline Syenite (%)
Inferred	110	0.5	1.7	0.05	65

**Notes:*

1. *Effective date 18 August 2021.*
2. *Qualified Person Mr Martin Pittuck MSc C.Eng*
3. *Mineral Resources are not Mineral Reserves until they have Indicated, or Measured confidence and they have modifying factors applied and they have demonstrated economic viability based on a Feasibility Study or Prefeasibility Study.*
4. *There is no guarantee that Inferred Mineral Resources will convert to a higher confidence category after future work is conducted.*
5. *The Mineral Resources reported have been constrained using an open pit shell assuming the deposit will be mined using open pit bulk mining methods and above a cut-off grade of USD150/t., including a 30% premium on projected commodity prices and unconstrained by commodity production rates and the 260m highway buffer zone.*
6. *The Mineral Resources reported represent estimated contained metal in the ground and has not been adjusted for metallurgical recovery.*
7. *Total Rare Earth Oxides (TREO) includes: La_2O_3, Ce_2O_3, Pr_2O_3, Nd_2O_3, Sm_2O_3, Eu_2O_3, Gd_2O_3, Tb_2O_3, Dy_2O_3, Ho_2O_3, Er_2O_3, Tm_2O_3, Yb_2O_3, Lu_2O_3, Y_2O_3.*
8. *Heavy Rare Earth Oxides (HREO) include: Eu2O3, Gd2O3, Tb2O3, Dy2O3, Ho2O3, Er2O3, Tm2O3, Yb2O3, Lu2O3, Y2O3*
9. *HREO is 52% of TREO*

Table ES 2: Norra Kärr Rare Earth Element Distribution

Light REO proportion of Total REO					Heavy REO proportion of Total REO									
La_2O_3	Ce_2O_3	Pr_2O_3	Nd_2O_3	Sm_2O_3	Eu_2O_3	Gd_2O_3	Tb_2O_3	Dy_2O_3	Ho_2O_3	Er2O_3	Tm_2O_3	Yb_2O_3	Lu_2O_3	Y_2O_3
0.100	0.210	0.030	0.110	0.030	0.004	0.030	0.007	0.050	0.010	0.034	0.005	0.033	0.005	0.340
0.48					0.52									

5 MINING METHODS

The mine planning work was carried out using a mining model, which was generated from the resource model through a regularization process to a block size of 5 m x 5 m x 5 m. A pit optimization was undertaken with a 260 m exclusion from the highway. SRK identified revenue factor 56% as the optimal shell to carry forward based on the highest average discounted cashflow assuming a production rate of 1.15 Mtpa of plant feed and a discount rate of 10%. The selected shell contains 29.7 Mt of mineralized rock and 9.2 Mt of waste for a strip ratio of 0.31.

The pit designs have been divided into four stages, with starter pits in both the north and south followed by ultimate pits in both areas which combined to form one final pit. The tonnage and grade differences between the selected shell and pit design are less than 2%. The ultimate pit inventory is 29.3 Mt of mineralized rock and 9.4 Mt of waste for a strip ratio of 0.32. There is one external waste rock storage facility and a backfill storage facility in the north pit to limit external waste storage.

The mine schedule targeted 1.15 Mtpa of plant feed and sought to maximize waste backfill quantities. The sequence starts in Stage 1, with Stage 2 commencing in Year 2. Stage 3 begins in Year 3, while Stage 4 is delayed until Year 16 to maximize backfill options. Total production averages 1,625 ktpa from Year 3 to 9, after which total material movement decreases as the strip ratio in Stage 3 decreases. Waste stripping requirements increase starting in Year 16 as Stage 4 begins, averaging 1.8 Mtpa until Year 20. The delay of Stage 4 allows for 1.9 Mt or 21% of total waste to be backfilled in the pit void.

The mine equipment includes two 5.5 m^3 excavator with up to six 46.8 t payload haul trucks. There will also be a stockpile loader and two 110 mm drills.

6 PROCESSING

Process evaluation has focused on the following approaches:

On Site Processing

- Magnetic separation in two stages to separate eudialyte from the bulk of the mineralized rock. The bulk of this non-magnetic residue is nepheline which will be sold as an industrial mineral. Second stage magnetic separation will separate fine eudialyte from aegirine that will be a waste on site.

Off-Site Processing

- Two stage sulfuric acid leaching to separate Zr, Nb and TREO from eudialyte.
- Separation and recovery of Zr, Nb and TREO into saleable products

Future work will look at the feasibility of utilizing hydrochloric acid in place of sulfuric in a closed circuit with nanofiltration recovery of acid and potential to recover Hf

The process will be undertaken at two sites (Figure ES 2).

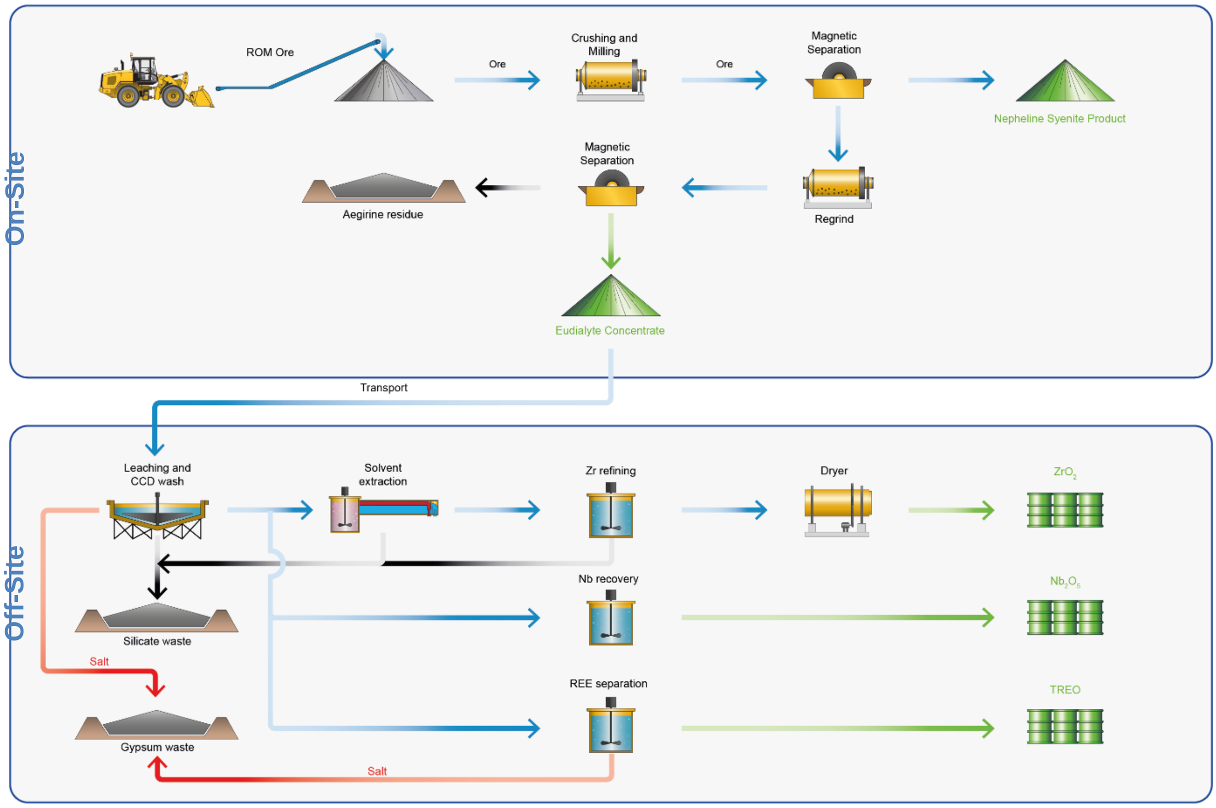

Figure ES 2: Overview schematic of modified process for Norra Kärr deposit

At Norra Kärr, primary and secondary crushing followed by two stage magnetic separation. No reagents will be used in this process. A concentrate will then be shipped for further processing off site. Approximately 7.5 million tons of aegirine residue will be stockpiled on site over the Life of Mine. Further development work will be undertaken to determine if this can be utilized as an additive in high temperature cement.

At Luleå in a brownfields industrial area the leaching separation in a hydrometallurgical facility will be undertaken along with metal separation and recovery. It is proposed that Zr will be produced as chemical grade zirconium oxide, Nb as niobium oxide and the REE as combined oxide (REO) for further refining. Nepheline syenite will be produced at the mine site as an industrial mineral.

In future work the potential to generate Hafnium as a by-product should be evaluated once a resource is estimated. Also, the Luleå processing facility will produce approximately 2.4 million tonnes of gypsum over the estimated life of project. This may also be a potential by-product.

7 WATER MANAGEMENT

General

Management of water at the mine is important both to meet make-up requirements and to limit the potentially adverse effects of run-off, groundwater flow and elevated pore pressures in the pit and its environs on the day-to-day operations of the mine.

Water Management Design

The present pit water management design uses the schedule of pit inflows produced by the WAI groundwater model but extends the period of peak flow (180m^3/h) by an additional 6 years to 26 years. With the exception of Year 1, Stage 1 of pit development, when there will be one duty pump and one sump, it is estimated there will be a requirement for two sumps, one for the southern pit and one for the northern pit, and two pairs of skid-mounted duty and stand-by pumps capable of lifting water against a head of between 20m and 160m, depending on the stage of the project. Each sump and pump shall have a rising main to lift the water out of the pit to a settling pond located outside the excavation. Pit water will be discharged in to the Adelovan stream, although where possible it should be used in the process to limit fresh make-up water from external sources and discharges to the stream should be regulated so there is minimal disruption to normal stream flow patterns.

Surface run-off towards the pit and waste facilities from the surrounding catchment will be diverted using berms and ditches positioned around the periphery of these structures.

Most of the rock mass to be mined is strong enough not to require active pit dewatering using peripheral wells and gravity drains. However, the poor ground associated with the fault in the SE corner of the pit and, possibly some parts of the west wall where fenitised granite will be exposed may require some gravity drains to lower in-situ pore pressures.

The impact of pit dewatering on the surrounding aquifers will be monitored using approximately 8 standpipe wells and up to 3 VWPs installed both inside and beyond the periphery of the pit.

Water supply will depend on pit inflow (although this should be used to keep reliance on external water supply to a minimum), captured runoff and a small quantity from Lake Vättern. The supply assumes a lake-side offtake with a duty and a stand-by pump, each capable of pumping 5-40 m^3/h water along approximately 1900m of buried HDPE pipe to the mine. This is considerably less than the originally predicted average of 100m^3/h (Davidson et al 2015).

8 INFRASTRUCTURE AND LOGISITICS

The Norra Kärr Project has three project areas:

- The Norra Kärr site where there is open pit mining and comminution plant;
- Luleå Processing Facility (“LPF”) where further chemical processing is undertaken; and
- The rail load-out facility, where Eudialyte “intermediate” product is received from Norra Kärr by truck and loaded to rail hopper wagons for transport to the LPF.

All sites require buildings and installations, utilities connections and power to support the mining and processing operations. This infrastructure is described in this section. Site wide water management is covered within the Water management section.

8.1 Norra Kärr

A new access road to the project site will be required. It is proposed to utilise the existing Stava junction (E4 / Road 500 intersection) and construct a 4.0 km access road between the junction and the site. Three preliminary options have been considered and will be assessed further at the next stage of study.

The project will be connected to the national grid via new grid connection to the Swedish National grid system involving connection works at the connection substation (e.g. at Tranås, Ödeshög, or Gränna), overhead transmission lines (e.g. circa 10-25 km, at between 20 - 35 kV and a Project Main Substation with a stepdown to site distribution voltage.

At Norra Kärr, the make-up water to the plant will be via a surface water abstraction pump and pipeline and once operational, much of the make-up water to the plant will be sourced from dewatering boreholes at the open pit.

Buildings and installations are required and are designated as supporting either mining, processing, product logistics, or the project overall. These facilities are positioned into compounds or areas for the purposes of management and control. Each area will be independently fenced.

Site wide utilities and services are required including MV / LV electrical distribution, area lighting, potable water storage and reticulation (e.g. for ablutions, kitchen, dining), raw water storage and reticulation from the main raw water tank (fire water, vehicle washing, dust suppression), fire water reticulation to fire water tanks in plant, compounds, and areas, stormwater / surface water management and pollution control, sewerage and wastewater reticulation and treatment, and Security systems (including front gate), alarm, CCTV and movement detection systems.

8.2 Off-Site Processing Facility

Eudialyte mineral will arrive at an off-site processing facility (“LPF”) for refinement to final products. The LPF is expected to be situated on an industrial park whose exact location is yet to be confirmed. The selected industrial park is anticipated to be a multi-occupancy site designed for potentially energy intensive industrial usage such as chemical and process plants. It is envisaged that the LPF will be constructed on a parcel of “brownfield” land and for the purposes of this study a bulk power and water connection is required from local connection points as well as for security and other utilities functions. It has been assumed that an existing railway line is located adjacent to the site. As a basis for this PEA, a location in the Luleå area within an existing industrial area and within easy reach of rail facilities is assumed as a viable option for consideration in this report.

Because an exact parcel of land is yet to be determined, a schematic layout was prepared as a basis for the PEA to ensure all technical and cost aspects are captured and the likely footprint is understood. There are four main asset areas:

- Processing facility building, equipment and materials handling;
- Waste storage facility;
- Support Infrastructure
- Logistics and warehousing.

All buildings and installations will support the processing and logistics operations These facilities are positioned within the general plant compound. In general buildings will be pre-engineered steel portal framed or column and beam style buildings with insulated panel roofs and cladding and with all necessary internal electrical, piping, fixtures and fittings, and architectural details. It's likely that some auxiliary buildings will be prefabricated and pre-fitted, modular, or converted container style buildings.

Logistics and warehousing at the LPF will be important. There will be a number of different consumables being imported by road and rail as well as the Eudialyte mineral concentrate. The products will be exported in twenty-foot containers ("TEUs") by road or rail.

8.3 Product Logistics

Key logistics operations are as follows:

- Transport of "Eudialytemineral concentrate"; and
- The REO, Zircon and Niobium final products.

The Eudialyte mineral concentrate (approximately 105ktpa) needs to be transported to the Company's off-site processing facility, which is planned for the Luleå area. The following logistics system is proposed:

- Road haulage via the National Road E4 to the Company's rail siding using a road haulage Contractor
- Loading of rail wagons at the rail siding, which is likely to be located at Mjölby or Nassjö;
- Transport via national rail system using a rail freight company who provide wagons, locomotives and arrange access to the rail system to the Company's off-site processing facility

The Company's rail siding will consist of a warehouse, dedicated sidings and loading equipment. The logistics provider shall provide shunting capabilities.

The REO, Zr and Nb oxide final products will be containerised and transported by rail or sea-freight from the Company's offsite processing facility to destination points.

The nepheline syenite will be sold at "mine gate" and therefore only a summary of the possible transport options is presented.

9 ESG

The Norra Kärr Project is in south-central Sweden, approximately 300 km south-west of Stockholm. It is located 1.5 km east of Lake Vättern- one of the largest lakes in Sweden and a Natura 2000 site, a nature protection ecologically sensitive area designated at European level to safeguard Europe's major habitat types and endangered species. There are two other Natura 2000 protected sites on the shores of Lake Vättern– Holkaberg and Narbäck. The Project site itself does not overlap any Natura 2000 areas. The Project site and the surrounding area is characterised by alternating agricultural land, scattered homesteads and forests. The main north-south E4 highway runs approximately 500 m to the west of the project area with the site itself accessed by rural roads.

The project site itself does not overlap any European designated nature protection sites or Swedish National Parks but there are several protected sites in the vicinity. Lake Vättern has a variety of different environmental designations. The entire lake is protected under the European Habitats Directive and the north eastern portion is also designated as a Special Protection Area, a European protection designation specific to birds under Directive 2009/147/EC, referred to as the Birds Directive. The water protection zone extends up various streams draining into the lake; these are separate from the Natura 2000 protected areas but are connected. The areas are shown on Figure ES 3 along with the main Project components.

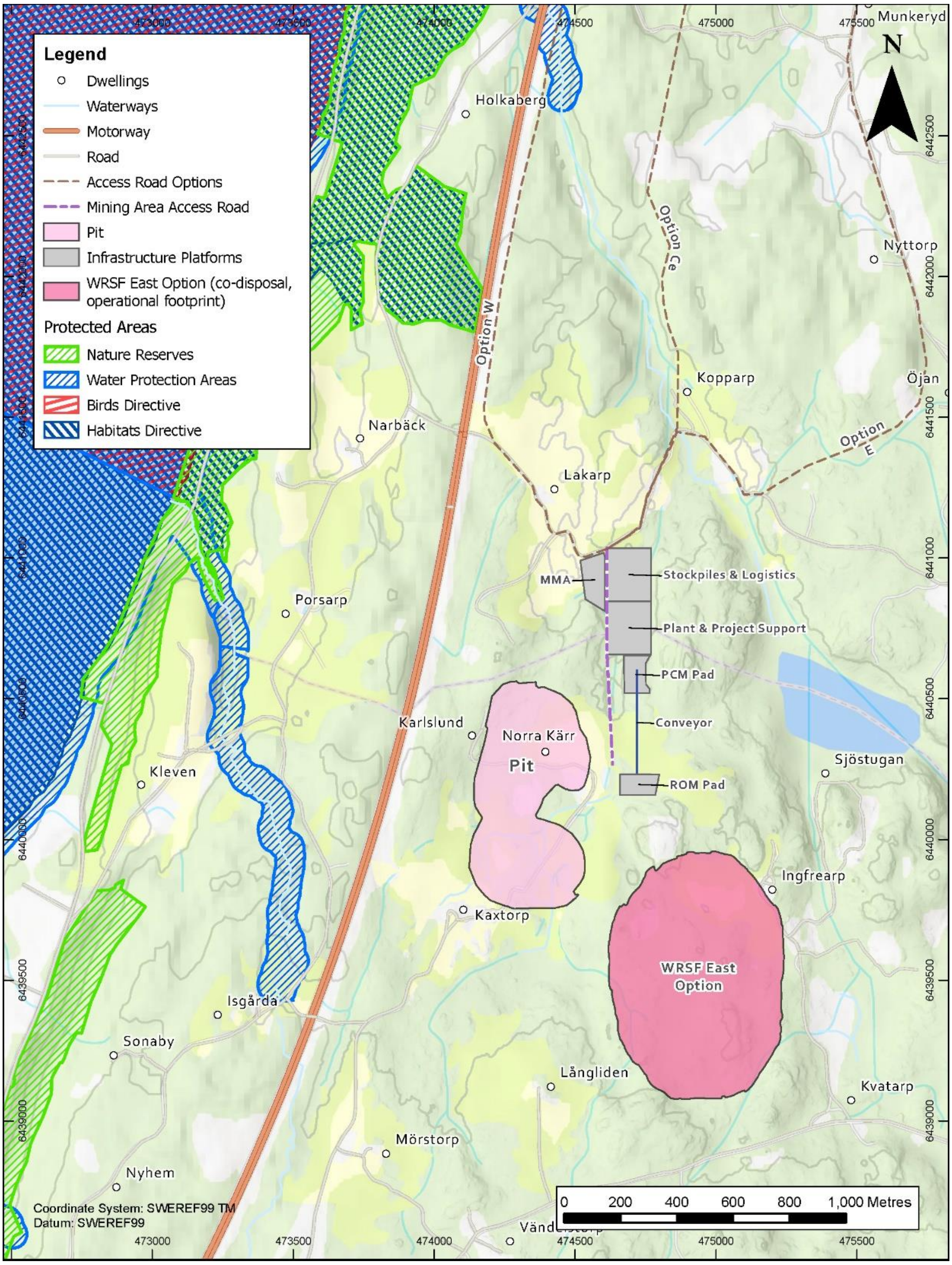

Figure ES 3: Project location and infrastructure in relation to protected sites.

The current PEA work being undertaken by SRK has resulted in a substantive update to the design of the project. This includes transport of a concentrate by road to a railway siding and then taken to Luleå by train before further processing at a brownfield industrial site near Luleå. This is a new development relative to the work carried out as part of the previous PFS in 2015 and Golder EIA produced in 2012. The layout of various project components at the proposed mine site has also evolved with the consolidation of the footprint of the infrastructure and waste management facilities reducing the number of sub-catchments potentially impacted by the infrastructure. As is normal at the PEA stage of a project, there are still a number of different project development options being considered. ESG risks and opportunities will inform the choice of the final project design.

The Norra Kärr deposit is listed as a mineral deposit of national interest by the Geological Survey of Sweden. This designation is linked to the potential for the deposit to provide a supply of rare earth elements to Sweden and Europe.

Sweden is a member of the European Union and as such is subject to the Directives and Regulations of the European Parliament and its Commission. European Directives must be transposed into member states legislation that often merely reference the text of the Directives.

There is a single Exploration Permit (Norra Kärr No.1) covering the area of the proposed pit.

The Exploration Permit, and additional perimeters now lapsed, were first granted to Tasman Metals AB in 2009 and was valid for three years. The Exploration Permit was renewed twice, and a request for a further five-year extension was submitted to the Mining Inspectorate in August 2019. The extension was granted in June 2020 with the Exploration Permit being valid until August 2024. Subsequently the Swedish parliament passed legislation to mitigate the impacts of COVID-19 by giving exploration companies an additional year to carry out their work which extends the Norra Karr exploration license to August 31, 2025. The five-year extension of the exploration license was appealed, and the administrative court of Lulea rejected the appeal in March 2021, upon which the case has been appealed to the next instance which is pending decision to grant leave of appeal. The extension of the exploration license remains in force until a final ruling in the case has been made, and remains in force until a final ruling has been made on the mining lease application (see below).

A 25-year Mining Lease was granted to the Company's Swedish subsidiary Tasman Metals AB, recently renamed to GREENNA Mineral AB, covering Norra Karr in 2013 by the Mining Inspectorate with approval of the local CAB. In 2014 the Government of Sweden upheld the granting of the mining lease after an appeal. In 2016, following an appeal to the Supreme Administrative Court (SAC) in Sweden regarding the decision-making process of the Bergsstaten under the Minerals Act, the Norra Karr mining lease reverted from granted to application status. On May 5, 2021, The Mining Inspectorate of Sweden ("Bergsstaten") rejected the mining lease application with the motivation that since the Company had not acquired a Natura 2000 permit for the Project, they were not able to rule on the mining lease application.

The Company subsequently lodged an appeal to the Government to cancel Bergsstaten's rejection of the mining lease application and continue the evaluation of the application once the SAC has ruled whether a Natura 2000 permit should be a pre-condition for the granting of a mining lease or not. This is based on the fact that this is not an isolated incident and similar case outcomes are still pending for other mining companies in Sweden too.An Environmental Permit has not yet been applied for. An application for this permit will begin after granting of a Mining Lease.

Baseline studies have been undertaken encompassing the area around the exploration permit. However further studies are expected to be needed for environmental permitting. These will need to take into consideration the changes in the project footprint and processing options.

Key ESG elements of the project that will require active management going forward include:

- Stakeholder engagement. - this appears to have been proactively managed in the early stages of the project with positive results. The lack of activity in recent years has allowed opposition voices to dominate stakeholder perceptions. LEM will need to develop a comprehensive stakeholder engagement programme to manage the interface with stakeholders at all levels of the project.
- Mine waste: - the project will need to develop a detailed waste management plan in line with EU Best Available Techniques guidance. This includes the requirement to test and classify all wastes from the process to inform the waste management plan and closure options.
- Water management: - water quality and water management will be a focus for the future project permitting. The updated project design and detailed EIA will need to clearly demonstrate there is no adverse impact on protected areas or downstream water uses as a result of the project development.
- Climate change: all new developments are being scrutinised in relation to their carbon footprint and also in relation to their climate change resilience. The Norra Kärr project will need to demonstrate that all aspects of the project have been made as energy efficient as practicable and that the design has taken climate change into consideration.
- Landscape and visual: - the Project is in an area that is important for tourism and recreation. As well as minimising the project footprint and protecting the water quality of the surrounding water bodies, the project will need to be sensitively designed to blend with the landscape. This will require consideration to be given to the location, height, shape and colour of key infrastructure and waste facilities.
- Biodiversity: - various ecological baseline studies have been conducted as part of the earlier project work but these will require updating. It is likely the scope of the studies will require review and updating in light of the increased scrutiny on biodiversity management, the Man and Biosphere Reserve boundary areas, Natura 2000 areas and the SAC ruling on the Mining Lease.
- Land use and land acquisition: - the new project design currently being considered has significantly reduced the infrastructure footprint, however, LEM will have to acquire or lease the land from the existing property owners. The lead time for this will need to be accounted for in the overall project schedule

10 ECONOMIC ASSESSMENT

The capital and operating cost estimates are considered overall to have a achieved a Scoping Study / PEA level of accuracy of ±40-50%. Costs are taken from in-house databases and recent budget quotes or benchmarks.

The capital cost estimate includes direct and indirect costs and a 20% contingency. Table ES 3 presents a summary of the initial capital cost for the Project which totals USD487M and is split between the Norra Kärr and Lulea sites. In additional to this, allowance is made for sustaining capital costs which total a further USD84M over the life of mine. An allowance of USD35M is also included for closure related costs at the end of the life of mine.

Table ES 3: Initial Capital Cost Summary

Project Capital Cost Summary	Units	Project	Norra Kärr	Luleå
Mining	(USDk)	12,748	12,748	-
Processing	(USDk)	261,220	65,305	195,915
Water Supply	(USDk)	1,007	1,007	-
TSF/Waste Management	(USDk)	8,168	3,607	4,561
Transport/Handling	(USDk)	8,352	8,352	-
Infrastructure/Utilities	(USDk)	43,980	19,920	24,060
Owners/General	(USDk)	15,000	7,500	7,500
Sub-total Direct	(USDk)	**350,475**	**118,439**	**232,036**
EPCM	(USDk)	31,543	10,659	20,883
Indirect	(USDk)	35,047	11,844	23,204
Contingency	(USDk)	70,095	23,688	46,407
Sub-total Indirect	(USDk)	**136,685**	**46,191**	**90,494**
Total	(USDk)	**487,160**	**164,630**	**322,530**

The operating cost estimate includes mining, processing, waste management, infrastructure/transport and owners' costs. In addition, a toll treatment charge of USD19/kg of REO sold is applied as a sales cost and included in the operating costs in the assessment. Table ES 4 presents a summary of the total operating costs (excluding by-product credit).

Table ES 4: Operating Cost Summary

Operating Cost Summary	Units	LoM	Av Annual	USD/t ore	USD/kg REO
Mining	(USDk)	164,960	6,345	5.63	1.19
Processing - Norra Kärr	(USDk)	525,617	20,216	17.93	3.79
Processing - Luleå	(USDk)	975,599	37,523	33.28	7.03
G&A	(USDk)	146,577	5,638	5.00	1.06
Transport	(USDk)	144,544	5,559	4.93	1.04
Sales	(USDk)	2,638,378	101,476	90.00	19.00
Royalty	(USDk)	21,898	842	0.75	0.16
Total	(USDk)	**4,617,572**	**177,599**	**157.51**	**33.25**
By-product revenue credit				**-**	**18.68**
Total (net of by-product credit)					**14.57**

SRK has constructed an Excel based Technical Economic Model (TEM) to assess the viability of the Norra Kärr project based on assumptions derived by SRK for the technical and economic aspects of the Project. In summary this includes the following:

- A mining/processing schedule based on the MRE, with a mining/processing rate of some 1.15Mtpa at steady state which has been sized primarily based on the REO market.
- Processing mass yields and recoveries to final products.
- Mining operating and capital costs.

- Processing operating and capital costs.
- Waste management capital and operating costs.
- Owners development and business services costs ('G&A').
- Infrastructure/transport operating and capital costs.
- Closure costs.

SRK has constructed a pre-finance and post-tax TEM on an annual basis and assumed:

- The currency is USD in Q1 2021 real terms.
- Construction starts in 2023 and continues over a 2-year period with processing of mineralized rock commencing in 2025.
- A base case discount rate of 10% to derive a NPV at 1 January 2023 (i.e. the start of construction).
- All numbers presented in this chapter are on a 100% ownership basis.

Revenue is assumed to be generated from a TREO product and also by-products: Zr product, Nb product and nepheline syenite product. Commodity price forecasts have been provided by Adamas (TREO) and other industry specialists. Some 74% of the LoM gross revenue is from the TREO product with the remining 26% from by-products (11% from the Zr product, 11% from the NS products and 5% from the Nb product). The assumed LoM average TREO basket price equates to USD53/kg.

Figure ES 4 below illustrates the Project yields an average LoM net operating margin of USD38.48/kg REO after taking into account credit from by-product revenue and with an operating cost net of by-product credit of USD14.57/kg REO.

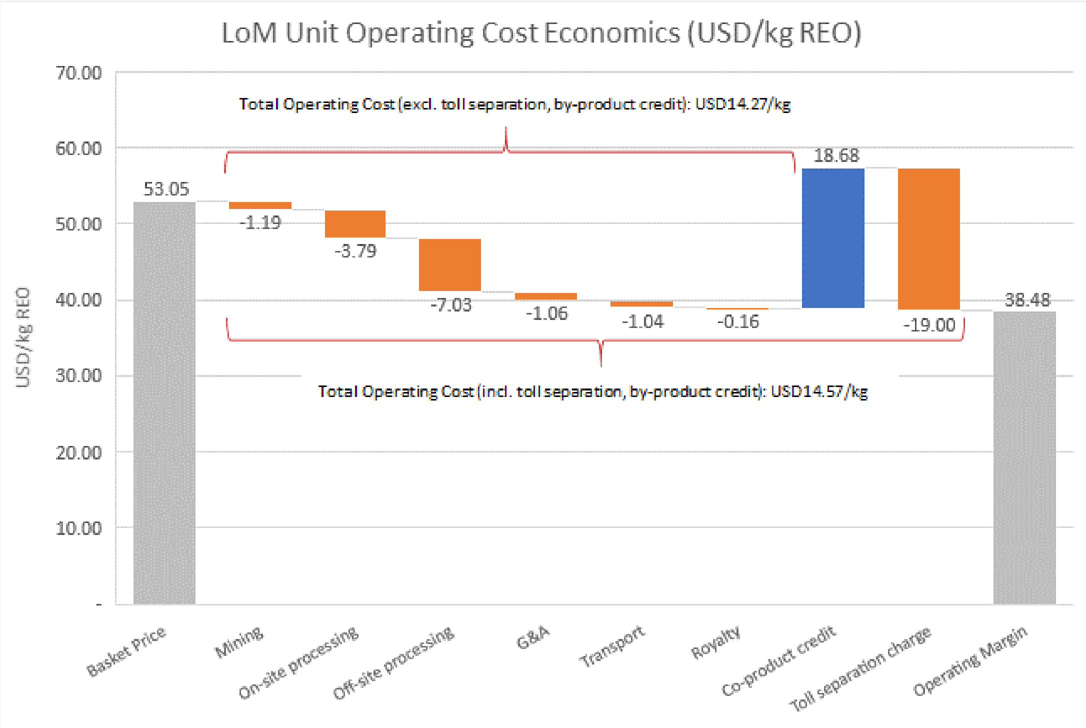

Figure ES 4: LoM Unit Operating Economics

Table ES 5 presents a summary of the LoM cashflow over a 26-year mine life. At a 10% discount rate the Project yields a robust post-tax NPV of some USD762M and a post-tax IRR of 26.3%

Table ES 5: LoM Cashflow Summary

Cashflow Summary	Units	LoM
Gross Revenue	(USDk)	9,961,593
Operating Costs	(USDk)	(4,617,572)
Net Operating Cashflow	(USDk)	**5,344,021**
Capital Costs	(USDk)	(606,354)
Working Capital	(USDk)	0
Net Project Cashflow (Pre-Tax)	(USDk)	**4,737,667**
Corporation Tax	(USDk)	(985,074)
Net Project Cashflow (Post-Tax)	(USDk)	**3,752,593**

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PRELIMINARY ECONOMIC ASSESSMENT OF NORRA KÄRR RARE EARTH DEPOSIT AND POTENTIAL BY-PRODUCTS, SWEDEN

1 INTRODUCTION

1.1 Purpose of Technical Report

SRK Consulting (UK) Limited ("SRK") is an associate company of the international group holding company, SRK Consulting (Global) Limited (the "SRK Group"). SRK was requested by Leading Edge Materials Corporation ("LEM", hereinafter also referred to as the "Company"), to undertake an updated Preliminary Economic Assessment ("PEA") for the Norra Kärr Rare Earth Element ("REE") deposit in Sweden ("Norra Kärr or the "Project").

Development work on the project was commissioned by Tasman Metals AB ("Tasman"), who are a 100%-owned subsidiary of LEM and owner of the Project. LEM's predecessor (Flinders Resources Limited) acquired Tasman Metals AB in 2016, thereafter changing the name to Leading Edge Materials Corporation and therefore all Tasman Metals AB sourced data is wholly owned by LEM and is stated as such in the report. For simplicity all work by Tasman and LEM is referred to as work owned or commissioned by LEM.

This report presents a re-evaluation of the Project to increase resource utilization and minimize local footprint by incorporating process improvements and changes in the mining and processing approach. The main changes are;

- Mining and non-chemical processing at the Norra Kärr site to produce a multi-element mineral concentrate and a nepheline syenite by-product. The mineral concentrate will be shipped to a process facility conceptually proposed to be in the industrial centre of Luleå where production of REE (also referred to in this report in oxide form as "TREO" or "REO"), niobium ("Nb") and zirconium ("Zr") products through leaching will occur. The overall result is a significant increase in resource utilization compared with the 2015 PFS where only REEs were considered to be produced.
- Smaller footprint for the mine resulting in 75-80% reduction in land area use at the Norra Kärr site compared with the 2015 PFS.
- All chemical processing will be undertaken at a conceptually chosen brownfield location in Luleå. With no chemicals required for processing at the Norra Kärr site this also results in a smaller plant footprint consisting of only crushing, milling, magnetic separation and storage facilities for concentrates. In addition, water requirements and discharge volumes become significantly reduced vs the 2015 PFS from 100 m^3/h to less than 40 m^3/h and no discharge to surface water is anticipated at the mine site with utilization of all dewatering water and runon
- Control of water on site and management to prevent impacts to catchment of Lake Vättern

- Total on-site mine waste has been reduced from approximately 42 Mt to 16.9 Mt (9.4Mt waste rock and 7.5Mt magnetic separation reject) of which 21% of mine waste rock will be backfilled into the pit during later stages of mining and as part of reclamation. The reduced amount of waste enables a switch to dry tailings leaving a significantly smaller and m benign waste footprint during mine life and on closure at the Norra Kärr site relative to the 2015 PFS.
- At Luleå process waste will be separated into neutralized leach waste and gypsum to be placed in geomembrane lined impoundments and. The gypsum also offers the potential to be processed for the construction industry, even further increasing resource utilization, and minimizing the total amount of waste from the Project.

The main aim of the PEA is to demonstrate a more sustainable use of all mined resources and shifting the chemical processing to a more suitable location which will reduce environmental risk profile of the Project at the Norra Kärr site whilst adding potential revenue additions by doing so.

Also, the high-level redesign of the Project has considered the recovery of zirconium and niobium to ensure improved resource recovery and lastly evaluated moving the hydrometallurgical processing to a more suitable off-site location.

SRK has relied on the Mineral Resource estimate ("MRE") completed by Wardell Armstrong International Limited ("WAI") in 2014 as part of the latest Prefeasibility Study ("PFS") as the basis for the updated statement and PEA. The latest PFS report is entitled "Amended & Restated Prefeasibility Study - NI43-101 - Technical Report for the Norra Kärr Rare Earth Element Deposit", dated July 2015 (Davidson *et al*, 2015b). SRK was not requested to review the quality of the work completed for this PFS and cannot comment on the validity of the conclusions. This updated Technical Report is focussed on the economic viability of the addition of nepheline syenite, Zr and Nb products as by-products of mining and processing the REE mineralised material.

This PEA has been prepared in a Technical Report format in compliance with Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F guidelines - Standards of Disclosure for Mineral Projects.

In this report, SRK has considered all aspects of the Project including geology, mineral resources, mining, processing and environmental and social issues in accordance with the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Mineral Resource and Mineral Reserve definitions and in conformity with generally accepted CIM "Exploration Best Practices" guidelines.

1.2 Independent Consultants

SRK is an associate company of the international group holding company SRK Consulting (Global) Limited. The SRK Group comprises over 1,400 staff, offering expertise in a wide range of resource engineering disciplines with 45 offices located on six continents. The SRK Group's independence is ensured by the fact that it holds no equity in any project. This permits the SRK Group to provide its clients with conflict-free and objective recommendations on crucial judgement issues.

The SRK Group has a demonstrated track record in undertaking independent assessments of resources and reserves, project evaluations and audits, Mineral Experts' Reports, Competent Persons' Reports, Mineral Resource and Ore Reserve Compliance Audits, Independent Valuation Reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide.

The SRK Group has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs. SRK also has specific experience in commissions of this nature.

1.3 Sources of Information

An initial Pre-Feasibility Study was published on Norra Kärr by the predecessor company ofLEM (Davidson et al 2015 a,b).

Subsequent to this the company has participated in European Union (EURARE program) into the processing of eudialyte-rich mineralized rock.

All information contained in this Technical Report has been supplied by LEM. The authors have relied upon this information from LEM, in addition to the PFS (2015) and internal reports covering the areas of exploration, infrastructure, environmental and legal matters and the results of metallurgical work on improving metal extraction and on the production of nepheline syenite by-products have also been utilized.

1.4 Site Visit

The SRK project manager, Dr Robert Bowell, made a site visit to Norra Kärr and to view core held at Woxna mine from June 28th to July 3rd 2021.

2 RELIANCE ON OTHER EXPERTS

SRK has relied upon the latest PFS technical report produced by GBM Minerals Engineering Consultants Ltd ("GBM", Davidson *et al*, 2015b) and metallurgical research groups, principally from Aachen university with some sections herein replicated from this report. SRK has ensured that all chapters authored by WAI and GBM are highlighted to ensure transparency. This information has been taken in good faith and SRK has not been provided with original analytical forms so cannot validate correct cross over of primary to processed information.

In addition, the marketing section (section 19) has been overseen by LEM utilizing recent reports from Adamas Intelligence and other specialists as cited in that section.

3 PROPERTY DESCRIPTION AND LOCATION

The following chapter was prepared by WAI and GBM in the latest PFS report (Davidson *et al*, 2015b) and has been modified by SRK to ensure it reflects the current status of the Project.

3.1 Location

The Project is located close to the eastern shore of Lake Vättern- one of the largest lakes in Sweden (Figure 3-1). It is 15 km northeast of the town of Gränna, 45 km to the north-north-east of the city of Jönköping, approximately 240 km south-west of Stockholm and 160 km east-north-east of Göteborg (Gothenberg). The Project occurs along the border of two counties (Lan), Jönköpings Lan in the south and Östergötlands Lan in the north. Approximately 75% of the intrusive is located in Jönköping County. The Project is centred at coordinate 474428 E by 6440268 N (approximately) by the SWEREF99TM coordinate system (58° 6'10'' North and 14° 33' 58'' East based on the WGS 84 datum).

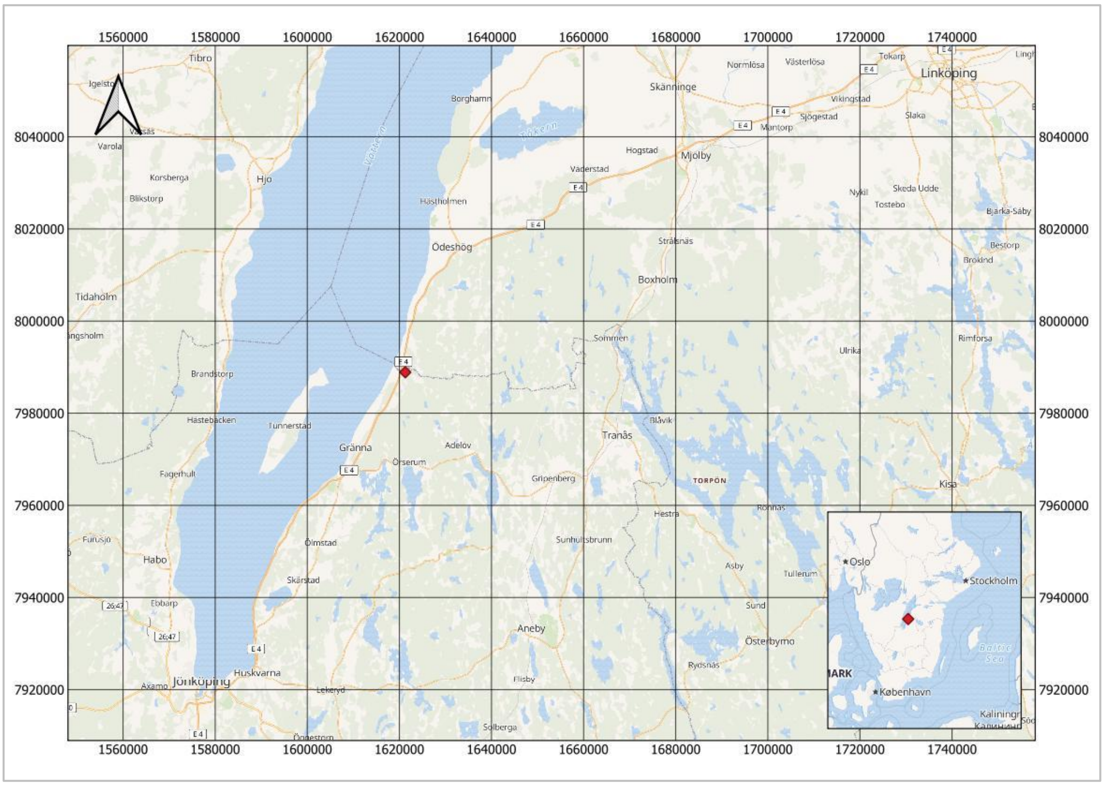


Figure 3-1: **Location of the Norra Kärr Project (red diamond), Sweden**

3.2 Mineral Title

3.2.1 Swedish System

Rules and regulations pertaining to mining exploration in Sweden are clearly outlined in the "Guide to Mineral Legislation and Regulations in Sweden" (2006) available from the offices or the website of the Swedish Geological Survey ("SGU", www.sgu.se). The Mining Inspectorate of Sweden (Swedish: *Bergsstaten*) provides clear directives, available from the Mining Inspectorate website (www.bergsstaten.se), for conducting exploration. Another useful link that summarizes these laws and guidelines is "A Guide to Mineral Legislation and Regulations in Sweden" published in 1995: (http://www.geonord.org/law/minlageng.html).

The Company addressed all the requirements before undertaking any exploration activities, and has the rights to access the property, and no restrictions or limitations as defined for work on the projects are evident. The Company has the obligation to outline a work programme and gain permission from landholders prior to accessing the properties, and to provide compensation for any ground-disturbing work conducted.

Exploration Permits (Swedish: *undersökningstillstånd*) are granted for specified areas that are judged by the Mining Inspectorate to be of suitable shape and size that they are capable of being explored in "an appropriate manner." The current rules do not require annual minimum expenditures on claims, but an exploration fee is due upon first application for an exploration permit and subsequent extension applications.

It is possible to extend the time an Exploration Permit is held to a total of 15 years after the date of the original granting. No further extension of Exploration Permit is allowed after year 15. The high fees in the later years discourage excessive claim holdings deemed to be of little value by the holder. However, application for Mining Lease is not linked to Exploration Permit. The exploration report, with results (raw data), must be submitted to the Mining Inspectorate. An Exploration Permit gives access to the land and an exclusive right to explore within the permit area. It does not entitle the holder to undertake exploration work in contravention of any environmental regulations that apply to the area. Applications for exemptions are normally made to the County Administrative Board (the "CAB").

3.2.2 Exploration Permit

The Project is contained within one Exploration Permit, Norra Kärr No. 1, which covers an area of 139 hectares (1.4 km^2). A map of the Exploration Permit from the SGU website (as of 05/02/2021) is provided in Figure 3-2.

In August 2019 an application was made to extend the exploration permit for another 5 years. This extension was granted by the Mining Inspectorate in June 2020. The Mining Inspectorate decision was subsequently appealed to the Administrative Court. In March 2021 the Administrative Court upheld the granting of the extension of the exploration permit. The ruling of the Administrative Court has subsequently been appealed to the Administrative Court of Appeals. The Administrative Court of Appeals will first have to grant leave of appeal to hear the case; this is currently ongoing. The extension of the exploration license remains in force until a final ruling in the case has been made, and remains in force until a final ruling has been made on the mining lease application (see below).

3.2.3 Mining Lease

A 25-year Mining Lease for a portion of the perimeter of the Norra Kärr Exploration Permit 1 was granted to the company in May 2013 by the Mining Inspectorate with approval which covers an area of 47 hectares (0.47 km^2). An appeal was made regarding the decision to award the Mining Lease, the decision was upheld by the Government of Sweden in 2014 but ultimately reverted to an application in February 2016 by the Supreme Administrative Court of Sweden ("SAC"). The SAC has determined that the decision by the Government to uphold the granting of the Mining Lease was incorrect, as the decision to grant the Mining Lease was '*not adequately supported by environmental studies into the potential impact of surrounding facilities of future mining operation*'. The Mining Lease is currently in the application stage, as shown on Figure 3-2. The status of the Mining Lease is described further in Section 18.3 The application is to extract REE along with zirconium, niobium and nepheline syenite.

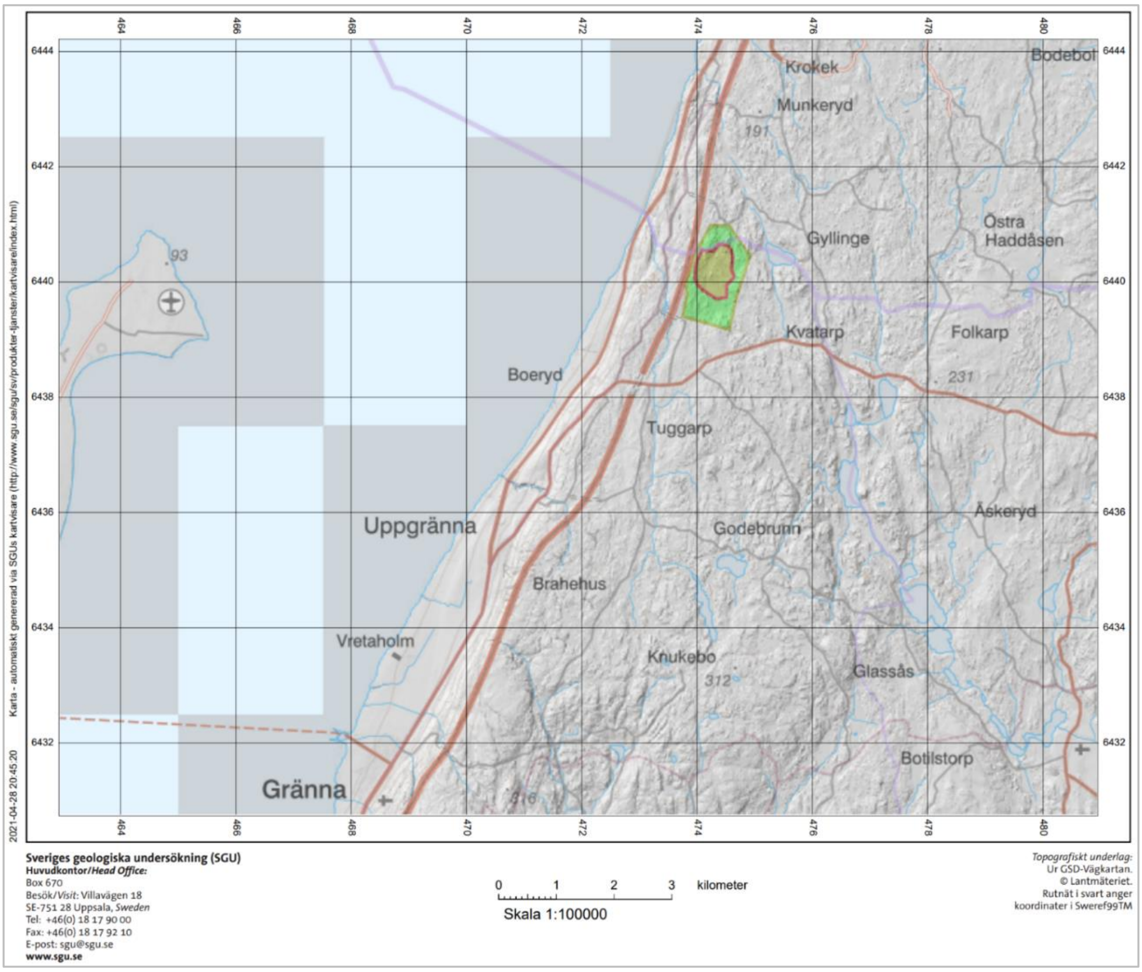


Figure 3-2: Regional location of Norra Kärr exploration permit (green) (Source: SGU map viewer, as of 05/02/2021)

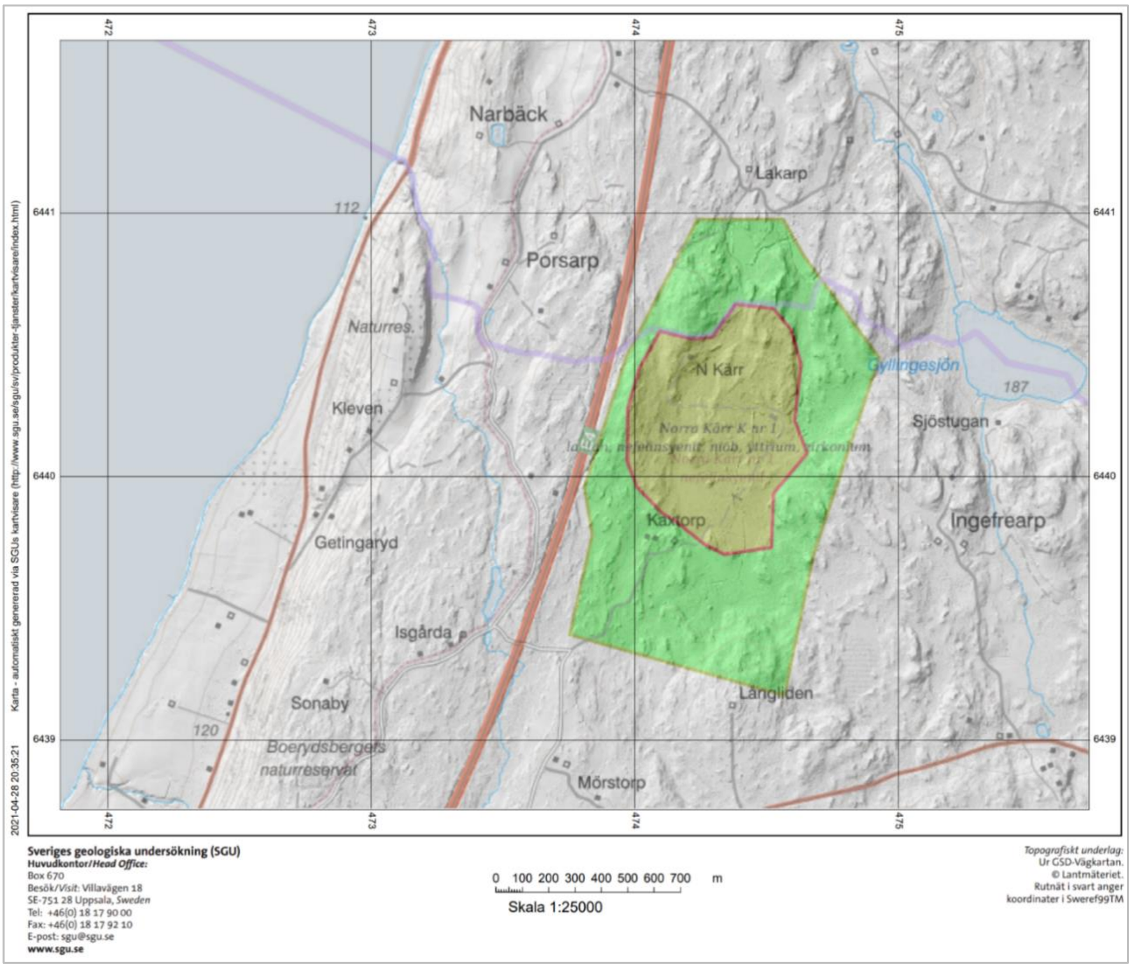


Figure-3-3: Local location of Norra Kärr mining lease application perimeter (yellow), within the exploration permit (green) (Source: SGU map viewer, as of 05/02/2021)

3.3 Legal Survey

The properties were not legally surveyed during this study.

3.4 Location of Mineral Occurrences and Historical Workings

The Norra Kärr intrusive complex has been known for more than a century having been discovered in 1906 by the Swedish geologist Alfred Törnebohm but was not seriously investigated for REE until the Company claimed title to the project in 2009.

Swedish company Boliden has held title to the project in the past and did some preliminary work on the zirconium potential initially in 1949 through some bulk sampling works and again in 1974 for nepheline through the development of two trenches. Boliden did not progress the project as it was reported at the time that the project was of low economic potential due to inefficient beneficiation technologies available at those times.

3.5 Environmental Liabilities and Permitting

There has been no mining activity in the Project area and therefore there are no environmental liabilities associated with the Project. More details on the environmental permitting status are described in Section 18.3.

4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The following chapter was prepared by WAI and GBM in the latest PFS report (Davidson *et al*, 2015b) and has been modified by SRK to ensure it reflects the current status of the Project.

4.1 Accessibility

The Norra Kärr project is readily accessible being within two hours' drive of Gothenburg and three hours from Stockholm. The city of Jönköping is 45 km to the south-west at the southern end of Lake Vattern. The town of Gränna is located on the eastern edge of Lake Vättern approximately 11 km south of the Norra Kärr project and is the closest main urban settlement to the project. From Gränna a secondary road heads north following the lake shore and then easterly under the E4, linking it with a gravel road that accesses the property.

4.2 Climate

The climate is a typical gulf stream modified northern European climate with warm summers and cool winters. The Köppen classification for this area of Sweden is Dfb. The temperature and rainfall regime for the area is provided in Figure 4-1 and Figure 4-2.

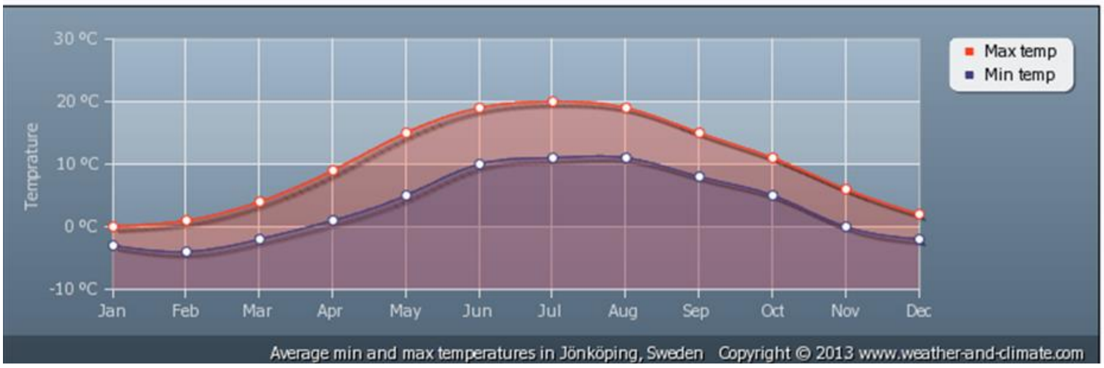


Figure 4-1: Temperature Regime for Jönköping

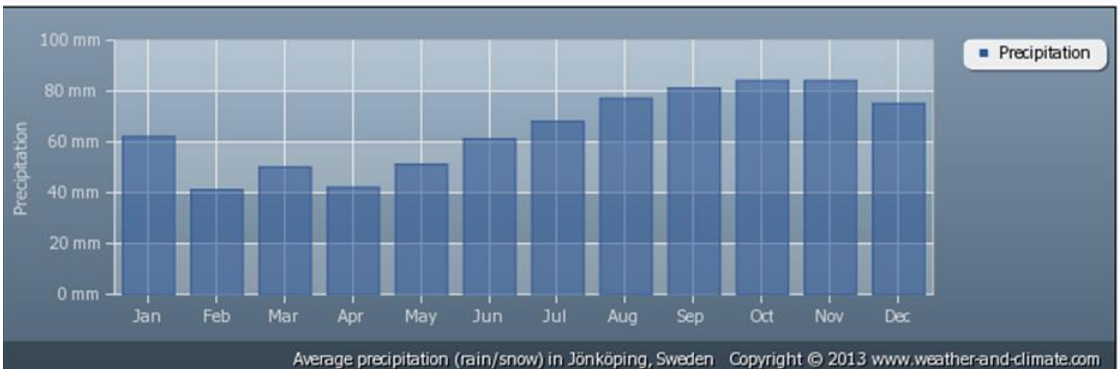


Figure 4-2: Rainfall Regime for Jönköping

Modelling by the Swedish Meteorological Institute suggests that the effects of climate change on rainfall for the project area is very limited. This element will be explored in more detail as part of the planned updates to the ESIA.

4.3 Local Resources & Infrastructure

Jönköping, 45 km to the south of the Project, is a city of some 100,000 inhabitants with all the usual services available while the city of Linköping has 115,000 people and is some 75 km to the north-east along the E4 highway. Gränna, the nearest centre some 11 km to the south-south-west, is a small town with many amenities. The location of the project relative to the E4 highway and the urban conurbations of Gränna, Jönköping, and Jönköping allows for good access to infrastructure, labour, electricity and service providers.

Given its favourable location within a country well known for its mining industry, the infrastructure is excellent.

Sweden has a long history of mining and metal refining stretching back more than a thousand years. Its metal ore and other mineral resources, and knowledge about how to use them, have been major factors in building the prosperity the country enjoys today.

Sweden is currently one of the EU's leading metal producers. It is, for example, by far the biggest producer of iron in the EU, and among the leading ones when it comes to the base metals copper, zinc and lead and the precious metals gold and silver. Sweden also has a number of universities including the Luleå University of Technology ("LTU") and the University of Gothenburg running specific mining related courses providing the industry with well educated professionals.

4.4 Physiography and Vegetation

The terrain encompassing the Norra Kärr property is relatively gentle with an average elevation of about 200 m AMSL. The deposit is approximately one and a half kilometres to the east of Lake Vattern, the general trend of the drainage is to the west towards the lake – although with many local variations. There are numerous rock outcrops which have assisted in the geological mapping of the area and there are many small lakes and ponds scattered throughout the countryside.

Figure 4-3 illustrates the topography on a Project scale showing the boundary of the Norra Kärr exploration permit. Later figures (see Section 6) provide more detail of the geological context and its topographic expression.

Vegetation consists of a mixture of conifers (dominantly spruce) and deciduous types like birch, alder and oak. Modest amounts of shrubby undergrowth occur along with sphagnum moss.

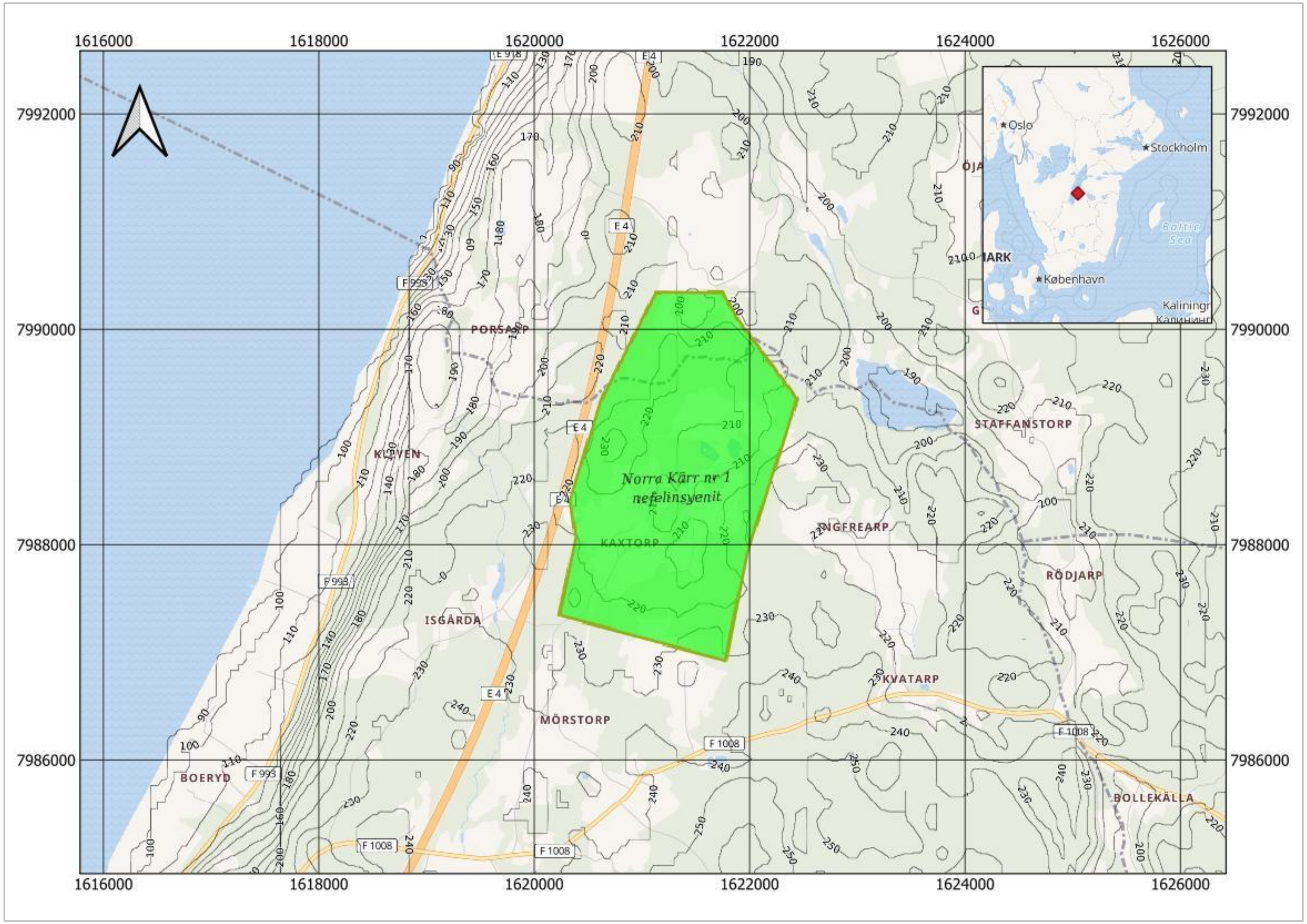


Figure 4-3: Topographic Map of Norra Kärr showing outline of the Norra Kärr exploration permit

5 HISTORY

The following chapter has been taken from the 2015 PFS prepared by WAI and GBM in the PFS report (Davidson *et al*, 2015b) and has been modified by SRK to ensure it reflects the current status of the Project.

5.1 Summary Ownership

There is a single Exploration Permit (Norra Kärr No.1) covering the area of the proposed pit. The Exploration Permit, and additional perimeters now lapsed, were first granted to Tasman Metals AB in 2009. A mining lease was granted in 2013 to Tasman Metals AB. Tasman Metals Ltd., the owner of Tasman Metals AB, was acquired by Flinders Resources in 2016. Flinders Resources changed name to Leading Edge Materials Corp. in 2016. The mining leasewas reverted to application status in 2016 after a ruling by the Supreme Admininstrative Court (see 18.3.1 ????). Tasman Metals ABrecently changed name to GREENA Mineral AB.

5.2 Summary of Regional Exploration and Mining History

Norra Kärr was initially discovered by Dr K.E. Norman in 1906 during SGU mapping in the area and identified the Norra Kärr alkaline igneous complex. Following the geological mapping, mineralogical studies were carried out by Professor Törnebohm on a green fine grained rock obtained from Norra Kärr. The principal constituents of this rock were ascertained to be nepheline with lesser amounts of zirconosilicates, eudialyte, and catapleiite. Professor Törnebohm named the rock type as “catapleiite-syenite”, however, the name was revised by later workers to “grennaite” in reference to the nearby town of Gränna.

In the 1940s extensive scientific investigations of the Norra Kärr complex was undertaken by O.J. Adamsson (1944) with works including detailed petrographical descriptions of the various rock types as well as additional geochemical data.

In the mid to latter parts of the 1940s during and post-World War II the Norra Kärr complex was investigated by Boliden AB. In 1948 agreement regarding mining rights between Boliden and the landowners was secured and in 1949 Boliden commenced some bulk sampling and concentration test work. Boliden reported difficulties in separating the nepheline and feldspar from the pyroxene aegerine resulting in elevated Fe levels in the final concentrate. The works carried out by Boliden ceased as prices for zirconium dropped due to increased global supply from placer and monazite deposits.

Exploration activity at Norra Kärr resumed in 1974 with Boliden carrying out two large E-W trenches (Figure 5-1) with the aim of focussing on the nepheline. The northern most trench was split into 8 geological intervals with between 4 to 30 samples taken in each interval with a total of 151 samples having been taken with a combined total length of 398 m. The southern trench was also split into 8 intervals with the number of samples per interval ranging between 8 and 34 and totalling 169 samples with a combined length of 382 m.

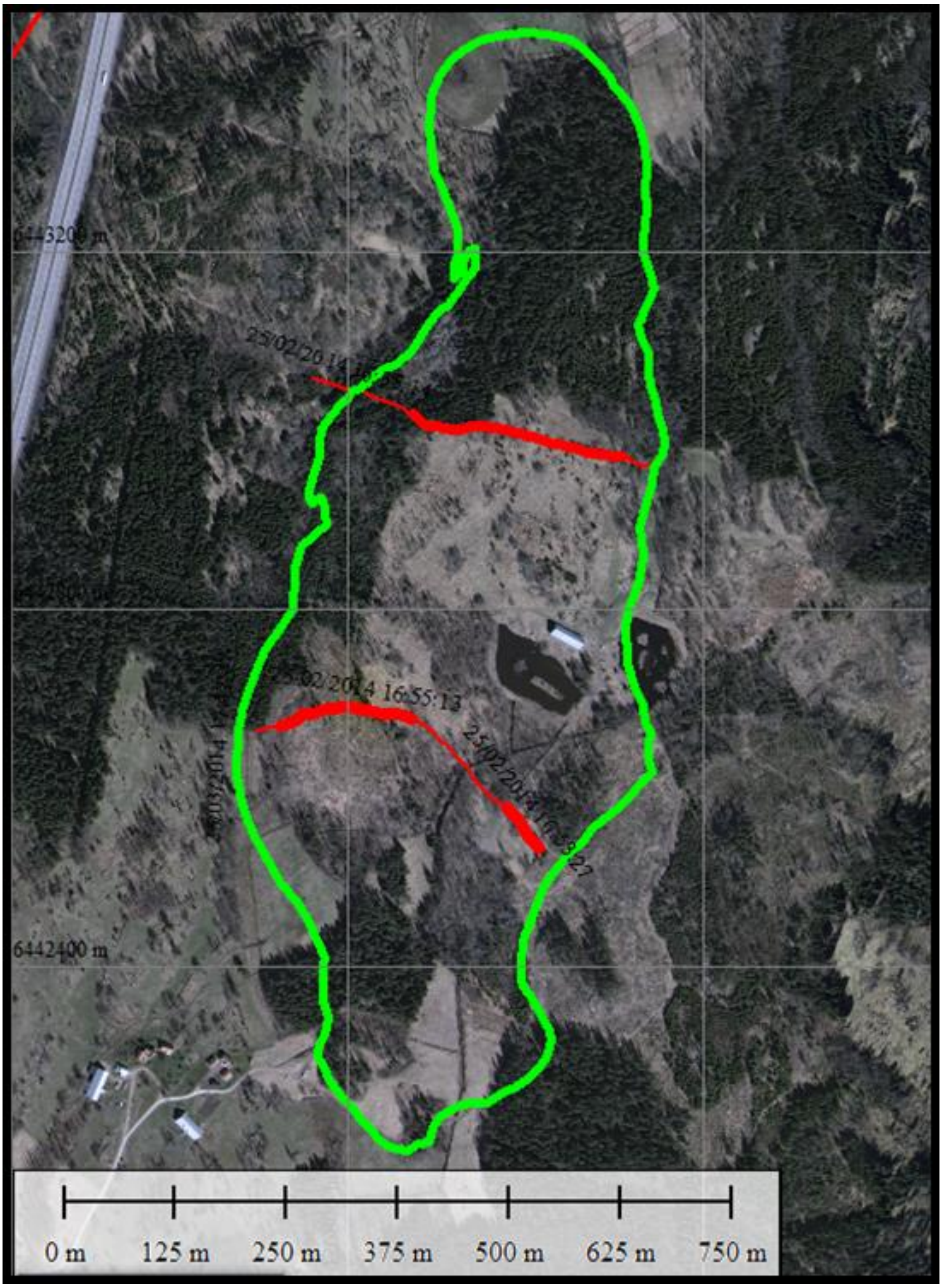


Figure 5-1: Location of the Boliden 1974 Trenches* (Source: Davidson *et al,* 2015a)

**Notes: Green track – outline of Norra Kärr intrusive complex; Red tracks – Boliden (1970's) trenches. [Heavier tracks are the higher grade REE zones.]*

A calculation of the composited Boliden samples (Gates *et al*, 2012) yielded the following weighted averages:

- North Trench: 244 m @ 1.92% zirconium oxide ("ZrO_2"), 0.37% Total Rare Earth Element Oxides ("TREO"); and

- South Trench: 149m @ 1.51% ZrO_2, 0.50% TREO, and 52 m @ 1.47% ZrO_2, 0.44% TREO.

Whilst Boliden reported TREO grades, six of the Heavy Rare Earth Element Oxides (“HREO”) were not assayed. Table 5-1 and Table 5-2 summarise the results of the Boliden trenching testwork.

Table 5-1: Boliden 1974 North Trench Results (Gates *et al*, 2012)

Interval	IB	II	III	IV	V	VI	VII	VIII
No. Samples	4	8	16	30	30	30	28	5
Length (m)	47	22	53	67.5	60.5	60	56	32
ZrO_2 (%)	0.17	0.59	0.48	2.15	2.00	1.94	1.52	0.55
Hf (%)	0.004	0.014	0.010	0.040	0.043	0.049	0.031	0.009
TREO (%)	0.06	0.13	0.16	0.35	0.40	0.45	0.28	0.10

Table 5-2: Boliden 1974 South Trench Results (Gates *et al*, 2012)

Interval	IX	X	XI	XII	XIII	XIV	XV	XVI
No. Samples	11	10	29	28	33	34	8	16
Length (m)	25.0	21.5	63.5	64.0	66.0	90.0	17.0	35.0
ZrO_2 (%)	0.90	1.40	1.66	1.40	0.47	0.35	1.47	1.47
Hf (%)	0.017	0.026	0.028	0.020	0.008	0.006	0.020	0.027
TREO (%)	0.07	0.22	0.42	0.67	0.38	0.24	0.71	0.31

The geology of the area has been subject to several academic studies have been undertaken by various workers (Zetterqvist, 1991; Holtstam, 1998; Strand 1999; Jonsson, 2010; Sjöqvist et al. 2013, 2017; Oberti et al 2015; Atanasova et al 2017; Tranefors, H and Olsson K, 2017; Sjöqvist 2020). These studies have addressed various facets of the Norra Kärr complex including:

- Mineralogy – mainly of the three eudialyte types present;
- Geochronology;
- Fenitisation of the contact zone; and
- Various geological excursion guides.

5.3 Technical Studies

A number of technical reports and studies have been completed on the Project to date:

- First NI 43-101 technical report on the Norra Kärr deposit released in 2009 by LEM was completed by Pacific Geological Services (“PGS”), entitled ‘Report on the Geology, Mineralization and Exploration Potential of the Norra Kärr Zirconium-REE Deposit, Gränna, Sweden’, dated November 2009 (Nebocat, 2009).
- Updated NI 43-101 report including the maiden MRE for the Project was produced in 2011 by Pincock Allen and Holt (“PAH”), entitled ‘NI 43-101 Technical Report, Norra Kärr REE - Zirconium Deposit Gränna, Sweden’, dated January 2011 (Reed, 2011).
- A preliminary economic assessment was produced in 2012 by PAH, entitled ‘Preliminary Economic Assessment NI 43-101 Technical Report for the Norra Kärr REE Zirconium Deposit, Gränna, Sweden’, dated May 2012 (Gates *et al*, 2012).

- A Prefeasibility Study was prepared by GBM Minerals Engineering Consultants Limited ("GBM") in 2015, titled 'Prefeasibility Study - NI 43-101 - Technical report for the Norra Kärr Rare Earth Element Deposit.', dated January 2015 (Davidson *et al*, 2015a).
- A re-stated and amended Prefeasibility Study was prepared by GBM Minerals Engineering Consultants Limited titled 'Amended & Restated Prefeasibility Study - NI 43-101 - Technical report for the Norra Kärr Rare Earth Element Deposit.', dated July 2015 (Davidson *et al*, 2015b).
- A re-stated and amended Prefeasibility Study was prepared by GBM Minerals Engineering Consultants Limited titled 'Amended & Restated Prefeasibility Study - NI 43-101 - Technical report for the Norra Kärr Rare Earth Element Deposit.', dated July 2015 (Davidson *et al*, 2015b).
- The PEA described herein used the results from the latest amended PFS by GBM as the basis for the latest technical information with the exception of work completed on the nepheline syenite conducted directly by Tasman Metals/LEM and the EURARE program at Aachen on improvements to processing.

5.4 Historical Resource Estimates

5.4.1 PAH 2011

The maiden modern Mineral Resource estimate ("MRE") for the Project, valid as of January 2011, was produced by PAH as part of the 2011 NI 43-101 Technical Report (Reed, 2011). The block model and grade estimation were carried out by Mr Geoff Reed, Senior Consulting Geologist, and the Mineral Resource constraining pit optimization was undertaken by Mr Paul Gates, PE and Principal Mining Engineer.

The MRE was based on diamond drilling from the 2009 - 2010 exploration programmes with the drillhole database being complete as of January 2011. A total of 26 drillholes totalling 3,276 m was supplied in the database.

Modelling works were carried out using Gemcom Surpac® software. Geological cross sections were produced along 5 drillhole section lines spaced approximately 200 m apart. At the southern end of the deposit the geological wireframes were extended 100 m to the south whilst at the northern end the geological wireframes were extended 250 m past the last drillhole profile reflect the surface geological interpretation.

The estimation of grades into the Mineral Resource block model was carried out using inverse weighting squared ("IDW^2"). A density of 2.70 t/m^3 was applied to the block model based on 178 density measurements and taking the average.

Classification of the Mineral Resources by PAH was carried out in accordance with the CIM. Only mineralisation falling within the conceptual open pit were classified as Mineral Resources with the classifications based on elevation. All modelled mineralisation from surface classified as Inferred Mineral Resource and reported >0.4% TREO (no pit optimization or depth constraints applied).

A copy of the 2011 Mineral Resource statement is shown below in Table 5-3.

Table 5-3: Norra Kärr Mineral Resource Statement (January 2011)*

Mineral Resource Classification	Tonnage	TREO	HREO /TREO	ZrO_2	HfO_2	Contained TREO
	(Mt)	%	%	%	%	(t)
Measured	-	-	-	-	-	-
Indicated	-	-	-	-	-	-
Meas+Ind	**-**	**-**	**-**	**-**	**-**	**-**
Inferred	60.5	0.54	53.0	1.72	0.034	326,700

**Notes:*

1. *Mineral resources that are not mineral reserves do not have demonstrated economic viability.*
2. *Total Rare Earth Oxides (TREO) includes: La_2O_3, Ce_2O_3, Pr_2O_3, Nd_2O_3, Sm_2O_3, Eu_2O_3, Gd_2O_3, Tb_2O_3, Dy_2O_3, Ho_2O_3, Er_2O_3, Tm_2O_3, Yb_2O_3, Lu_2O_3, Y_2O_3.*
3. *Heave Rare Earth Oxides (HREO) includes: Eu_2O_3, Gd_2O_3, Tb_2O_3, Dy_2O_3, Ho_2O_3, Er_2O_3, Tm_2O_3, Yb_2O_3, Lu_2O_3, Y_2O_3.*
4. *"In-Pit" Mineral Resources were estimated using the Whittle pit optimization software and preliminary economic parameters for commodity prices, metal recoveries and current operating expenses as presented in the PEA.*
5. *Mineral Resources are reported at a marginal cut-off grade of 0.4% TREO.*
6. *HREO is stated as a ratio of HREO/TREO, so 53% represents 53% of the 0.54% TREO.*
7. *Resource estimate assumed mining recovery of 95% and dilution of 5%.*

5.4.2 PAH 2012

A MRE valid as of March 2012 was produced by PAH as part of the 2012 PEA (Gates *et al*, 2012). The block model and grade estimation were carried out by Mr Geoff Reed, Senior Consulting Geologist, and the Mineral Resource constraining pit optimization was undertaken by Mr Paul Gates, PE and Principal Mining Engineer.

The MRE was based on diamond drilling from the 2009 - 2011 exploration programmes with the drillhole database being complete as of 19 August 2011. A total of 49 drillholes totalling 7,375 m was supplied in the database.

Modelling works were carried out using Gemcom Surpac® software. Geological cross sections were produced along 10 drillhole section lines spaced approximately 100 m apart. At the southern end of the deposit the geological wireframes were extended 100 m to the south whilst at the northern end the geological wireframes were extended 250 m past the last drillhole profile reflect the surface geological interpretation.

The estimation of grades into the Mineral Resource block model was carried out using inverse weighting squared ("IDW^2"). A density of 2.70 t/m^3 was applied to the block model based on 579 density measurements and taking the average. For the purpose of reporting Mineral Resources, the block model was constrained by a conceptual Whittle® open pit. The pit optimization is based on a basket price for TREO of USD 51.00 per kg which was discounted by 38% to USD 31.60 per kg TREO to reflect the proposed concentrate to be produced.

Classification of the Mineral Resources by PAH was carried out in accordance with the CIM. Only mineralisation falling within the conceptual open pit were classified as Mineral Resources with the classifications based on elevation. Mineralisation from surface to a depth of 80 m RL was assigned a classification of Indicated Mineral Resource whist any mineralisation between 80 and 0 m RL was classified as Inferred Mineral Resource.

A copy of the 2012 Mineral Resource statement is shown below in Table 5-4.

Table 5-4: Norra Kärr Mineral Resource Statement (March 2012)*

Mineral Resource Classification	Tonnage	TREO	LREO	HREO	HREO /TREO	ZrO_2	Contained TREO
	(Mt)	%	%	%	%	%	(t)
Measured	-	-	-	-	-	-	-
Indicated	41.6	0.57	0.28	0.29	51	1.71	237,120
Meas+Ind	**41.6**	**0.57**	**0.28**	**0.29**	**51**	**1.71**	**237,120**
Inferred	16.5	0.64	0.33	0.31	49	1.70	105,600

**Notes:*

1. *Mineral resources that are not mineral reserves do not have demonstrated economic viability. The Preliminary Economic Assessment includes Inferred mineral resources which are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the results projected in the Preliminary Economic Assessment will be realised and actual results may vary substantially.*
2. *Total Rare Earth Oxides (TREO) includes: La_2O_3, Ce_2O_3, Pr_2O_3, Nd_2O_3, Sm_2O_3, Eu_2O_3, Gd_2O_3, Tb_2O_3, Dy_2O_3, Ho_2O_3, Er_2O_3, Tm_2O_3, Yb_2O_3, Lu_2O_3, Y_2O_3.*
3. *Heave Rare Earth Oxides (HREO) includes: Eu_2O_3, Gd_2O_3, Tb_2O_3, Dy_2O_3, Ho_2O_3, Er_2O_3, Tm_2O_3, Yb_2O_3, Lu_2O_3, Y_2O_3.*
4. *"In-Pit" Mineral Resources were estimated using the Whittle pit optimization software and preliminary economic parameters for commodity prices, metal recoveries and current operating expenses as presented in the PEA.*
5. *Mineral Resources are reported at a marginal cut-off grade of 0.17% TREO.*
6. *HREO is stated as a ratio of HREO/TREO, so 51% represents 51% of the 0.57% TREO (for 0.29% total HREO for Indicated)*
7. *Resource estimate assumed mining recovery of 95% and dilution of 5%.*

5.4.3 GBM 2015

In the Pre-Feasibility Study additional drilling results were undertaken and the resource estimate updated and modified reporting only the REE resource. Wardell Armstrong International (WAI) were provided with a geological database with sample data from surface diamond drilling containing a total of 119 holes (20,420 m) from which 9,986 samples were assayed.

Wireframes to represent the Norra Kärr alkaline igneous intrusive body and associated REE mineralisation were constructed based on the geological logging carried out by Tasman. WAI was provided with geological cross sections and plans by Tasman showing the interpreted structure based on the detailed lithology logs, geochemical assay results and the foliation measurements. In total six key lithologies were modelled:

- (KAX) – Kaxtorpite (Microcline-Pectolite-Amphibole-Aegirine-Nepheline Syenite);
- (GTM) – Grennaite (Recrystallised to in part migmatitic);
- (PGT/GT) – Grennaite with pegmatitic zones and Grennaite (Fine grained aegirine
- rich nepheline syenite);
- (GTC) – Grennaite (Catapleiite porphyritic – low grade TREO);
- (ELAK) – Lakarpite with eudialyte; and
- (MAF) – Mafic dyke material.

Based on the mineralisation wireframes sample data was selected and coded according to the host lithology and reviewed statistically. Very few outlier grades were identified that could present an issue during the variography or grade estimation stages, those that presented a problem were top-cut.

Grade estimation was carried out using Ordinary Kriging (OK) as the interpolation method with Inverse Power of Distance Squared (IDW2) and Nearest Neighbour (NN) also used for comparative purposes for each element. Following the grade estimation process, statistical and visual model validation assessments were undertaken. Globally no indications of significant over or under estimation are apparent in the model nor were any obvious interpolation issues identified. In terms of conformance to the drill hole composite data, WAI considered the OK interpolation method to most closely represent the drill hole data.

The Mineral Resource classification for the Norra Kärr REE deposit is in accordance with the guidelines of the CIM Definition Standards for Mineral Resources and Mineral Reserves [CIM (2010). Criteria for defining Mineral Resource categories were based on geostatistical studies, QA/QC data review and the overall degree of confidence in the geological and grade continuity exhibited at the deposit. WAI classified the Norra Kärr deposit as Indicated.

In order to report a Mineral Resource in accordance with CIM for disclosure in an NI 43-101 report, there needs to be the reasonable prospect for eventual economic extraction. Thus, the resource was constrained by a pit optimisation and only mineralisation falling within the open pit disclosed as a Mineral Resource as shown in Table 5.5. WAI constrained the Mineral Resource reported to the size of a 20-year pit.

Table 5-5: Norra Kärr Mineral Resource Statement (July 2015)

Mineral Resource Classification	Tonnes (Mt)	Density (t/m^3)	TREO (%)	HREO (% of TREO)
Measured	-	-	-	-
Indicated	31	2.7	0.6	52.6
Measured+Indicated	**31**	**2.7**	**0.6**	**52.6**
Inferred	-	-	-	-

**Notes:*

1. *Mineral resources that are not mineral reserves do not have demonstrated economic viability.*
2. *Mineral Resources are reported inclusive of any Mineral Reserves*
3. *Mineral Resources have been constrained on the basis of a 20 year pit*
4. *Mineral Resources are reported for the combined GTM, PGT, GTC and ELAK mineralization only*
5. *Mineral Resources reported represent estimated contrained metal in the ground and has not been adjusted for metallurgical recovery*
6. *Total Rare Earth Oxides (TREO) includes: La_2O_3, Ce_2O_3, Pr_2O_3, Nd_2O_3, Sm_2O_3, Eu_2O_3, Gd_2O_3, Tb_2O_3, Dy_2O_3, Ho_2O_3, Er_2O_3, Tm_2O_3, Yb_2O_3, Lu_2O_3, Y_2O_3.*
7. *Heave Rare Earth Oxides (HREO) includes: Eu_2O_3, Gd_2O_3, Tb_2O_3, Dy_2O_3, Ho_2O_3, Er_2O_3, Tm_2O_3, Yb_2O_3, Lu_2O_3, Y_2O_3.*
8. *Preferred Base Case Mineral Resources are reported at a TREO % cut-off grade of 0.4% TREO*

6 GEOLOGICAL SETTING AND MINERALISATION

The following chapter has been taken from the 2015 PFS prepared by WAI and GBM in the PFS report (Davidson *et al*, 2015b) and has been modified by SRK to ensure it reflects the current status of the Project.

6.1 Regional Geology

While the regional setting within Scandinavia is not a critical factor in the exploration process, it is considered that there are several significant features that are worth considering.

The Sveconorwegian orogen in southern Scandinavia is the result of a collision between Fennoscandia (the southwestern continental segment of Baltica) and another continent in late Mesoproterozoic time. The orogenic province is composed of five distinct Proterozoic gneiss segments that were displaced and reworked during a succession of compressional (and extensional) orogenic phases at between 1.14-0.96 Ga. Figure 6-1 shows the generalised litho-tectonic domains of southern Scandinavia.

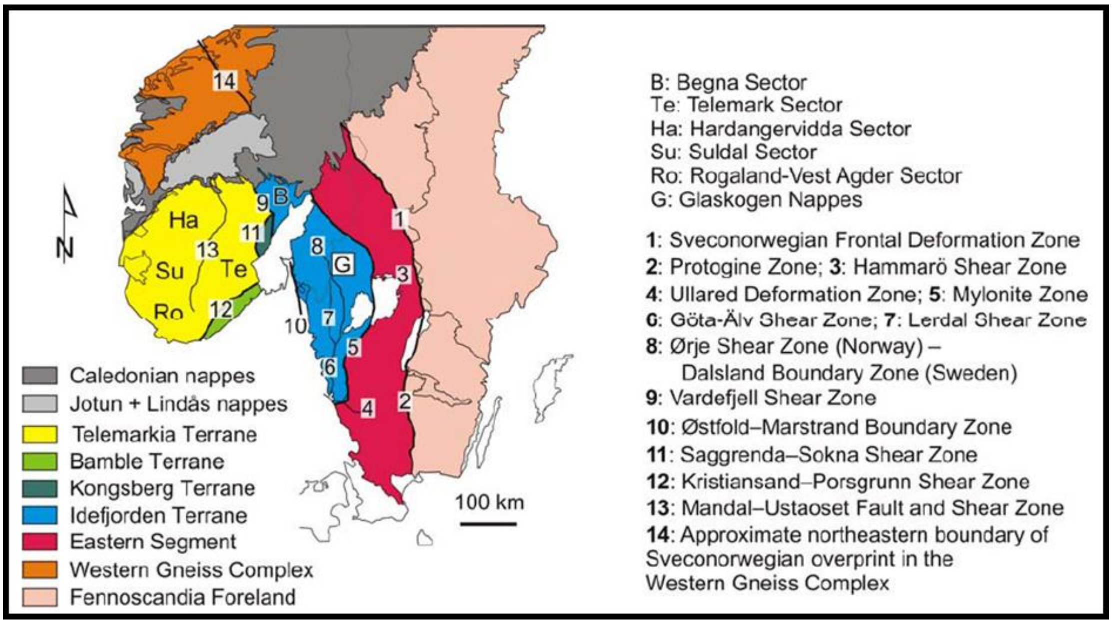


Figure 6-1: Litho-tectonic map of the Sveco-Norwegian orogeny (Source: Davidson *et al,* 2015a)

The Norra Kärr complex has been emplaced in a rift setting (Lake Vattern) within the Trans Scandinavian Igneous Belt ("TIB").

The Trans Scandinavian Igneous Belt comprises a giant elongated array of batholiths extending c. 1400 km across the Scandinavian Peninsula from southeasternmost Sweden to northwestern Norway. The TIB is traditionally divided into four main units based on tectonic and lithological criteria:

- Småland-Varmland belt in the south and west (the host region of the Norra Kärr complex);
- Dala Province.
- Ratan Batholith in the centre; and
- Revsund granitoid suite in the north.

The rocks of TIB dominantly consist of coarse (porphyritic) monzodiorites to granites of alkali-calcic chemistry, with transitions to calc-alkaline or alkaline rocks.

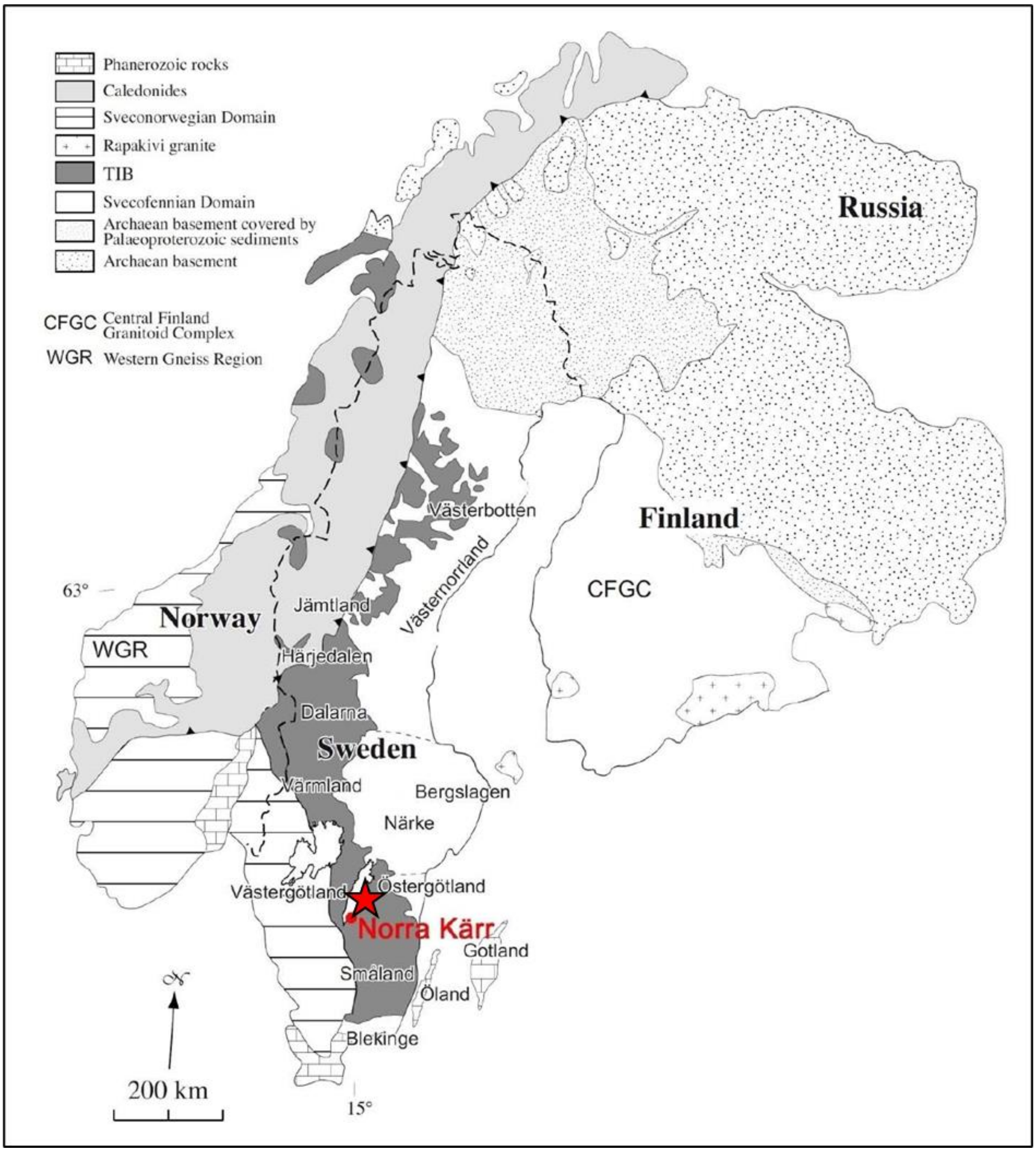


Figure 6-2: Generalised Geological Map of the Fennoscandian Shield Showing the TIB; Norra Kärr indicated by the red star (Source: Högdahl *et al,* 2004)

It is apparent that the positioning of the Norra Kärr complex is at least partially controlled by the shearing separating the Fennoscandia Foreland from the eastern segment of the Proterozoic gneiss sequences to the west. This shearing known variously as the Sveconorwegian Frontal Deformation Zone to the north of Lake Vättern and the Protogine Zone to the south has been intensively studied over the years (Sørensen, 2003; Högdahl *et al,* 2004). This is also well shown in the regional aeromagnetics (see Figure 6-3).

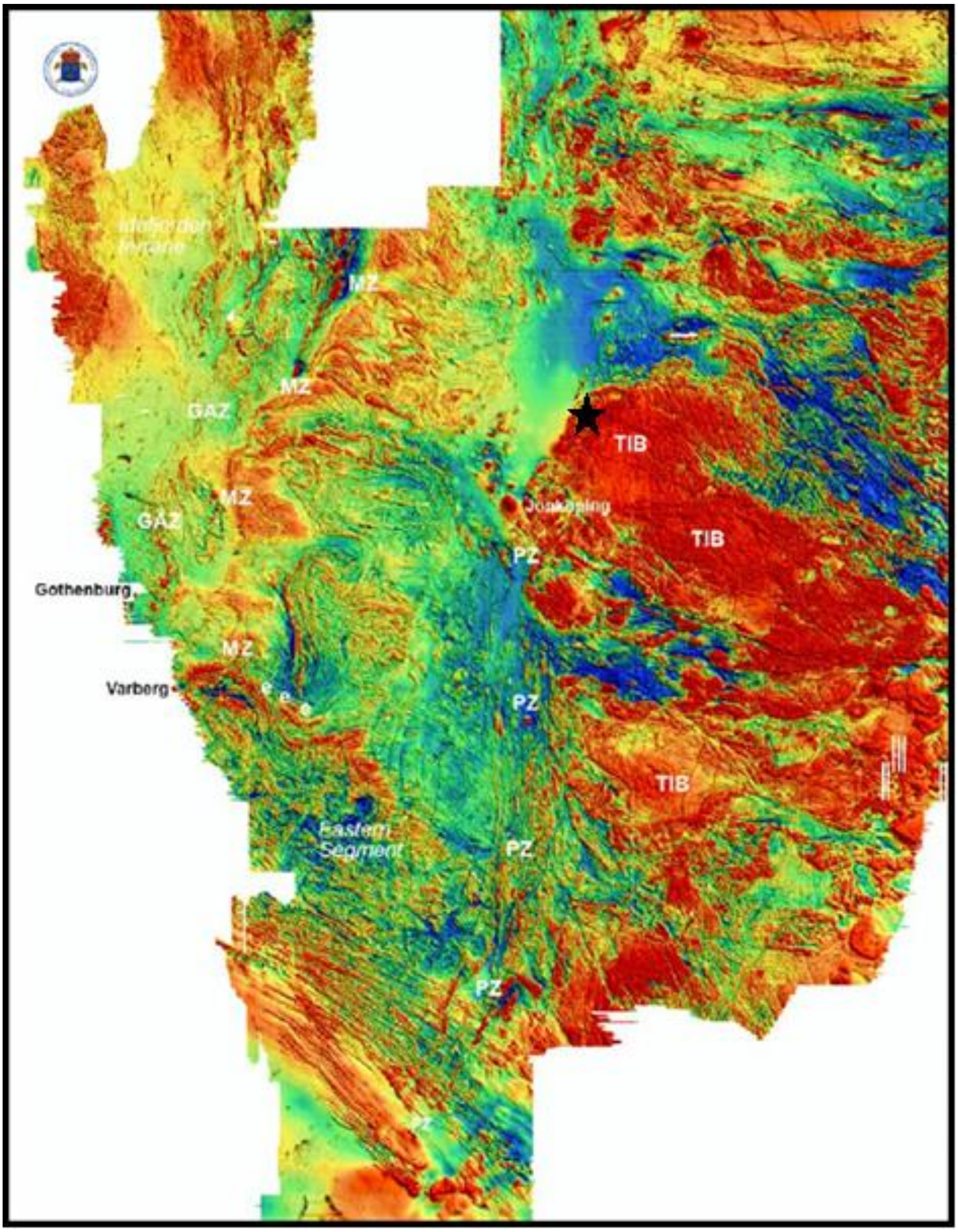


Figure 6-3: Aeromagnetic Anomaly Map of Southern Sweden (Source: Davidson *et al,* 2015a)*

**Notes: Abbreviations: PZ = Protogine Zone, MZ = Mylonite Zone, GAZ = GötaAlv Zone, TIB= Transscandinavian Igneous Belt. Norra Kärr indicated by the black star.*

A further regional-scale factor worth considering is the parallelism of the Lake Vättern rift and the Oslo rift to the eastwest; this is shown in Figure 6-4.

The significance of this is illustrated by reference to the work of Sørensen (2003) in which he notes the occurrences of nepheline syenite in the Gardar and Oslo rift systems in south Greenland and south-east Norway. The comparison with Norra Kärr and the Vattern rift is compelling.

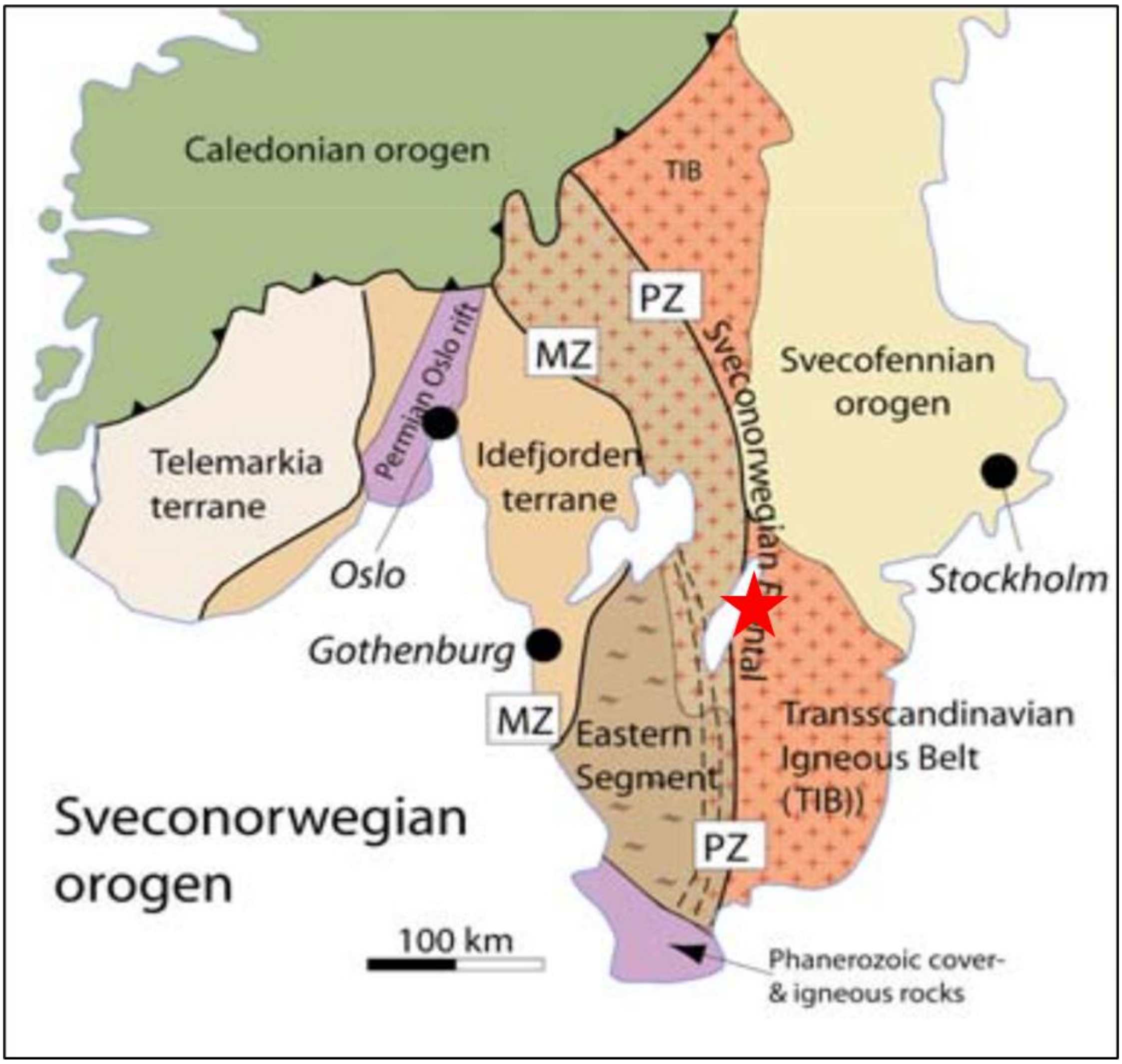


Figure 6-4: Simplified geological map of the Svecconorwegian Orogeny; Norra Kärr indicated by the red star (Source: 33 IGC excursion No 51, 2008)

6.2 District Geology

The Norra Kärr intrusive complex is located close (1.52 km) to the eastern shore of Lake Vattern, a large rift-controlled lake. Following the LEM exploration efforts, the Norra Kärr complex is known to be an ovoid mass with an aerial extent of some 1.3 km by 450 m.

The Norra Kärr complex is intruded into rocks of the Transscandinavian Igneous Belt – specifically the Vaxjö Granite of the Småland-Varmland belt (see Section 6.1 above). This has been dated at ~1,800 Ma while the fenitisation flanking the western side of the Norra Kärr complex has been dated at 1,489 ± 8 Ma (Sjöqvist et al, 2013; Christensen, 2013) thus establishing a reliable age for this intrusive body. The question of “provenance” of the Norra Kärr rocks is still debated.

6.3 Property Geology

6.3.1 Setting

Geologically, the Norra Kärr complex is best defined as a zoned agpaitic, peralkaline, nepheline syenite - similar in many respects to other well-known peralkaline complexes, e.g. Ilimaussaq in Greenland, Dubbo in Australia, Kipawa in Canada and Lovozero on the Kola Peninsula of Russia.

The term “agpaitic” is restricted to peralkaline nepheline syenites containing minerals such as eudialyte and rinkite, that is complex silicates of zirconium, titanium, the rare earth elements, and fluorine and other volatiles.

Agpaitic nepheline syenites per definition do not contain simple zirconium (“Zr”) minerals like zircon and baddeleyite, despite generally high Zr contents (up to 1–2 wt.% ZrO_2). Instead, Zr is mainly hosted in rock-forming complex sodium(“Na”)-calcium (“Ca”)-Zr silicate minerals such as members of the catapleiite, eudialyte, rosenbuschite, and wöhlerite groups.

There has historically been considerable discussion about whether or not the Norra Kärr complex has been deformed and metamorphosed. New Ar-Ar step heating ages on sodic amphibole from Norra Kärr and muscovite and biotite from the country rocks give plateau ages at 1.1 Ga and 0.94 Ga, which correspond to ages derived for Sveconorwegian shear zones in the area (Sjöqvist et al, 2013). Together with textural and crystal chemical evidence, these ages make a compelling argument for some form of Sveconorwegian overprint of the Norra Kärr alkaline complex. This is borne out both by a report on the structural geology of the complex (Rankin, 2011) and by observations by the LEM geologists in the field. Rankin states that:

“Gross magmatic layering and orientation of early deformation fabrics suggest the body was emplaced as a sill (possibly lopolithic) within Palaeoproterozoic granite / gneissic basement of the Fennoscandian Shield.

The intrusive underwent 3 significant phases of deformation:

a) D1 – N-S to NE-SW compression with development of subhorizontal to shallow-dipping foliation (variable intensity). S1 is sub-parallel to So magmatic layering. Folding appears restricted to small-scale intrafolial folds.

b) D2 – E-W compression with development of a regional N-S F2 synform. The synform dips moderately to the west and plunges shallow to moderate to the SW. The Norra Kärr intrusive is preserved in the synformal keel. A variable intensity F2 foliation overprints S1. This is generally a flattening fabric but is locally associated with reverse – thrust mylonite development on the overturned western limb (along the intrusive contact).

c) D3 – N-S compression with development of E-W and minor conjugate NE- / NW- trending kink folds across the intrusive. The kink folding produces localised complexities in the orientation of the dominant S0 / S1 fabric.”

6.3.2 Lithologies

The basic geology of Norra Kärr is relatively straightforward within the undoubted complexities of the alkaline intrusive environment as noted above.

The size and shape of the Norra Kärr intrusive complex is now well established from the exploratory drilling carried out by LEM in their evaluation of the body. The intrusive is elongated in a NNE-SSW direction and is approximately 1,300 m long by 450 m wide as is been illustrated in several of the figures.

As noted, it is intruded into rocks of the Transscandinavian Igneous Belt – specifically the Vaxjö Granite of the Småland-Varmland belt. There is evidence that the Norra Kärr intrusive suffered some deformation during the Sveconorwegian shearing episode(s) as there is evidence of cataclasis on the western side of the intrusive. This is backed up by an interpretation of the (recently obtained) ground magnetic data which suggests that some rotation, suggestive of a dextral shearing movement, has affected the intrusive body. It is possible that it acted as a "kernel" and was rotated clockwise slightly as a result of the shearing couple oriented NE-SW evident from the ground magnetic data (see Figure 6-5).

This interpretation would go some way towards explain the structural complexities evident in the southern portion of the intrusive body where the faulting has been observed and interpreted. The whole intrusive body dips to the west at angles between 35° and 40°.

Lithologically, the Norra Kärr intrusive demonstrates the same complexity of rock types as other similar agpaitic nepheline syenite intrusives, e.g. Ilimaussaq, Lovozero, with a number of more or less unique lithologies being present. Due to the age of some of the work - the workers were without other similar bodies to use as a comparison – "local" rock names were applied. They are mostly variations on a theme so the lithological complexity originally – and understandably – interpreted by the LEM geologists, while somewhat confusing at first sight, is actually relatively easy to understand. More recently, the LEM geologists have simplified the "stratigraphy" and WAI used this simpler approach in the MRE (see Section11). However, the full lithological table as compiled by the LEM geological team is given in Figure 6-7 in order to illustrate both the complexity and the well-thought out approach by LEM.

The Norra Kärr intrusive exhibits a clear "layering". It is not clear if this is primary igneous layering or not and, in this respect, there again similarities with the type localities of Ilimaussaq and Lovozero. By analogy, it is probable that there is at least a degree of igneous differentiation but the mechanics of this are not yet understood. Notably, if the original intrusion had a lopolithic shape then the later deformation has played a significant role. Some 75% of the intrusive consists of varieties of "grennaite" an aegirine-rich nepheline syenite carrying the rare, zircono-silicate minerals, eudialyte and catapleiite. In some respects, they resemble the lujavrites present at other agpaitic syenites, e.g. Ilimaussaq and Lovozero.

Regardless of the mechanism, the Norra Kärr intrusive exhibits a roughly concentric layering (Figure 6-6 with the detailed legend shown in Figure 6-7) with the following pattern being evident. From the centre to the flanks, the main zonal lithotypes are:

- A central (but off-set) core of kaxtorpite ("KAX"; microcline-pectolite-amphibole-aegerine-nepheline syenite) surrounded by:
- A migmatitic grennaite zone ("GTM") and;
- A "pegmatitic" grennatite ("GPG" or "PGT") ± lakarpite (LAK);
- Fine-grained grennaite containing catapleiite ("GTC"); and
- Various other relatively minor alkaline rocks and mafic intrusives.

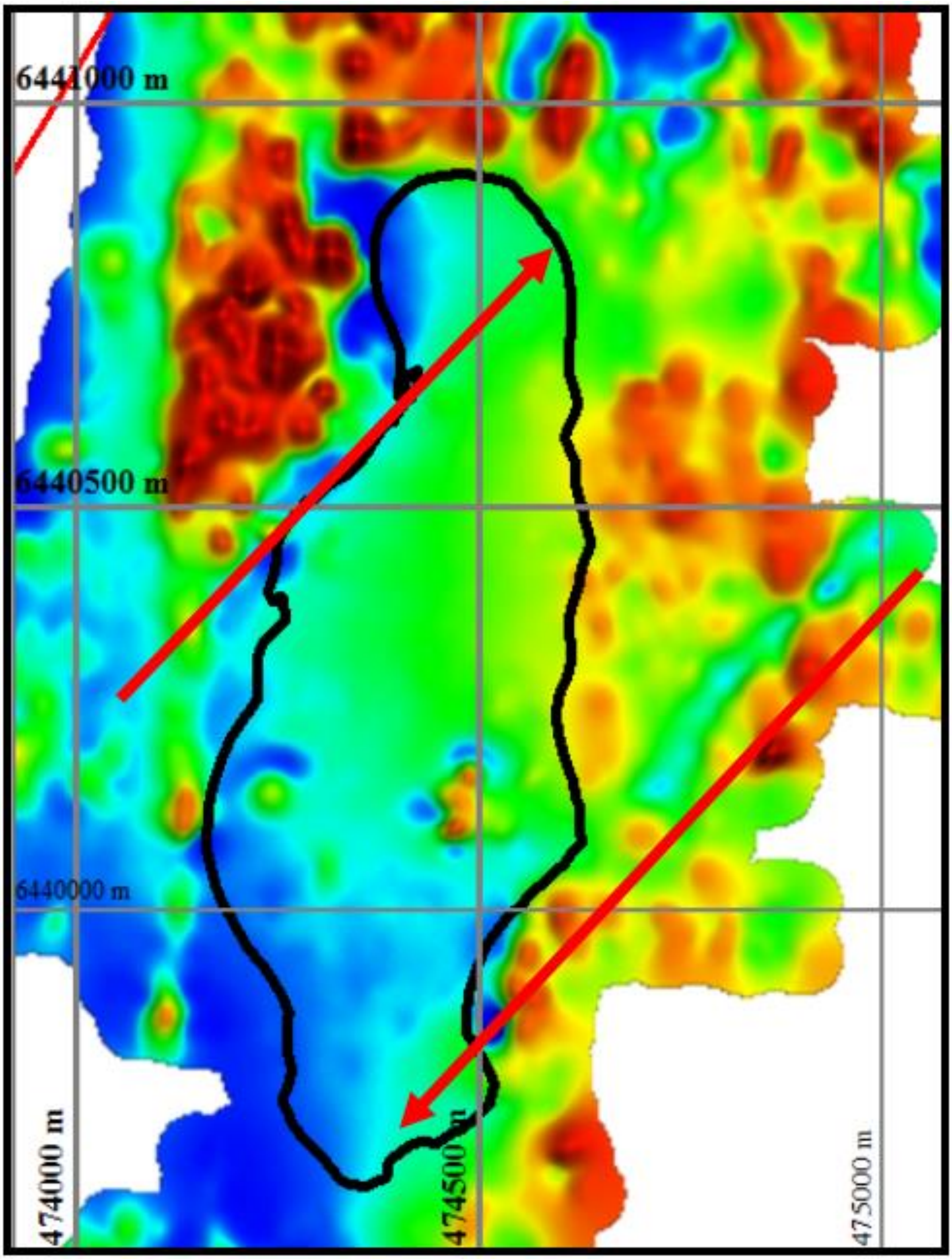


Figure 6-5: Possible Shearing Movement for the Norra Kärr Intrusive (Source: Davidson *et al,* 2015a)

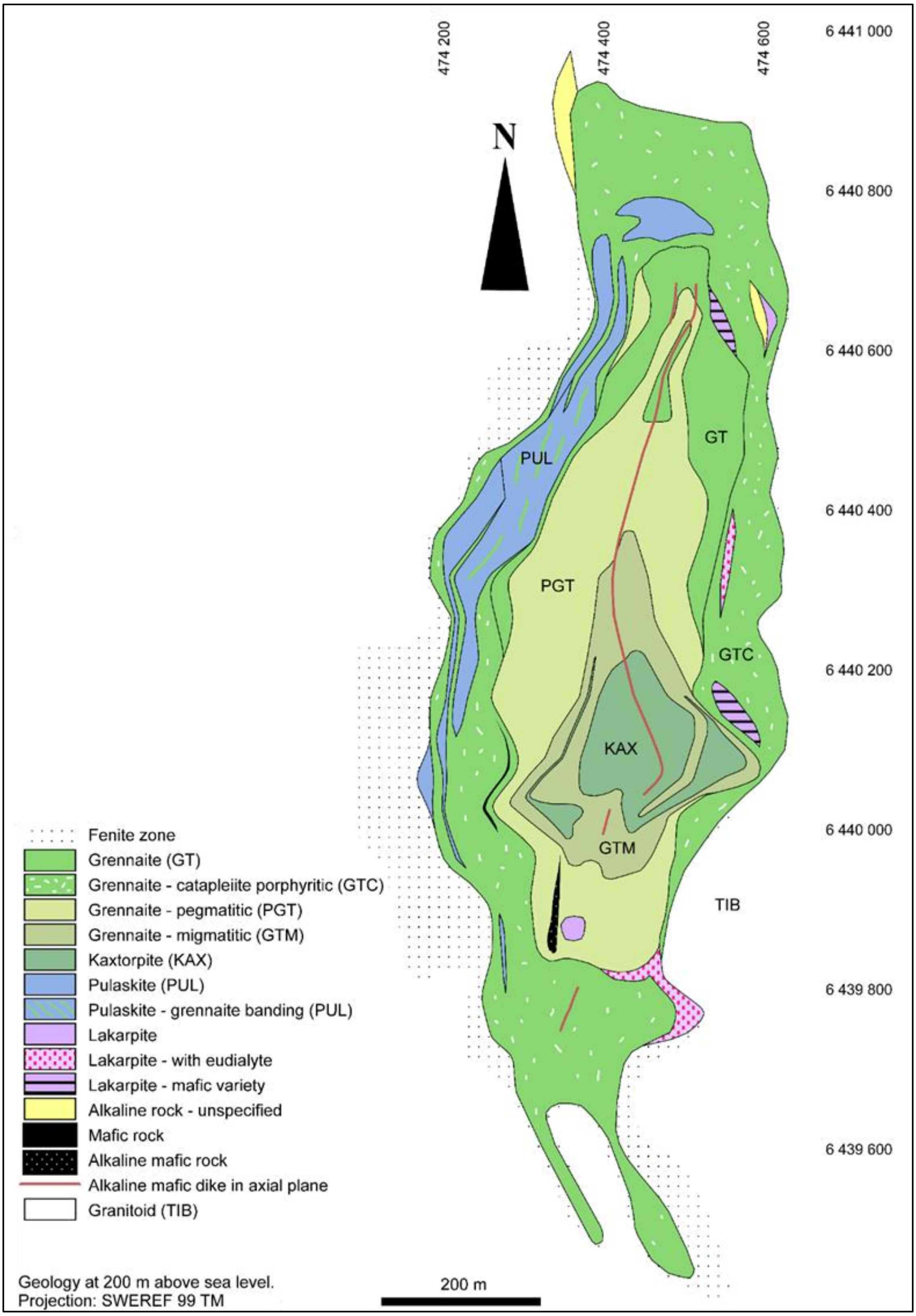


Figure 6-6: Geological Map of the Norra Kärr Intrusive (Source: LEM)

A brief description of the various rock types (and their sub-types) is given in Table 6-1.

Table 6-1: Norra Kärr Intrusive Body Lithology Types (Red outline = principal mineralisation zones that are included in the mineral resource)

Lithology Zone	Logging Code	Short Description	Average ZrO2%	Average TREO%	% of total samples in database
GTC	GT	Grennaite. Fine grained with no or low amount of larger Catapleiite grains and with less than 5% pegmatitoidal schlieren. Very fine grained to fine-grained ground-mass	1,37	0,31	10,8
	GTC	Grennaite with mineralized rock than 3% larger Catapleiite laths/needles. Schistose. Very fine grained ground-mass.	1,43	0,27	20,0
	GTCE	Grennaite with both Catapleiite and Eudialyte porph.	1,14	0,17	0,8
GPG	GT1	Grennaite with 5-10% pegmatitoidal schlieren or zones.	1,68	0,49	7,6
	GT2	Grennaite with 10-30% pegmatitoidal schlieren or zones.	1,87	0,58	8,0
	GT3	Grennaite with 30-50% pegmatitoidal schlieren and/or zones.	2,10	0,60	6,2
	GTP	Grennaite with 50-70% coarser, pegmatitoidal zones and schlieren.	2,14	0,66	3,4
	PGT	70-90% Pegmatitic Grennaite with 10-30% fine-grained Grennaite zone/slabs.	2,30	0,69	3,1
	NEF	Nepheline-syenite (Grennaitic) pegmatite >90%	2,15	0,67	5,6
	GTR	Evenly medium grained "Grennaite". Only in part coarser pegmatitoidal.	2,01	0,64	2,5
GTM	GTM	"Migmatitic", possibly cooked/re-crystallized Grennaite.	1,42	0,48	6,0
	GTMi	Slightly cooked, re-crystallized Grennaite. In part showing crenulated folding.	1,52	0,51	3,2
PUL	PUL	Pulaskite, Coarse-medium grained. Microline augen in Alb-Aegirine-Amph ground-mass. Minor Nepheline-Biotite.	0,35	0,14	3,2
	PULF	Pulaskite? or possibly strongly fenitizied granitoide.	0,70	0,25	0,5
	PULG	Pulaskite with Grennaite zones/bands.	0,61	0,19	1,2
KAX	KAX	Kaxtorpite. Microcline-Eckemannite-Aegirine+-Nepheline-Pectolite-Natrolite. Often strongly folded (in part isoclina).	0,22	0,14	2,2
	KAG	Kaxtorpite with some Grennaite bands. Often intensely folded.	0,40	0,22	2,8
	GTK	Grennaite with kaxtorpite bands.	0,77	0,30	1,4
MAF	MAF	Mafic dike. 10-100 cm wide, very fine grained, sometimes Amph porphyritic.	0,12	0,10	1,3
	MAA	Mafic, probably alkaline, fine to medium grained, dark, amphibole-rich, intrusive rock.	0,23	0,18	1,2
	MHYB	Mafic rock. Infiltrated by/brecciated by alkaline (fsp-eudialyte) veining.	1,51	0,66	1,0
	LAK	Lakarpite. Albite-Afredsonite-Nepheline dominated medium grained.	0,54	0,27	0,2
	AUN	Alkaline unspecified rock. Often pale, Fsp dominated.	0,75	0,25	3,7
	FEN	Fenite. Strongly bleached. Albite rich, very fine grained.	0,70	0,17	0,5
	MYL	Mylonite.	0,14	0,08	0,4
	PEG	Granite pegmatite.	0,34	0,30	0,2
GR	GR	Granitoide. In general very coarse grained. Often "fenitizied".	0,12	0,07	2,8

The key points on the distribution within the Norra Kärr complex of the main lithotypes shown above and on the geological map (Figure 9.2) are the lithologies are described below:

- **Kaxtorpite (KAX)** - the main (200 by 110 m) body of the kaxtorpite is in the (off-set) central core of the intrusive. It is a zirconium poor, coarse grained, often foliated, sheared, mafic alkaline rock commonly with larger microcline augen in a groundmass of dark alkali-amphibole, aegirine, pectolite and nepheline. It is relatively barren of REE mineralisation in comparison with the grennaite varieties discussed below.

Figure 6-7: KAX (KAG) - Dark Kaxtorpite Intensely Folded with Thin Bands of Green, Fine-grained Grennaitic Material

- **Migmatitic grennaite zone (GTM)** – surrounds the kaxtorpite and as the name suggests has a somewhat migmatitic texture with some kink folding evident – possibly due to the structural features noted above. This zone is well mineralised – probably due to the presence of pegmatitic schlieren hosting eudialyte. The contact with the enclosing pegmatitic grennaite is not sharp and interpretation of this contact depends to an extent on the percentage pegmatitic schlieren.

Figure 6-8: GTM - "Migmatitic" Grennaite Medium Grained

- **Pegmatitic grennaite zone (PGT or GPG)** - while not pegmatite, there are visual elements that give this impression. Notably the quantity and appearance of the eudialyte-bearing schlieren allow for this descriptive term to be used. The PGT zone is somewhat inhomogeneous and the various subdivision (GT1, GT2, etc.) are quite justified. However, it is also fully justified to take the PGT zone as a single unit based on the criteria utilised by the LEM technical team. The transition between the GTM and PGT domain is gradational. The eastern contact between the PGT and GTC domain is however often quite sharp.

Figure 6-9: PGT - Pegmatitic Schlieren/Veining in Grennaite, with Elongated Crystals of Catapleiite

The pegmatitic schlieren and zones generally contain the same main minerals as the fine grained grennaite but in different proportions. There is also a large variation in mineral composition and grain size between different pegmatitic schlieren and zones within the complex. Most commonly, the zones are dominated by microcline-albite and nepheline though darker aegirine rich varieties locally have been observed.

The distribution of eudialyte and catapleite are variable but geochemical analyses as well as visual observations suggest that both minerals are more abundant in the pegmatitic facies than elsewhere. The amount of eudialyte is seldom greater than ten volume percent though short intervals can be richer. Eudialyte occurs as rounded to subhedral up to a 2cm size grains are quite variable in colour from relatively dark brown-red to over clear red to pale pink. In places the eudialyte is pink reddish, semi-translucent in bands, veins or patches.

- **Catapleiite Porphyritic Grennaite (GTC)** – the catapleiite porphyritic variety of the grennaite occupies a large volume surrounding the PGT Zone. The colour varies between greyish green to light green or medium grey. The lighter variations are often found towards the granite contacts. The green colour arises from the presence of the sodium rich pyroxene aegerine in the groundmass.

Figure 6-10: GTC – Fine-Grained Grennaite with Bluish-White Elongated Catapleiite Grains

While all the major rock units at Norra Kärr are REE-mineralised to a greater or lesser degree, it is clear from the work that LEM and predecessors have undertaken that the pegmatitic and migmatitic varieties of the grennaite are the main targets.

6.3.3 Mineralisation

The mineralisation at Norra Kärr is at once both simple and complex. Simple in mineralogical terms in that nearly all the REE mineralisation is hosted in the complex zircono-silicate mineral eudialyte. There are minor amounts of the (probably) secondary Ca-LREE-F-silicate (britholite) and trace mosandrite. The eudialyte at Norra Kärr is, relatively rich in REE compared to most other similar deposits and also contains a very high proportion of HREO.

Various studies (e.g. Sjöqvist *et al*, 2013), have shown that within the resource, the TREO content of the eudialyte varies between 6-10% (Figure 6-12). The percentage of HREO also varies from about 30% in the more central GTM zone to above 70% in more distal GTC zone. The total content and also distribution of the REEs within the eudialyte also varies throughout the deposit. Sjöqvist *et al* defined three compositional varieties of eudialyte group minerals at Norra Kärr:

- Fe-rich, REE-poor, classical pink eudialyte;
- Fe-Mn-bisected, HREE-rich eudialyte from "pegmatitic" grennaite; and
- Mn-rich, LREE-rich eudialyte from "migmatitic" grennaite.

This study supports the field and analytical evidence that there are varying types of eudialyte which can, to an extent, be differentiated by colour with the more vibrant pink variety being poor in REOs and the duller, brownish-red varieties containing higher values of REOs.

The overall distribution of the REOs within the deposit is illustrated in Figure 6-12, Figure 6-13, and Figure 6-14 below.

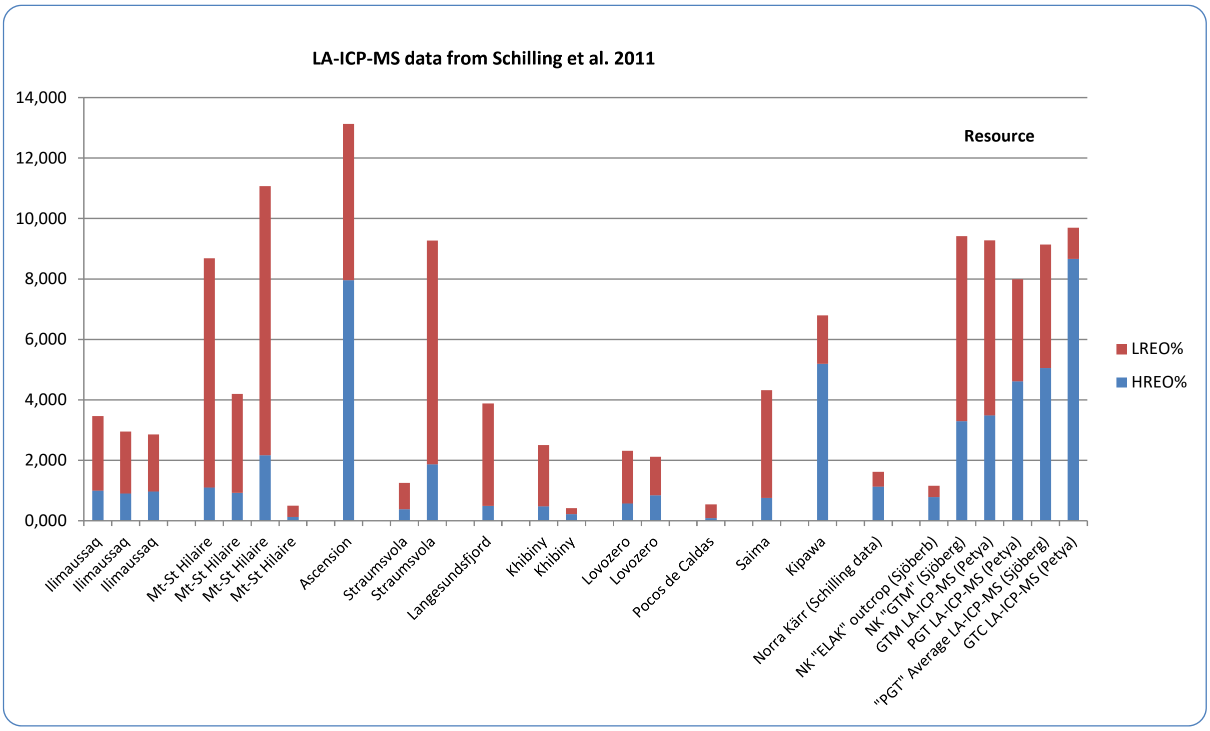


Figure 6-11: Laser ablation ICP-MS analyses of eudialyte from various locations worldwide (Source: LEM)

Equally, it may be shown that ~60% of the zirconium is hosted by eudialyte with the remaining 40% hosted by the catapleiite ($Ca/Na_2ZrSi_3O_9 \cdot 2H_2O$). Consequently, whilst the GTC domain contains LREO and HREO and in particular zirconium, the greater concentrations of LREO and HREO found in the GTM and PGT domains, coupled with the improved liberation characteristics of these domains make the PGT and GTM the main economic targets at Norra Kärr.

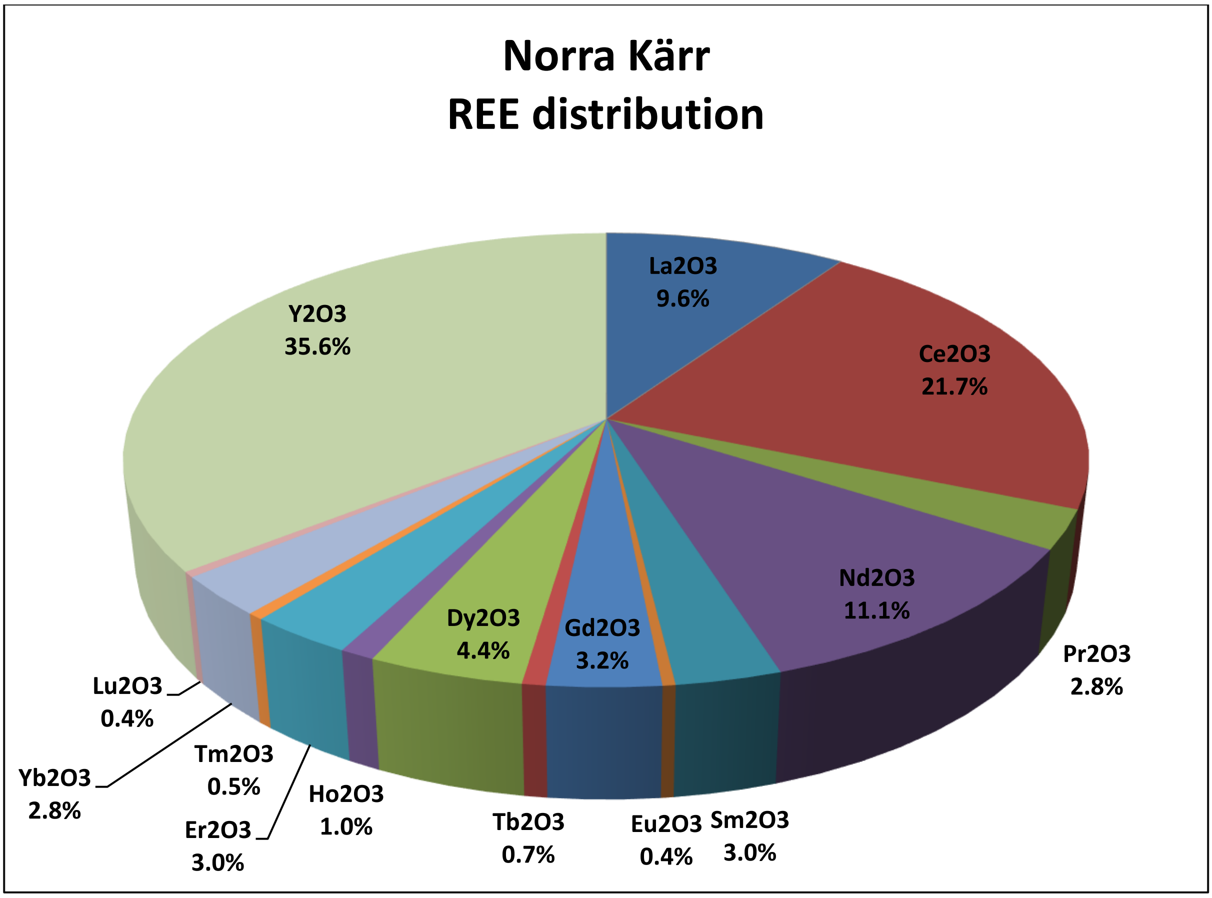


Figure 6-12: Norra Kärr REE distribution (Source: LEM)

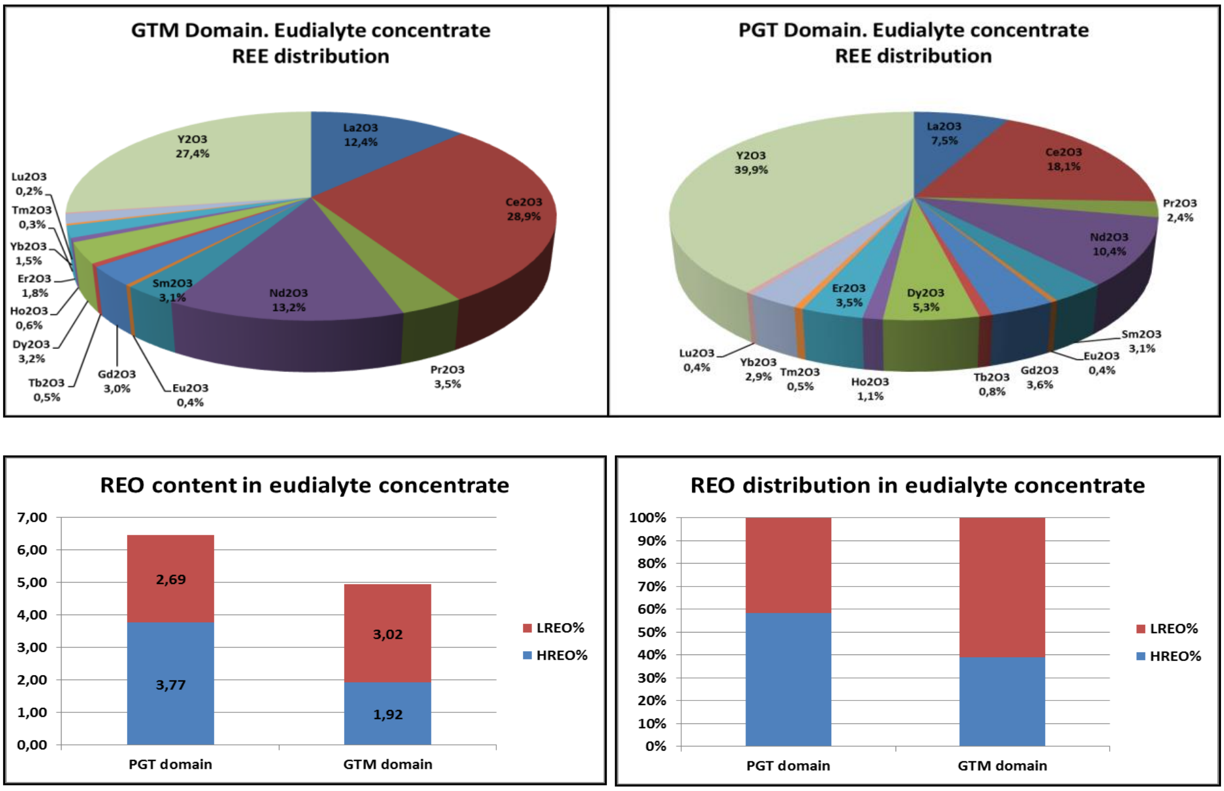


Figure 6-13: Norra Kärr REO distribution by major mineralisation types (Source: LEM)

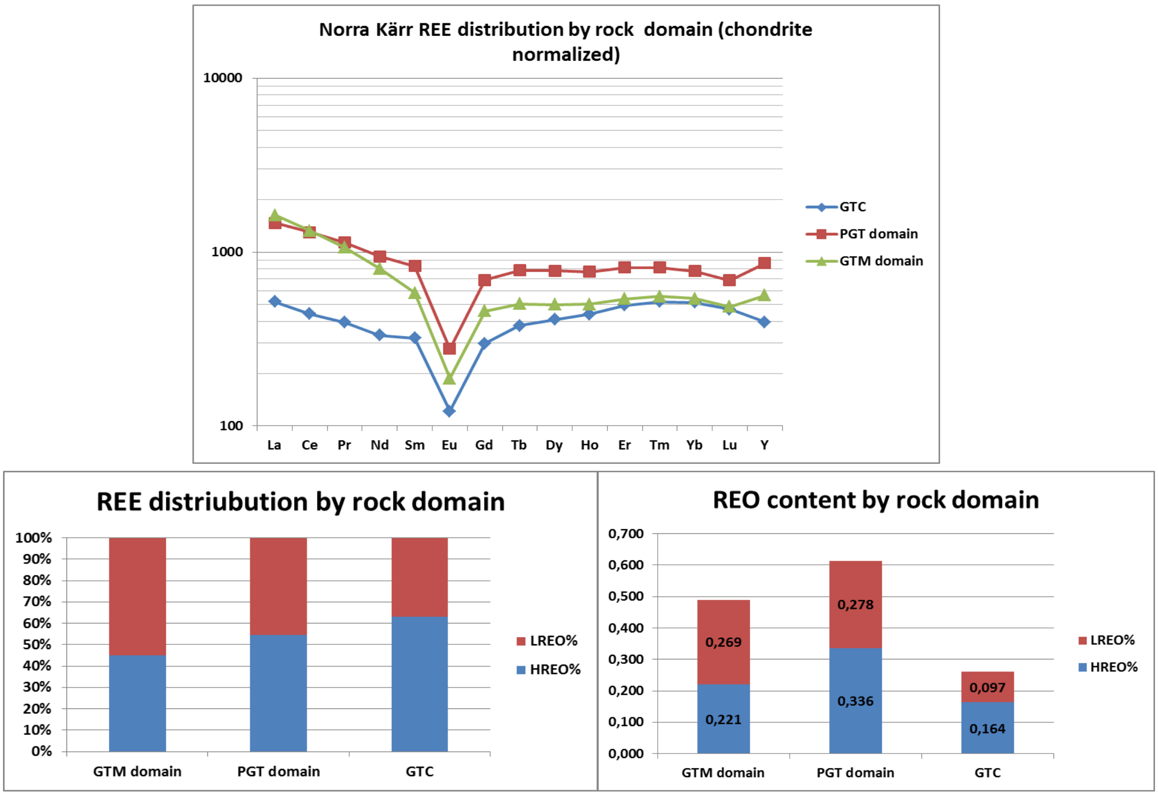


Figure 6-14: Norra Kärr REO distribution by rock domain (Chondrite Normalised) (Source: LEM)

7 DEPOSIT TYPES

The following chapter has been taken from the 2015 PFS prepared by WAI and GBM in the PFS report (Davidson *et al*, 2015b) and has been modified by SRK to ensure it reflects the current status of the Project.

7.1 General

There are several different types of REE mineralisation that are of varying importance due to a number of factors – including some which are not geological. A key factor affecting the importance of REE is the strategic nature of the minerals and the current dominance in supply from Chinese REE.

Of the “primary” deposits, i.e. not including the highly important ion adsorption clay deposits of China, the British Geological Survey (“BGS”) has produced the following classification:

Table 7-1: Primary REE Deposit Types - Norra Kärr-type outlined in red (Source: BGS, 2011)

Deposit type	Brief description	*Number documented	Typical grades and tonnage	Major examples
Primary deposits				
Carbonatite-associated	Deposits associated with carbonate-rich igneous rocks associated with alkaline igneous provinces and zones of major faulting	107	A few 10s thousands of tonnes to several hundred million tonnes, 0.1-10% REO e.g. Bayan Obo: 750 million tonnes at 4.1% REO	Mountain Pass, USA; Bayan Obo, China; Okorusu, Namibia; Amba Dongar, India; Barra do Itapirapuã, Brazil; Iron Hill, USA
Associated with alkaline igneous rocks	Deposits associated with igneous rocks characterised by abundant alkali minerals and enrichment in HFSE	122	Typically <100 million tonnes (Lovozero >1000 million tonnes), grade variable, typically <5% REO e.g. Thor Lake: 64.2 million tonnes at 1.96% REO	Ilimaussaq, Greenland; Khibina and Lovozero, Russia; Thor Lake and Strange Lake, Canada; Weishan, China; Brockman, Australia; Pajarito Mountain, USA
Iron-REE deposits (iron oxide-copper-gold deposits)	Copper-gold deposits rich in iron oxide and diverse in character and form	4	e.g. Olympic Dam: 200 million tonnes at 0.3295% REO (Orris and Grauch, 2002)	Olympic Dam, Australia; Pea Ridge, USA
Hydrothermal deposits (unrelated to alkaline igneous rocks)	Typically quartz, fluorite, polymetallic veins and pegmatites of diverse origin	63(a)	Typically <1 million tonnes, rarely up to 50 million tonnes, grade variable, typically 0.5-4.0%, rarely up 12% REO e.g. Lemhi Pass: 39 million tonnes at 0.51% REO (Orris and Grauch 2002)	Karonge, Burundi; Naboomspruit and Steenkampskraal, South Africa; Lemhi Pass and Snowbird and Bear Lodge, USA; Hoidas Lake, Canada

The other critical factor in assessing REE mineralisation is the “split” between the light rare earth oxides (LREO) and the heavy rare earth oxides (HREO) within the overall grade of the total rare earth oxides (TREO). This is critical as many of the HREO are considerably scarcer and attract higher prices than the LREO. The split of LREO/HREO is generally taken as elements with atomic numbers 63-71 and includes yttrium that whilst not part of the lanthanide group from which the other 14 REE metals occur - are included in the HREO category as they are often found together in mineralized rock deposits, Norra Kärr is an example of this although there is no anomalous Sc.

The lighter REE are more incompatible – due to their larger ionic radius – and are consequently more strongly concentrated in the continental crust than the HREE. They are thus more commonly available, e.g. in monazite, a common mineral in heavy mineral sand deposits. However, monazite is also heavily enriched in thorium a (radioactive) deleterious component.

In terms of defining rare-earth minerals such as eudialyte they can be described as contain one or more rare-earth elements as major metal constituents. Rare-earth minerals are usually found in association with alkaline to peralkaline igneous complexes, in pegmatites associated with alkaline magmas and in or associated with carbonatite intrusives. Perovskite mineral phases are common hosts to rare-earth elements within the alkaline complexes. Mantle-derived carbonate melts are also carriers of the rare earths. Hydrothermal deposits associated with alkaline magmatism contain a variety of rare-earth minerals (Jones et al 1996).

However, it is important to note that the mineralisation in the agpaitic syenite complexes are often dominated by eudialyte group minerals which are particularly favourable for their HREO content. This is particularly the case at Norra Kärr.

Another important feature is the "criticality" matrix. This is based on the currently perceived availability both now and in the foreseeable future as shown in Figure 7-1.

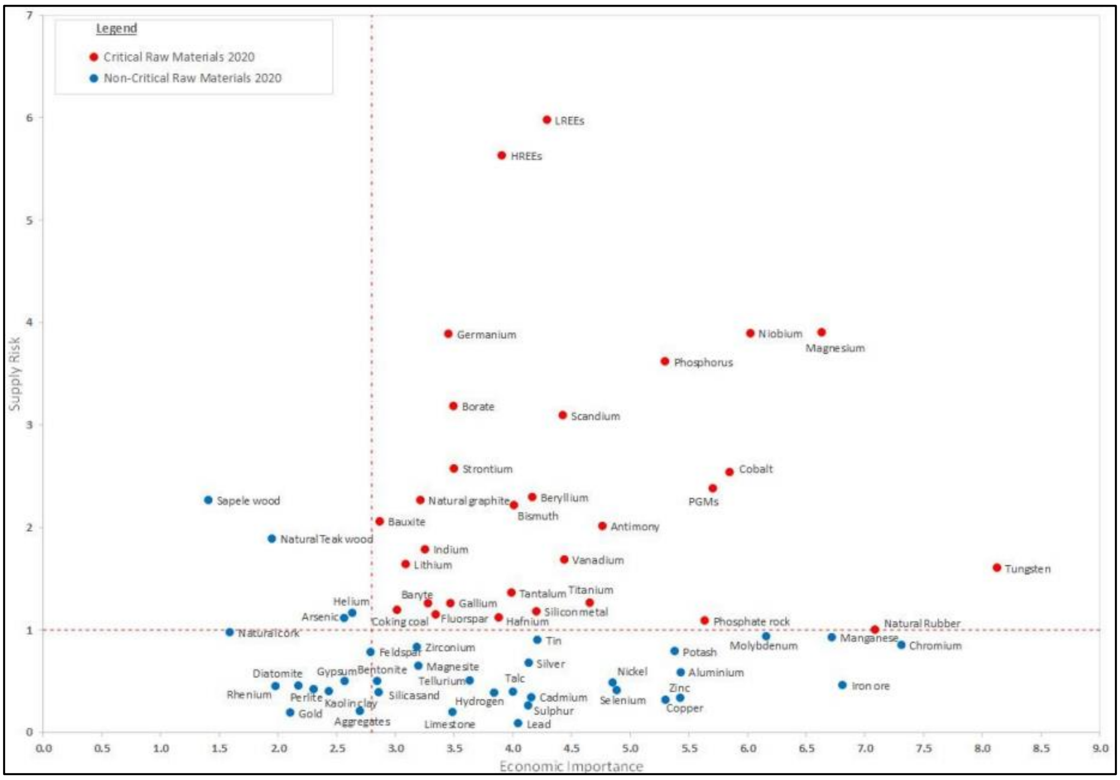


Figure 7-1: REE Criticality Matrices (Source: European Commission, 2020)

There are a very large number of REE minerals which further complicate the exploration/evaluation process when considering the economics of REE deposits. A selection of "common" REE-bearing minerals is shown in Table 7-2.

Table 7-2: REE-bearing minerals; main REE-mineral at Norra Kärr outlined in red (Source: BGS, 2011)

Minerals	Formula	Approximate REO%
Aeschynite-(Ce)	$(Ce,Ca,Fe,Th)(Ti,Nb)_2(O,OH)_6$	32
Allanite-(Ce)	$(Ce,Ca,Y,La)_2(Al,Fe^{+3})_3(SiO_4)_3(OH)$	38
Apatite	$Ca_5(PO4)_3(OH,F,Cl)$	19
Bastnãsite-(Ce)	$(La, Ce, Y) CO_3F$	75
Brannerite	$(U,Ca,Y,Ce)(Ti,Fe)_2O_6$	9
Britholite-(Ce)	(Ce,Ca)5(SiO4)3OH,F	32
Eudialyte	$Na_{15}Ca_6(Fe,Mn)_3Zr_3SiO(O,OH,H_2O)_3$ $(Si_3O_9)_2(Si_9O_{27})_2(OH,Cl)_2$	9
Euxenite-(Y)	(Y,Ca,Ce,U,Th)(Nb,Ta,Ti) 2O 6	24
Fergusonite-(Ce)	(Ce,La,Y)NbO4	53
Gadolinite-(Ce)	Ce,La,Nd,Y) 2FeBe 2Si 2O 10	60
Kainosite-(Y)	Ca2(Y,Ce) SiO4O12(CO3)(H2O)	38
Loparite	(Ce,Na,Ca)(Ti,Nb)O 3	30
Monazite-(Ce)	(Ce,La,Nd,Th)(PO4,SiO4)	65
Parisite-(Ce)	Ca(Ce,La)2(CO3)3F2	61
Xenotime	YPO_4	61
Yttrocerite	CaF2 + (Y,Ce)F3	53
Huanghoite-(Ce)	BaCe(CO3)2F	39
Cebaite-(Ce)	Ba3Ce2(CO3)5F2	32
Florencite-(Ce)	(La,Ce)Al3(PO4)2(OH)6	32
Synchysite-(Ce)	(Ce,La)Al3(PO4)2(OH)6	51
Samarskite-(Y)	(YFe3+Fe2+U,Th,Ca)2(Nb,Ta)2O8	24
Knopite	(Ca,Ce,Na)(Ti,Fe)O3	na

7.2 Sveccofennian Deposits – Discussion

There are a number of more or less well known REE deposits in Europe including two of the better-known deposits – Ilimaussaq and Lovozero.

Ilimaussaq is the type locality for the agpaitic nepheline syenite deposits as well as several mineral species, including eudialyte, and has been extensively described in the literature – partly due to its excellent exposures in a glacially smoothed environment. It consists of a large, at least pseudo-layered intrusion of a variety of rock types. Of particular significance with respect to the REO are the kakortotites which consist of a layered sequence at the base of the complex, it consists of separate layers of feldspar (white), arfvedsonite (black) and eudialyte (red). There are over twenty-nine such horizons known over the outcropping thickness of approximately 400 m. This forms the main bulk of the REE mineralisation at Ilimaussaq. However, the ratio HREO/TREO is only about 0.15 lowering the value of this deposit.

The Lovezero Complex (together with the related Khibina Complex) on the Khola Peninsula of Russia form the largest agpaitic massif in the world. The Lozovero Complex totals some 650 km² and has the following characteristics (Laznicka, 2006):

> The intrusion has a subcircular outline, sharp contacts, a stock-like shape in depth topped by a laccolith-like differentiated sequence. The concentrically zoned intrusion resulted from four successive magmatic phases. Metamorphosed and metasomatized nepheline syenites of Phase 1 are preserved mostly as rafts and xenoliths. Phase 2 produced a cyclic, rhythmically layered sequence of (from bottom to top of each cycle) urtite, foyaite, lujavrite. Phase 3 comprises coarsely crystalline layered lujavrite with a prominent zone of eudialyte lujavrite in the central part of the massif. Phase 4 emplaced alkaline dikes (monchiquite, camptonite, tinguaite).
>
> Layered eudialytic lujavrite (mesocratic nepheline syenite) constitutes 18% of the intrusion and forms a rhythmic sequence 150 to 500 m thick, with an average content of 1.36% ZrO_2. Each rhythm consists of successive bands of leucocratic, mesocratic and melanocratic varieties, with gradual transition from one to the other. Dark red bands, highly enriched in eudialyte, contain up to 75% of eudialyte crystals in nepheline matrix, and they form lenticular intercalations 0 to 40 cm thick. In the Chivruai valley 13 eudialytite bands form about 40% of a 3 m thick section of lujavrite. Eudialytites contain 6.76 to 8.68% ZrO_2, 0.39-0.93% $(Ta,Nb)_2O_5$ and 1.01-1.56% REE_2O_3. This represents a resource of 10 to 100 kt Ta, 1 mt Nb, 100 kt to 1 mt REE and some 4 to 40 mt Zr

Both the Ilimaussaq and the Lozovero are interpreted as having had (originally, at least) a lopolithic or laccolithic shape (Figure 7-1) and it is hypothesised that this is possibly the case at Norra Kärr.

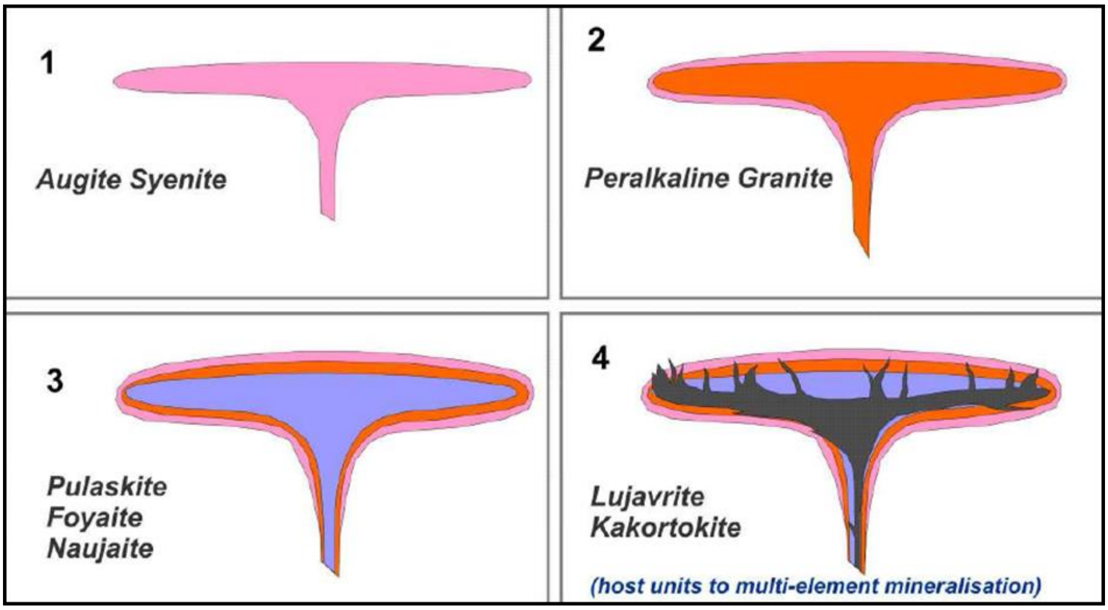


Figure 7-1: Possible Mode of Formation of the Ilimaussaq Intrusion (Source: Larsen & Sørensen, 1987)

8 EXPLORATION

The following chapter has been taken from the 2015 PFS prepared by WAI and GBM in the PFS report (Davidson *et al*, 2015b) and has been modified by SRK to ensure it reflects the current status of the Project.

8.1 Initial Works

LEM (as Tasman) initiated their exploration of Norra Kärr by investigating Boliden's historical work noted in Section 5 above. The Boliden trench samples were archived at the SGU and were made available. From this, LEM was able to select and assay rocks that were representative of various lithological units along the two trenches that were excavated (see Section 5) and sampled by Boliden. While the samples were only grab samples meant to be representative of each of the lithotypes sampled by Boliden, they served to give a very good indication of the REE potential of the Norra Kärr intrusive complex. With the Boliden results for the trench composites available a comparison showed that there was considerable potential for a significant REO deposit, as shown in Table 8-1.

Table 8-1: Comparison of LEM grab samples vs Boliden composite trench samples

TASMAN SAMPLE	BOLIDEN SPECIMEN	INTERVAL	WIDTH (M)	TASMAN ZrO2%	BOLIDEN ZrO2 %	TASMAN Hf %	BOLIDEN Hf %	TASMAN TREO%	BOLIDEN TREO%
400017	3	I	47	0.05	0.17	0.001	0.004	0.04	0.06
400016	9	II	22	0.67	0.59	0.014	0.014	0.17	0.13
400015	25	III	53	0.56	0.48	0.010	0.010	0.24	0.16
400014	33	IV	67.5	3.26	2.15	0.045	0.040	0.26	0.35
400013	38	IV	67.5	0.97	2.15	0.018	0.040	0.16	0.35
400012	45	IV	67.5	1.63	2.15	0.030	0.040	0.46	0.35
400011	54	IV	67.5	1.84	2.15	0.035	0.040	0.64	0.35
400010	62	V	60.5	1.65	2.00	0.030	0.040	0.33	0.40
400009	70	V	60.5	0.06	2.00	0.001	0.040	0.10	0.40
400008	79	V	60.5	1.69	2.00	0.031	0.040	0.54	0.40
400007	93	V	60.5	1.75	2.00	0.032	0.040	0.70	0.40
400005	117	VI	60	1.87	1.94	0.037	0.040	0.43	0.45
400004	122	VII	56	1.76	1.52	0.034	0.032	0.27	0.28
400003	129	VII	56	1.33	1.52	0.025	0.032	0.19	0.28
400002	136	VII	56	4.78	1.52	0.052	0.032	0.35	0.28
400001	152	VIII	32	0.05	0.55	0.001	0.012	0.04	0.10
400018	4	IX	25	1.11	0.90	0.023	0.017	0.05	0.07
400019	16	X	21.5	1.35	1.40	0.033	0.026	0.21	0.22
400020	20	XI	63.5	1.09	1.66	0.021	0.027	0.18	0.42
400021	41	XI	63.5	1.10	1.66	0.019	0.027	0.49	0.42
400022	59	XII	64	1.49	1.40	0.023	0.029	0.57	0.67
400023	73	XII	64	1.26	1.40	0.022	0.029	0.46	0.67
400024	86	XIII	66	1.43	0.47	0.024	0.011	0.46	0.38
400025	108	XIII	66	0.04	0.47	0.001	0.011	0.26	0.38
400026	129	XIV	90	0.59	0.35	0.011	0.009	0.24	0.24
400027	139	XIV	90	0.02	0.35	0.000	0.009	0.10	0.24
400028	151	XV	17	0.82	1.47	0.012	0.029	0.32	0.71
400029	159	XVI	35	0.92	1.47	0.020	0.018	0.35	0.31
400030	167	XVI	35	1.03	1.47	0.022	0.018	0.32	0.31

In 2009, LEM (as Tasman) also submitted five rock specimens for petrographic analysis. The findings supported the observations made previously about Norra Kärr intrusive, in that as a whole, the rocks could be classified as peralkaline nepheline syenites. Small amounts of eudialyte, catapleiite or rosenbuschite, fluorite and apatite were also noted. Foliation, defined by aligned mineral grains, is attributed to either regional deformation or as a primary magmatic flow texture (Ashley, 2009).

Also in 2009 LEM (as Tasman) contracted Mr John Nebocat of Pacific Geological Services to prepare an NI43-101 technical report. This report summarised the pre-drilling history of the property, recommended further exploration, and encouraged LEM to continue advancement of the Project (Reed, 2011).

In keeping with the recommendations of Mr Nebocat, drilling commenced on the Norra Kärr Project during the winter 2009 continuing until spring 2010. LEM (as Tasman) drilled 26 diamond drillholes totalling 3,276 m in five E-W orientated profiles across the Norra Kärr intrusion.

Various techniques have been applied when considering the geological mapping of the Norra Kärr igneous body. These include:

- Topographic mapping (including orthophotography and a laser survey);
- Geological mapping (including structural mapping); and
- Ground Magnetometry.

8.2 Topographic Mapping

Various topographic maps of the area are readily available, as documented below:

- Standard Swedish survey maps - at various scales, e.g. 1:50,000 sheet 56375 DinKarta;
- 1 metre contours – a topographic map with 1 m contours has also been produced by LEM (see Fig 8-2).
- Notes: red shape is the exploration licence boundary

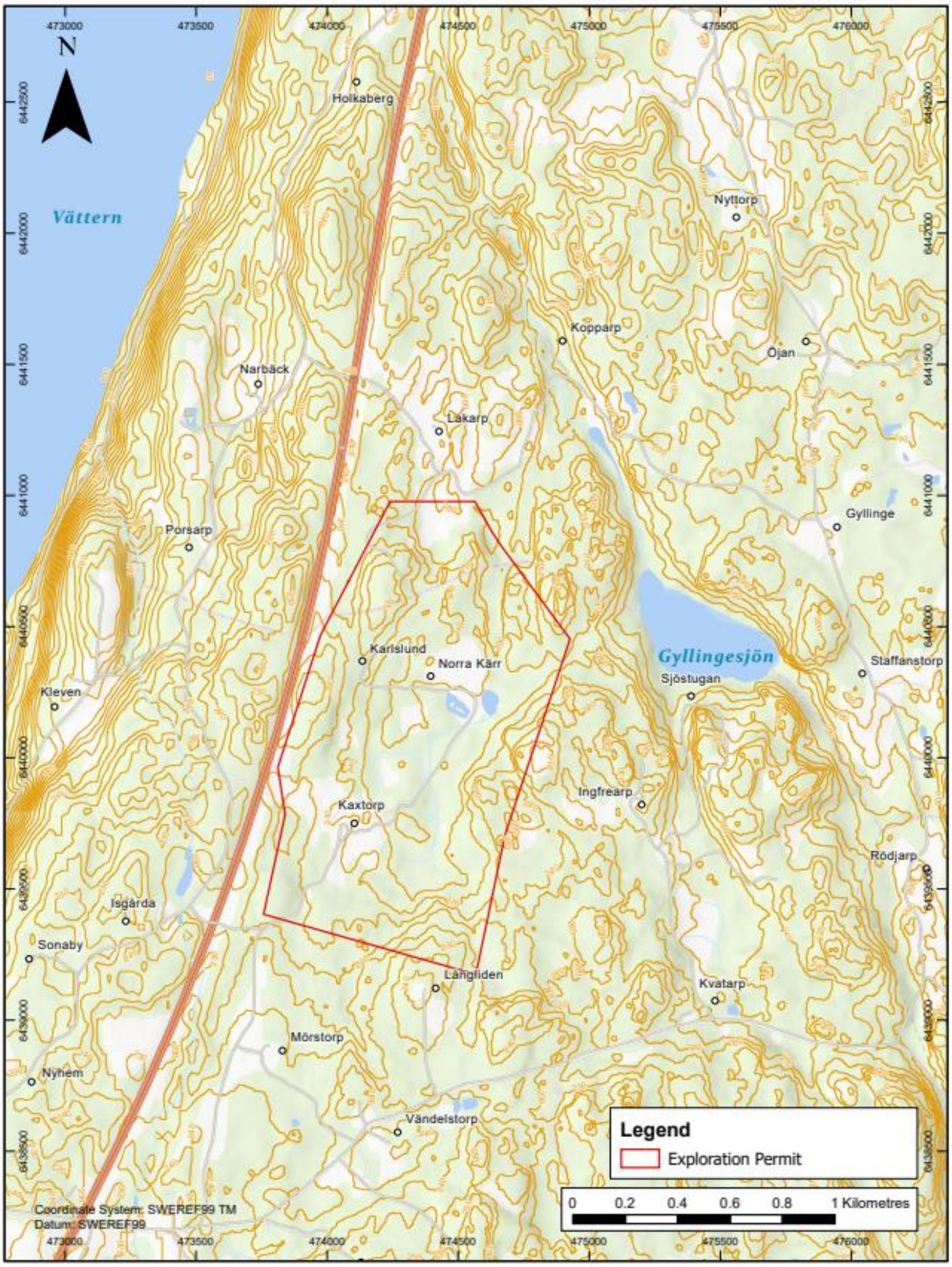


Figure 8-1: Swedish Survey 1:50,000 Topographic Sheet (Source: 56375 DinKarta)

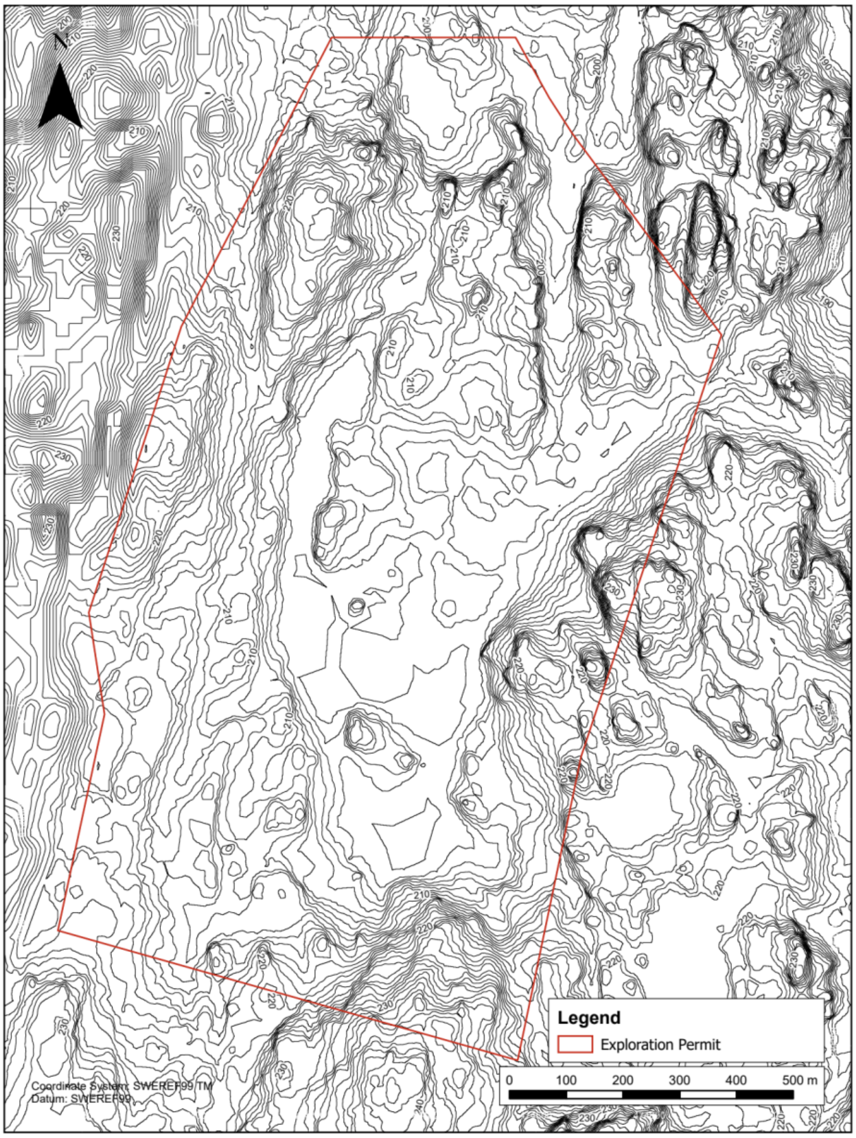


Figure 8-2: Contour Plan of Norra Kärr at 1m Intervals (Source: Davidson *et al,* 2015a)

8.3 Geological Mapping

The geological mapping of the Norra Kärr complex has been an iterative process as many elements have been utilised to produce the current geological map incorporating the following:

- Surface geological mapping;
- Ground Magnetometry;
- Detailed structural mapping; and
- Projection of the drillholes to surface which has confirmed the geology.

8.3.1 Surface Geological Mapping

While there is not a large amount of outcrop, there is sufficient to be able to produce an outcrop map which confirms the lithotypes - better identified in drill core - and certainly to confirm the overall lithostructure of the intrusive. Figure 8-5 shows the approximate outline of the Norra Kärr complex with the surface, geological "observation" points showing the dip and strike and the LEM coding of the outcrops. It is apparent that there is some considerable outcrop in the north of the intrusive, but it is somewhat sparser in the south where it is relatively marshy – as indicated by the topography.

The structure of the Norra Kärr complex is well indicated by the dip and strike observations with the overall westerly dip being apparent. This is better shown in the geological sections which are included in Section 8.3.3 below.

These observations immediately give a very reasonable impression of the geology of the Norra Kärr intrusion and the layering/zonation of the complex. This is further enhanced by the other techniques that LEM has applied.

8.3.2 Magnetometry

While the ground Magnetometry has been the most useful tool in elucidating the tectonics that have affected Norra Kärr, the regional airborne data has also been applied. This has allowed for a better understanding of the overall tectonic setting.

The ground magnetometry provided backing for the interpretation of the structural geology. In terms of lithotypes, one other point to note is that the relatively mafic (and barren of mineralisation) kaxtorpite forming the (offset) centre of the complex shows up very well as a positive magnetic feature – as is to be expected (see Figure 6-5).

8.3.3 Detailed Structural Mapping

LEM commissioned an experienced structural geologist to undertake a study of the intrusive and its surrounds (Rankin, 2011), Rankin concluded that:

"Emplacement and distribution of REE mineralisation is dominantly controlled by early pegmatite (comagmatic?), with some pre- to syn-D1 partial melt remobilisation possible.

The intrusive was emplaced along a regional, structurally-anomalous N- to NNE- trending fault/shear corridor within a Palaeoproterozoic granite/gneiss terrane dominated by a strong WNW- to NEW-trending structural grain. The emplacement locus may be coincident with an intersecting deep-crustal WNW-trending fault corridor."

In total 119 drillholes have been drilled and this has been the main tool in elucidating the geology of Norra Kärr as well as providing an excellent database for the estimation of a Mineral Resource (Section 11)

Nearly all the holes are angled holes drilled in an easterly direction at 50° to allow for the overall structure of the intrusion as illustrated in Figure 8-3 (the drillhole positions are shown as blue dots and the traces as the black lines).

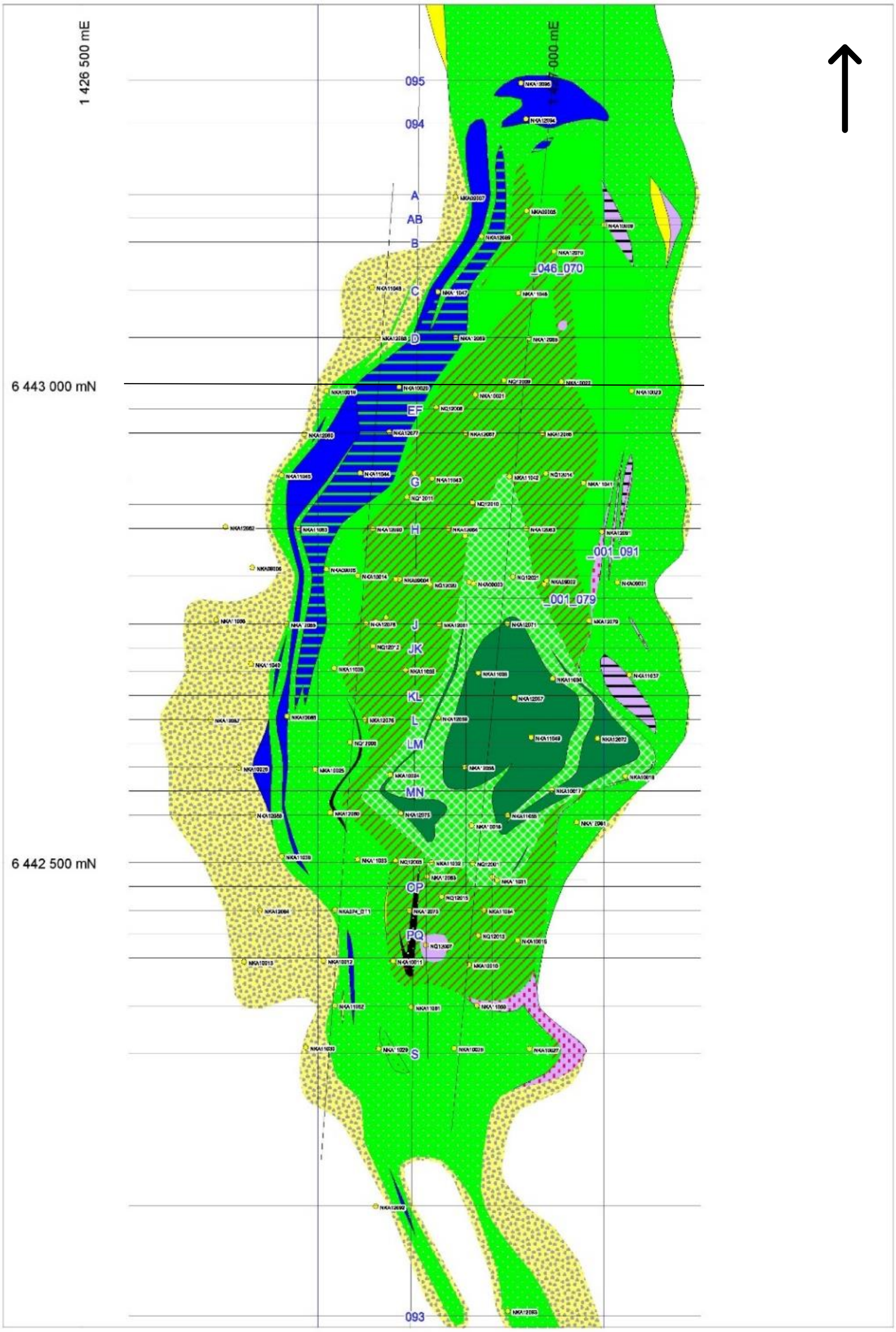


Figure 8-3: Geological Map of the Norra Kärr Intrusive showing drillhole collars and cross-section lines (map legend shown in Figure 8-7) (Source: LEM)

The Norra Kärr intrusive is a keel-shaped, synformal body, possibly a folded lopolithic intrusion, more or less layered and subsequently deformed. The geological interpretation is presented as a geological map (Figure 8-3 above) and as various sections across the intrusive (Figure 8-8 to Figure 8-11). This interpretation holds together well when rigorously interrogated and thus forms a robust model for the resource calculations and mine planning which are presented in later Sections of this document.

There are various observations from the geological map and sections that are worth noting as they both strongly confirm the geological interpretation and the quality of the drilling database:

- Axial plane mafic dyke – this dyke is a consistent feature both in (limited) outcrop and in many of the drillholes and appears to have been intruded (prior to or more probably during the major deformation episode) in the axial plane of the intrusive. It forms a good "marker horizon" and also shows up the strong deformation towards the southern end of the complex (previously noted) where it is strongly folded and somewhat broken up – probably by the late rotational (dextral) shearing.
- "Foliation" – this feature has been noted previously (see Section 8.3.1 and Figure 8-5 and Figure 8-8) and is clearly demonstrated on several "levels", i.e. plan slices through the intrusive constructed from the sections. Two illustrations of this are given as Figure 8-3 and Figure 8-4.
- Zonation – the obvious zonation of the differing grennaite types is clear. This aids considerably in the economic assessment of the REO mineralisation of Norra Kärr.
- Drilling density – the drilling density (±40 m) allows for the construction of quite accurate geological sections and consequently, an accurate map both surface and sub-surface.

All of these features are well noted and described in the LEM work as well as being verified by WAI and confirmed by SRK (Section 11).

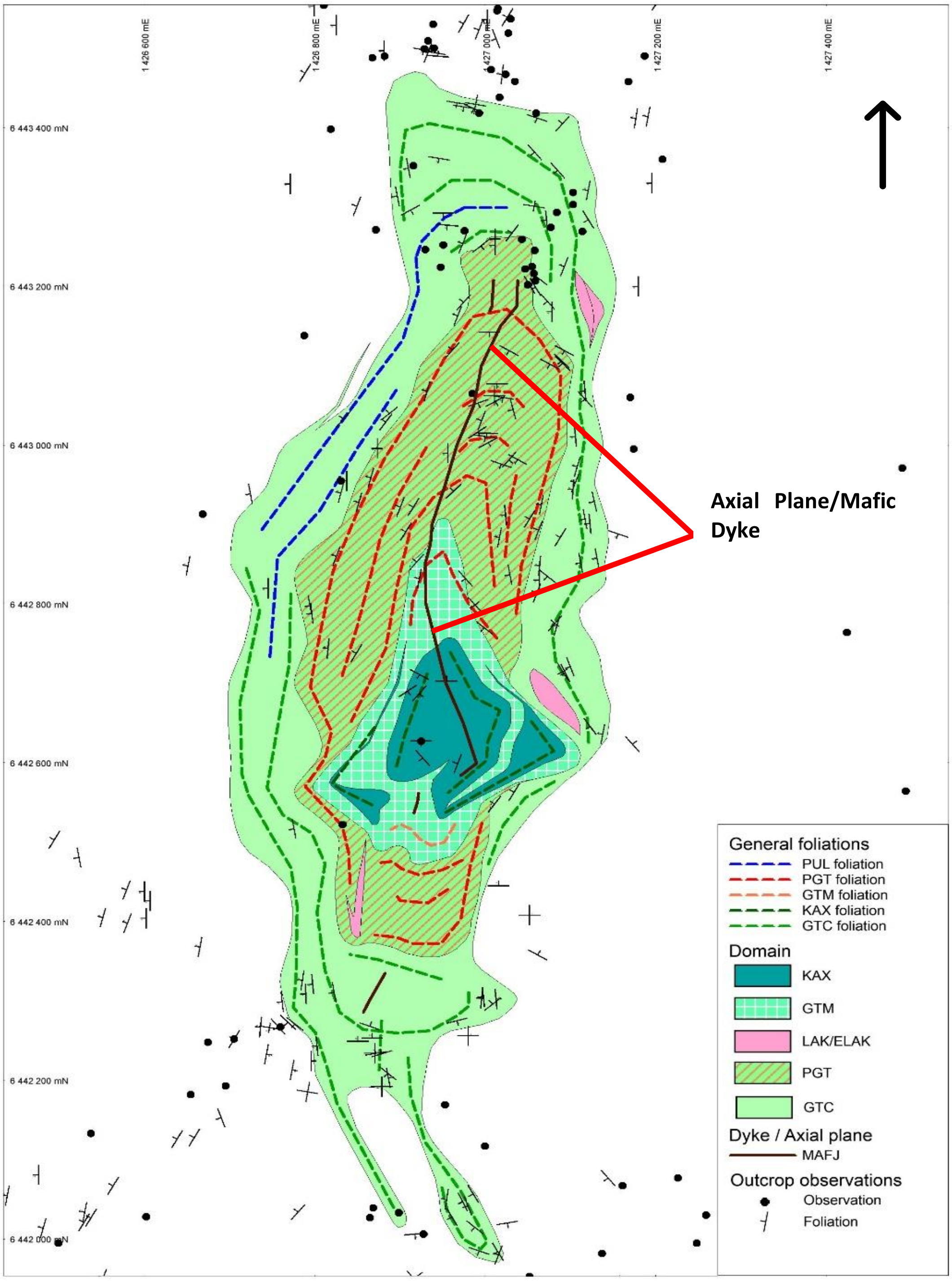


Figure 8-4: Geological Plan of Norra Kärr showing foliation trends (Source: LEM)

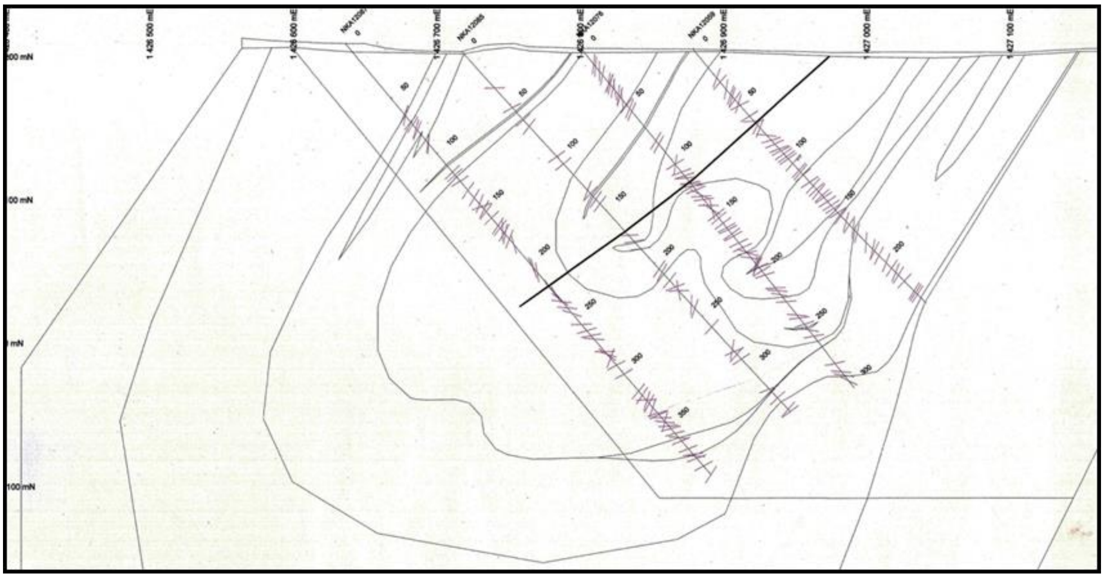


Figure 8-5: "Typical" Geological Section of Norra Kärr Showing Foliation Trends (Source: LEM)

Obviously, as a consequence of the drilling and to provide the data necessary to inform the resource calculations, geological sections have been constructed and a selection of these are presented as Figure 8-8 to Figure 8-11.

The geology of the representative sections presented here (and others) was confirmed during the site visit with the WAI geologist inspecting the core and accompanying the logging with the LEM geologists. As well as the sections, the detailed logs, the strip logs and the core photography created by the LEM staff also provide valuable tools.

The strip logs also include the assay values with an example in Table 8-2 showing the TREO and the HREO. The full strip logs show all the REO assays by individual element.

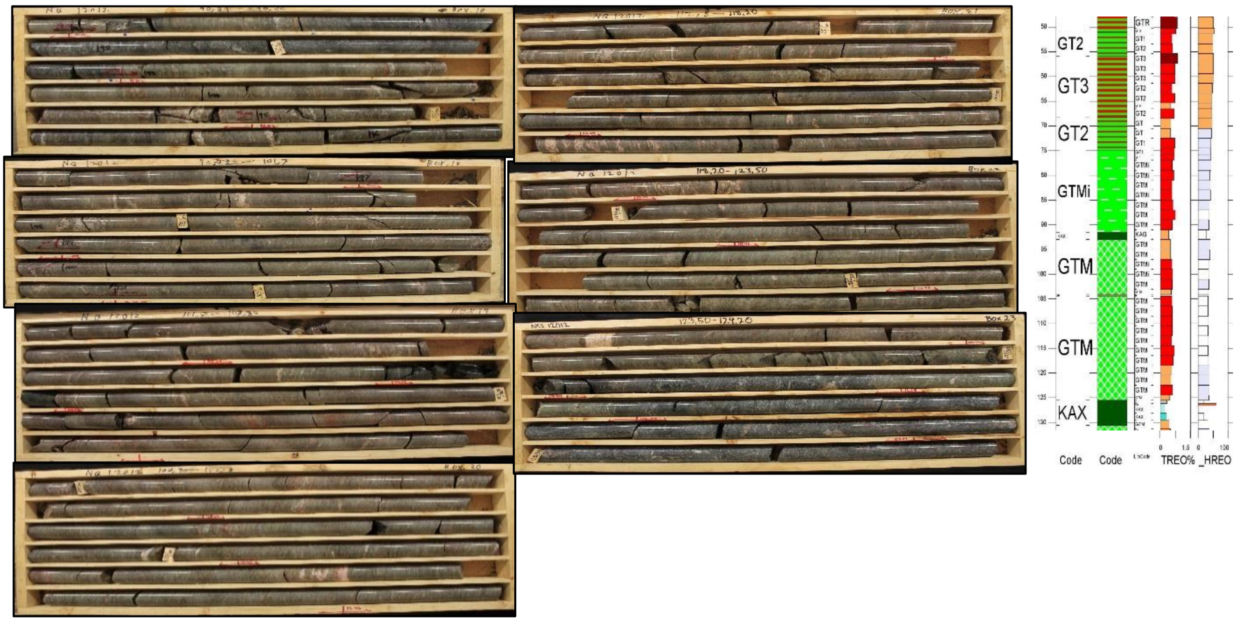


Figure 8-6: Core Photography and Corresponding Strip Log (Source: LEM)

Table 8-2: Example of Detailed Drillhole Log (Hole NKA10014)

Hole_ID	Sample_No	From	To	Width	Description
NKA10014	401033	1.95	2.60	0.65	Grennaite with 40% peg zones rich in bluish Cat xx:s.
NKA10014	401034	2.60	4.60	2.00	Grennaite slightly cooked, re-crystallized with only occasional cm wide migmatitic-pegmatitoidal schlieren.
NKA10014	401035	4.60	6.60	2.00	Grennaite as above with 3-5% bluish Cat laths.
NKA10014	401036	6.60	8.25	1.65	Grennaite as above with 3-5% bluish, small Cat laths. Occasional cm-wide migm-peg schlieren.
NKA10014	401037	8.25	9.85	1.60	Grennaite as above with 3-5% bluish, small Cat laths. Occasional cm-wide migm-peg schlieren.
NKA10014	401038	9.85	11.35	1.50	Grennaite as above with 3-5% bluish, small Cat laths. Occasional cm-wide migm-peg schlieren.
NKA10014	401039	11.35	13.35	2.00	Grennaite with 5-10% up to 7 cm wide peg zones/schlieren. Some Eu.
NKA10014	401040	13.35	15.40	2.05	Grennaite with 5% peg/migm schlieren
NKA10014	401041	15.40	17.45	2.05	Grennaite with 5-10% peg schlieren with some blue Cat and minor Eu.
NKA10014	401042	17.45	18.75	1.30	Almost 100% very coarse grained peg. Mic-Alb-Nef, some Aeg and relatively rich in Eu and also pink large (up
NKA10014	401043	18.75	20.00	1.25	70% Peg with some zones of Grennaite. Relatively Eu rich and also with large pink Cat xx:s.
NKA10014	401044	20.00	21.25	1.25	80% Peg with some zones of Grennaite. Relatively Eu rich and also with large pink Cat xx:s.
NKA10014	401045	21.25	23.00	1.75	25% irregular peg zones/schlieren with large orange-red Cat xx:s and some Eu.
NKA10014	401046	23.00	24.75	1.75	25% irregular peg zones/schlieren with large orange-red Cat xx:s and some Eu.
NKA10014	401047	24.75	25.42	0.67	100% Peg. Pale fsp dominated. Relatively rich in fine diss. PbS, minor blue Cat.. In part pink-red coloured fsp.
NKA10014	401048	25.42	27.40	1.98	25% peg/migm schlieren (Mic-Alb-Nef and some Eu).
NKA10014	401049	27.40	29.40	2.00	20% peg/migm schlieren as above. Tr PbS.
NKA10014	401050	29.40	31.20	1.80	10% peg/migm schlieren as above. Some Natrolite.
NKA10014	401052	31.20	33.00	1.80	10% peg/migm schlieren as above. Some Natrolite.
NKA10014	401053	33.00	33.45	0.45	100% very coarse grained peg with some Natrolite xx:s in open vugs. Some Eu.
NKA10014	401054	33.45	35.20	1.75	100% very coarse grained peg with some Natrolite xx:s in open vugs. Some Eu.
NKA10014	401055	35.20	36.90	1.70	30% peg and medium grained migm/peg zones. Some Eu-Cat. Tr. PbS.
NKA10014	401056	36.90	38.90	2.00	Grennaite with only occasional thin migm schlieren. Some Natrolite bandlets and vugs.
NKA10014	401057	38.90	40.90	2.00	30% peg and medium grained migm/peg zones. Some Eu-Cat. Tr. PbS.
NKA10014	401058	40.90	42.90	2.00	Grennaite with 25% up to 3 cm wide peg/migm bands rich in blue-grey Cat xx:s and minor Eu.
NKA10014	401059	42.90	44.90	2.00	Grennaite with 40% peg/migm zones. Rel rich in both Eu and Cat. Red-grey Cat xx:s.
NKA10014	401060	44.90	46.90	2.00	40-50% peg zones. Relatively Eu rich, some Cat. Tr PbS.
NKA10014	401061	46.90	48.90	2.00	25% peg schlieren/zones.
NKA10014	401062	48.90	50.90	2.00	30% peg schlieren/zones. Some Eu+Cat.
NKA10014	401063	50.90	52.90	2.00	30% peg schlieren/zones. Some Eu+Cat.
NKA10014	401064	52.90	54.90	2.00	40% zones of peg and more medium grained "granite textured" migm. Grennaite host.
NKA10014	401065	54.90	56.75	1.85	30% peg zones and schlieren with Eu and bluish Cat. Tr. PbS.
NKA10014	401066	56.75	58.65	1.90	30-40% peg zones and schlieren with Eu and bluish Cat. Tr. PbS + fluorite.
NKA10014	401067	58.65	60.55	1.90	20% peg zones and schlieren with Eu and bluish Cat.
NKA10014	401068	60.55	62.45	1.90	80% peg zones and more medium grained Grennaite. Relatively Eu rich, some Cat. Tr. PbS+fluorite.
NKA10014	401069	62.45	64.45	2.00	40% peg and medium grained zones. Relatively Eu rich. Minor PbS+fluorite.
NKA10014	401070	64.45	65.45	1.00	90-100% very coarse grained peg. Eu rich. Some vugs filled with Natrolite xx:s. Minor fluorite, tr. PbS.
NKA10014	401071	65.45	67.45	2.00	40-50% peg zones. Relatively Eu rich. Tr PbS+Fluorite.
NKA10014	401072	67.45	69.45	2.00	25-35% peg zones. Relatively Eu rich. Minor bluish Cat.
NKA10014	401073	69.45	71.45	2.00	50-60% Peg zones (in part medium grained). Relatively Eu rich, some Cat. Tr PbS+fluorite.
NKA10014	401074	71.45	73.45	2.00	30% peg (mainly one zone). In part weathered, vuggy. Some Natrolite xx:s in vugs. Some Eu, minor PbS.
NKA10014	401075	73.45	75.45	2.00	15% peg zones/schlieren. Minor Eu, tr. PbS.
NKA10014	401077	75.45	77.05	1.60	60% mainly medium grained Grennaite. In part real peg. Some Eu, minor PbS+Fluorite.
NKA10014	401078	77.05	78.55	1.50	10-20% Peg schlieren. In part somewhat cooked, re-crystallized Grennaite.
NKA10014	401079	78.55	80.55	2.00	80% Peg zones. Often more medium grained texture. Some Eu, tr. PbS + Fluorite.
NKA10014	401080	80.55	82.55	2.00	60% Peg zones. Often more medium grained texture. Some Eu, minor. PbS + Fluorite.
NKA10014	401081	82.55	84.55	2.00	70% Peg zones. In part more medium grained texture. Some Eu, tr. PbS + Fluorite.
NKA10014	401082	84.55	86.55	2.00	90% Peg zones. Often more medium grained texture. Aeg-amph rich. Some Eu, tr. PbS + Fluorite.
NKA10014	401083	86.55	88.45	1.90	90% Peg zones. Often more medium grained texture. Aeg-amph rich. Some Eu, tr. PbS + Fluorite.
NKA10014	401084	88.45	89.60	1.15	10-15% peg zones in slightly re-crystallized Grennaite.
NKA10014	401085	89.60	90.72	1.12	30% Peg zones.
NKA10014	401086	90.72	92.20	1.48	85% of the sample medium grained, "granite textured". Not real peg. Rich in Aeg. Some Eu.
NKA10014	401087	92.20	94.20	2.00	Somewhat re-crystallized Grennaite with only one 15 cm zone of Peg.
NKA10014	401088	94.20	96.20	2.00	Grennaite only rare thin peg schlieren.
NKA10014	401089	96.20	98.20	2.00	10% peg schlieren.
NKA10014	401090	98.20	100.20	2.00	5-10% Peg zones/schlieren. Somewhat re-crystallized Grennaite.
NKA10014	401091	100.20	102.20	2.00	Less than 5% peg schlieren.
NKA10014	401092	102.20	104.10	1.90	Only occasional peg schlieren. At 103.35m some pale yellow-red small xx:s in a peg vein (possibly strange Eu?
NKA10014	401093	104.10	106.15	2.05	25% irregular peg zones/schlieren. Minor Eu, Tr PbS+Fluorite.

All of the features mentioned in the descriptions above were checked by WAI and was satisfied that LEM staff exercised considerable diligence in their logging and that the sections produced – see below – are both accurate and well recorded. SRK reviewed and confers with WAI.

The complete suite of geological sections were utilised in producing the 3D geological wireframes and the MRE presented in Section 11.

The Legend for the geological sections is also reproduced here as Figure 8-7.

- Section E – has been constructed from six drillholes drilled at the 'standard" angle of ~50°. Careful logging has enabled the discrimination of the main lithotypes across the 400 metres of the section line and this is clearly seen on the section together with the TREO values and the percentage HREO. The individual REO values are included in the database and on the strip logs (illustrated above – see Figure 8-6 for an example).
- Section I – has been constructed from 12 drillholes drilled at the 'standard" angle of ~50°. Again the lithotypes are clear to see – this includes the axial plane mafic dyke on this section together with the first indications of the kaxtorpite "core" of the intrusive. The main mineralisation types – the PGT and GTM domains – are also clearly evident flanked by the GTC grennaite. During the site visit, these domains were clearly differentiated and confirmed by WAI.
- Section L – has been constructed from six drillholes drilled at the 'standard" angle of ~50°. On this section to the south of Section I, the kaxtorpite has developed further and clearly forms the core of the intrusive complex. The axial plane mafic dyke is clearly seen together with the two main mineralized rock domains. The Vaxjo Granite "host" rocks are also evident flanking the Norra Kärr alkaline body showing a degree of the fenitisation which was the subject of an academic work previously referenced (Christensson, 2013).
- Section O – has been constructed from nine drillholes drilled at the 'standard" angle of ~50°. At the southern end of the Norra Kärr body, the kaxtorpite core pinches out and this is well shown on this section. The GTM domain here forms two "tongues included in the massive PGT domain. The high values in the PGT are clearly shown. At the base of the section in drillholes NKA11038 and NKA11033 a tongue of the ELAK domain is to be seen. This is the eudialyte-bearing lakarpite previously described.

8.3.4 Surface Exploration Summary

Study of all the available data clearly shows the relatively simple structure and zonation of the Norra Kärr intrusive. The elongated, ovoid shape is well defined by the magnetometry, the surface mapping and the drill sections. The foliation indicates that the intrusive body may well "close off" at depth forming a synformal structure but this is not yet certain.

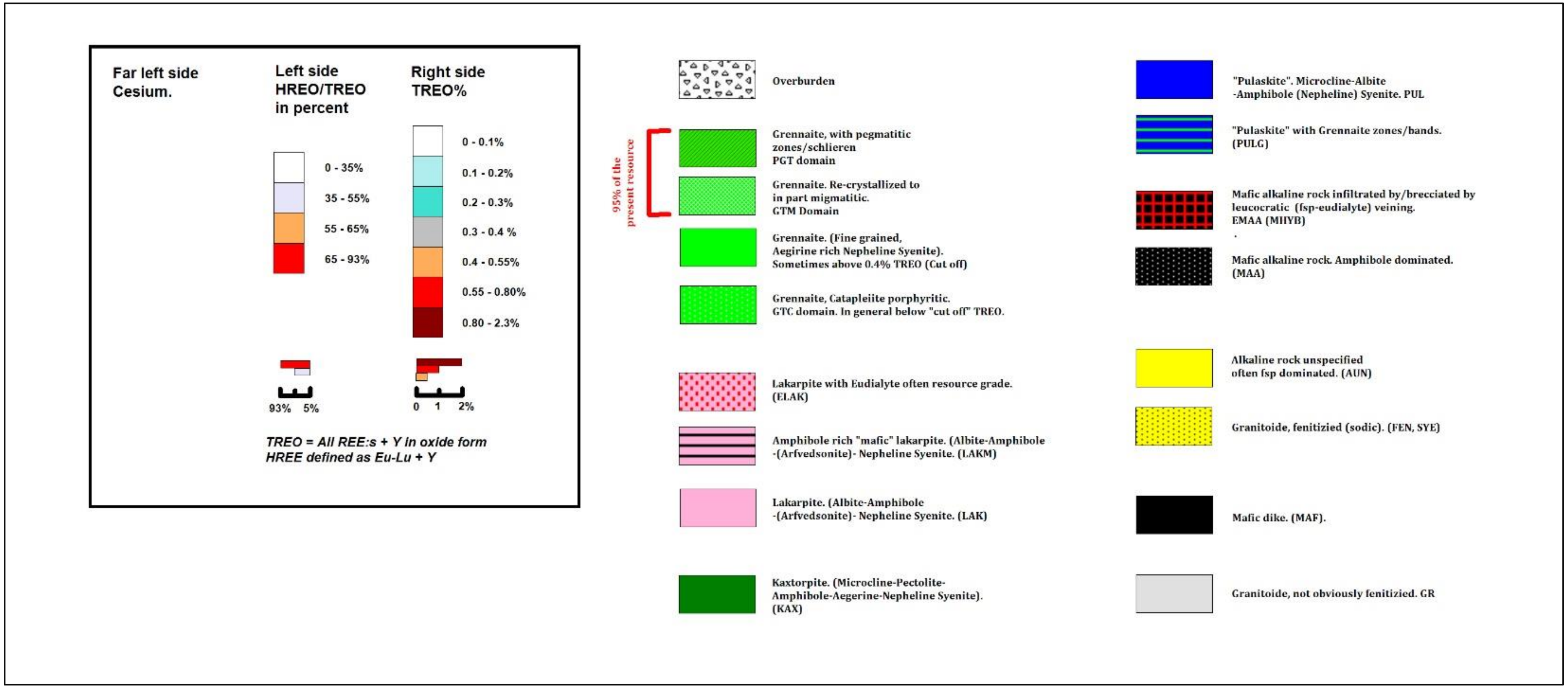


Figure 8-7: Legend for Geological Sections (Figure 8-8-Figure 8-11)

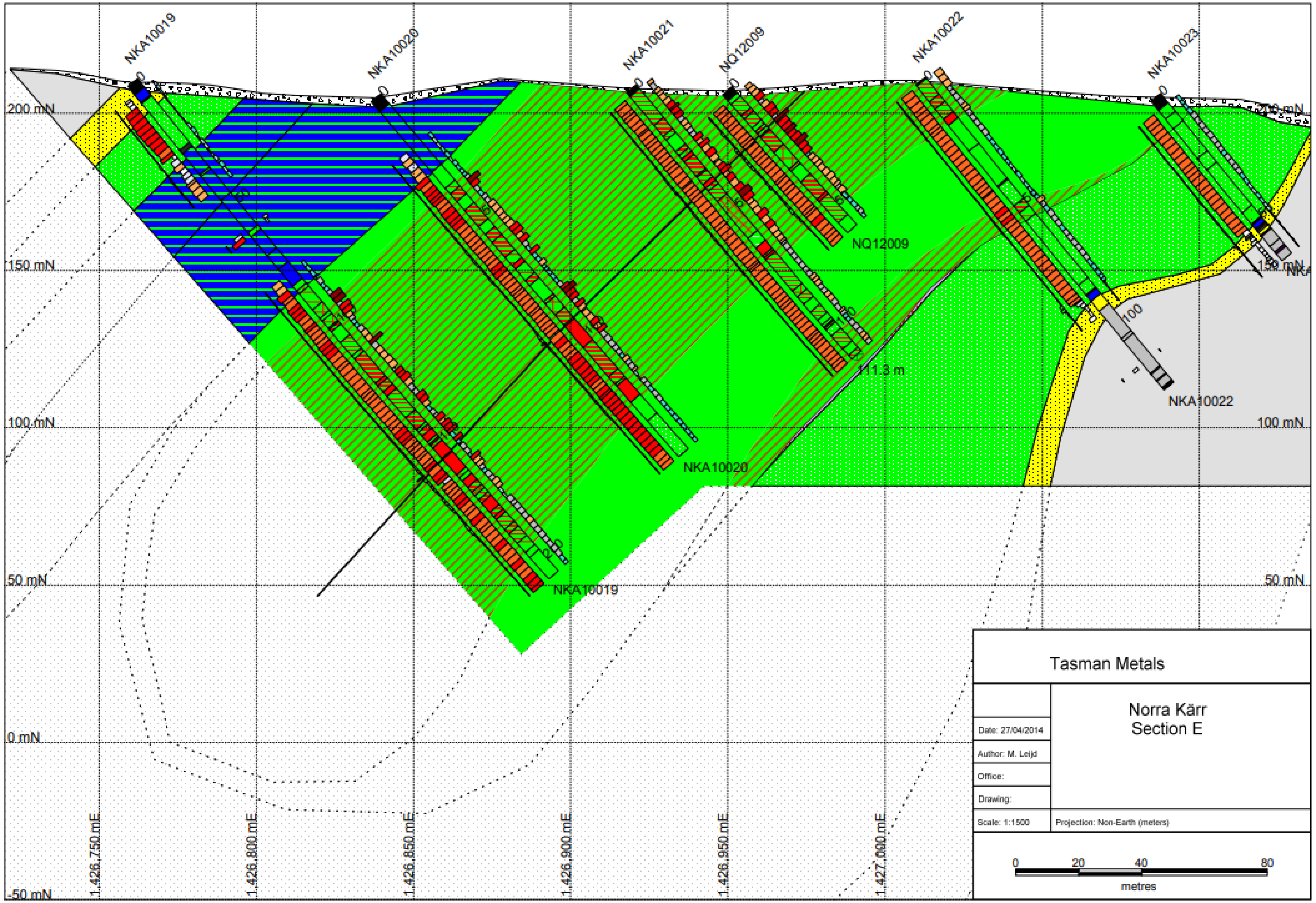


Figure 8-8: Section E, Geological Cross Section (See Figure 8-3)

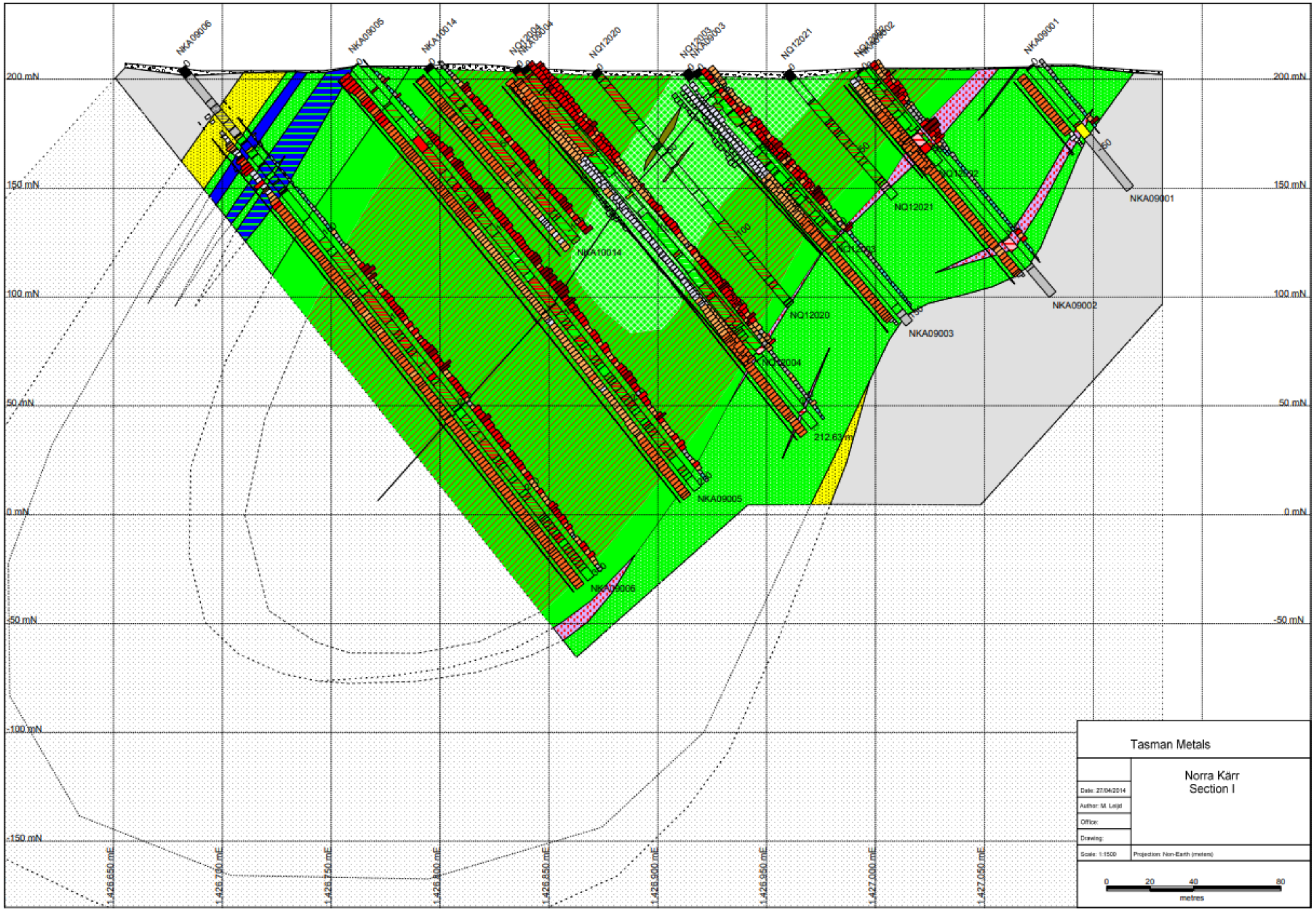


Figure 8-9: Section I, Geological Cross Section (See Figure 8-3)

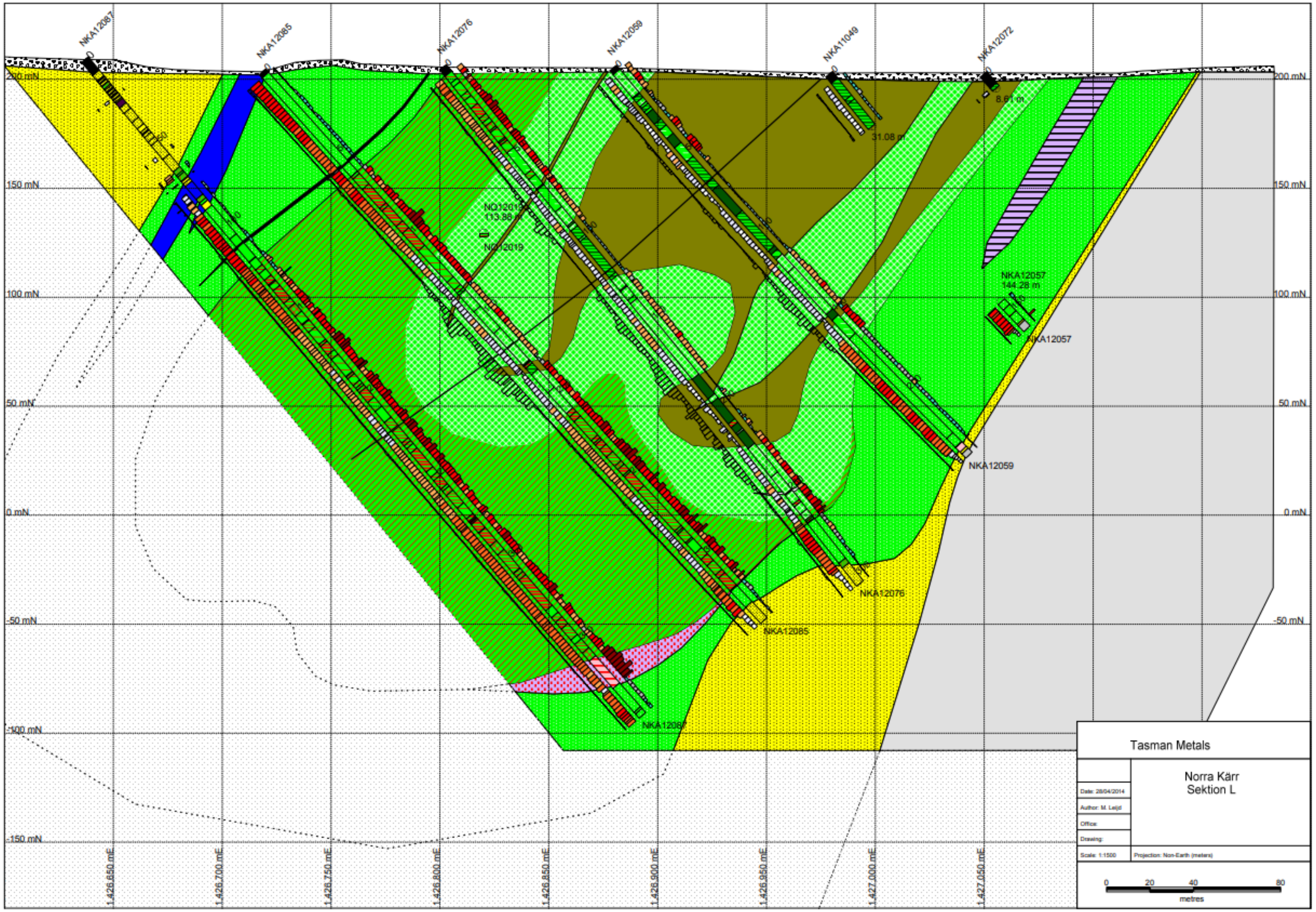


Figure 8-10: Section L, Geological Cross Section (See Figure 8-3)

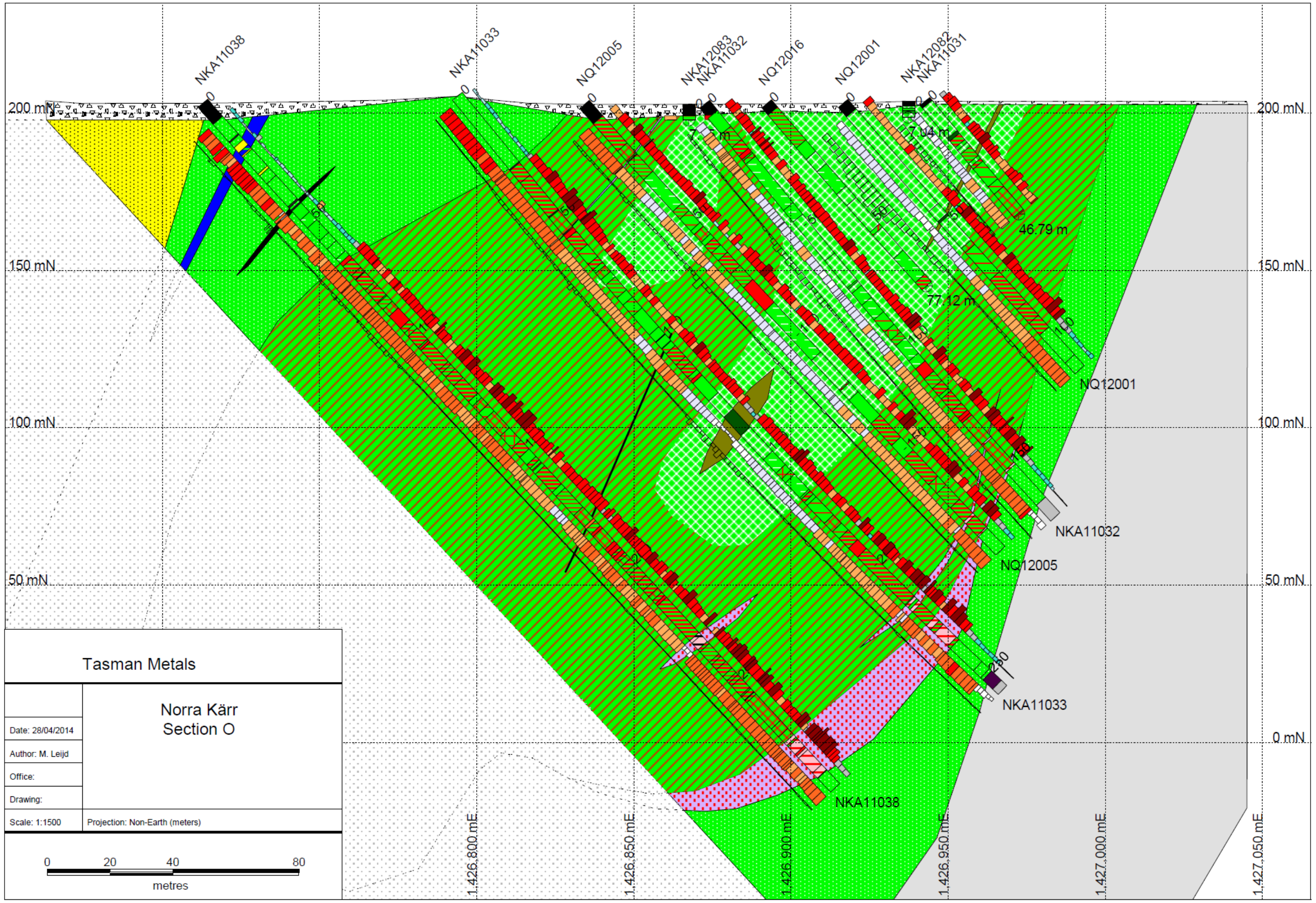


Figure 8-11: Section O, Geological Cross Section (See Figure 8-3)

9 DRILLING

The following chapter has been taken from the 2015 PFS prepared by WAI and GBM in the PFS report (Davidson *et al*, 2015b) and has been modified by SRK to ensure it reflects the current status of the Project.

9.1 Summary

LEM commenced drilling in December 2009 and as of the end of 2012 had completed a total of 119 drillholes totalling 20,420 m. Drilling has been carried out on east-west section lines with drilling inclined to the east at approximately 50°. Section lines are spaced approximately 50 m apart with drillholes spaced along the section lines at 60 to 80 m. Hole depths range from 30.5 to 395.6 m with an average depth of 172 m.

All drilling has been carried out using diamond core drilling, in total three drilling contractors have been used as summarised in Table 9-1. The initial phase of drilling comprising 11 drillholes in 2009-2010 was conducted by North Scandinavian Drilling ("NSD") who used a Diamec U6 (Atlas Copco) drill rig producing a core diameter of 42 mm (BGM). In February 2010 after NSD had drilled 11 holes drilling was taken over by Geo-Gruppen AB based in Göteborg Gothenburg who drilled 76 holes between February 2010 and May 2012 producing core with a diameter of 40.7 mm (BQ-TK).

The most recent drilling contractor employed was Olstam Borrteknik AB ("Olstam BT") using Sandvik drill rigs the company drilled 32 holes in 2012 with 9 of the holes yielding 39 mm (WL56) core diameter and 23 holes producing larger diameter 50.7 mm (NQ2) diameter core.

Table 9-1: Summary of LEM Exploration Drilling Activity

Drill Operator	No. Holes	Total Length (m)	Core rock Diameter	Drill Period	Orientated Core
NSD	11	1,793	BGM	2009-Jan 2010	Only for 3 Holes
Geo-Grupen	76	13,419	BQ-TK	Feb 2010-2012	47 holes
Olstam BT	32	5,209	WL56 (9 holes); NQ2 (23 holes)	2012	4 Holes
Total	**119**	**20,420**	**-**	**-**	**54**

9.1.1 Core Recovery

As part of the drilling and core logging works LEM has recorded core recovery based on length drilled verses the length recovered. The LEM drillhole database provided to WAI contained recovery details for 104 drillholes and comprises 11,295 core recovery records ranging from no recovery in the case of 6 holes through to >90% recovery for 99% of the sample database, core recovery averages 99.7%.

The geological database provided to WAI also recorded Rock Quality Designation ("RQD") for 103 drillholes and comprised 11,274 records with an average RQD of 86%.

SRK visited the site and the drill core storage at Woxna graphite mine located in Gävleborg County. The core is stored in an all-weather building in fully referenced and labelled covered wooden core trays and when visually reviewed appear in excellent condition and also display excellent recovery.

Overall, SRK is of the opinion that the Norra Kärr mineralisation is highly competent rock (that is not broken) and that no material losses of core have occurred. Further the high level of recovery and the assay results obtained preclude any preferential bias due to core loss.

9.1.2 Survey

Drillhole locations are initially laid out using a GPS based on the LEM geologists exploration plan. A final pick up of the drillhole collar co-ordinates is carried out post drilling by Metria AB, an independent Swedish land survey company. The collar survey is carried out using a Differential Global Positioning System ("DGPS") with an accuracy of 0.01 m in the horizontal and 0.2 m in the vertical.

Down hole surveys have been carried out for all drillholes using one of two different down hole survey instruments:

- DeviTool PeeWee; and
- Reflext EZ-Trac.

The DeviTool™ PeeWee is a downhole multishot magnetic survey instrument with an azimuth accuracy of ±0.5° and an inclination accuracy of ±0.1°. This particular downhole instrument was used for the initial phase of drillholes between 2009 and 2011.

In the later part of the exploration programme in 2001 and 2012 a Reflex EZ-Trac multishot magnetic downhole instrument was used with an azimuth accuracy of ±0.35° and an inclination accuracy of ±0.25°.

Overall, SRK is of the opinion that the survey works carried out by LEM (as Tasman) are robust and accurate.

9.1.3 Core Logging, Sampling and Storage

LEM (as Tasman) has taken a diligent approach to the core logging resulting in a detailed geological database containing driller information, survey data, detailed geological descriptions including lithological codes and geotechnical data comprising RQD results and foliation orientations.

During the drilling campaigns a core storage and preparation facility was utilized. A large warehouse (with a nearby office) was rented and drill core storage and preparation facilities installed. A picture of the facilities used for geological logging is shown in Figure 9-1.

SRK can verify the high quality of geological and sampling information. The underlying data supporting the Mineral Resource estimate are considered by SRK to be generated and input into the corresponding resource model in a satisfactory manner.

Figure 9-1: Core Logging Facility.

10 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following chapter has been taken from the 2015 PFS prepared by WAI and GBM in the PFS report (Davidson *et al*, 2015b) and has been modified by SRK to ensure it reflects the current status of the Project.

10.1 Core Cutting

At the LEM core facility in Gränna there is a "built in" core sawing shack which has an automated core saw to ensure accurate splitting of the core. The core is cut taking in consideration the main foliation/banding of the rock. When possible to reassemble the core, the same half of the core was submitted for assay. One half of the core is placed in a numbered plastic bag together with the corresponding sample ticket and the other half is left in the core tray.

Prior to the establishment of the LEM facility at Grannä core was cut at SGU facilities in Malå by an independent contractor. Details of the sample intervals are provided by LEM to the contractor ensuing that lithological boundaries are honoured. The core is then cut perpendicular to the foliation and banding of the rock using a diamond saw. The half cores for assay are placed into plastic bags with a sample ticket recording the individual sample numbers. Retained half core are returned to the core trays and stored at the SGU archive in Malå.

10.2 Sample Preparation

Samples are prepared at the ALS Chemex facility at Piteå in northern Sweden. ALS Chemex is an ISO accredited (ISO/IEC 17025) laboratory and the sample preparation facility is part of the main ALS laboratory set up.

Samples arriving at the ALS Chemex facility in Piteå are initially checked against the corresponding sample submission documents submitted by LEM's geological staff. Samples are then dried and weighed and recorded on the ALS Chemex database. Samples are dried at 110-120°C and then crushed with either an oscillating jaw crusher or a roll crusher.

The crushed sample, typically 250 g to 1 kg, is sub-sampled by use of a riffle splitter. A 250 g sub-sample is taken for further pulverising whilst the remaining crushed material is bagged up and returned to LEM. The 250 g sub-sample is then pulverized using a ring mill till >85% passes 75 µm (200 mesh screen).

For assay a sub-sample of the pulverised material of approximately 10 to 15 g is taken and shipped to the ALS Chemex assay laboratory in Vancouver, Canada for analysis. The remainder of the crush and pulp reject samples are stored at ALS Chemex in Piteå prior to being returned to LEM.

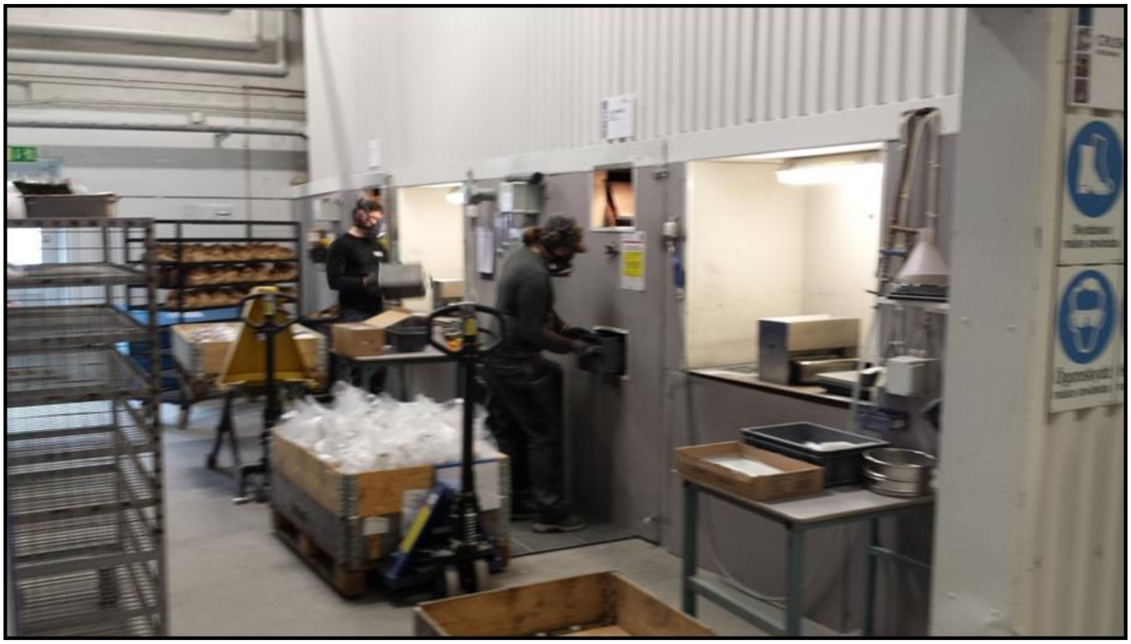


Figure 10-1: ALS Chemex Sample Preparation Facility Piteå

10.3 Analyses

LEM submitted all samples for assay at the ALS Chemex laboratory in Vancouver, Canada. ALS Chemex analysed the samples using the analytical method ME-MS81, an inductively coupled plasma ("ICP") mass spectrometry technique. In instances where samples returned assays with zirconium grades>10,000 ppm then these samples were also analysed using the ME-XRF10 method.

The analytical specification for the ME-MS81 method is:

A prepared sample (0.2 g) is added to lithium metaborate flux (0.90 g), mixed well and fused in a furnace at 1000°C. The resulting melt is then cooled and dissolved in 100 mL of 4% HNO_3 / 2% HCl solution. This solution is then analysed by inductively coupled plasma ("ICP") mass spectrometry."

Details of the detection limits for analytical method ME-MS81 are provided in Table 10-1 below:

SRK are satisfied that the analyses have been carried out to the highest standards and that the results may be used in a preparing a Mineral Resource in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves.

Table 10-1: ALS Chemex ME-MS81 Detection Limits

Element	Detection Range (ppm)		Element	Detection Range (ppm)	
	Min	Max		Min	Max
Ag	1	1000	Ga	0.1	1000
Ba	0.5	10000	Gd	0.05	1000
Ce	0.5	10000	Hf	0.2	10000
Co	0.5	10000	Ho	0.01	1000
Cr	10	10000	La	0.5	10000
Cs	0.01	10000	Lu	0.01	1000
Cu	5	10000	Mo	2	10000
Dy	0.05	1000	Nb	0.2	10000
Er	0.03	1000	Nd	0.1	10000
Eu	0.03	1000	Ni	5	10000
Pb	5	10000	Tm	0.01	1000
Pr	0.03	1000	U	0.05	1000
Rb	0.2	10000	V	5	10000
Sm	0.03	1000	W	1	10000
Sn	1	10000	Y	0.5	10000
Sr	0.1	10000	Yb	0.03	1000
Ta	0.1	10000	Zn	5	10000
Tb	0.01	1000	Zr	2	10000
Th	0.05	1000			
Tl	0.5	1000			

10.4 QA/QC

As part of the LEM exploration works a Quality Assurance/Quality Control ("QA/QC") programme has been implemented to provide support and credence to the sample assay data. The QA/QC programme has been applied to the drill core samples and comprises the submission of field duplicates, laboratory duplicates, external laboratory duplicates, blanks and certified reference materials ("CRM"). QA/QC samples use the same numbering sequence as standard samples; so as to ensure the samples are submitted blind to the laboratory and that no specialist treatment of the samples is carried out.

WAI were provided with the results of the field duplicates, laboratory duplicates, external laboratory duplicates, blanks and CRM in Excel format. The full analysis by WAI has been published previously (Davidson *et al*, 2015b). SRK has reviewed the data and agrees with WAI.

10.4.1 Field Duplicates

A total of 426 field duplicate results were analysed by WAI. The field duplicate samples were quarter core which was submitted to the sample preparation facility prior to being sent to the laboratory. In sending quarter core field duplicates the results allow LEM to ascertain the potential sum analytical, sub sampling and sampling error. Any issues associated with the representivity of the sampling methods, errors occurring due to the sample preparation methods or problems relating to analytical precision will most likely show up in the field duplicate results.

If an error is identified then further QA/QC procedures such as the use of crushed and pulverised duplicates, blanks and CRM are required to better delineate the source of the error. No material issues were identified by SRK in review.

10.4.2 Laboratory Duplicates

To test the analytical precision of the laboratory LEM submitted duplicate pulp samples. A total of 601 pulp duplicate sample results were analysed by WAI; no material issues were identified by SRK in review.

The results of the LEM laboratory duplicate samples show excellent levels of repeatability and thus demonstrate the laboratory is operating with a high level of analytical precision and that the sample post pulverising during the sample preparation stages is homogenous and representative.

10.4.3 External Duplicates

To provide support for the analytical precision and accuracy of the principal ALS Chemex laboratory duplicate pulverised pulp samples were submitted to external laboratories. For the 2010 and 2011 exploration works a total of 198 samples were submitted to ACME laboratory of which 77 samples were submitted in 2010 and 121 samples in 2011. The results of the external assays for the 2010 and 2011 exploration works shows an excellent correlation with the primary ALS Chemex assay results.

From the 2012 exploration programme a total of 323 pulp samples were submitted to Actlabs laboratory for check assays. As with the 2010 and 2011 external sample submissions a good correlation was demonstrated with the original ALS Chemex assays.

10.4.4 Blanks

As part of the QA/QC procedures adopted by LEM blank samples were submitted to the sample preparation facility/laboratory to test for sample contamination during the preparation process or the analytical stages. To comply with Swedish requirements LEM submitted a granite blank material.

A total of 234 blank sample results were provided to WAI. The results show that there are a number of samples reporting above the detection limit of 0.05ppm Dy, 0.5ppm Y, and 0.1ppm Nd that the laboratory test method is capable of producing. The blank samples show elevated grades across the suite of elements tested. In using a granite as a blank sample there will inherently be elevated levels of rare earth elements as granites are typically more enriched in these elements than most other rock types.

Whilst it is noted that the blank samples do contain mineralisation this is due to the function of using a granite as a blank material and is not a function of contamination issues at the sample preparation stage. The grades exhibited are minor typically <3% of the mean grade of the overall Norra Kärr intrusive mineralized rock body grades and if contamination issues were significant then the error would likely show up in the field duplicate results.

10.4.5 Certified Reference Material ("CRM")

LEM has carried out extensive checks with regards to the accuracy of the analytical methods used on the Norra Kärr samples through the use of CRM. CRM samples have been supplied by Ore Research and Exploration Pty Ltd ("OREAS"), a few other purchased CRM have been submitted but only in very low numbers. A summary of the certified values for the relevant CRM as well as the calculated standard deviations is reported in Table 10-2.

As well as using pre-made CRM samples LEM produced two of their own CRM; NKA01 and NKA02 both of which have been certified by OREAS as matrix-matched certified reference material ("MMCRM"). The two MMCRM represent different TREO concentrations with NKA01 having a TREO grade of 0.4% and NKA02 having a grade of 0.6%. Both MMCRM were produced from crushed reject material from the sample preparation stage of the original core samples. To certify the samples ten commercial laboratories took part in a round robin of lithium borate fusion ICP-MS test work with five 20g samples being submitted to each laboratory per MMCRM. The certified values and the calculated standard deviations for each tested element are shown in Table 10-2 below.

The CRM results show a good level of accuracy with the majority of results for both the OREAS CRM and the NKA01 and NKA02 CRM falling within ±2 standard deviations. In the case of CRM OREAS 45c the results for dysprosium show a lower level of accuracy. Given the low Dy grade in CRM OREAS 45c The OREAS 45c Dy CRM results are considered reasonable.

10.4.6 WAI Duplicate Assays

As part of the WAI due diligence processes for the 2015 PFS a total of 14 samples were selected by Greg Moseley (WAI Qualified Person) during the May 2014 site visit, of the 14 samples 8 samples were duplicate core samples, 4 samples were pulverised duplicates, 1 samples was CRM NKA02 and 1 sample was a blank.

A good correlation was exhibited between the original assays and the core and pulverised duplicates particularly with regards to the core duplicates. Pulverised duplicates show lower levels of precision for some elements, but this may be related to the sample not being fully homogenised using the cone and quarter method prior to submission to the laboratory.

10.4.7 QA/QC Summary

The WAI audit of the QA/QC data identified a number of risks within the sample data. These risks are summarised in Table 10-3. It should be noted that WAI considered the QA/QC risk for the Project to be "Low" and that the sample preparation procedures and analytical methods are suitable.

Table 10-3 does not provide a quantitative risk assessment but gives an indication as to where WAI considered the risk lie within the sampling data.

A six-score classification has been employed where:

- 1-2 ('low' risk): Little or no perceived risk, or low uncertainty.

- 3-4 ('moderate' or 'medium' risk): Risk present which could lead to small material error in the resource model; and
- 5-6 ('high' risk): This feature could lead to material error in the resource model (high uncertainty).

Based on review SRK consider the QA/QC risk for the Project to be **"Low"** and that the sample preparation procedures and analytical methods are suitable.

Table 10-2: CRM Sample Summary

CRM ID	Element	Target CRM Grade	±1 SD	±2 SD	Number of CRM Submitted	Mean Assay Grade	% of Assay Results within ±2 SD of Target CRM Grade
OREAS100a	Dy	23.2	0.9	0.18	36	23.04	92
	Y	142	6	12		134.24	78
	Eu	3.71	0.36	0.72		3.88	100
	La	260	13	26		255.64	100
	Nd	152	14	28		149.81	100
	Ce	463	29	58		453.92	100
	Gd	23.6	2.2	4.4		24.96	100
	Tb	3.80	0.34	0.68		3.83	100
	Pr	47.1	4.0	8.0		47.03	100
	Sm	23.6	0.7	1.4		23.86	83
OREAS102a	Dy	18.1	1.0	2.0	50	17.46	92
	Y	105	5	10		101.31	84
	Eu	3.89	0.35	0.7		3.97	100
	La	323	16	32		315.10	92
	Nd	180	16	32		177.45	100
	Ce	587	48	96		563.86	98
	Gd	20.9	1.6	3.2		22.50	40
	Tb	3.05	0.23	0.46		3.13	96
	Pr	58	4	8		56.67	100
	Sm	24.7	1.3	2.6		24.65	100
OREAS104	Dy	7.11	0.14	0.28	85	7.31	58
	Eu	1.29	0.08	0.16		1.30	100
	La	48.8	2.7	5.4		48.38	100
	Nd	49.5	1.4	2.8		50.16	87
	Ce	102	7	14		102.80	100
	Gd	9.4	0.9	1.8		9.09	100
	Tb	1.39	0.14	0.28		1.33	100
	Pr	12.8	0.7	1.4		12.89	100
	Sm	10.5	0.7	1.4		10.82	100
OREAS146	Dy	224	16	32	360	220.29	97
	Y	905	53	106		922.32	98
	Eu	127	9	18		129.28	96
	La	2513	185	370		2527.15	98
	Nd	2182	192	384		2183.37	98
	Ce	4691	360	720		4758.69	99
	Gd	359	23	46		339.96	92
	Tb	47.2	3.4	6.8		45.21	98
	Pr	548	36	72		555.31	99
	Sm	441	36	72		449.92	99
NKA01	Dy	175	9.7	19.4	161	168.13	95
	Y	1131	62.2	124.4		1136.02	99
	Eu	10.8	0.49	0.98		10.93	95
	La	313	24.6	49.2		315.84	99
	Nd	307	18.6	37.2		309.73	99
	Ce	626	58.2	116.4		634.76	100
	Gd	111	7.7	15.4		107.46	98
	Tb	24	1.68	3.36		23.48	100
	Pr	79	3.1	6.2		79.14	92
	Sm	88	2.3	4.6		87.57	81

CRM ID	Element	Target CRM Grade	±1 SD	±2 SD	Number of CRM Submitted	Mean Assay Grade	% of Assay Results within ±2 SD of Target CRM Grade
NKA02	Dy	241	11.9	23.8	162	228.19	85
	Y	1569	94.7	189.4		1659.26	90
	Eu	18.2	0.63	1.26		18.09	83
	La	520	28.2	56.4		517.96	99
	Nd	576	22	44		575.31	89
	Ce	1120	81.2	162.4		1135.18	100
	Gd	176	12.4	24.8		169.17	99
	Tb	34.8	2.12	4.24		33.81	98
	Pr	143	6	12		145.75	88
	Sm	152	3.8	7.6		153.65	73
OREAS 45P	Dy	4.1	0.2	0.4	28	3.94	86
	Y	18	1.4	2.8		17.31	100
	Eu	1.2	0.1	0.2		1.17	82
	La	24.8	1	2		24.31	89
	Nd	21	0.6	1.2		20.75	71
	Ce	48.9	1.6	3.2		47.69	71
	Gd	4.0	0.2	0.4		4.29	75
	Tb	0.69	0.04	0.08		0.69	100
	Pr	5.42	2.1	4.2		5.61	100
	Sm	4.51	0.14	0.28		4.37	57
OREAS 45C	Dy	4.0	0.1	0.2	74	4.03	61
	Y	18.1	1.4	2.8		18.77	100
	Eu	1.18	0.08	0.16		1.19	99
	La	26.2	1.1	2.2		26.35	89
	Nd	22.8	1.7	3.4		22.14	97
	Ce	50.2	1.3	2.6		50.92	65
	Gd	4.2	0.2	0.4		4.09	81
	Tb	0.70	0.04	0.08		0.66	95
	Pr	5.61	0.12	0.24		5.89	30
	Sm	4.7	0.1	0.2		4.74	54

Table 10-3: Risk Matrix: QA/QC Sample Auditing

QA/QC Sampling	Risk	Comments
Field Duplicates	1	The field duplicates display an excellent correlation and level of repeatability. Such results show that the combined sampling, sample preparation and analytical errors are minor and do not materially impact the precision of the samples.
Lab Duplicates	1	As with the field duplicates the laboratory pulverised duplicates display an excellent level of precision showing that the sample material following pulverisation is homogenous and that the laboratories analytical systems are precise.
Blanks	3	LEM used a granite blank for their QA/QC procedures due to state requirements. The results of the analysis of the blanks shows that there are low grades of mineralization above the analytical detection limit. Whilst the blank material has displayed mineralization WAI believes this is natural levels of mineralization within the granite rock rather than a contamination issue at the sample preparation facility. The blank sample results shows a fairly uniform trend, if there was a contamination issue WAI would expect to see a more erratic distribution of results. If there were contamination issues at the sample preparation stage such errors would be exhibited in the field duplicate samples as the field duplicate results are a sum of the sampling, sample preparation and analytical errors. The high level of correlation in the field duplicates therefore supports the view that the grades in the blank samples are natural grades within the granite.
Certified Reference Materials	2	LEM submitted a number of CRM included CRM samples bought from Ore Research and Exploration Pty Ltd (OREAS) as well as two CRM (NKA01 and NKA02) which LEM have developed from crushed sample material from Norra Kärr which has been subject to round robin testing and has subsequently been certified by OREAS. The CRM results for a range of elements and at a range of grades shows a good level of analytical accuracy at the laboratory.

QA/QC Sampling	Risk	Comments
WAI Check Assays	2	WAI chose 14 duplicate samples during their May 2014 site visit by Greg Moseley (Qualified Person) for submission to the ALS Chemex laboratory for verification purposes. From the 14 samples a total of 8 core duplicates were submitted showing a reasonable level of repeatability to the original assays. A further 4 pulverized duplicates were submitted and showed a lower level of precision although this may be attributable to the cone and quarter method of mixing the pulverized sample not adequately homogenizing the samples. Whilst the samples show some deviation WAI does not consider it material and is of the opinion that the LEM assays display both precision and accuracy.
External Duplicates	1	WAI was provided with external pulverized pulp sample duplicate results for the 2010-2012 exploration works. A total of 198 duplicate samples were submitted to the ACME laboratory in 2010-2011, and a further 323 were submitted in 2012 to Actlabs. For all the key payable elements the level of precision was shown to be excellent with a good correlation between the duplicate assays and the original ALS Chemex assays.
Overall Rating	**2**	**Low Risk**

10.5 Density Determination

A total of 1,692 bulk density tests have been carried out by LEM with the resultant database supplied to WAI. The database contains the recorded drillhole ID, depth from and to, dry and wet weights as well recording the lithological rock type. Density values range between 2.2 t/m^3 and 3.4 t/m^3 and average 2.7 t/m^3.

10.6 SRK Comment

Although the above QA/QC analysis indicates no material issues, this accounted for analysis of the major REE elements only. No QA/QC assessment of Zr, Nb or nepheline syenite data was completed.

11 DATA VERIFICATION

The following chapter has been taken from the 2015 PFS prepared by WAI and GBM in the PFS report (Davidson *et al*, 2015b) and has been modified by SRK to ensure it reflects the current status of the Project.

11.1 Introduction

SRK visited the Norra Kärr project in June 2021, including the logging facilities, drill core, surface exposures as well reviewing and discussing the geology, exploration works, testwork and QA/QC procedures with LEM personnel. The following aspects were inspected during the visits:

- Geological and geographical setting of the Norra Kärr alkaline intrusive complex;
- Extent of the exploration work completed to date;
- Inspection of the core logging, sampling and storage facilities;
- Inspection of the core and a review of the logging procedures;
- Review of the sampling and sample preparation procedures;
- Discussions with the geological staff regarding geological interpretation; and

IWAI with LEM undertook a number of steps to verify the quality and robustness of the sample data. The data verification is split into three sections:

- Field verification;
- Sample treatment; and
- Laboratory treatment.

SRK has reviewed this information and is agreement with the findings of WAI audit.

11.2 Field Verification

11.2.1 Surface Mapping

Some aspects of the field verification, e.g. surface geological mapping, are covered in Section 8.3.1 above, where the location of outcrops and old Boliden excavations was noted and the trajectory of WAI's site visit illustrated. WAI visited a considerable number of the important outcrops and photographs of these some of the sites are shown in Figure 11-1 to Figure 11-4.

Figure 11-1: LEM Geologists at a Typical Norra Kärr Outcrop

Figure 11-2: Site of Former Boliden Excavation

Figure 11-3: "Migmatitic" Grennaite (GTM Domain)

Figure 11-4: "Discovery" Outcrop Showing Pink Eudialyte

11.2.2 Drillhole Surveys

During the May 2014 site visit, WAI undertook a review of drillhole collar positions. This was achieved during the various traverses undertaken as part of the surface mapping discussed previously. Approximately 65% of the more than 100 drillhole collar sites were visited and their positions confirmed by GPS.

While there is a consistent small error – in general less than five metres – this can be put down to "experimental error" and the conversion used from the GPS reading to the SWEREF99TM projection via the Global Mapper (V15.2) software utilised by WAI. This error is not considered significant, and the collar positions were accepted by WAI as being accurate for all practical purposes.

11.3 Sampling Verification

WAI considered that the sampling and assay information to be reliable. WAI chose 8 core sample duplicates and 4 pulverised sample duplicates for assay (see Section 10.4.6). The results of the WAI duplicate samples shows support for the robustness of the sample data on which the Mineral Resource estimate reported in Section 13 is based.

The plastic bags containing the samples where then packed in cardboard boxes and sent by bus to the ALS Chemex preparation laboratory in Piteå in the north of Sweden prior to being sent to the ALS Chemex laboratory in Vancouver.

The sample quality is considered to be adequate in all respects – from the logging of the core (reviewed above and in the data base considerations below) to the core splitting and sample despatch to the ALS Chemex facility in Piteå. The handling of the core and the treatment and despatch of the samples by LEM has fully conformed to industry "best practice" and WAI was satisfied with all these aspects.

WAI discussed core and sample handling procedures with LEM's key geological and technical personnel. On the basis of these discussions, WAI believed that all split core was well and securely packed and stored prior to transportation to the laboratory for processing. As a result, PAH considered sample security to be adequate.

WAI also understands that other than the core splitting procedure at no time was an officer, director or associate of LEM involved in the sample preparation or analytical work and an independent laboratory was employed for sample preparation and analysis. It was therefore WAI's belief that it is highly unlikely that an officer, director or associate would have had the opportunity to contaminate the sample data.

11.4 SRK Comment

The exploration data has been collected and quality controlled using industry standard methods, SRK agrees with the findings reached by WAI, that the data is sufficiently robust and detailed to be used for an MRE.

12 MINERAL PROCESSING AND METALLURGICAL TESTING

12.1 Introduction

The following chapter has been taken from the 2015 PFS prepared by WAI and GBM in the PFS report (Davidson *et al*, 2015b) and has been modified by SRK to ensure it reflects the current status of the Project.

The original mineral processing and hydrometallurgical plant design were prepared based on information generated by the testwork conducted by, JKTech, Geological Survey of Finland, Dorfner ANZAPLAN and ANSTO Minerals as well as design experience provided by GBM. This was published previously (Davidson et al 2015 a,b). The previous process design was considered preliminary in nature and subject to change based on additional testwork programmes.

The design philosophy for the process plant was a zero liquid discharge policy to reduce environmental risks. A key consideration of the project was the proximity to sensitive environmental receptors. Much of the work relied on in this PEA was supplemented with work undertaken in the European Union Rare Metal Research (EURARE) research program.

12.2 Historic Testwork

Relevant pre-PFS technical material generated as part of historical programmes has previously been reported in the PEA prepared by PAH (Gates *et* al, 2012) with subsequent work reported by Davidson et al (2015).

SGS Lakefield conducted limited leach testing on whole mineralized rock employing sulfuric acid. The tests demonstrated the efficacy of the method however the acid consumption was uneconomic. Beneficiation work was subsequently undertaken by GTK employing both negative flotation and magnetic separation either separately or in combination. The programme confirmed that the eudialyte could be beneficiated by means of magnetic separation. The flotation work demonstrated that aegirine could be also separated from the eudialyte, subject to sufficient liberation, by means of negative flotation whereby aegirine was selectively floated from the remaining mineral mass. Finally, hydrometallurgical testing was undertaken by John Litz and Associates where tests employed both sulfuric and hydrochloric acid leaching on mineral concentrate prepared by GTK where a majority of acid consuming gangue minerals were rejected. This further confirmed the efficacy of sulfuric acid leaching and demonstrated an improved acid consumption profile compared with the whole mineralized rock treatment.

12.3 Recent Testwork

Testing completed since the 2012 PEA covers quantitative mineralogical characterisation, beneficiation and hydrometallurgical campaigns. Two primary material composites were prepared to undertake various aspects of the testing programmes. The first composite was prepared in April 2012 to facilitate investigative testing of various beneficiation methods at the research facilities of Dorfner ANZAPLAN (ANZAPLAN, Germany). The second composite was prepared in November 2012 to facilitate a broader testing campaign developing from the ANZAPLAN work and for the preparation of mineral concentrate for further hydrometallurgical testing.

The methodology and results are discussed in detail in the 2015 PFS (Davidson *et al*, 2015b) and are not repeated herein.

12.4 GBM Proposed Process in 2015

This section summarises the proposed process in the 2015 study and is provided here for completeness. Based on the work reported by Davidson et al (2015) the process approach initiated with comminution involving conventional crushing and grinding. Very little variability work was completed. Extensive analysis and modelling of the magnetic beneficiation process was undertaken and indicated the importance of minimising the production of fine material less than 20 μm as recovery performance deteriorates significantly. Equally important is the reduction of particles larger than 100 μm as these particles carry much of the acid consuming gangue into the mineral concentrate. It was therefore recommended by GBM that a wet screening process be employed in the grinding circuit to minimise over grinding and eliminate large particles (Davidson et al 2015).

Continuous piloting of the hydrometallurgical processes (leach, neutralisation, solvent extraction and precipitation) was recommended to fully determine the product quality, deportment of radionuclides and improve the accuracy of the utility demand and reagent consumptions. The implementation of the continuous testing is also required for optimization of the process s.

No integrated testing was been performed on the raffinate management system and this is also required for future feasibility study to confirm its feasibility as well as accurately estimate utility and reagent costs.

12.5 SRK Conclusions

Davidson et al (2015) proposed an acceptable approach to REE separation and concentration for the Norra Kärr project. However, due to developments made under the European Commission co-funded research project EURARE and a change in REE pricing and environmental considerations for the project there has been a need to revise this approach.

Based on the additional testwork described below, SRK proposes the recoveries given in Table 12-1) and considers the conceptual process will provide saleable products at a suitable level of purity whilst providing the most sustainable approach to the exploitation of the Norra Kärr mineralized rock.

Table 12-1: Proposed Step Recoveries, Norra Kärr project

Mass Balance	Magnetic Separation	Leach Recovery	Solution Separation	Overall Recovery
Ce_2O_3	93%	91%	99%	84.1%
Dy_2O_3	93%	91%	99%	84.1%
Er_2O_3	93%	91%	99%	84.1%
Eu_2O_3	93%	91%	99%	84.1%
Gd_2O_3	93%	91%	99%	84.1%
Ho_2O_3	93%	91%	99%	84.1%
La_2O_3	93%	91%	99%	84.1%
Lu_2O_3	93%	91%	99%	84.1%
Nd_2O_3	93%	91%	99%	84.1%
Pr_2O_3	93%	91%	99%	84.1%
Sm_2O_3	93%	91%	99%	84.1%
Tb_2O_3	93%	91%	99%	84.1%
Tm_2O_3	93%	91%	99%	84.1%
Y_2O_3	93%	91%	99%	84.1%
Yb_2O_3	93%	91%	99%	84.1%
ZrO_2	86%	65%	87%	48.6%
Nb_2O_5	93%	91%	96%	81.6%

Comminution with conventional crushing and grinding techniques will be suitable for the process. Further testing on domain specific material should be undertaken to determine the variability particularly with respect to nepheline syenite, hafnium, niobium and zircon.

Integrated testing needs to be performed during the next phase of work once the various steps have been fully assessed to optimize the process as a whole and generate representative samples of products and wastes for further evaluation.

12.6 Geometallurgy

The main focus for mining is a grennaite lithological unit that is an aegirine-rich nepheline syenite. This unit hosts the main economic mineral, eudialyte ($Na_{15}Ca_6(Fe,Mn, Ti, Cr,Y,REE)_3(Zr,Ti,Nb)_3Si(Si_{25}O_{73})(O,OH,H_2O)_3(Cl,OH)_2$) or more correctly the formula can be broken out as (Johnsen et al 2003);

$[N(1)N(2)N(3)N(4)N(5)]_3[M(1.1)M(1.2)]_3[M(2.1)M(2.2)M(2.3)]_{3-6}[M(3)M(4)]Z_3[Si_{24}O_{72}]$
$\Theta_{4<}$b>X(1)X(2)

N(1–5) = Na, H_3O^+, K, Sr, REE, Ba, Mn, or Ca;

M(1) = Ca, Mn, Y, REE, Na, Sr, or Fe;

M(2) = Fe, Mn, Na, Zr, Ta, Hf, Ti, K, Ba, or H_3O^+

M(3) and M(4) = Si, S, Nb,Ti, W, or Na;

Z = Zr, Ti, or Nb;

Θ = O, OH, or H_2O;

X(1) and X(2) = Cl, F, H_2O, OH, CO_3, SO_4, AlO_4, or MnO_4.

The structural and chemical complexity of the mineral perhaps reflects challenges with processing.

Apart from eudialyte there are also some other Zr and REE minerals that will be recovered in magnetic separation, these are present as mineral inclusions in aegirine and eudialyte as well liberated grains such as:

- Jinshajiangite $BaNaFe^{2+}_4(Ti,REE,Y,Zr)_2(Si_2O_7)_2O_2(OH)_2F$
- Låvenite $(Na,Ca)_2(Mn^{2+},Fe^{2+})(Zr,Ti)(Si_2O_7)(O,OH,F)_2$
- Mosandrite $Ca_3REE[(H_2O)_2Ca_{0.5}\square_{0.5}](Ti,Fe)(Si_2O_7)_2(OH)_2(H_2O)_2$
- Rinkite-(Ce) $(Ca_3CeFe)Na(NaCa)Ti(Si_2O_7)_2(OF)F_2$
- Rosenbuschite $Na_6Ca_6Zr_{3(}Ti,Fe)(Si_2O_7)_4O_2F_6$

There will also be some that are non-magnetic and will be lost in magnetic separation and be concentrated in the nepheline syenite waste and these include;

- Catapleiite $Na_2Zr(Si_3O_9) \cdot 2H_2O$
- Zircon $Zr(SiO_4)$

Of these catapleiite is the only one of material importance and it hosts approximately 50% of the Zr (and presumably Hf) in the deposit.

In terms of metal concentrations an example of the typical head grade (based on metallurgical samples) is shown in Table 12-2. All target elements show significant enrichment from average crustal abundance (CRC 2016-17). Of these the most pronounced enrichment is for zirconium, yttrium and dysprosium.

From previous work reported by GBM the main economic units were demonstrated to have consistent mineralogy with only minor variations observed (Figure 12-1). As such all units can be processed alongside each other.

Table 12-2: Head Grade of Mineralized rock concentrate from ANSTO testwork, Norra Kärr project (ANSTO, 2012)

Element	Concentration (%)	Enrichment vs Crustal Abundance
Erbium	0.039	111
Holmium	0.013	100
Ytterbium	0.035	100
Zirconium	1.508	91.4
Yttrium	0.2005	60.8
Niobium	0.1	50
Dysprosium	0.02532	48.9
Terbium	0.00371	30.9
Gadolinium	0.01825	29.4
Samarium	0.01647	23.4
Cerium	0.1156	17.4
Praseodymium	0.01504	16.4
Neodymium	0.06025	14.6
Lanthanum	0.0519	13.3
Europium	0.00203	10
Hafnium	0.0025	10
Thulium	0.005	10
Lutetium	0.00227	2.84

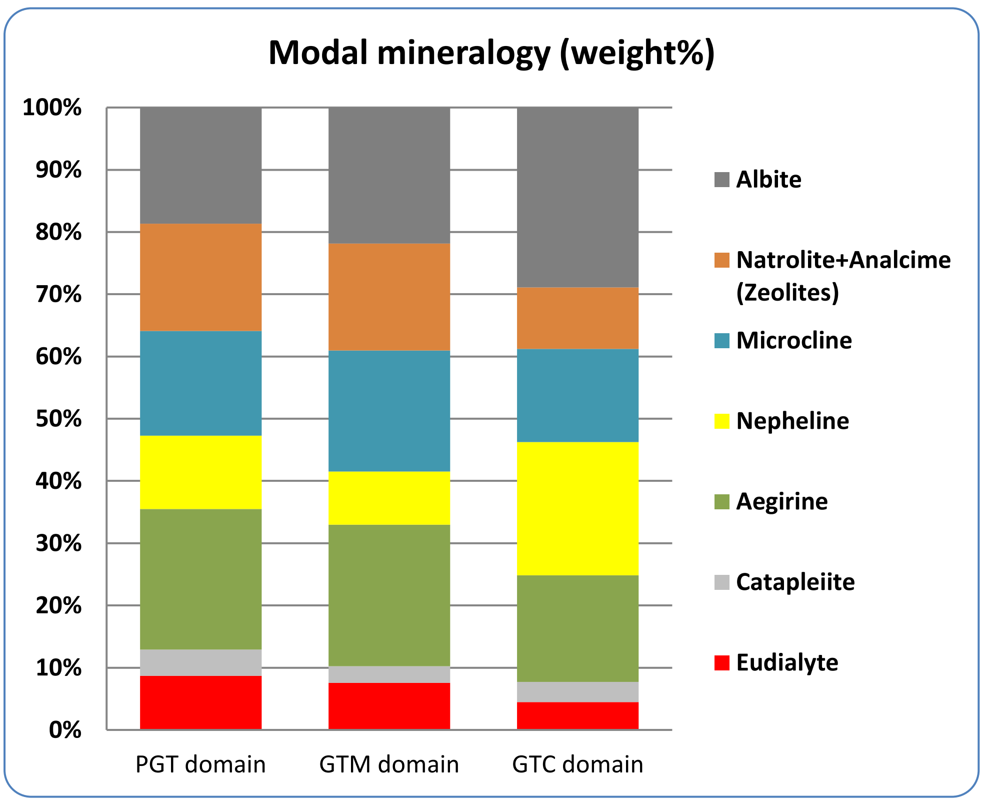


Figure 12-1: Model mineralogy by ANZAPLAN, reported by Forrester et al (2015)

The mineralogy studies demonstrated that eudialyte was well liberated and as such should have good separation through the proposed comminution routes.

12.7 Testwork on Comminution

LEM completed several studies utilizing magnetic separation, flotation and gravity (Davidson et al 2015). Based on the work undertaken by GTK (Maksimainen, 2012), magnetic separation using low and then high-grade material separation appears to be the best approach.

Mineralogical analysis of crushed samples identified eudialyte separation at 95% or more in material crushed to less than 200 µm fractions (p80 ~ 145 µm). Testing by LEM focussed on optimization of magnetic beneficiation testing with the aim of maximising the recovery of eudialyte whilst rejecting acid consuming gangue. The Metso high grade material separation ("HGMS") was identified as the superior technology to recover eudialyte and reject gangue material using multi-stage magnetic separation.

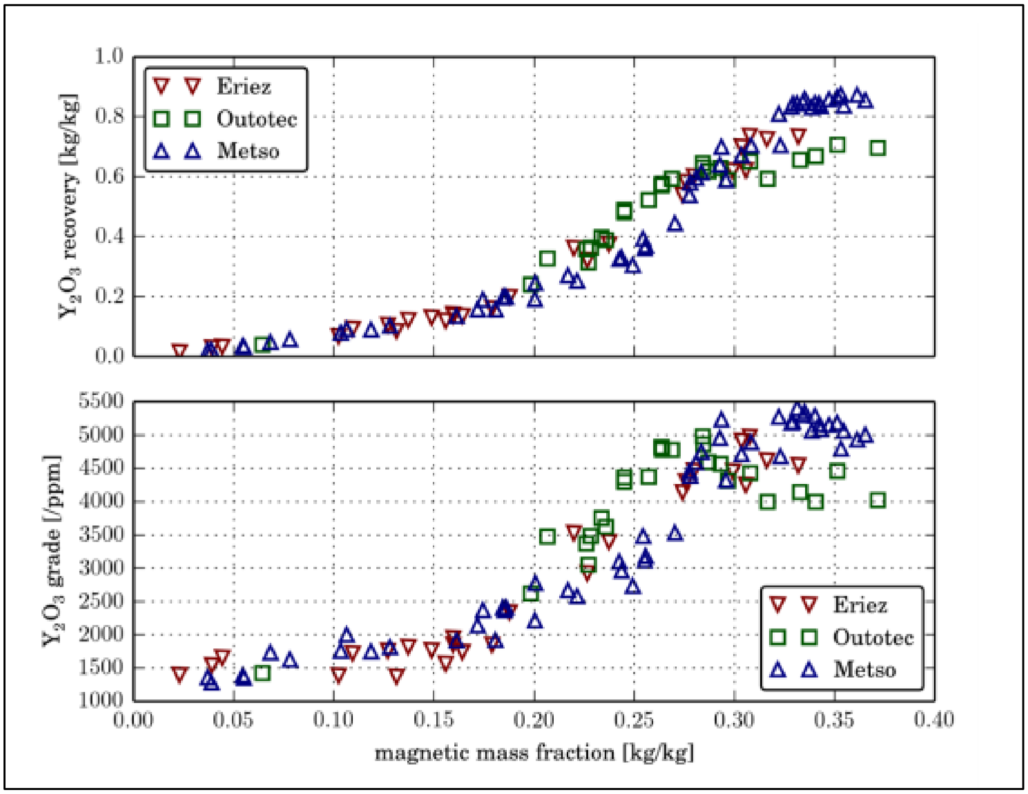


Figure 12-2: Yttrium grade and recovery by Magnetic Separation (Forrester et al 2016)

The best separation with overall recovery of REE at more than 95 to 99% utilized a multi-stage magnetic separation approach (GTK, 2012, test 18). In this test by Metso, was also verified in later work by GTK using a two-stage approach essentially undertaking a rougher and cleaner stage separation, three magnetic separations and re-grinding for the first magnetic concentrate. In the first magnetic separation the field strength was 1.96T field with XRO -matrix. After 15 min re-grinding 0.3T and 1.0T fields were used with XMO -matrix. The block diagram of test 18 is presented in Figure 12-3.

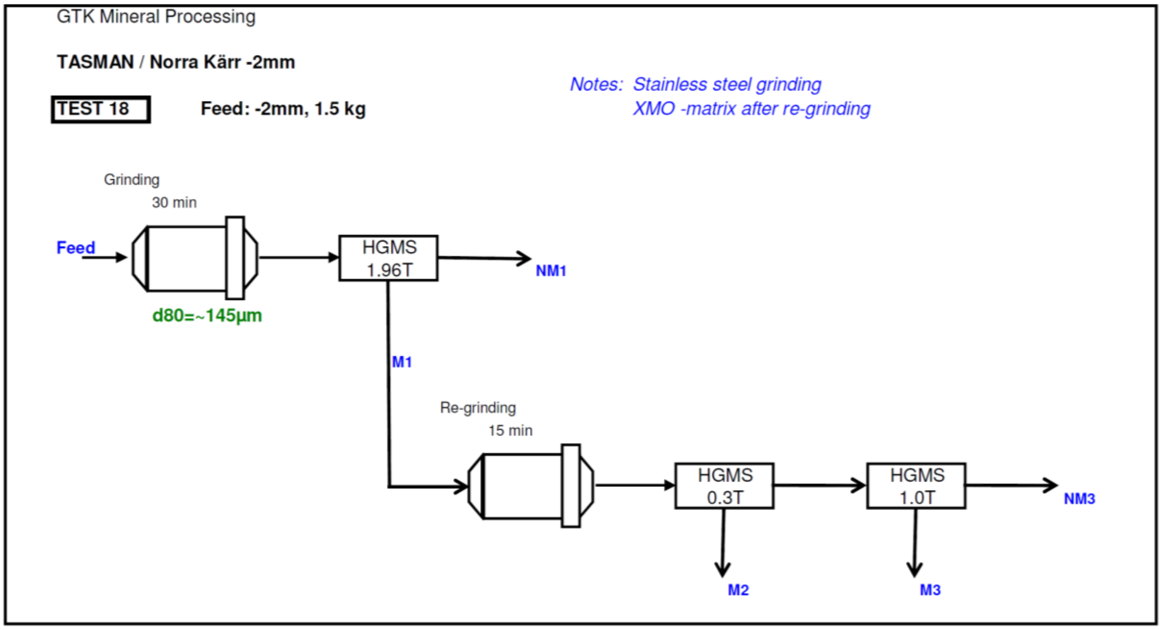


Figure 12-3: Block diagram of magnetic separation process (GTK, 2012)

This could be replaced by two stages of magnetic separation at high field strength. For purpose of the mass balance, 95% recovery to the magnetic concentrate assuming high gauss (1.96T) was assumed. Later optimization work by Stark et al (2017) reported an improvement up to 99% recovery of eudialyte. In terms of Zr and Hf only 60% was recovered and approximately 88% of Nb (Stark *et al* 2016).

Flotation was tested and should produce similar results of eudialyte concentration there will be some loss of eudialyte to aegirine concentrate. In order to streamline the processing on site the proposal is to move forward with a two-stage magnetic separation at high gauss.

12.8 Testwork on Metal Leaching

The pyro- and hydrometallurgical methods developed for the eudialyte concentrates have several major challenges, the main hydrometallurgical challenge is the presence of significant (50 wt.%) silica content (Johnsen et al. 2003; Zhang and Edwards 2013; Balomenos et al. 2017; Karshigina et al. 2018; Ma et al., 2018, 2020). During acid leaching, silicon passes into solution in the form of silicic acid, which undergoes a polycondensation process with the formation of gels of various composition and structure. It reduces the recovery of valuable components, decreases the filtration performance, and complicates the further processing of solutions by extraction and sorption-based methods (Jha et al., 2016; Ma et al., 2018, 2020). The critical issue therefore is suppression of silica gel formation to avoid encapsulation of REE. Different approaches are proposed for the solution of the silica gel formation problem.

In the 2015 amended PFS study, ANSTO studies were utilized to assess mineral solubility, silica solubility and silica stability as well as monitoring the concentration of deleterious elements in preparation of a pregnant leach solution ("PLS") to be used for the recovery of REE (testwork by ANSTO and reported in Davidson et al 2015b). Diagnostic leach tests were conducted to determine the minimum acid activity required to dissolve eudialyte. It was determined that a relatively mild acidic environment, around pH 1, was required to dissolve the eudialyte whilst leaving intact most of the accompanying minerals. Additionally, tests were conducted at various temperatures to determine the impact on leaching kinetics and pulp handleability. The results of which indicated that a leach temperature of 30 °C is sufficient to maintain satisfactory leach kinetics and pulp condition. Follow-up tests further examined the mineral solubilities at constant acidities, around the determined range, at various pulp densities to determine the operating limits for silica in solution. When operating at higher pulp densities a threshold limit was exceeded whereby the silica would precipitate from solution causing the pulp to gel. The threshold limit of silica in solution was quantified resulting in an optimal pulp density being determined as approximately 14 % solids on a weight basis. A sub-set of the leaching results are shown in Figure 12-4 that show the leach acid strength has a significant control on metal leaching from the mineralized rock .

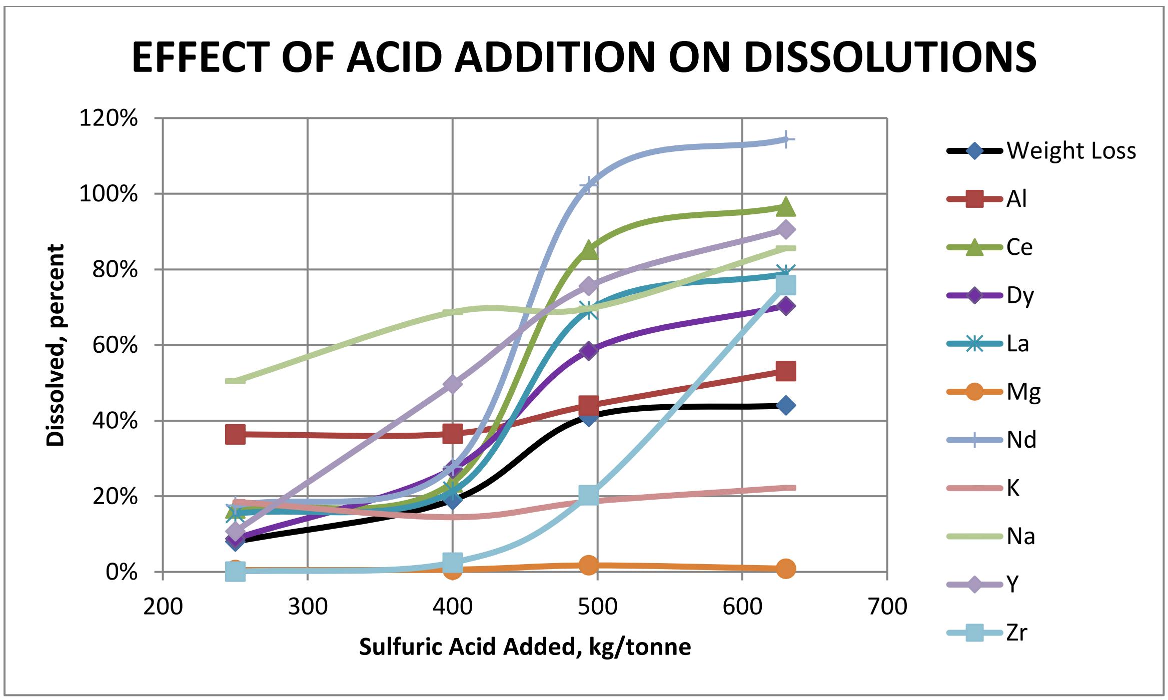


Figure 12-4: Influence of sulfuric acid on leaching of various metals from Norra Kärr mineralized rock (from GTK database, GTK, 2011).

From large scale leach tests the addition of solids to the leach showed greater rate of leaching with the higher addition of solid Figure 12-4. The leaching above 100% for Nd indicates variance in concentration of Nd between head analysis and the sum of the leachate concentrations.

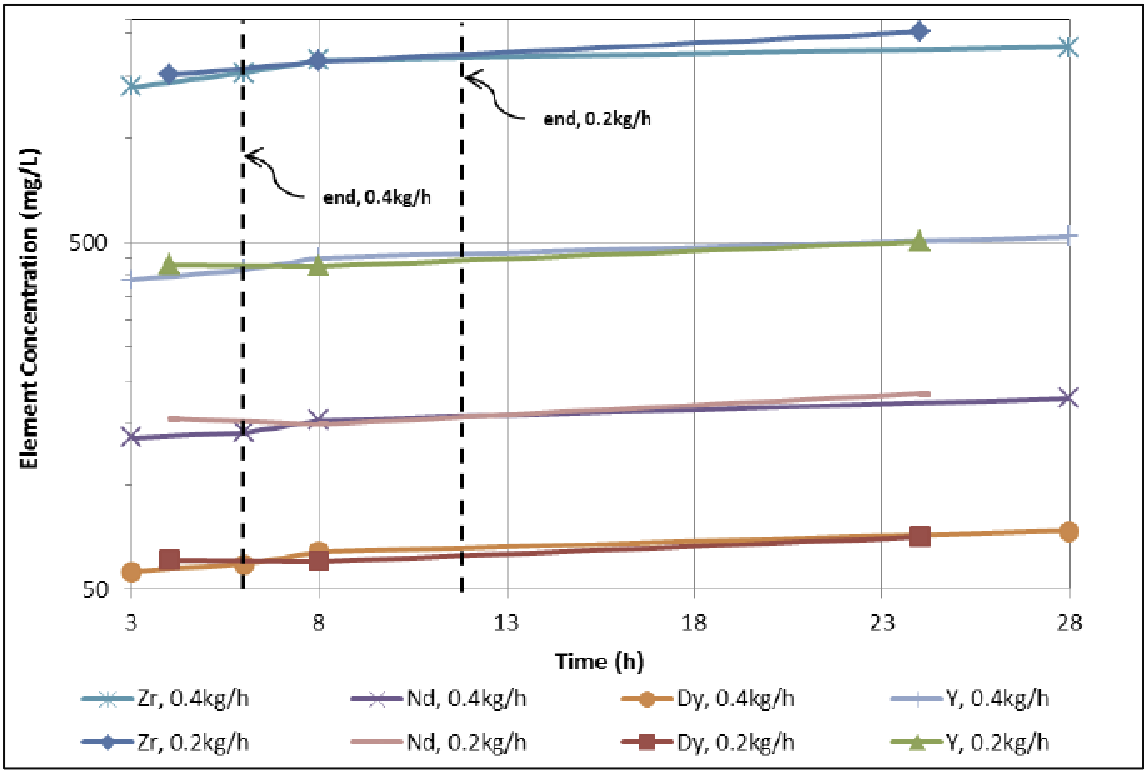


Figure 12-5: Selected elements concentration during leaching at variable solid addition rates (ANSTO, 2014)

The more rapid addition of solids resulted in slightly lower acid consumption, presumably due to lower time for gangue acid interaction Figure 12-6.

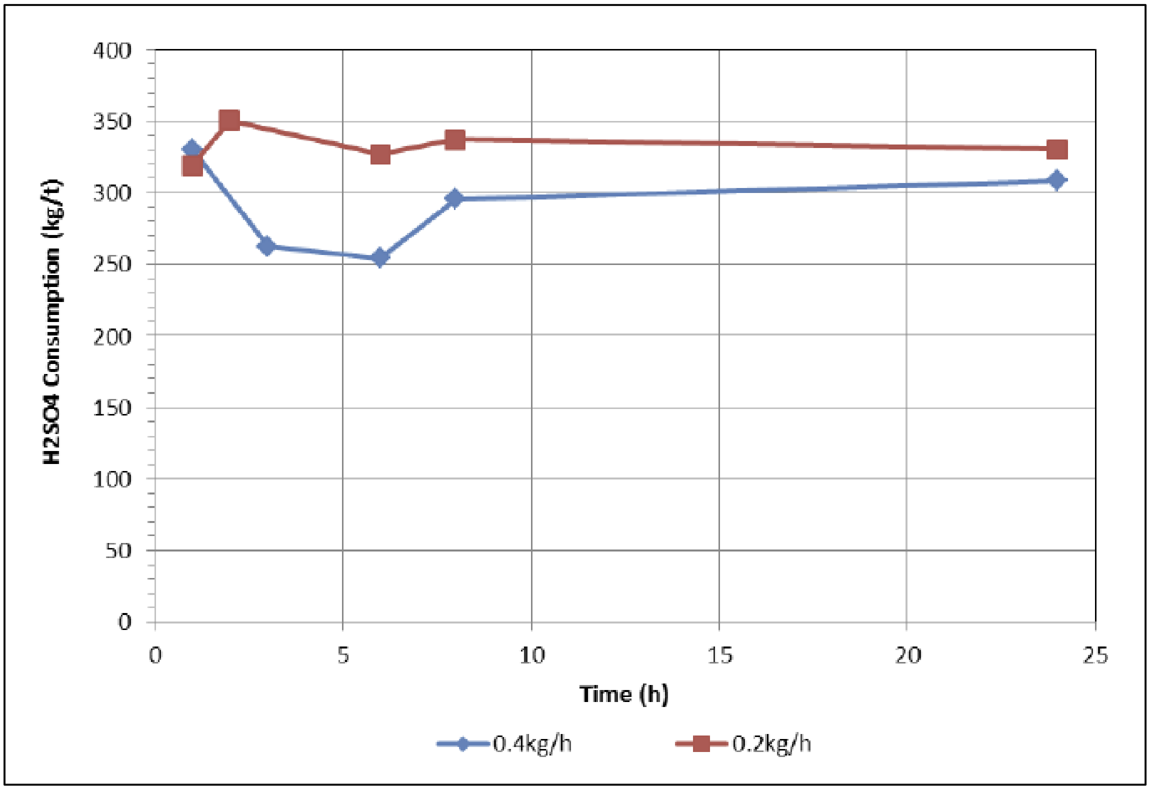


Figure 12-6: Acid consumption during leaching at variable solid addition rates (ANSTO, 2014)

In addition, the consumption at various pH values and pulp densities. It is also evident that most of the acid consumption occurs in the first two hours of the leaching process. This correlates strongly with the leach extraction of REE into solution where extraction efficiencies were routinely in excess of 90 % Figure 12-7

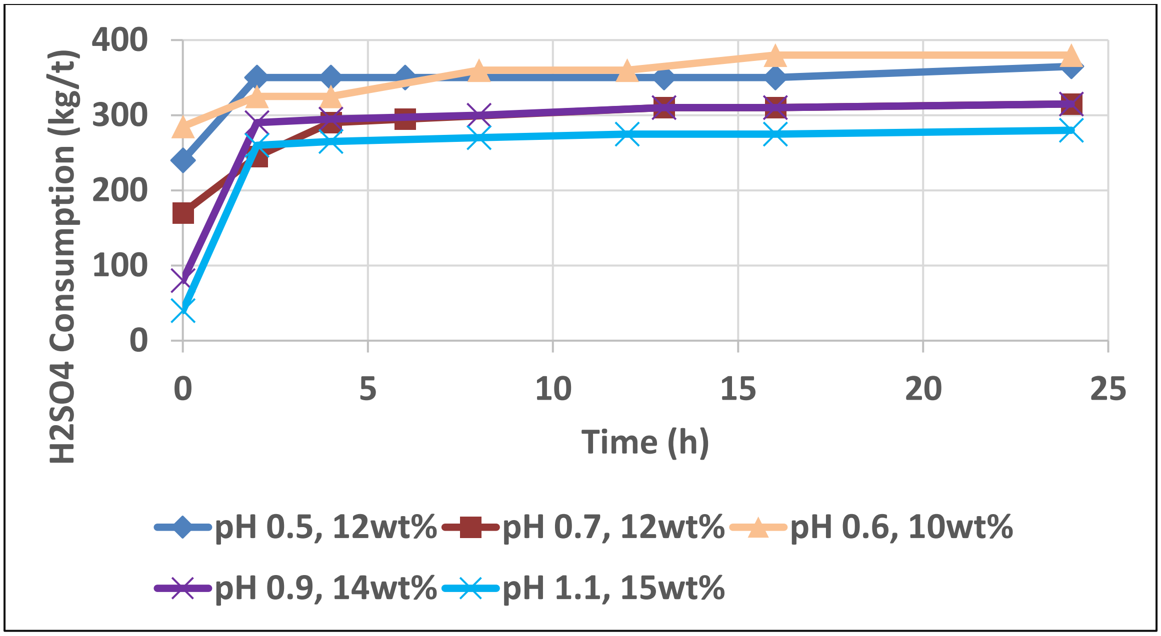


Figure 12-: Acid Consumption as a function of pH, time and pulp densities (source: Davidson et al 2015)

Further leaching with concentrate produced by GTK using an optimized beneficiation method focused on the rejection of gangue resulting in the acid consumption decreasing to approximately 250 kg/t (per tonne concentrate). Although the concentrate was prepared from the improved method further mineralogical analyses indicate further improvements in gangue deportment are possible with improvements in the particle size control, in particular the removal of particles greater than 100 µm as these contribute most gangue minerals as identified in the mineralogical studies (section 12.6).

The formation of silica gel can be avoided by applying a combination of mechanical comminution, reagent and timing approaches including the following.

- Energy treatment (ultrasonic ("US"), high-power nanosecond electromagnetic pulses ("HPEMP") and electrochemical ("ETC")) and mechanochemical ("MA") of the mineral suspensions, which improves the unlocking and breakage performance of fine-grained mineral complexes and the recovery of micro- and nano-sized particles of non-ferrous and precious metal (Chanturia et al. 2017, 2018a, 2018c, 2019b; Bogatyreva et al. 2018a, 2018b; Khokhlova 2018);
- Extraction of REE with nitric acid/tri-n-butylphosphate (Chizevskaya et al. 1994);
- Extraction of REE with hydrochloric acid (Ma et al 2019)
- Introduction of F− ions during leaching (Dibrov et al. 2002; Litvinova and Chirkist 2013)
- Two-stage hydrometallurgical process (Davris et al. 2017), including preliminary acid treatment of the heated concentrate followed by weak acid aqueous leaching. rare earth elements and suppressing silica gel formation,

The proposed hydrometallurgical step will incorporate a pre-processing step called ''fuming", which is the same as "acid pugging" and involves the addition of a sulfuric acid solution to eudialyte concentrate heated at boiling temperatures, followed by the leaching step, where the treated concentrate is leached at ambient or low temperature (Davris et al 2017). Fuming pre-treatment is with sulfuric acid, and subsequently water or weak acid leaching of the treated concentrate, resulted to >90% recovery of rare earths avoiding silica gel formation.

An initial step in the assessment of hydrochloric acid leaching was undertaken in 2014 (RPC, 2014). The study found that it is possible to depress silica gel formation. Utilizing $AlCl_3$ at 90 °C for 12 hours, 65.0 % total REE and Y (TREE +Y) extraction was obtained while 99.9 % of the Si remained in the residue. A mineralogical analysis of the final residue after 12 hours of $AlCl_3$ leaching at 90°C indicated that rare exceptions of silica gel was present but mostly the silicate grains and lithic roc fragments retained their sharp grain and fragment boundaries, without evidence of weathering or corrosion.
Additional work on this was undertaken through the EURARE study whose goal was to examine the interactions between the eudialyte concentrate minerals and the solutions of nitric, sulfuric, and hydrochloric acids conventionally used in leaching.

In addition, the aim was the investigation of various energy impact effects on the processes and quantitative properties of silicate gel formation and its absorption of valuable components from pregnant solutions. Moreover, the final task was to present the optimal process conditions for the recovery of Zr and REE at a minimal loss with silica gel based on the obtained results (Davris et al., 2016; Balomenos et al., 2017). Much of this work was undertaken on Norra Kärr mineralized rock.

It is proposed to modify the leach circuit to that proposed by Davris et al (2017) and apply a two-stage acid leach process to reduce impact of silica gel. This avoids the addition of magnesium oxide and improves REE leaching to over 90%. By this process, sulfuric acid is added to a heated concentrate at 110ºC, S/L ratio of 1/4 followed by water leaching of the treated concentrate at ambient temperature, S/L ratio 1/20 for 30 minutes, resulting in 91% REE recovery. This will result in high gypsum production but is considered to have more confidence at this stage than hydrochloric acid leach circuit that is likely to provide a higher REE recovery (97% or more) but at significantly higher acid consumption cost. The use of hydrochloric acid would also reduce silica gel formation potential but requires further evaluation.

Utilization of hydrochloric acid would result in 97% recovery of REE but the recycle although possible is yet to be proven or tested. Both hydrochloric acid and nitric acid should be evaluated in the next phase of work.

The resulting solution can be further treated for REE purification and separation by known methods. The proposed process flowsheet can achieve high REE recovery yields with negligible Si dissolution avoiding silica gel formation and minimizing the reagents input in the process.

Initial leaching of Zr and Hf will result in approximately 60% leaching during the two-stage process described above and approximately 80% Nb (Davris et al 2016, 2017). An additional leaching step would recover further metal content with sulfuric acid and possibly addition of fluoride ion (Litvinova and Chirkist 2013). The additional step will increase liberation of Nb, Zr and Hf from secondary oxides formed to liberate a total of 98-99% of these metals.

The two-stage processing of eudialyte via a dry digestion and subsequent leaching establishes a suitable prevention of silica gel formation with high recoveries of REE, Nb and Zr. The main parameters are the type of acid (HCl is favoured than H_2SO_4 in case of REE recoveries but currently has less demonstration), the concentration of acid ([4 mol/L) and the digestion time ([10 min). With the optimal choice of parameters (main parameters for dry digestion: solid liquid ratio: 1:1.8, acid-concentration: 10 mol/L, reaction time: 2 h) maximum metal yields of 91-95 % REE, and 65-75 % Zr are achieved (Davris et al, 2017; Voßenkaul et al, 2017; 2018).

12.9 Testwork on REE Recovery

The principal method reported in the 2015 PFS to recover the rare earth elements from the PLS was by means of solvent extraction (GBM, 2015). ANSTO Minerals conducted the preliminary definition work using a mix of pregnant leach solutions generated as part of the leaching and neutralisation definition programmes. This feed stock was deemed satisfactory for determination of operating conditions and equilibrium and initial engineering of the solvent extraction plant. Phase disengagement studies determined that an elevated operating temperature was required to ensure satisfactory phase disengagement behaviour within the system. Kinetic loading studies were also conducted which determined the rate in which equilibrium was attainment was faster than the phase separation kinetics therefore the system design will be dependent upon phase separation kinetics.

Equilibrium loading isotherms were generation by ANSTO utilising a primary amine solvent to determine its loading capacity and the likely number of equilibrium stages required to extract the REE's from the aqueous PLS into the organic solvent. The solvent preferentially loaded the light REE's relative to the heavy REE's where yttrium had the lowest affinity (see Figure 12-7).

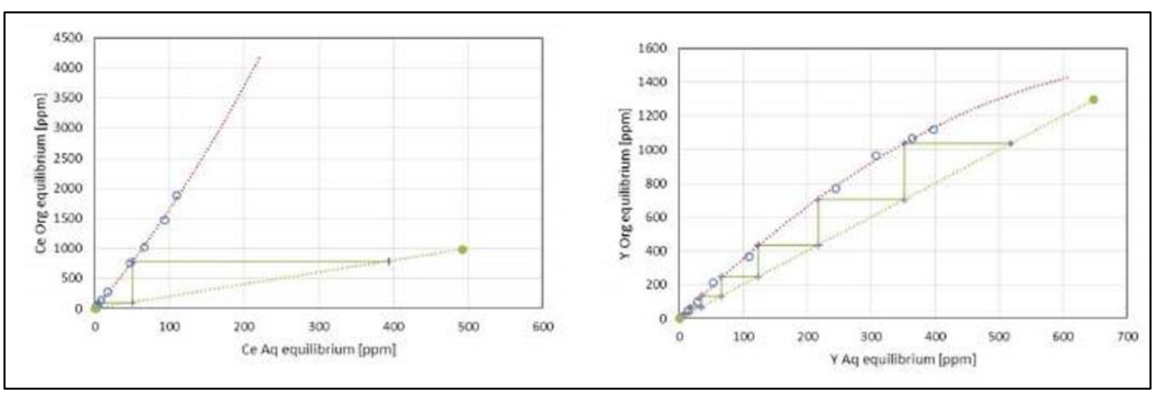


Figure 12-7: Cerium and Yttrium equilibrium loading isotherm (source, ANSTO, 2012)

The equilibrium loading work was successful in enabling the determination of the organic to aqueous ratio as well as the number of stages to affect an acceptable level of extraction. The REEs were stripped from the loaded solvent with various concentrated strong acid solutions however the REEs were very strongly complexed with the solvent and resulted in higher acid requirements for sulfuric acid versus hydrochloric acid to achieve satisfactory stripping efficiency.

12.10 REE Precipitation Testwork

Reactive stripping process for REE separation and recovery utilized oxalic acid contacting the loaded organic causing the formation of a rare earth oxalate precipitate at the interface of the two phases precipitate was then separated from the two phases by centrifugation producing a mixed oxalate concentrate. The concentrate has a very high REE concentrate with low impurity concentrate that is saleable as a mixed REE concentrate (Table 12-3).

Table 12-3: REE oxalate concentrate (Davidson et al 2015), Norra Kärr project

Element	Weight, %
La	4.935
Ce	10.750
Pr	1.458
Nd	6.763
Sm	1.650
Eu	0.269
Gd	1.657
Tb	0.351
Dy	2.191
Ho	0.491
Er	1.288
Tm	0.182
Yb	1.138
Lu	0.149
Y	12.210
Impurity	0.171
TREE	45.48
LREE	23.91
HREE	18.00

The balance of the assay mass is the oxalate anion and hydration water from the oxalate salts. SRK proposes that to ensure the efficient separation of REE to a concentrate the specialised solvent, either CY-272 or CY-923 that can be utilized in high acidity solutions will be applied.

Using oxalic acid is not efficient with organic stripping and would not be required with hydrochloric acid leaching and provides further motivation for assessing this in the next phase of work.

12.11 Testwork on Zr, Hf Recovery

For zirconium and hafnium, solvent extraction is employed to separate zirconium and hafnium on a commercial scale although some alternative processes, such as ion exchange, fractional crystallization, electrorefining and vapor phase chlorination have been proposed (Wang et al 2013).

In the GBM (2015) study no collection or separation of Zr and Hf was considered but various solvent options have been proposed for these elements for collection and segregation of the two elements.

Solvent extraction experiments have been performed. Single acidic organophosphorus extractant (D2EHPA,PC88A, Cyanex272, Cyanex 572) and its mixture with TOPO from acid solution in the concentration range from 1 to 4 mol/dm^3 has been assessed in various studies from bench scale to full operational application. Tertiary amines (such as Alamine 336) have also been utilized commercially.

Solvation mechanisms have been verified for the extraction of Zr and Hf. Using single separation has been shown to present the best separation efficiency and the highest separation factor value of 9.8. Mixtures of TOPO and acidic organophosphorus extractants exhibited synergistic enhancement on the extraction of Zr and Hf (Wang et al 2013). Alamine 336 in shellsol/isodecanol has been shown to be an efficient approach (ANSTO 2013b,c; 2014).

A clean zirconium oxide product would be produced.

ANSTO (2013c) assessed the recovery of Zr using Alamine 336. In this work the starting Zr concentration was 0.9 g/L and sulfate 60 g/L. The solution was saturated with respect to Si,

The effect of pH on Zr, Nb and Ti extraction is presented in Figure 12-9. There was no significant extraction of rare earths, Al, Mn, P, Si and Zn. Less than 10% of the Fe loaded. The concentrations of Th, U and Hf in the feed were too low to calculate extractions in these tests but the loading isotherms will give clearer data. Both Hf and U are expected to load.

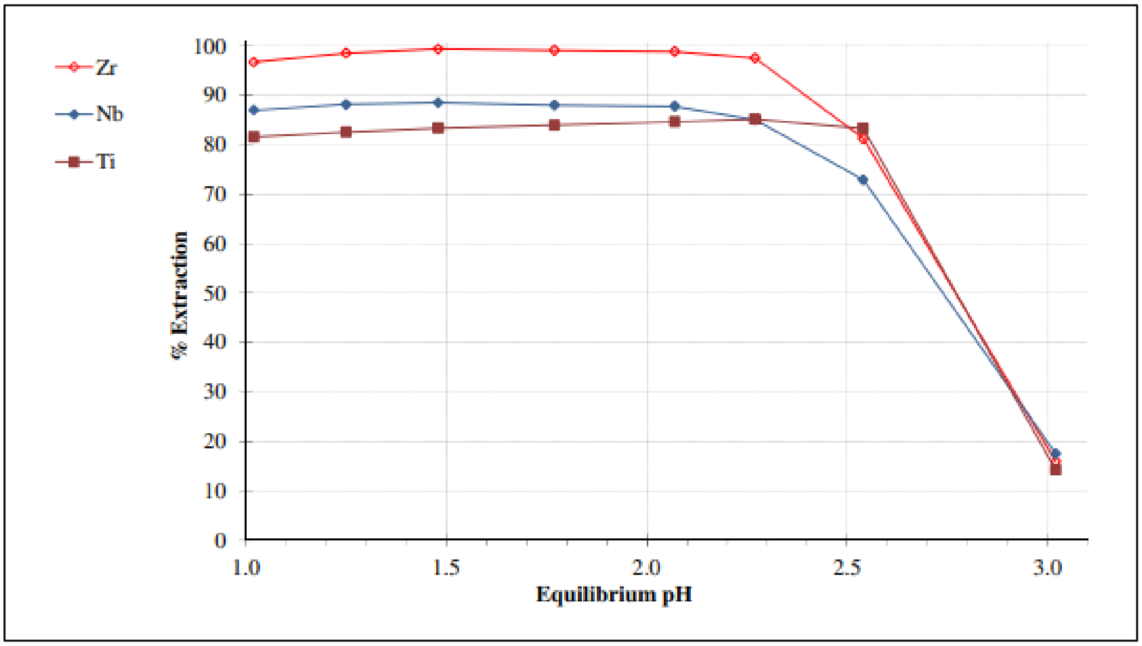


Figure 12-8: pH curve for 10% Alamine 336/5% isodecanol (source, ANSTO, 2013)

The pH profile demonstrated that pH < 2 is required for extraction of Zr without Nb. Hence the loading isotherms were conducted at pH 2. Phase separation in the tests was good at O:A = 10.

Hafnium will be stripped out and currently rejected to the neutralized residue at Luleå. In the next phase recovery of this product will be considered. Although not tested on Norra Kärr, SRK has evaluated this route on similar ores/PLS from Russia, Brazil and Canada and it is proposed to undertake testwork in the next phase of work. For both metals' precipitation will be as oxides, in the case of zirconium as a fused zirconia product.

Two loading isotherms, one with 10 vol% Alamine 336 and one with 15 vol% Alamine 336 have been completed. Analytical results are currently only been available for the 10 vol% Alamine 336 isotherm.

The feed for these tests was the combined and neutralised PLS (Table 2). The organic and aqueous phases were contacted at varying O/A ratios for ten minutes at 30°C and the pH of the aqueous feed was maintained at pH 2 as required with 5 M H_2SO_4. The loaded organic was stripped and the strip liquors analysed. Two different strip solutions were trialled – 1 M Na_2CO_3 and 1 M NaCl/0.1 M HCl. Stripping with 1 M NaCl/0.1 M HCl resulted in better accountabilities compared to stripping with 1 M Na_2CO_3.

The equilibrium isotherm for Zr is presented in Figure 12-9. The data show very good agreement between the Zr solvent concentrations calculated by difference between aqueous feed and equilibrium and those directly determined by stripping with chloride (ANSTO, 2013). It indicates that zirconium may be extracted from the neutralised PLS in 4 stages at an A:O ratio of 4.8, aiming for a solvent loading of 4 g/L Zr.

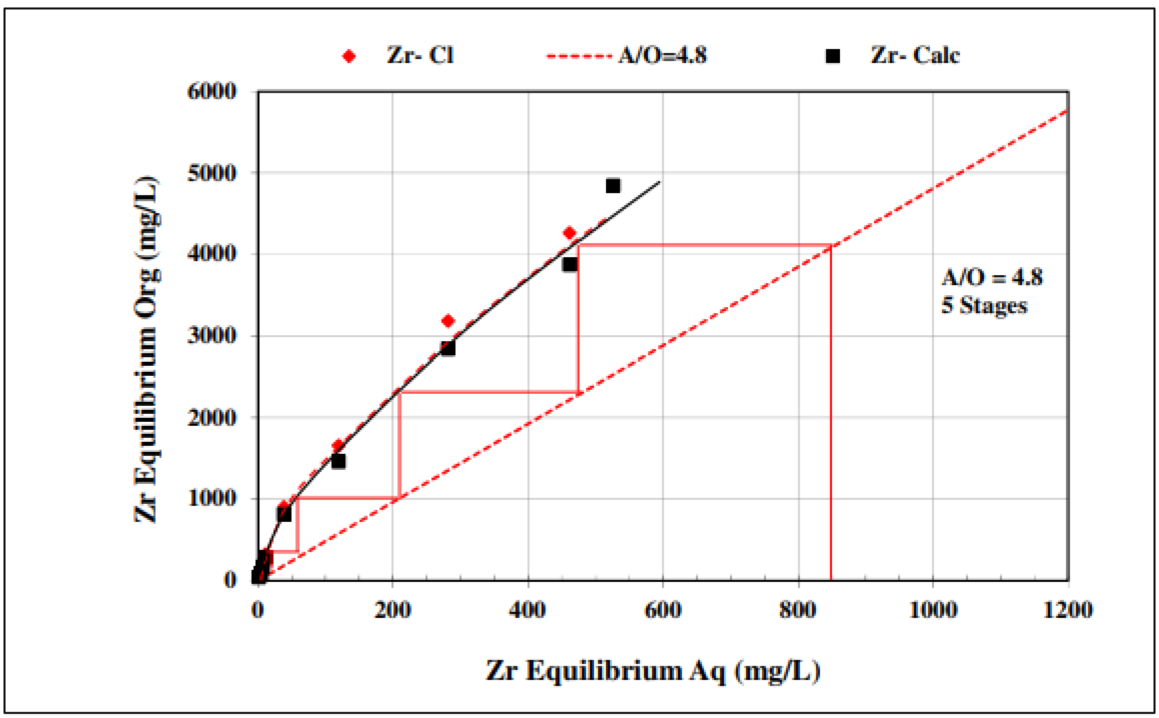


Figure 12-9: McCabe-Thiele Diagram for Zirconium Extraction with 10% Alamine 336/5% isodecanol at 30°C (source, ANSTO, 2013)

There was no significant extraction of rare earths, silicon, thorium, zinc and manganese. Niobium, titanium, hafnium, uranium and a small amount of iron were found to extract with the zirconium at pH 2 Table 12-4

Table 12-4: Solvent Element Concentration at high Zr loading

A :O = 10 10% Alamine 336/ 5% Isodecanol, pH 2

Element	Feed concentration, mg/L	Organic concentration, mg/L
Zr	850	4260
S	19600	4230
Fe	708	162
U	2	19
Nb	11	42
Ti	8	50
Hf	4	13

Phase separation was good for A:O < 2. At higher loadings a white precipitate was formed.

12.12 Testwork on Nb Recovery

For Niobium recovery from sulfuric acid, the standard method of separation involves the use of EDTA (Ethylenediaminetetraacetic acid), solvent extraction or thermal precipitation (Lerner, 1962; Walter, 1963; Shigematsu et al 1964; Ribagnc et al 2017; Hong & Lee, 2018; Tkaczyk et al 2018). EDTA is a chemical that binds and holds on to (chelates) minerals and metals. In addition, work by ANSTO demonstrated the benefit of separation from PLS by Alamine 336 and then sequential separation after Zr is separated.

As no additional testwork has been undertaken on Norra Kärr mineralized rock to SRK's knowledge a two-step separation is assumed at present to ensure a saleable niobium oxide product. This will require verification in the next phase of work.

12.13 NORM Testwork

In the ANSTO study (ANSTO, 2014) an initial assessment or deportment of U and Th through the leach neutralisation steps for the conceptual flowsheet was examined.

There are several points to note regarding the deportment of U and Th:

- Some U and Th is solubilised during leaching, however, the resultant concentrations in the PLS are extremely low at 1.3 and 1.7 ppm, respectively.
- The neutralisation step removes a majority of U and Th from the PLS such that the liquor for further processing to recover REE by SX contains non-detect concentrations <0.01 mg/L.
- During neutralisation, U and Th precipitates along with Al, Fe, and Si, resulting in very low overall (concentrate to liquor) extraction.

The resultant neutralised waste will contain negligible of U and Th and would be classed under IAEA and EU guidelines as being non-radioactive.

12.14 Testwork on Nepheline Syenite

The GTK metallurgical testwork on samples T14 and T15 included major oxide assays for the non-magnetic concentrates ("NMC") and three splits of each based on size fraction. These have been compared with the chemistry of some commercially traded nepheline syenite products reported in the public domain.

There is a high content of nepheline syenite in the Norra Kärr deposit based on mineralogical observation and definition of the lithology types. Once eudialyte and aegirine have been separated as a magnetic concentrate the remaining non-magnetic product is mainly nepheline syenite. LEM considers there to be potential to identify offtakers to take all of this material from site particularly given that it is desirable to have relatively fine grind for some such products and given the likelihood that it will meet the high alumina, high alkali, low iron chemistry criteria which are key considerations for most end users.

Alumina grade in commercially traded products is 21 – 28 % Al_2O_3; the Norra Kärr NMC is in the range of 21-23 % Al_2O_3. Total alkali grade in commercially traded products is 14 – 17 % (Na_2O+K_2O); the Norra Kärr NMC is in the range of 14 – 15 % (Na_2O+K_2O).

Iron content is undesirable in commercial products, the range of values in commercially traded products is 0.1 – 0.2 % Fe_2O_3. For the NMC iron concentration is in this range except for the -20 µm fraction where it is typically in the range of 0.6 – 0.8 % Fe_2O_3. Consequently, this fraction needs diluting or separating out. There is an opportunity to reduce iron content further with additional magnetic separation steps that were not taken in the historical testwork.

13 MINERAL RESOURCE ESTIMATION

13.1 Introduction

The following section describes the geological and block modelling methodology used by WAI for its 2014 Mineral Resource statement as presented in the 2015 PFS. SRK has reviewed and adopted the mineral resource model and has then modified the reporting method to incorporate:

- updated metal price forecasts,
- the contributing value from by-products (Zr, Nb and nepheline syenite),
- updated operating costs and product recoveries,
- reclassification of the MRE to reflect uncertainty around biproduct grades
- a holistic assessment revenue generating potential resulting in a USD/t variable for a reporting cut off grade
- new pit optimisations based on SRK technical-economic model using an appropriate metal price premium

This work is based on the 2015 PFS prepared by WAI and GBM in the PFS report (Davidson *et al*, 2015b) as modified by SRK to incorporate new REO prices and the work undertaken to include nepheline syenite and the Zr and Nb by-products in SRK's Mineral Resource Estimate ("MRE").

13.2 Topography

A high-resolution LIDAR topographic survey was provided to WAI in AutoCAD dwg format. WAI trimmed the topographic data to cover only the area of interest so as to make the topographic data file easier to manage. The Norra Kärr project is a greenfield site and as such there are no operating pits in the area and the topography as of 30 June 2014 remains valid today for the purposes of this PEA and MRE. Figure 13-1 shows the topographic contours for the project area as well as the distribution of exploration drillholes.

13.3 Database Compilation

WAI was supplied with a geological database (LEM_WAI.mdb) in Microsoft Access® format with an effective date of 06 May 2014. The database comprises:

- Diamond drillhole collar surveys in both RT90 and SWEREF99TM co-ordinate systems;
- Down hole surveys;
- Multi-element Assays
- Whole-Rock Assays;
- Geological log;
- RQD results;
- Drillhole recoveries;
- Density testwork results by drillhole interval and lithology; and
- QA/QC results.

In total, 119 diamond core drillholes for 9,986 samples (20,420 m) was used by WAI for the geological and block model. No drilling has been undertaken on the Norra Kärr project since 2014.

Drilling to date extends over a strike length of approximately 1,280 m with drillhole section lines orientated east-west typically on 50 m spacing. The distribution of drillholes along the section lines is typically 80 m with drillholes drilled inclined to the east (with the exception of five drillholes; NQ12019, NQ12015, NKA12083, NKA12082, and NKA074-GT1) with an average inclination of 48°. A plan showing the spatial distribution of the diamond drillholes is shown in Figure 13-1, a long section of the drillhole distribution looking towards the west is provided in Figure 13-2.

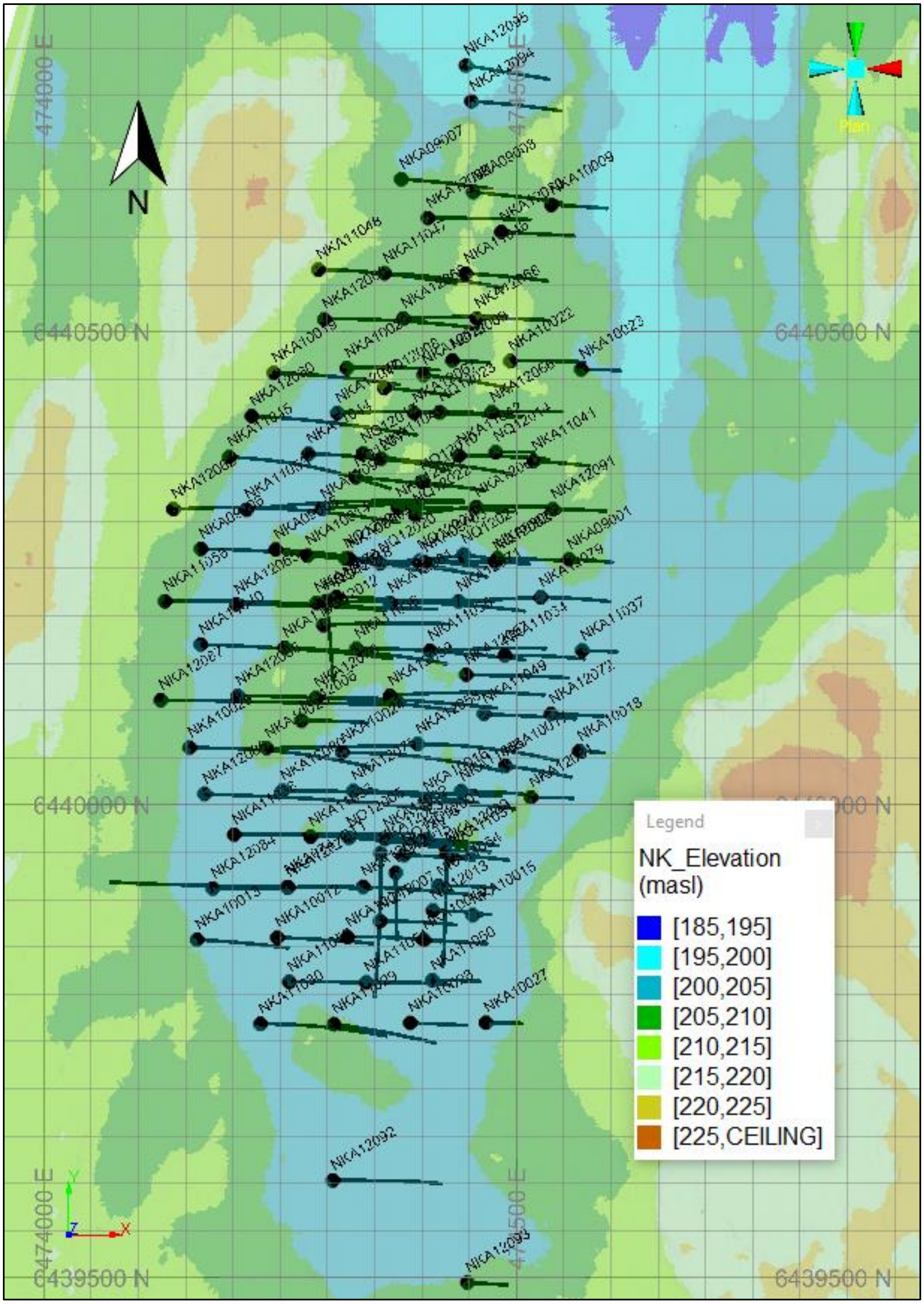


Figure 13-1: Drillhole layout and topographic survey

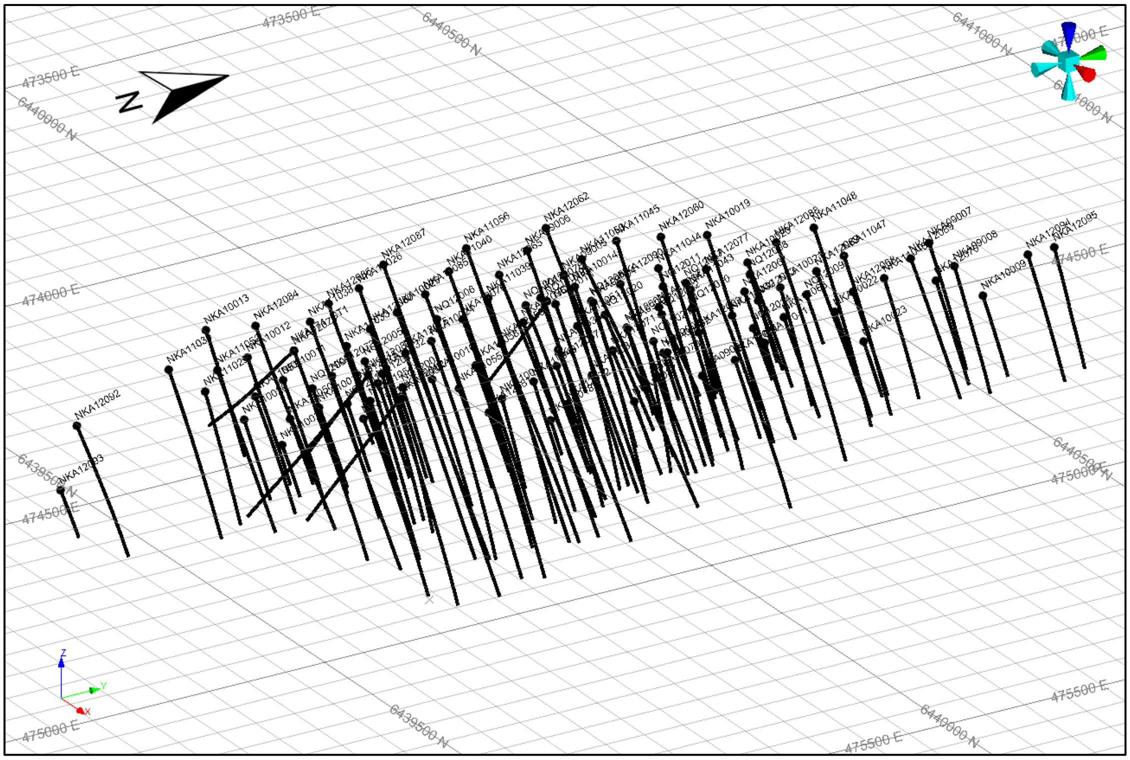


Figure 13-2: 3D view of Norra Kärr Drillholes – looking northwest

13.3.1 SRK Comments

The samples in the drilling database were almost all assayed using a multi-element method. However only a small percentage was additionally assayed using a major oxide (Whole Rock) suite which is required to allow a modal reconstruction of mineral proportions which in turn can be used to make a meaningful assessment of eudialyte, catapleiite, aegirine and nepheline syenite content for each sample.

Further evidence for nepheline syenite grade comes from the bulk samples used for metallurgical testwork where major oxide assays were performed on head samples, magnetic concentrates and non-magnetic concentrate. The quality and quantity data are presented in Section 12.14 and Section 16.2.2; in summary SRK's estimate of the nepheline syenite average grade is approximately 65%.

13.4 Geological Interpretation-Wireframe Modelling

Wireframes to represent the Norra Kärr alkaline igneous intrusive body and associated REE mineralisation were constructed by WAI based on the geological logging carried out by LEM. WAI was provided with geological cross sections and plans by LEM showing the interpreted structure based on the detailed lithology logs, geochemical assay results and the foliation measurements.

In total, WAI defined 6 key lithologies for wireframing and subsequent modelling, these being:

- LTYPE 1 (KAX) – Kaxtorpite;

- LTYPE 2 (GTM) – Grennaite (recrystallised to in part migmatitic);
- LTYPE 3 (PGT/GT) – Grennaite with pegmatititc zones and grennaite;
- LTYPE 4 (GTC) – Grennaite (catapleiite porphyritic – low grade TREO);
- LTYPE 5 (ELAK) – Lakarpite with eudialyte; and
- LTYPE 6 (MAF) – Mafic dyke material.

The lithology units LTYPES 2-4 do not reflect distinct individual units but observe more diffuse boundaries. Where there is uncertainty in the recording of a lithological unit LEM has reviewed the geochemical assay results to ascertain where the transition occurs.

Within the geological database provided to WAI there are sub-divisions of the 6 main lithology groups. Due to the diffuse nature of LTYPES 2-4 and the complexity of the geology it was decided by LEM and WAI that the level of detail in the six lithology divisions was sufficient for a Mineral Resource model, and that there is little benefit in modelling in greater detail.

As well as the 6 main lithologies LEM has also logged additional units at the periphery of the mineralisation which are not of economic interest, these units comprising:

- Pulaskite (microcline-albite-amphibole);
- Pulaskite (with grennaite zones/bands);
- Alkaline rock (unspecified); and
- Granitoid (fenitisied).

These additional lithological units are not of economic interest and as such for the purpose of the MRE have not been modelled.

Upon completion of a preliminary set of mineralisation wireframes, WAI submitted the wireframes to LEM geologists for review. Following the review WAI and LEM geologists worked through the wireframes and geological sections adjusting the wireframes where necessary to ensure they honoured the geology.

Within the geological database provided to WAI includes a record of the depth of overburden. WAI created a digital terrain model ("DTM") surface of the base of overburden and used this to constrain mineralisation.

Images of the WAI lithology-based domain wireframes are shown in Figure 13-4 to Figure 13-6.

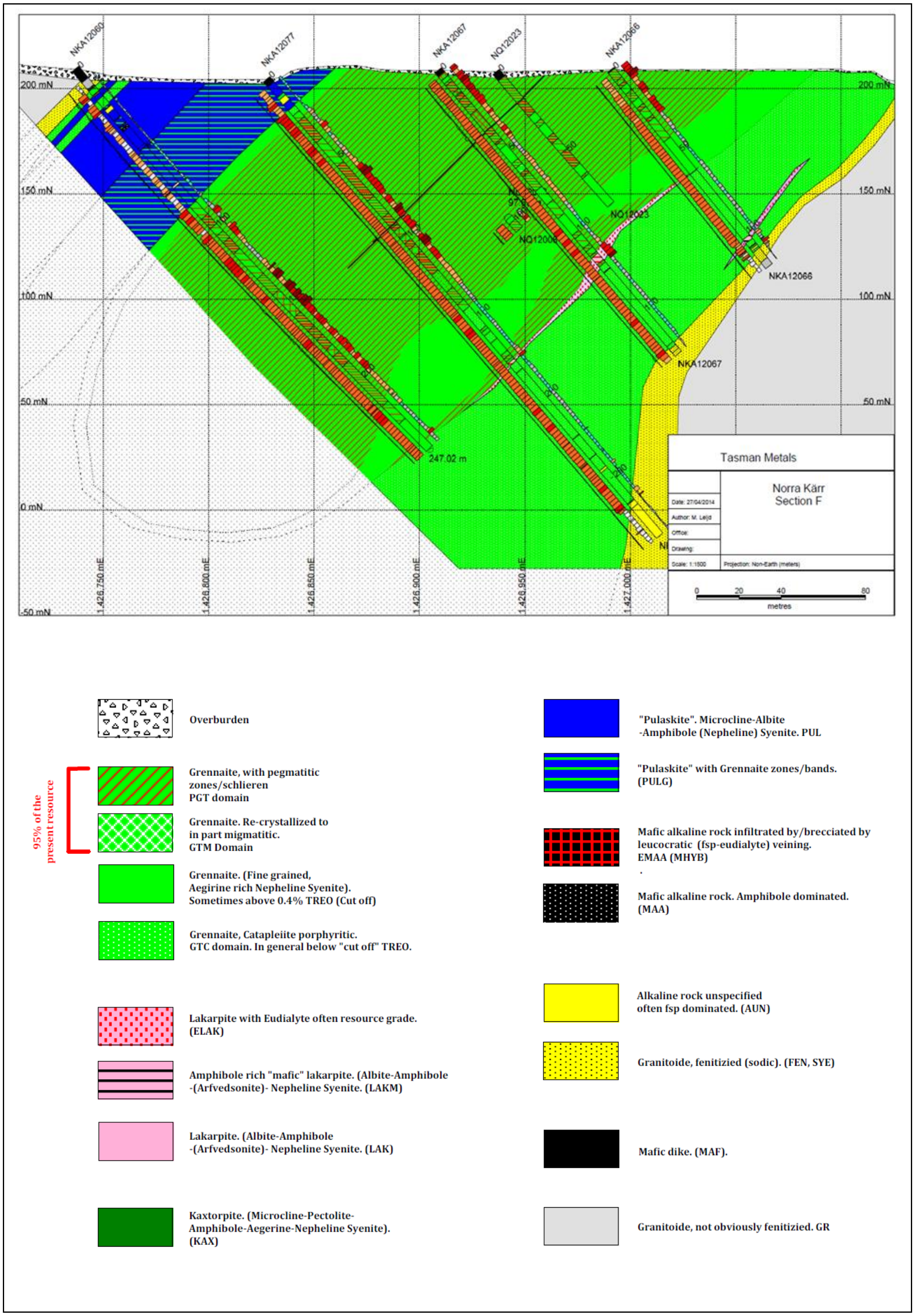


Figure 13-3: LEM geological interpretation cross section – Northing 6440410

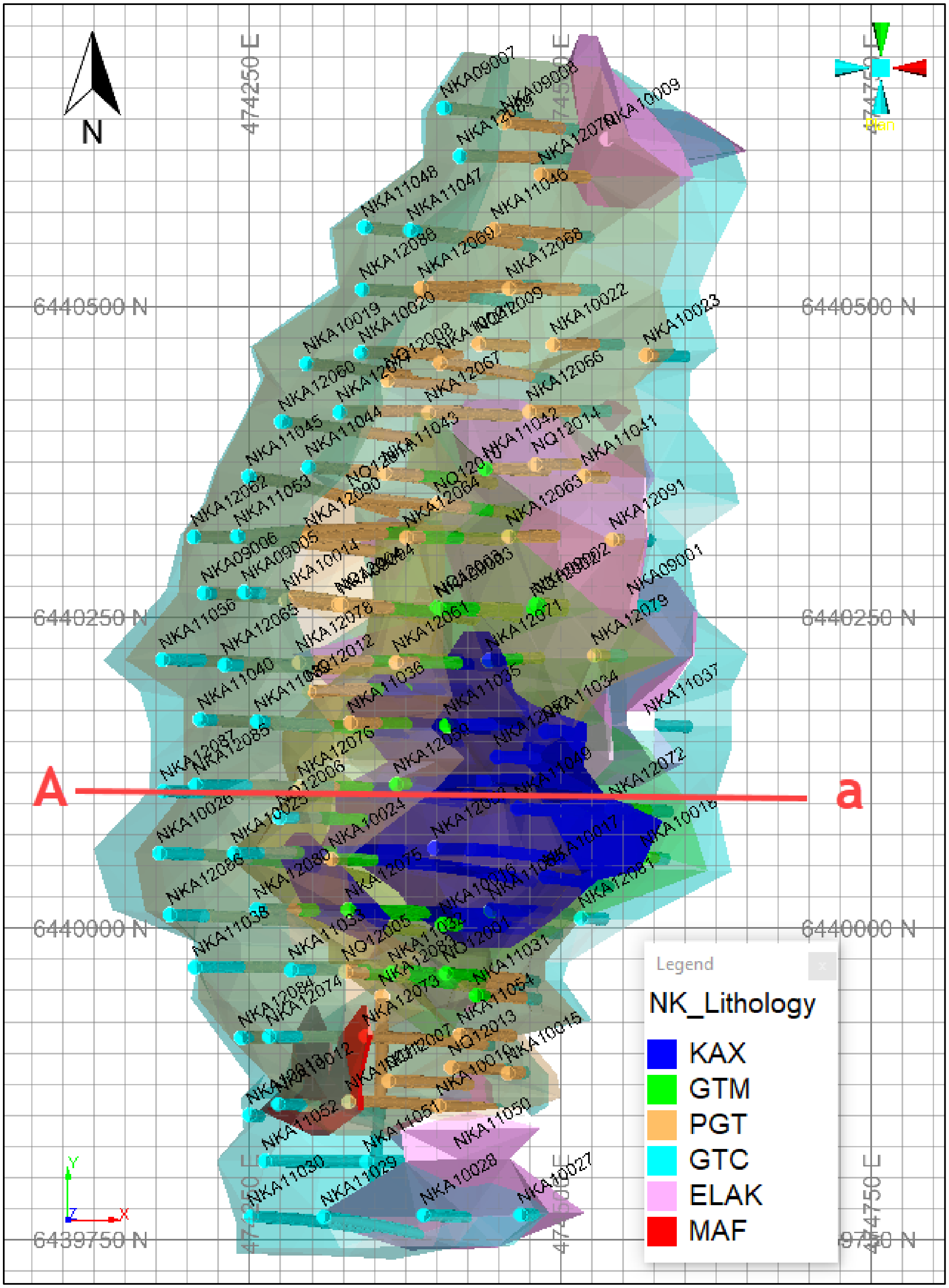


Figure 13-4: WAI lithology wireframes – plan view (cross-section line A-a on Figure 13-5)

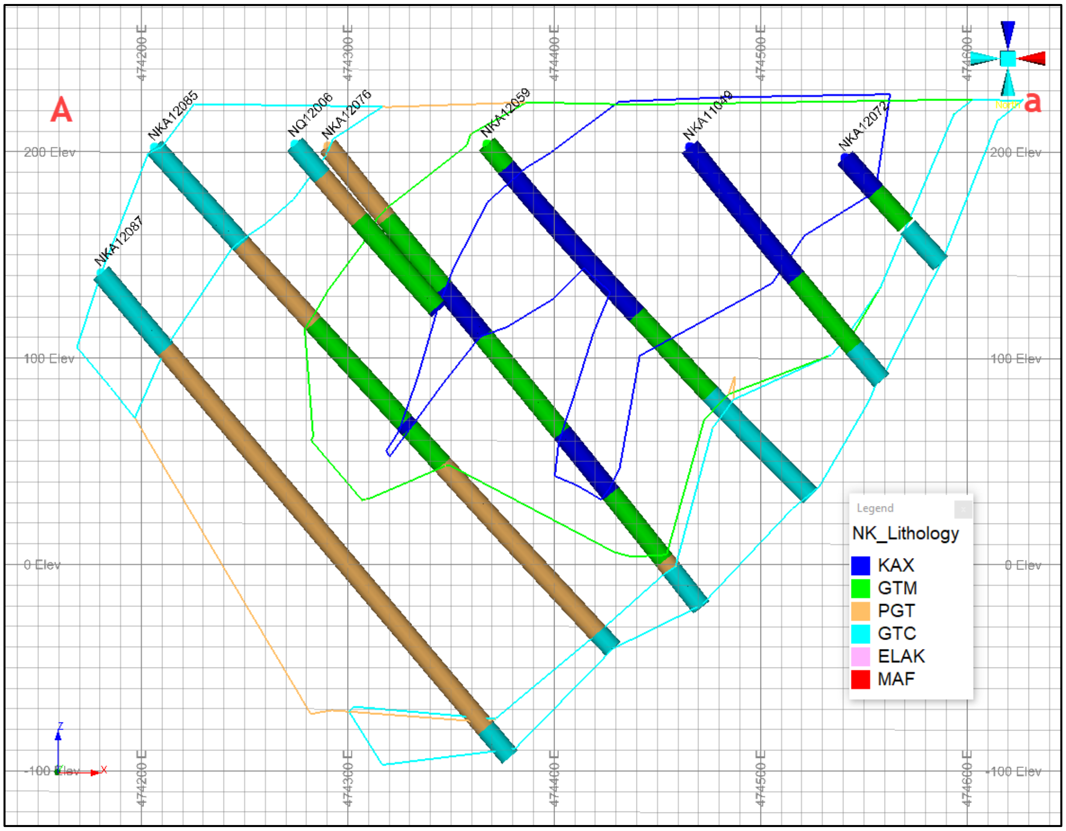


Figure 13-5: WAI lithology wireframes – cross-section (reference line shown in Figure 13-4)

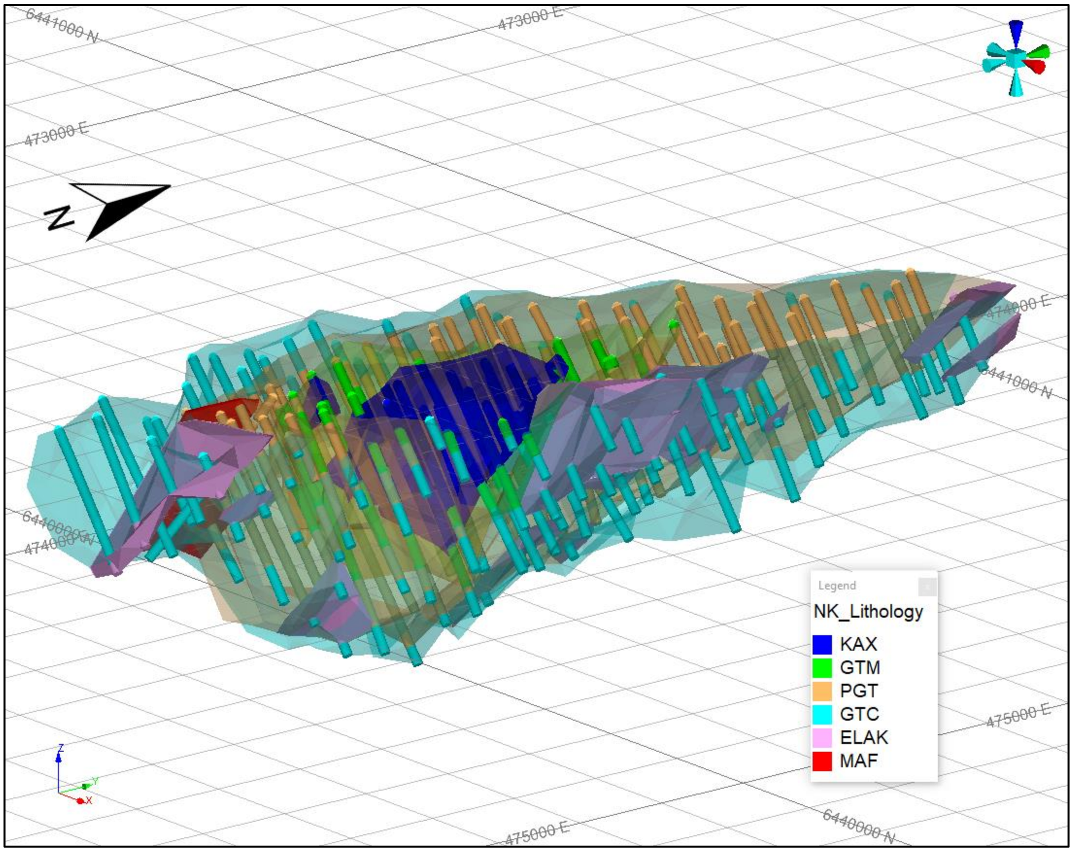


Figure 13-6: WAI lithology wireframes – 3D view looking northwest

13.4.1 SRK Comments

SRK has reviewed the wireframes and agrees that these provide a robust 3D geological interpretation of the deposit; furthermore, these are suitable to be used for grade estimation domains for the purposes of the MRE and classification given in this report. The wireframes honour the lithologies logged in the drillholes after an appropriate simplified grouping of the detailed logging.

13.5 Sample Data Processing

13.5.1 Sample Selection

The geological wireframes were used to select the drillhole sample data. The selected sample data was coded according to lithology to correspond with the wireframe envelope LTYPE codes (as described above in Section 13.4). A summary of the selected sample data by LTYPE is shown in Table 13-1.

Table 13-1: Selected Sample Summary by LTYPE

LTYPE	No. of holes	No. of samples	Total length (m)
1	22	461	804
2	42	1,340	2,423
3	85	4,895	8,499
4	95	2,134	3,808
5	42	266	415
6	4	31	40

WAI undertook a graphical review of the sample data for the 11 main payable REEs: Dy, Y, Eu, La, Nd, Ce, Gd, Tb, Pr, Sm and Lu.

The graphical review was to ascertain the grade populations of each lithology and ascertain the need for additional domaining and the potential for top-cutting of the sample dataset. Figure 13-7 and Figure 13-8 below show the variogram grade distributions for Dy and Y in LTYPES 2-5. Overall LTYPES 2, 3, and 4 show normal distributions with some positive skew suggesting that additional domaining is not required but there may be a requirement for top-cutting to prevent high grade outlier bias.

LTYPES 5 and 6 show more log normal grade distributions with possible sub populations necessitating domaining. However, in the case of LTYPE 5 the variable grade population is reflective of the different lenses of ELAK material through the deposit, during the estimation works these ELAK zones are treated separately of each other effectively sub-domaining the LTYPE 5 (ELAK) material. The very low number of samples in LTYPE 6 (MAF) means that it is impossible to discern sub-populations as such no additional domaining of this material has been carried out.

LTYPE 1 displays a log normal trend for the sample data with a slight negative skew, in reviewing the data WAI determined that no additional domaining was required.

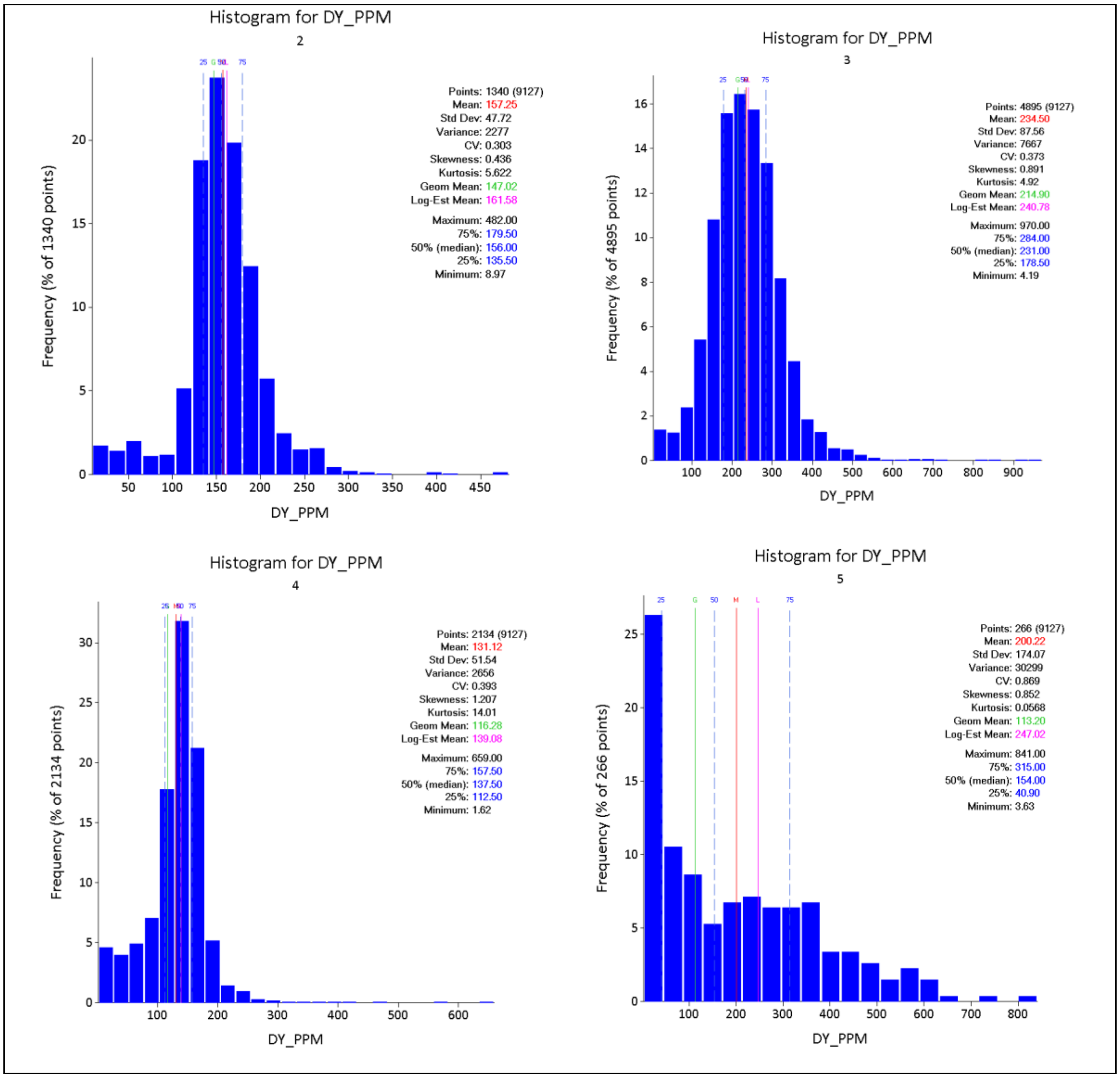


Figure 13-7: Dy Histogram Plots for LTYPE 2-5

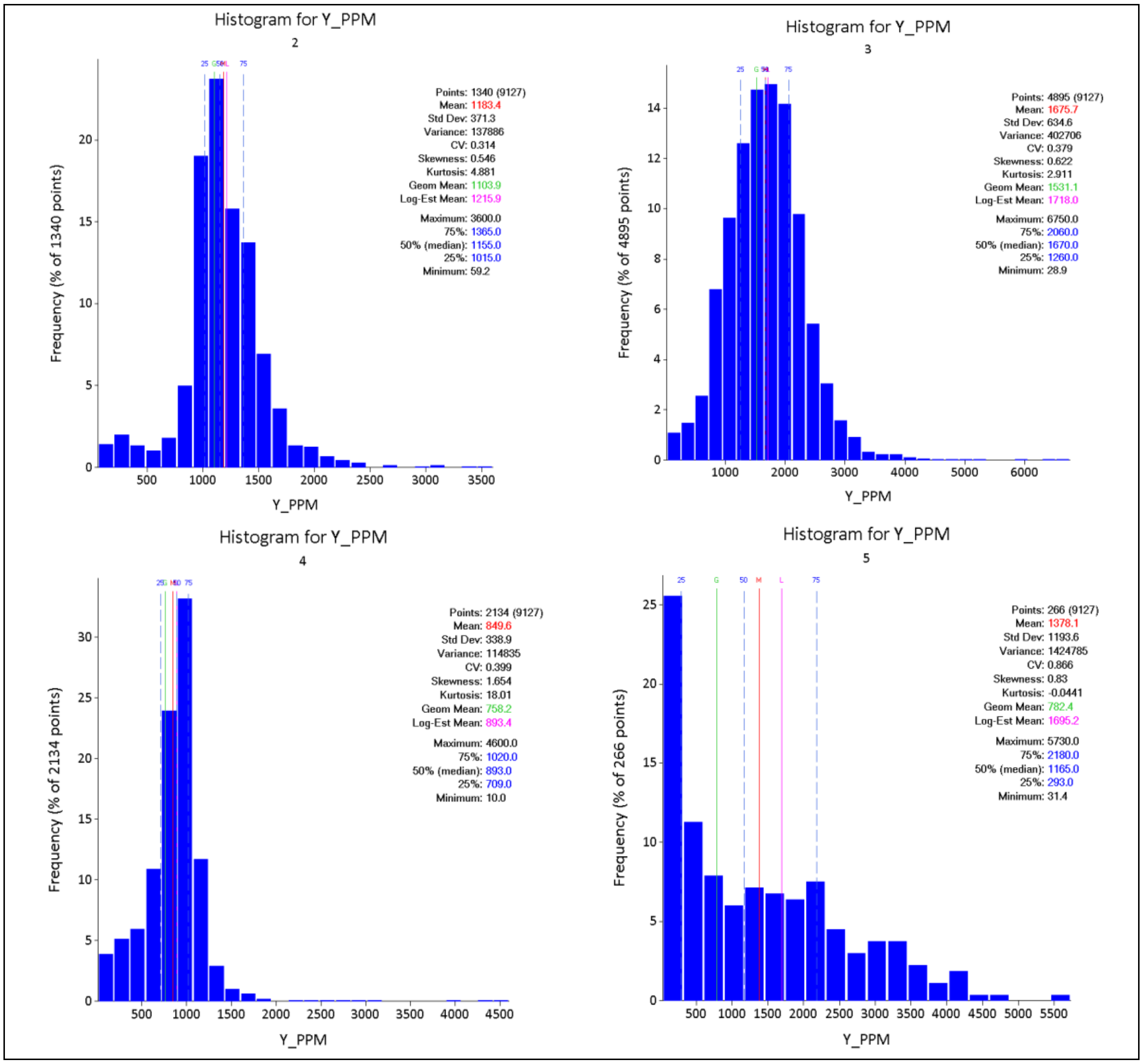


Figure 13-8: Y Histogram Plots for LTYPE 2-5

13.5.2 Top Cutting/Grade Capping

The presence of high-grade outliers in a sample dataset can exert a bias on a grade estimation leading to overestimation of grades in the block model. To help ascertain the presence of high grade outliers in the selected sample database WAI undertook a decile analysis and compared the results to the grade distribution plots as shown in Figure 13-7 and Figure 13-8 above. An example table of results for a decile analysis is shown in Table 13-2.

Table 13-2: Decile analysis – samarium LTYPE 4

Quantile % FROM	Quantile % TO	No of Samples	Mean (ppm)	Minimum (ppm)	Maximum (ppm)	Metal (ppm x length)	Metal%
0	10	213	14.31	1.66	26.4	3048.82	2.26
10	20	213	36.76	26.6	46.4	7830.5	5.79
20	30	214	51.12	46.4	55.2	10938.8	8.09
30	40	213	58.41	55.2	61.4	12441	9.21
40	50	214	63.85	61.4	65.8	13664	10.11
50	60	213	68.40	65.8	70.7	14568.3	10.78
60	70	213	72.75	70.7	74.7	15495.6	11.47
70	80	214	76.78	74.7	79.1	16430	12.16
80	90	213	82.48	79.2	86.6	17567.5	13.00
90	100	214	108.26	86.8	335	23168.1	17.14
90	91	21	87.45	86.8	88.1	1836.5	1.36
91	92	21	88.48	88.1	89	1858.1	1.37
92	93	22	89.79	89.1	90.4	1975.3	1.46
93	94	21	91.11	90.4	91.8	1913.3	1.42
94	95	22	92.89	91.9	94.2	20440	1.51
95	96	21	95.59	94.4	97.7	2007	1.49
96	97	21	100.19	97.8	104	2104	1.56
97	98	22	110.43	104.5	117	2430	1.80
98	99	21	128.38	119	142	2,696	1.99
99	100	22	195.66	142	335	4,305	3.18
0	**100**	**2134**	**63.33**	**1.66**	**335**	**135,153**	**100**

In reviewing the decile analysis, WAI assessed whether there is a jump in the proportion of contained metal in the top 1% (Quantile 99%-100%) of the sample data, such jumps in the grade distribution are also often visible in the grade distribution histograms and probability plots. Such inflections signify possible high-grade outliers that may bias the grade estimation.

In general, there are very few outlier grades that may present an issue during the variography or grade estimation stages, those that presented a problem have been capped and a summary of these is shown in Table 13-3.

Any absent assay values within the sample database have been assigned grades of half the analytical detection limit (see Table 10-1).

Table 13-3: Grade capping summary

LTYPE	Element	Top-Cut (ppm)
4	Eu	27.5
2	Gd	288.0
3	Gd	472.0
4	Gd	282.0
4	Ho	112.0
5	Lu	66.4
4	Sm	222.0
3	Sm	481.0
4	Tb	63.1
5	Tm	67.3
4	Yb	341.0
5	Yb	495.0

13.5.3 Compositing

To ensure all samples in the variography and grade estimation stages have equal support the sample data was composited. The sample dataset was reviewed for sample length to ascertain the optimum sample composite interval. An average composite length of 2 m was applied to sample data for all lithologies ("LTYPE") corresponding with the median sample length. Figure 13-9 below shows the sample length distribution.

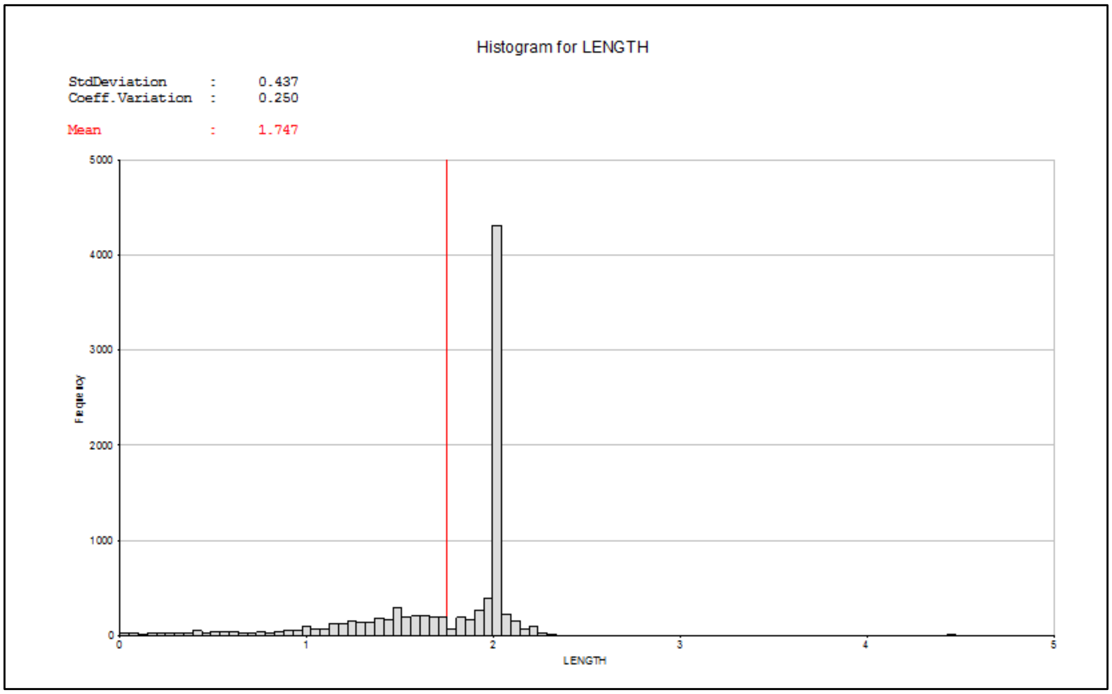


Figure 13-9: Average Sample Length Histogram

13.5.4 SRK Comments

SRK considers the compositing and capping approach to be reasonable, although it is not clear whether capping was applied pre-compositing or post-compositing. Some of the highest grades are associated with shorter intervals which is indicative of selective sampling and in SRK's opinion these should first be composited before assessing capping levels. In SRK's view the main purpose of capping is to prevent isolated extreme grades in the block model grade estimation and SRK is satisfied that this has been achieved.

It is also not clear whether any capping was applied to Zr and Nb datasets; SRK notes a few high grades in the drilling data which tend to be located in the vicinity of each other or at least in a similar position within the deposit and that the highest-grade blocks in the block model correspond with these locations which means there are few or no instances of unsupported extreme high grade values.

Overall, SRK considers there to be no sign of any undesirable estimation effects based on the approach to capping.

13.6 Variography

13.6.1 Introduction

Variography was undertaken:

- To determine the presence of any anisotropy in the deposit;
- To derive the spatial continuity of mineralisation along the principal main anisotropic orientations;
- To produce suitable variogram model parameters for use in geostatistical grade interpolation; and
- To assist in selection of suitable search parameters upon which to base the resource estimation.

Variographic analysis was performed using Supervisor 8.2. Semi-variogram analysis was attempted for each variable in each domain.

13.6.2 Analysis

Semi-variogram analysis was attempted for 20 components in each of the 6 mineralised domains using the 2 m composited samples contained within the mineralised domain wireframes.

The methodology for producing experimental variograms and variogram models was as follows:

- Experimental variograms with small lag distances were generated downhole to aid in estimation of nugget effect. Nugget effect is the variance between sample pairs at the same direction containing components of inherent variability, sampling error and analytical error;
- Omni-directional variograms were generated to assist with determining optimal lag distances;
- Variogram maps were generated in 18 directions in the horizontal plane, across strike and in the dip plane at each stage selecting the direction of greatest continuity; and
- Experimental variograms were generated and variogram models fitted using the spherical scheme for three orthogonal directions defining the principal directions of anisotropy: the major, semi-major and minor axis.

13.6.3 Variogram Interpretation

During this process it was found that the kaxtorpite (LTYPE=1) and lakarpite (LTYPE=5) did not contain enough sample pairs to calculate robust directional experimental variograms and so for these zones only omni-directional variogram models were fitted. For the mafic dykes (LTYPE=6) not enough sample pairs were present for any meaningful experimental variograms to be interpreted. For all other zones; grennaite (LTYPE=2), grennaite with pegamtitic zones (LTYPE=3) and porphyritic grennaite (LTYPE=4) enough sample pairs were present to generate robust directional experimental variograms for which models were fitted. It was found that as a result of the good positive correlation between most components that experimental variograms were quite similar for many of the individual components. Regardless of these individual variogram models were completed for each component in each zone.

Figure 13-10 shows the variogram models for Dy for the porphyritic grennaite (LTYPE=4). When calculating the experimental variograms variable lag distances were used to reflect the varying drillhole spacing in different orientations for the different domains. For downhole variograms a lag distance of 2 m was used, equal to composite length.

Directional anisotropy is seen in all cases where robust experimental variograms can be calculated. Zonal anisotropy is seen for some elements and the models are fitted appropriately. The experimental variograms are, in general, reasonably well structured with large ranges along strike. The nugget values are generally seen to be low reflecting that, at short ranges, little variation is seen.

The spherical model scheme was used for all variables. Variogram models were created for each variable for each domain. A summary of the variogram models was provided in the PFS report (Davidson *et al*, 2015b) and is not repeated herein.

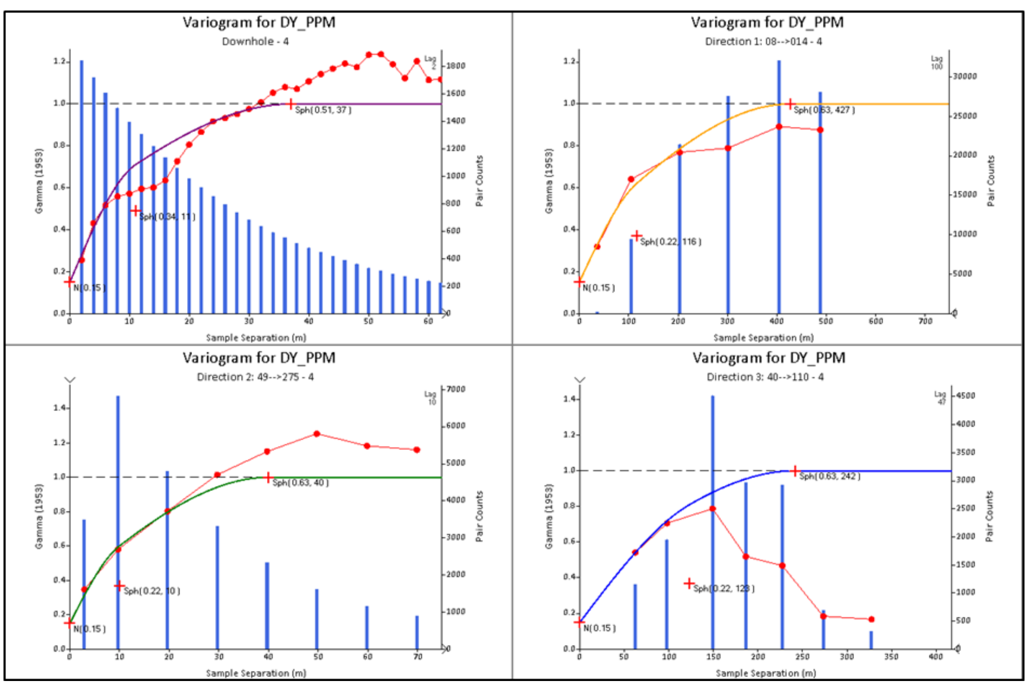


Figure 13-10: Variogram models for Dy – Porphyritic Grennaite (LTYPE=4)

13.6.4 Ellipses

Ellipses orientated and sized to represent the variogram model ranges were created and compared to the mineralised zone wireframes to ensure that the variogram models reflected the geological interpretation. An example of an ellipse in relation to the grennaite domain wireframe is shown below in Figure 13-11.

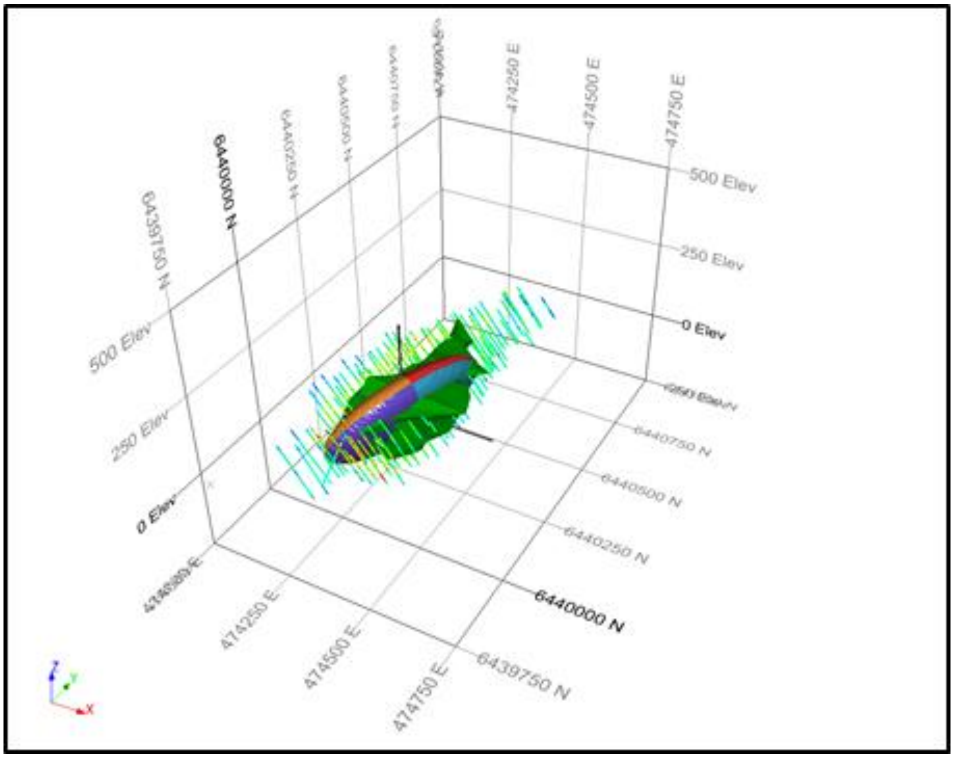


Figure 13-11: Search ellipse representing variogram ranges and grennaite wireframe

13.6.5 SRK Comments

WAI's variography is conventional and in SRK's view does not present a risk to the grade estimation,

13.7 Volumetric Modelling

13.7.1 Block Model Parameters

An initial empty block model was created inside the mineralised zone wireframes. A summary of the block model parameters is listed in Table 13-4 below. A parent cell size of 25 x 25 x 5 m (northing x easting x vertical) was selected. Key fields were established within the block model to identify and separate the individual mineralised zones for control on grade estimation as described below. The model was not rotated. A minimum sub-cell size of 5 x 5 x 1 m was allowed to ensure accurate volumes compared to the wireframe surfaces.

Table 13-4: Summary of Block Model Parameters

Property	Direction	Metres (m)
Model Origin	X	473,600
	Y	6,439,300
	Z	-134
Parent Cell Size	X	25
	Y	25
	Z	5
No. of Cells	X	62
	Y	72
	Z	79

13.7.2 Dynamic Anisotropy

Dynamic anisotropy is the process of reflecting small changes in dip and dip direction of a deposit during grade estimation by slightly altering the orientations of search ellipses used to select samples. At Norra Kärr a significant database of foliation directions measured from drill core were used to interpret these subtle changes in orientation. Dip and dip directions were estimated into the volumetric block model using IDW^3 and the foliation information supplied LEM which was converted to point 'sample' locations with consistent lengths of 0.02 m.

Estimation of dip and dip direction was carried out using a single pass estimation run, i.e. a single non-expanding ellipse was used in order to achieve a block model that had no 'steps' in results.

Estimation was carried out using the mineralised domains as key fields, i.e. estimation of dip and dip direction in a domain only used samples within that domain for the estimation. The size and orientations of the search ellipses used were based on the variogram models interpreted for Dy grades as these roughly followed the orientation of the interpreted fold axis.

The size of the final search ellipses used were calculated based on normalising the longest range of these variograms to 100 m and determining the size of the other axis as a factor based on the anisotropy. The estimation was then carried out as an iterative process with the search ellipse sizes being increased for each zone until all blocks in that zone were estimated in a single pass. A minimum of 4 samples was specified in order to 'average out' any inconsistent dip and dip directions recorded in the database.

The parameters used for the estimation of dip and dip direction are listed in Table 13-5. The estimated dip and dip directions were validated visually against the input data.

Table 13-5: Estimation Parameters for Dip and Dip Direction (using IDW^3)

Parameter	Domain 1	Domain 2	Domain 3	Domain 4	Domain 5	Domain 6
Search Radii 1 (m)	200	300	800	900	400	200
Search Radii 2 (m)	200	210	80	90	400	200
Search Radii 3 (m)	200	75	160	495	400	200
Rotation (Z,X,Z)	0,0,0	-70,60,10	90,50,0	110,50,170	0,0,0	0,0,0
Minimum Composites	4	4	4	4	4	4
Maximum Composites	16	16	16	16	16	16
Minimum Drillholes	2	2	2	2	2	2

13.7.3 SRK Comments

The approach used by WAI for the block model framework and the sample search parameters as described is reasonable in SRK's opinion.

13.8 Density

A total of 1,692 bulk density tests were carried out by LEM with the resultant database supplied to WAI. To facilitate assigning densities by lithology type WAI incorporated the density data into the drillhole file and then extracted the density records by the wireframe lithology (LTYPE). The average density values shown Table 13-6 by LTYPE have been calculated and subsequently assigned to the block model.

Table 13-6: Density values applied by Zone

LTYPE	Rock Type	Density (t/m^3)
0	Granite	2.69
1	Kaxtorpite	2.74
2	Grennaite	2.67
3	Grennaite with pegmatitic zones	2.72
4	Porphyritic grennaite	2.71
5	Lakarpite	2.74
6	Mafic dykes	2.74

13.8.1 SRK Comments

SRK considers these values to be reasonable and the differences between rock types to be of no material significance, so has used a single density value of 2.7 t/m^3 for the MRE presented in this report.

13.9 Grade Estimation

13.9.1 Kriging Neighbourhood Analysis (KNA)

Kriging Neighbourhood Analysis ("KNA") is a method used for optimizing block sizes and search parameters used during grade estimation. The KNA was carried out using Supervisor v8.2 software which enables a series of tasks to be carried out in turn to determine:

- optimum block sizes;
- minimum and maximum samples to be used during estimation;
- optimum search ellipse size; and
- optimum block discretisation.

For each of these tasks a series of estimation runs are carried out using the variogram parameters calculated as shown above on a block centre location selected for its representivity of the deposit. Due to the strong positive correlation between many of the variables under examination the KNA was run on Dy as being representative of mineralisation across the deposit. KNA studies were carried out on the two largest domains: the grennaite with pegmatitic zones and the porphyritic grennaite. Similar results were obtained both of these zones.

13.9.2 Estimation Plan

Grade estimation was carried out using Ordinary Kriging ("OK") as the principal interpolation method. IDW^2 and Nearest Neighbour ("NN") were also used for comparative purposes for each element. For the mafic dykes only IDW^2 and NN were used due to a lack of robust variogram models in this zone. The OK method used estimation parameters defined by the variography and refined after KNA studies as described above. The estimation was performed on mineralised material domains defined during the domaining process described above and only drillhole composites contained within a domain were used in the grade estimation of that domain.

For the mineralised zones, the OK estimation was run in a three-pass estimation plan, the second and third passes using progressively larger search radii to enable the estimation of blocks not estimated on the previous pass. The search ellipse sizes and orientations were derived from the variographic analysis, with the first search distances corresponding to the distance at roughly 2/3 of the variogram range value and the second search distance approximating the variogram range and the third search ellipse approximating twice the variogram range. Search ellipses were orientated to fit the directions of continuity as defined by the variography study. The directional control settings defining the local variation in the strike and dip of the mineralised zones that were defined during the block model creation process were used during estimation. The dip and dip directions were used as vectors to interpolate dip directions and dip values into the block model. These orientations were subsequently used during grade estimation to orient the search ellipses independently for each block. This dynamic anisotropy procedure gives a more realistic reflection of the local variations in the strike and dip of the deposit.

Sample weighting during estimation was determined by variogram model parameters for the OK method. Block discretisation was set to 5 x 5 x 2 points to estimate block grades. Sub cells received the same estimate as the parent cell. A summary of the estimation parameters is shown in Table 13-7 and summary of the search ellipse sizes is shown in Table 13-8.

Table 13-7: Grade estimation parameters

Item	Search Ellipse		
	1	2	3
Search Ellipse Size Increase of ellipse #1	x 1	x 1.5	x 3
Minimum Composites	20	15	8
Maximum Composites	40	40	16
Minimum Octants	4	4	-
Min Composite per Octant	1	1	-
Max Composite per Octant	5	5	-
Minimum Drillholes	3	3	2

Table 13-8: Initial search ellipse dimensions (m) for each domain

Element	Domain 1	Domain 2	Domain 3	Domain 4	Domain 5	Domain 6
Dy	25,25,25	160,115,45	300,60,35	280,160,25	100,100,100	100,100,100
Y	25,25,25	145,115,45	230,65,50	260,130,25	100,100,100	100,100,100
Eu	25,25,25	155,80,65	170,65,55	315,175,30	100,100,100	100,100,100
La	45,45,45	110,80,35	150,40,35	235,220,30	70,70,70	100,100,100
Nd	25,25,25	120,70,60	175,110,65	245,200,30	80,80,80	100,100,100
Ce	60,60,60	130,65,65	145,155,50	245,180,30	70,70,70	100,100,100
Er	25,25,25	130,80,35	220,70,35	265,180,25	95,95,95	100,100,100
Gd	25,25,25	120,75,65	230,75,50	285,160,25	95,95,95	100,100,100
Ho	25,25,25	120,85,25	265,60,30	280,160,30	95,95,95	100,100,100
Lu	25,25,25	85,65,25	215,80,35	260,165,25	90,90,90	100,100,100
Pr	55,55,55	130,75,65	170,110,65	255,195,30	75,75,75	100,100,100
Sm	25,25,25	155,75,65	170,170,55	280,170,25	100,100,100	100,100,100
Tb	25,25,25	145,70,30	300,60,35	315,160,30	100,100,100	100,100,100
Tm	30,30,30	130,65,25	255,80,35	260,175,20	95,95,95	100,100,100
Yb	25,25,25	130,70,25	240,80,35	285,145,25	100,100,100	100,100,100
Zr	25,25,25	80,45,20	100,45,45	60,60,60	90,90,90	100,100,100
U	25,25,25	90,65,25	80,65,35	75,75,75	35,35,35	100,100,100
Th	15,15,15	30,30,30	170,40,35	25,25,25	40,40,40	100,100,100
Hf	25,25,25	80,35,20	130,65,40	60,60,60	90,90,90	100,100,100
Nb	25,25,25	150,115,75	265,50,15	60,60,60	95,95,95	100,100,100

SRK Comments

WAI's approach to grade estimation follows good industry practice in SRK's opinion, it optimises the search strategy to balance the quality of local and global estimation and also employs dynamic anisotropy which allows for grade estimates to follow the orientation trends of the controlling lithologies.

Note a simple grade estimation for nepheline syenite was undertaken by SRK using a grade of 65% as described in Section 13.3.

13.10 Validation

Following grade estimation, a statistical and visual assessment of the block model was undertaken to 1) assess successful application of the estimation passes 2) to ensure that as far as the data allowed, all blocks within mineralisation domains were estimated and 3) the model estimates performed as expected. The model validation methods carried out included:

- visual assessment of grade.
- global statistical grade validation; and
- grade profile analysis.

13.10.1 Visual Assessment

A visual comparison of composite sample grade and block grade was conducted in cross section and in plan, as shown in an example section in Figure 13-12. Visually the model was generally considered to reflect the composite grades and the domain boundaries are honoured.

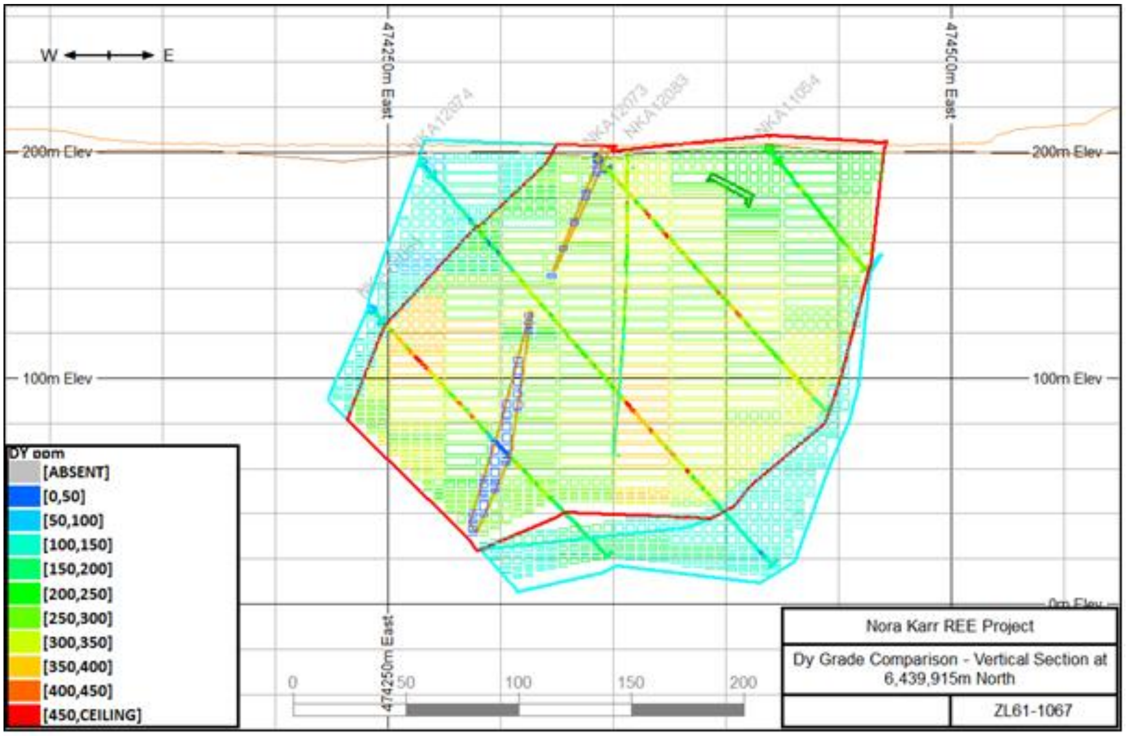


Figure 13-12: Example East-West Vertical Section Showing Composite and Estimated Grades at Northing=6439915

13.10.2 Global Grade Comparison

Statistical analysis of the block model was carried out for comparison against the composited drillhole data. This analysis provides a check on the reproduction of the mean grade of the composite data against the model over the global domain. Typically, the mean grade of the block model should not be significantly different from that of the samples from which it has been derived. The mean block model grade for each mineralized rock zone and its corresponding mean composite grades were analysed in detail in the PFS report (Davidson *et al*, 2015b), which is not repeated herein. A good match is seen between input sample/composite grades and the estimated block grades for each domain.

13.10.3 Local Grade Profile Comparisons

Grade profile (swath) plots are a graphical display of the grade distribution derived from a series of bands, or swaths, generated in several directions through the deposit. The plot compares the grade within these bands of the composite samples and the block estimated grades for the different methodologies used. Where the composite grades and the estimated grades show a good correlation, greater confidence can be placed on the estimate.

These plots were generated in three directions, by easting, northing and vertically to fit with the general form of the mineralised zones. Comparison of the grade was made against the composited grades with block grades estimated by OK, IDW^2 and NN methods. The swath plots were generated by averaging the blocks and composites using 25 m swaths in easting and northing and 5 m swaths by elevation. Grade profile plots were generated for the OK, IDW^2 and NN estimation methods and should exhibit a close relationship to the composite data upon which the estimation is based.

A generally close relationship was observed between composite and block grade across the model for all variables. Some deviations between the composite and estimated block grade occur at the edges of the deposit where reduced tonnages or wide sample spacing accentuate the differences in grade.

An example of grade profile plots are shown in Figure 13-13 for Dy in domains LTYPE=2; grennaite. Overall, the plots illustrated a good correlation between the composites and the block grades.

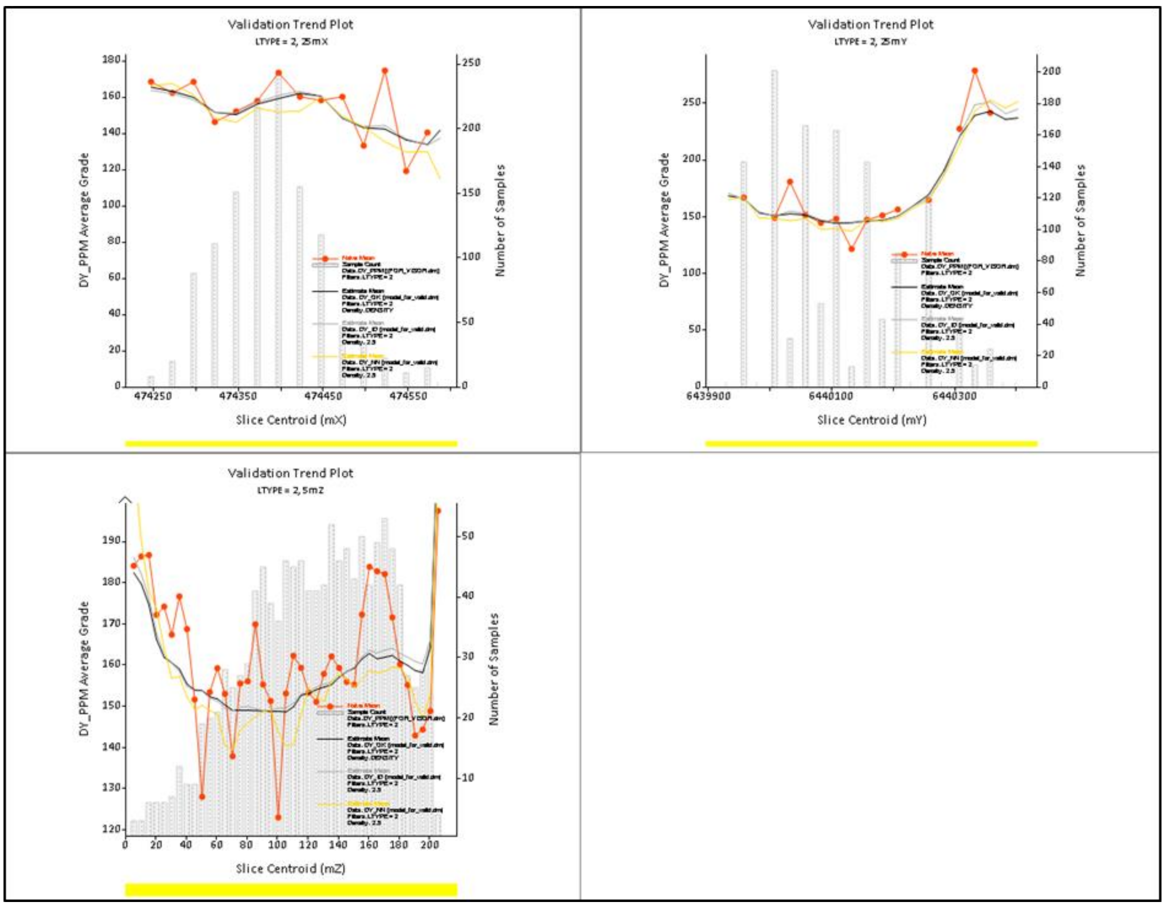


Figure 13-13: Swath plots for Dy in LTYPE=2, grennaite

13.10.4 Model Reconciliation

The Norra Kärr project is a greenfield site and, as such, no reconciliation studies have been carried out.

13.10.5 Validation Summary

Globally no indications of significant over or under estimation are apparent in the model nor were any obvious interpolation issues identified. From the perspective of conformance of the average model grade to the input data, WAI considered the model to be a satisfactory representation of the sample data used and an indication that the grade interpolation has performed as expected. In terms of conformance to the drillhole composite data WAI considered the OK interpolation method to most closely represent the drillhole data. The Mineral Resource estimate is therefore based upon the OK grade estimation.

13.10.6 SRK Comments

SRK has also made a visual validation of the REO grade estimates, the Nb and Zr estimates, and the USD/t derived for each block in the model. SRK agrees the block grade distributions closely match the original drillhole sample data and an appropriate amount of smoothing is evident without any sign of significant bias.

13.11 Depletion

The Norra Kärr project is a greenfield site and, as such, no depletion of the Mineral Resource has been carried out.

13.12 Mineral Resource Classification

13.12.1 CIM Definition Standards

Under the CIM definition standards for Mineral Resources and Mineral Reserves “A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge including sampling”.

A ‘*Measured* Mineral Resource’ is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

An ‘*Indicated* Mineral Resource’ is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An '*Inferred* Mineral Resource' is that part of a Mineral Resource for which quantity and grade are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An *Inferred* Mineral Resource has a lower level of confidence than that applying to an *Indicated* Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of *Inferred* Mineral Resources could be upgraded to *Indicated* Mineral Resources with continued exploration.

13.12.2 SRK Comments

To classify the Norra Kärr deposit SRK considered the following factors:

- Geological continuity and complexity;
- Drill hole coverage, spatial grade continuity - results of geostatistical analysis;
- QA/QC results - quality of data;
- Quality of block model grade and tonnage estimation; and
- Consideration of reasonable prospects for eventual economic extraction ("RPEE") including operating parameters and market.

Geological Continuity and Drilling Coverage

There is a good coverage with diamond drilling for which there is good core recovery and consistent lithological logging and coding has been completed; this allows the 3D form of the lithologies and their arrangement in the overall mineral deposit to be well understood.

Quality of Data

Quality control processes and the data analysis resulting from this give assurance around the reliability of the sample preparation and assaying methods and consequent repeatability of sample assay results for REO grades; however, SRK wishes to see more detail around the QAQC pertaining to Zr and Nb results and to the density determinations to raise confidence in the accuracy or their estimates.

Of note, most of the drilling samples do not have whole rock (major oxide) assay results so it is not possible to calculate mineral proportions in the majority of drillhole samples. In the absence of this an alternative approach such as automated core scanning or quantitative X-ray diffraction on stored pulps may be used to quantify the mineralogy of each existing drillhole sample. Consequently, at this time, it is not possible to model the 3D variability of nepheline syenite grade; or for that matter the quality of the non-magnetic concentrate, both of which may vary within and most likely between lithology types.

In 2015, the WAI MRE reported only REO grades however their market prices have fallen in the intervening period and now the project relies on Zr, Nb and Nepheline Syenite sales to provide some 30% of the revenue in the PEA. Furthermore, the Nepheline Syenite sales need to be successful to achieve the much reduced amount of waste now planned to be disposed of on site.

Given the importance of achieving by-product sales for the economics of the project and the importance of finding a willing nepheline syenite offtaker, SRK would wish to advance these aspects before attaching an Indicated level of confidence to the USD/t calculation used for the determination of RPEEE.

Quality of Block Model

Validation of the block model has shown the estimated grades to be a good reflection of the input composite grades. Visual and statistical checks reveal no evidence of estimation bias affecting the TREO, Zr and Nb grades.

13.13 Reasonable Prospects for Eventual Economic Extraction

The cut-off grade for the MRE is based on a summary of SRK's technical-economic model which is presented in Table 13-9. The table illustrates the gross revenue received by the project (accounting for mineral recoveries but not treatment charges) which for the purpose of reporting a Mineral Resource in a holistic and inclusive manner, includes a premium of 30%.

Based on typical run of mine grade, the gross revenue per tonne milled is attributed to each of the four revenue drivers.

TREO provides some 70% of the gross revenue and the three additional products contribute 5 - 10% each to the total.

The unit operating costs are also provided, these are weighted towards processing and sales, the latter being cost of sales derived from the toll treatment charge associated with TREO that would be netted off by an offtaker; currently this is assumed to be USD19/kg TREO which is some 20% of the life of mine average forecast basket price of 53 USD/kg TREO.

The contribution to the revenue from the by-products emphasises the importance of the Zr, Nb and nepheline syenite to the economic prospects of the project overall. Hence SRK attaches importance to the lesser value products, particularly the confidence we have in the estimation and characterisation of those.

Table 13-9: RPEEE and Cut-off Grade

Annual Mill Feed	Ore	(t)	1,150,000
Typical Grades	TREO	(%)	0.53%
	ZrO_2	(%)	1.52%
	Nb_2O5	(%)	0.06%
	Nepheline Syenite	(%)	65%
Revenue after recovery losses	TREO	(USDk)	179,870
	ZrO_2	(USDk)	25,513
	Nb_2O5	(USDk)	14,331
	Nepheline Syenite	(USDk)	31,058
	Total	(USDk)	250,771
Recovered Revenue per t milled	TREO	($/t ore)	156
	ZrO_2	($/t ore)	22
	Nb_2O5	($/t ore)	12
	Nepheline Syenite	($/t ore)	27
	Total	**($/t ore)**	**218**
RecRev $/t ore multiplier on grade	TREO	per %	295
	ZrO_2	per ppm	0.0015
	Nb_2O5	per ppm	0.0218
	Nepheline Syenite	per t ore	27
Operating Costs LOM average	Mining	($/t ore)	4.3
	Processing	($/t ore)	51.2
	G&A	($/t ore)	5.0
	Transport	($/t ore)	4.9
	Sales	($/t ore)	90.0
	Total	**($/t ore)**	**155.4**

13.14 Mineral Resource Statement

SRK's Mineral Resource statement is based on the WAI block model which has been reviewed and adopted as fit for purpose. In addition to TREO, SRK has also reported Zr, Nb and nepheline syenite as by-products which are understood to be material to the project's economic margin and overall project feasibility.

The four revenue drivers are combined in the block model using a USD/t variable and the MRE is reported above a cut-off value of USD150/t which includes a 30% premium on the metal prices used in the technical-economic model provided in this report.

The MRE is constrained to an open pit shell (overall pit slope angles between 44.4 and 45.2 degrees) similarly based on premium metal prices and the recovered revenues and the operating costs given in Table 13-9, but notably unconstrained by commodity production rates and unconstrained by the 260m buffer zone either side of the nearby highway, which is a generous interpretation of the technically calculated blast fly-rock hazard distance. None of this is considered to be unreasonable for the purposes reporting an MRE, but these are all factors which impact the positioning and size of the pit used to generate the life of mine plan in the technical-economic model in this PEA.

The MRE is classified in the Inferred category due to SRK's awareness of the sensitivity of the tonnage and grade to change in cut-off grade. In terms of the REE composition the content is dominated by HREO (Table 13-11).

Table 13-10: SRK Mineral Resource statement, 18 August 2021

Mineral Resource Classification	Tonnes (Mt)	TREO (%)	ZrO_2 (%)	Nb_2O_5 (%)	Nepheline Syenite (%)
Inferred	110	0.5	1.7	0.05	65

**Notes::*

1. *Effective date 18 August 2021.*
2. *Qualified Person Mr Martin Pittuck MSc C.Eng*
3. *Mineral Resources are not Mineral Reserves until they have Indicated, or Measured confidence and they have modifying factors applied and they have demonstrated economic viability based on a Feasibility Study or Prefeasibility Study.*
4. *There is no guarantee that Inferred Mineral Resources will convert to a higher confidence category after future work is conducted.*
5. *The Mineral Resources reported have been constrained using an open pit shell assuming the deposit will be mined using open pit bulk mining methods and above a cut-off grade of USD150/t., including a 30% premium on projected commodity prices and unconstrained by commodity production rates and the 260m highway buffer zone.*
6. *The Mineral Resources reported represent estimated contained metal in the ground and has not been adjusted for metallurgical recovery.*
7. *Total Rare Earth Oxides (TREO) includes: La2O3, Ce2O3, Pr2O3, Nd2O3, Sm2O3, Eu2O3, Gd2O3, Tb2O3, Dy2O3, Ho2O3, Er2O3, Tm2O3, Yb2O3, Lu2O3, Y2O3.*
8. *Heavy Rare Earth Oxides (HREO) include: Eu2O3, Gd2O3, Tb2O3, Dy2O3, Ho2O3, Er2O3, Tm2O3, Yb2O3, Lu2O3, Y2O3.*
9. *HREO is 52% of TREO*

Table 13-11: Norra Kärr Rare Earth Element Distribution

Light REO proportion of Total REO (%)					Heavy REO proportion of Total REO (%)									
La_2O_3	Ce_2O_3	Pr_2O_3	Nd_2O_3	Sm_2O_3	Eu_2O_3	Gd_2O_3	Tb_2O_3	Dy_2O_3	Ho_2O_3	Er2O3	Tm_2O_3	Yb_2O_3	Lu_2O_3	Y_2O_3
0.100	0.210	0.030	0.110	0.030	0.004	0.030	0.007	0.050	0.010	0.034	0.005	0.033	0.005	0.340
0.48					0.52									

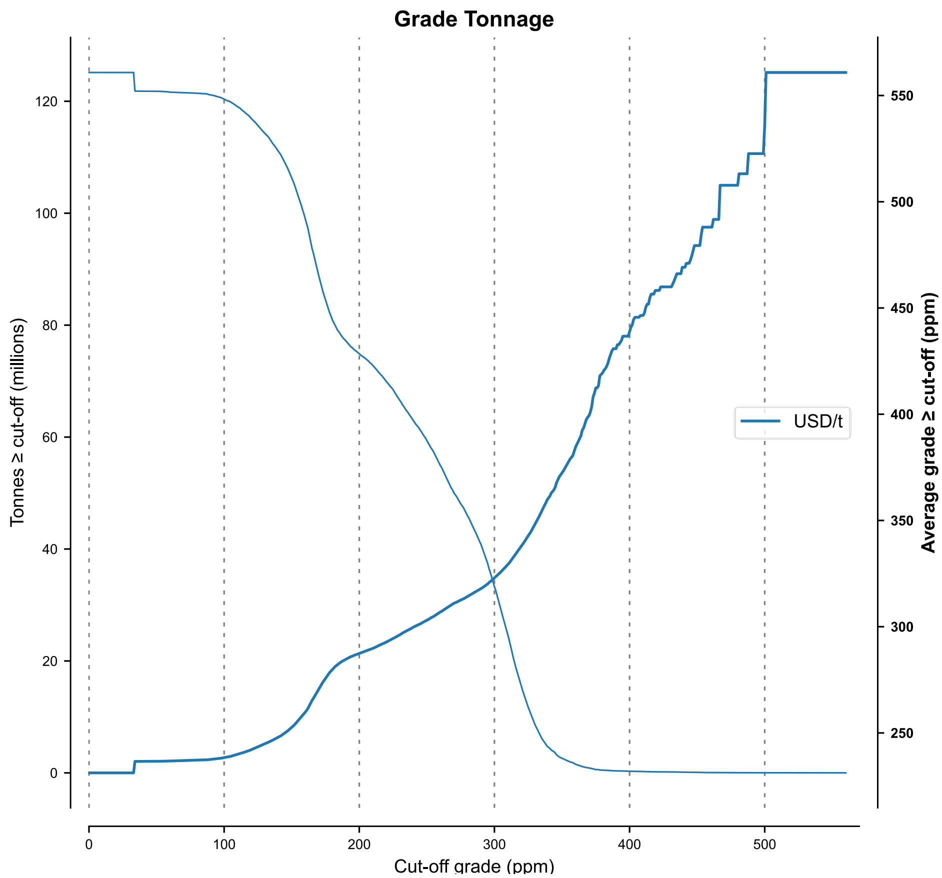


Figure 13-14: Grade-tonnage curve for all material within the SRK 2021 Mineral Resource pit shell

In Figure 13-14 below, the grade-tonnage curve for material inside the pit shell used to constrain the MRE is provided. It shows that the tonnage is very sensitive to cut-off grade at the value of USD150/t which is the cut-off grade used for SRK's MRE.

13.14.1 Comparison to Previous MRE

The SRK MRE is three times the size of the WAI MRE presented in the 2015 PFS owing to several contributing factors despite there being no additional drilling or sampling and the 3D block model utilised being the same.

Change in MRE Quantum

The pit shell used to constrain the current MRE is now much larger than used in the 2015 PFS MRE mainly because SRK has removed the annual and mine life production limit previously applied by WAI on the optimised pit shell. This is considered reasonable for the current MRE but a similar restriction is still used for the technical-economic model in this PEA. It demonstrates the actual size of the mineralised body which is relevant if greater production rates are considered or if a longer-term view of the project is taken.

The pit shell is also no longer constrained by the 260m distance from the highway which SRK understands to be a conservative interpretation of the technically calculated blasting safety buffer. The MRE pit does not meet the road, but it comes halfway into the buffer zone and a revised buffer distance will likely be applied to any mine planning pits generated in the future given that highway diversion is unlikely to be feasible.

Change in MRE Classification

TREO prices are now lower than was previously the case placing more importance onto Zr and Nb by-product and nepheline syenite sales. SRK has described the relatively low confidence associated with the accuracy of the grades and the variability of the nepheline syenite content and has moved the classification from Indicated to Inferred to reflect this.

Table 13-12: Norra Kärr Mineral Resource Statement (WAI, 2014)*

Mineral Resource Classification	Tonnes (Mt)	Density (t/m^3)	TREO (%)	HREO (% of TREO)
Measured	-	-	-	-
Indicated	31	2.7	0.6	52.6
Measured+Indicated	**31**	**2.7**	**0.6**	**52.6**
Inferred	-	-	-	-

14 MINERAL RESERVE ESTIMATES

Not applicable at this stage – no Mineral Reserves have been declared.

15 MINING METHODS

15.1 Mining Modifying Factors

The resource block model was provided for use in the mine planning process. Table 15-1 shows the resource block model properties.

Table 15-1: Resource Block Model Properties

Property	Unit	X	Y	Z
Origin	m	473600	6439300	-134
Number of Blocks	#	62	72	79
Parent Block Size	m	25	25	5
Sub Block Size	m	5	5	1
Rotation	°	-	-	-

The resource block model was regularized to a selective mining unit (“SMU”) of 5 m x 5 m x 5 m to estimate loss and dilution on a local basis. This represents a reasonable SMU for the selected mining equipment, blasting techniques and deposit characteristics. The regularization process involves combining blocks up to the SMU size.

The block value was estimated in the resource block model and the regularized mining block model to estimate the amount of loss and dilution on a value basis. Using the block value allows all elements to be assessed. The ore loss and dilution of the block value within the pit designs are described in Section 15.3.2. The parameters used to assess the block value are described in Section 15.2.1. The resulting loss and dilution for different cut-off values ("COV") for the entire model are shown in Table 15-2. The tonnage vs. value is shown at different COVs in Figure 15-1 for both the resource block model and regularized mining model.

Table 15-2: Loss & dilution by cut-off value

Cut Off Value (USD/t)	Loss	Dilution
10	0.2%	2.3%
20	0.3%	1.7%
30	0.4%	1.4%
40	0.7%	0.9%
50	1.1%	0.4%
60	1.4%	0.1%
70	1.5%	0.0%
80	1.0%	0.3%
90	0.7%	0.5%
100	1.5%	0.3%
110	2.2%	0.1%
120	2.5%	0.1%
130	2.7%	0.2%
140	3.4%	0.5%
150	5.4%	1.1%

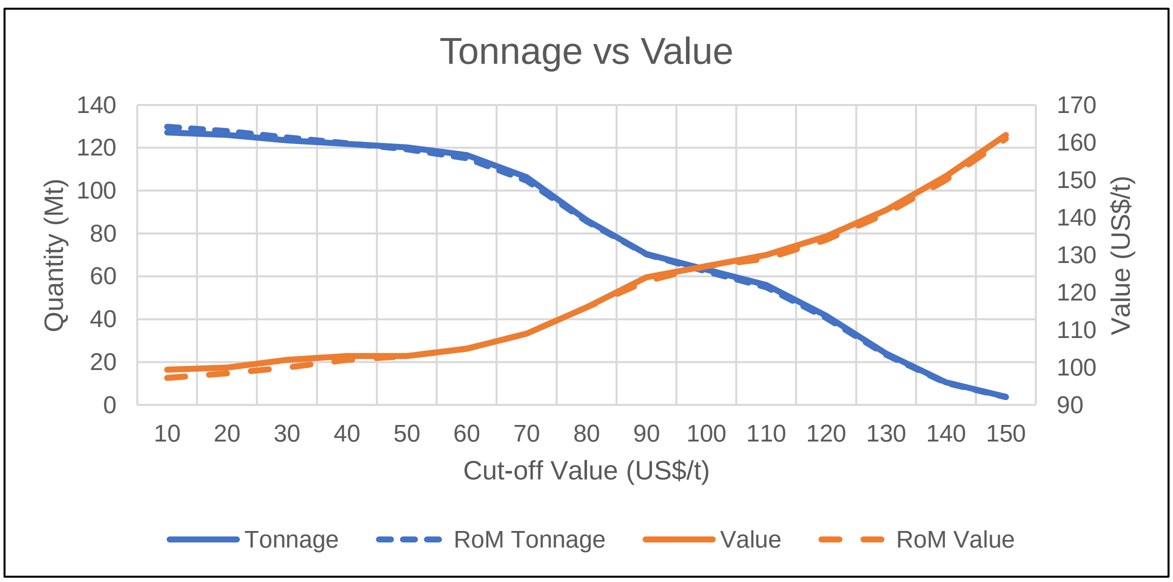


Figure 15-1: Resource and mining model tonnage vs value

15.2 Pit Optimization

15.2.1 Pit Optimization Parameters

The pit optimization parameters have been estimated based on the costs from the PFS Report (GBM Minerals Engineering Consultants Limited, 2015) which have been escalated to 2021 costs, up-to-date price forecasts, and estimates provided by the Client and SRK. Measured, Indicated, and Inferred classified Mineral Resources have been included in the pit optimization.

The pit optimization parameters used in the pit optimization are shown in Table 15-3 and Table 15-4. The slope parameters have been described in further detail in Section 23.4 and have been adjusted to include two ramps in the walls.

Table 15-3: Pit optimization commodity price & metallurgical recovery

Element	Commodity Prices (USD/kg)	Total Recovery
Ce_2O_3	2.02	84.1%
Dy_2O_3	340.81	84.1%
Er_2O_3	25.22	84.1%
Eu_2O_3	40.91	84.1%
Gd_2O_3	27.47	84.1%
Ho_2O_3	62.05	84.1%
La_2O_3	2.79	84.1%
Lu_2O_3	630.54	84.1%
Nd_2O_3	81.00	84.1%
Pr_2O_3	86.00	84.1%
Sm_2O_3	2.22	84.1%
Tb_2O_3	709.80	84.1%
Y_2O_3	4.75	84.1%
Yb_2O_3	16.86	84.1%
ZrO_2	4.00	48.6%
Nb_2O_3	35.00	81.6%

Table 15-4: Pit optimization parameters 2021

Parameters	Units	Value
Geotechnical		
West (225˚ to 0˚)	˚	44.4
East (0˚ to 135˚)	˚	45.2
South (135˚ to 225˚)	˚	45.2
Mining Factors		
Dilution[1]	%	-
Loss[1]	%	-
Limits		260 m from highway

Parameters	Units	Value
Processing Factors		
Magnetic Yield	%	9.1
Aegirine Waste Yield	%	25.9
Non-Magnetic Yield	%	65.0
Operating Costs		
Base Mining Cost	USD/t mined	3.50
Incremental Cost	USD/t/10m	0.020
Reference Level	mRL	211
G&A	USD/t plant feed	5.00
Processing	USD/t plant feed	51.64
Transport Cost	USD/t plant feed	5.70
Cut-off Value (COV)	**USD/t plant feed**	**62.34[3]**
Selling Costs		
Royalty	%	0.20
Treatment & Refining TREO[2]	USD/kg	19.00
Cashflow Analysis		
Production Rate	Mtpa	1.15
Discount Rate	%	10

[1]Regularized mining model (5 m x 5 m x 5m) used in pit optimization, no additional loss or dilution has been applied.
[2]TREO: Ce_2O_3, Dy_2O_3, Er_2O_3, Eu_2O_3, Gd_2O_3, Ho_2O_3, La_2O_3, Lu_2O_3, Nd_2O_3, Pr_2O_3, Sm_2O_3, Tb_2O_3, Tm_2O_3, Y_2O_3, Yb_2O_3
[3] Note this does not include external costs so is different to the MRE cut-off value of USD150/t

A 260 m exclusion zone has been established from the edge of the highway, as shown in Figure 15-2. The offset has been based on the analysis undertaken in the PFS Report (GBM Minerals Engineering Consultants Limited, 2015) which estimated the potential fly rock distance to be 160 m.

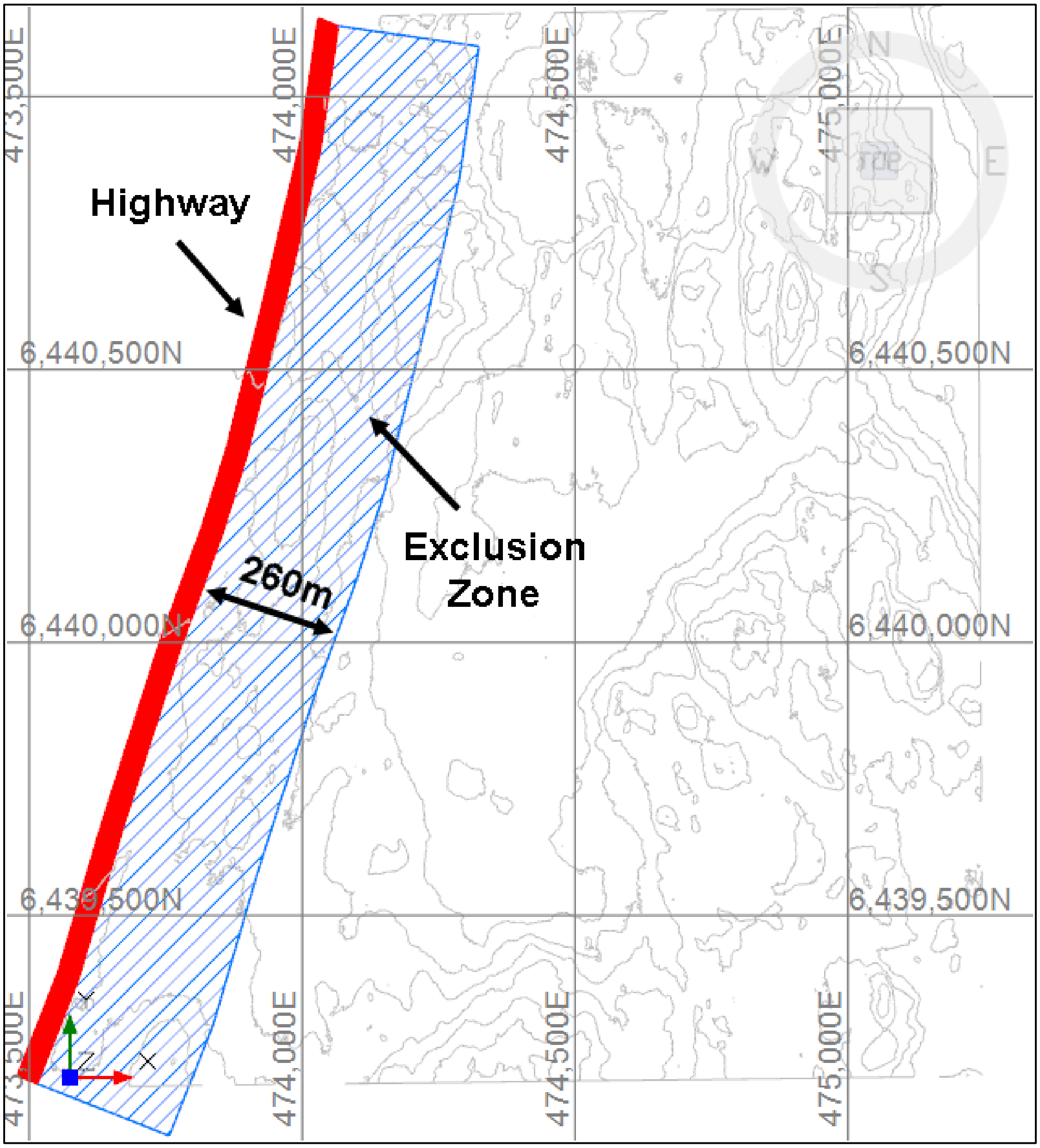


Figure 15-2: Pit optimization highway exclusion zone

15.2.2 Pit Optimization Results

The pit optimization results are shown for selected revenue factors ("RF") in Table 15-5 and Figure 15-3. SRK notes that the revenue, processing recoveries and selling costs have been calculated within the model and therefore the optimization RF results include variation of these three parameters and not solely the selling price.

The revenue sensitivity in Figure 15-3 shows a steep curve until RF 61%, where there is a jump to 62%. The remaining RFs are relatively flat and level off at RF greater than 90%. These results suggest that the deposit is sensitive at revenues less than 62%.

Table 15-5: Pit optimization results

Revenue Factor	%	55	56	58	62	67	80	100
Inventory	**Units**							
Total	**Mt**	**35.5**	**38.9**	**46.9**	**83.5**	**100.1**	**117.2**	**135.3**
Waste	Mt	8.5	9.2	11.1	23.8	29.7	35.2	42.2
Strip Ratio	t:t	0.32	0.31	0.31	0.40	0.42	0.43	0.45
RoM	**Mt**	**27.0**	**29.7**	**35.8**	**59.7**	**70.4**	**81.9**	**93.1**
Zr	ppm	13,984	13,931	13,818	13,171	13,040	13,022	12,717
Ce_2O_3	ppm	1,132	1,142	1,150	1,197	1,154	1,064	993
Dy_2O_3	ppm	267	265	260	247	241	232	224
Er_2O_3	ppm	185	183	180	171	168	163	159
Eu_2O_3	ppm	21	21	21	20	19	18	17
Gd_2O_3	ppm	191	190	187	180	175	165	157
Ho_2O_3	ppm	59	59	58	55	54	52	51
La_2O_3	ppm	502	507	513	541	521	481	448
Lu_2O_3	ppm	24	24	23	22	22	22	21
Nd_2O_3	ppm	605	608	606	613	591	547	513
Pr_2O_3	ppm	149	150	150	154	149	137	128
Sm_2O_3	ppm	170	169	167	165	159	149	141
Tb_2O_3	ppm	39	39	38	37	36	34	33
Tm_2O_3	ppm	28	28	28	26	26	25	25
Y_2O_3	ppm	2,105	2,091	2,057	1,972	1,920	1,829	1,751
Yb_2O_3	ppm	174	173	170	162	159	156	152
Nb_2O_5	ppm	581	578	571	552	537	509	482
Value	USD/t RoM	120.02	119.20	117.40	111.09	109.02	106.26	102.95
Operating Costs								
Mining	USDm	131	144	175	318	383	448	519
Processing, Transport & G&A	USDm	1,684	1,849	2,234	3,721	4,389	5,108	5,802
Unit Operating Costs								
Mining	USD/t mined	3.70	3.71	3.72	3.81	3.82	3.82	3.83
	USD/t RoM	4.87	4.86	4.88	5.32	5.44	5.47	5.57
Processing, Transport & G&A	USD/t RoM	62.34	62.34	62.34	62.34	62.34	62.34	62.34
Cashflow								
Revenue - Selling Costs	USDm	3,242	3,536	4,208	6,631	7,676	8,706	9,582
Cashflow	USDm	1,427	1,542	1,798	2,592	2,904	3,150	3,261
Best DCF	USDm	555.6	565.3	580.0	594.0	595.1	595.5	595.5
Avg DCF	USDm	518.6	522.7	525.9	513.9	499.2	480.6	471.3
Worst DCF	USDm	481.5	480.1	471.8	433.8	403.2	365.7	347.0
Production								
Production Rate	Mtpa	1.15	1.15	1.15	1.15	1.15	1.15	1.15
Mine Life	yrs	23.5	25.8	31.2	51.9	61.2	71.3	80.9

Notes: RoM run-of-mine, DCF: Discounted Cashflow, G&A: General & Administrative

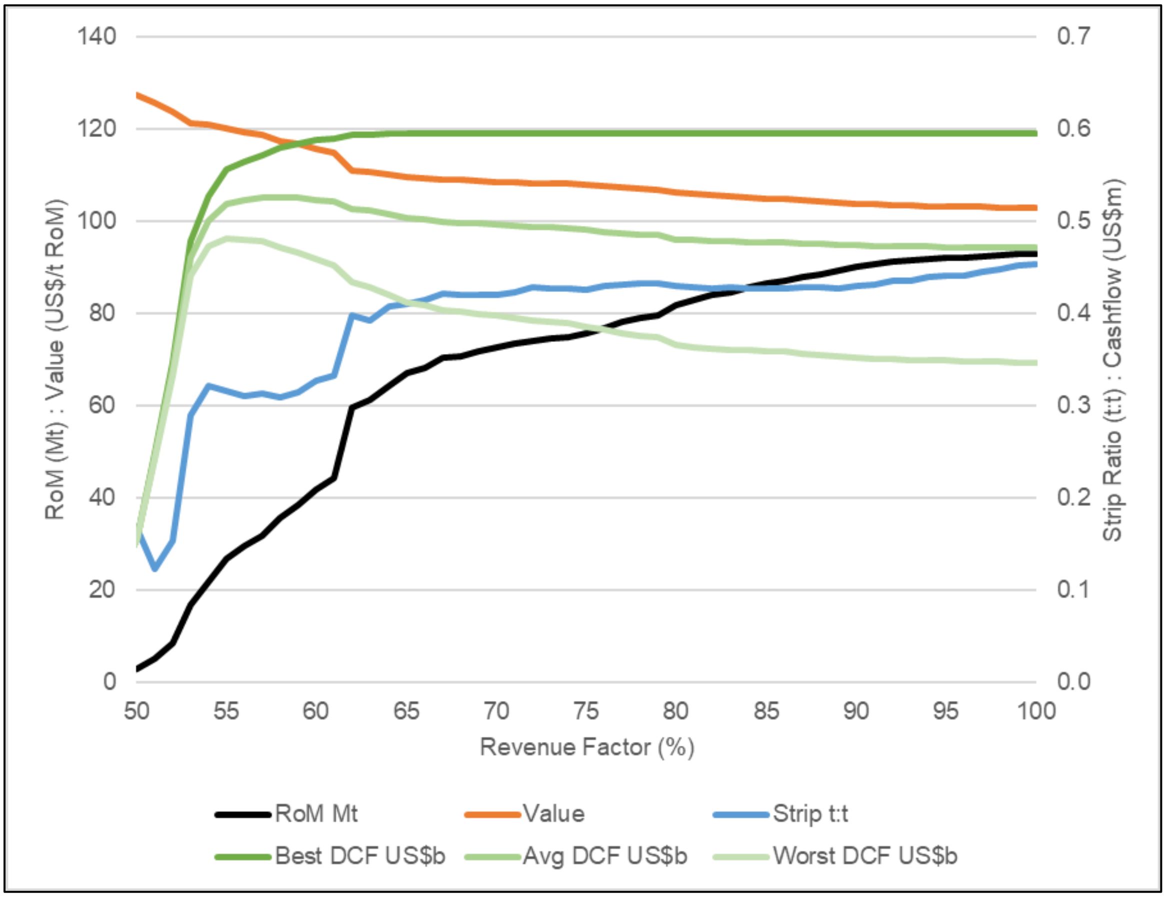


Figure 15-3: Pit optimization results (DCF: Discounted Cashflow, RoM: run-of-mine

The shell outlines for various RFs are shown in Figure 15-4.

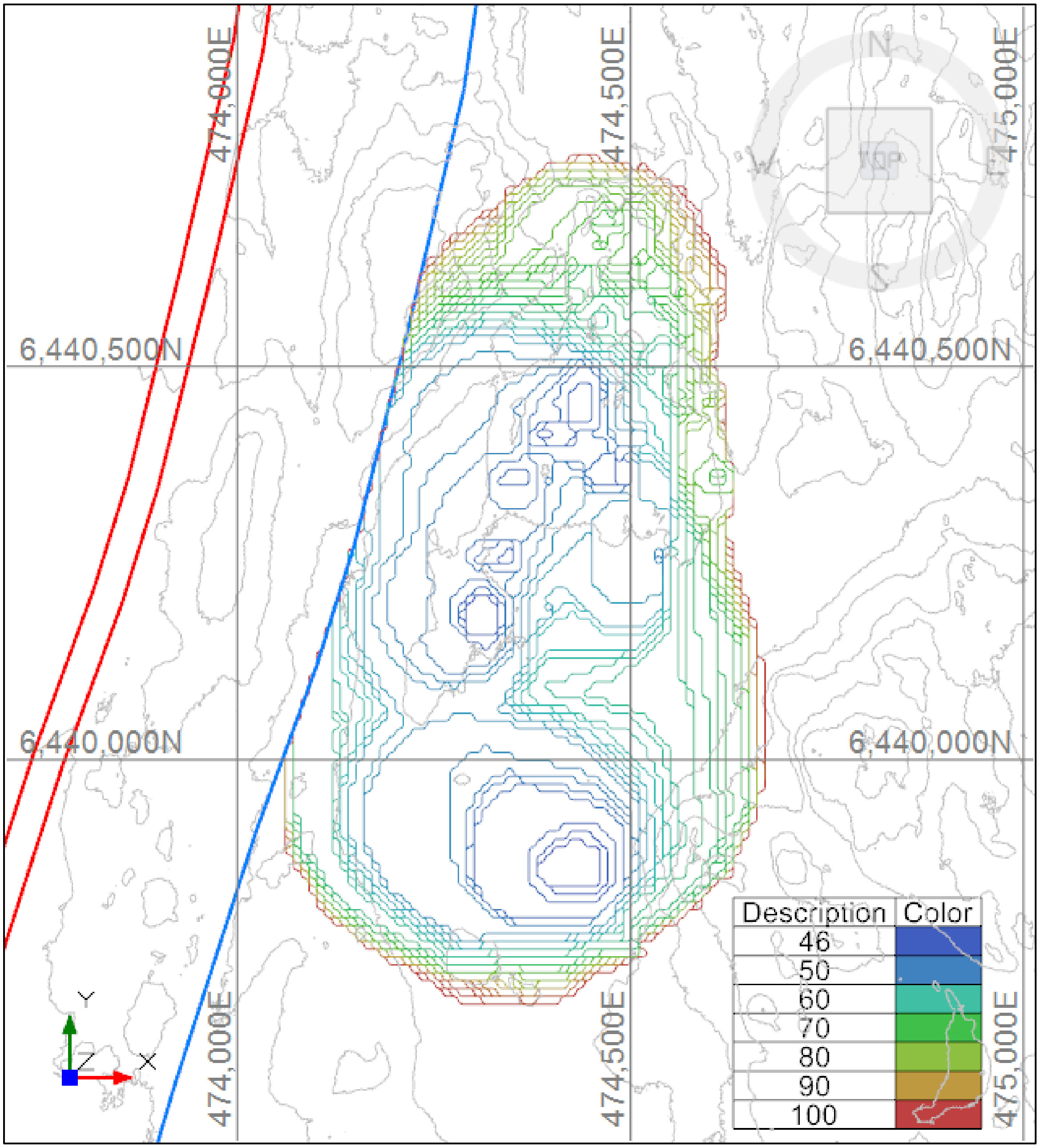


Figure 15-4: Pit optimization revenue factor outlines in plan view

A discounted cashflow analysis was undertaken on the pit optimization results based on a production rate of 1.15 Mtpa and a discount rate of 10%. The production rate has been based on the expected market. The best case cashflow assumes that each RF is extracted in sequence as a separate pushback. The worst case assumes the shell is extracted bench by bench. In general, the average between these two cases provides a more realistic scenario. RF 58% has the highest average discounted cashflow, however there is no significant difference in cashflows between RF 56% to 60%.

RF 56% was selected as the basis of the pit design, which provides a 25 ½ year mine life, 29.7 Mt of run-of-mine (RoM) and 9.2 Mt of waste for a strip ratio of 0.31.

15.3 Mine Design

15.3.1 Design Parameters

The geotechnical slope parameters are shown in Table 15-6 and described in more detail in Section 23.4. The road designs parameters are shown in Table 15-7. The road widths have been based on 3.5 times the width of a Caterpillar 773G haul truck. The pits have been designed with a 10 m mining bench height assuming a 5 m flitch.

Table 15-6: Geotechnical slope parameters

Slope Region	Bench Height (m)	Bench Face Angle (°)	Berm Width (m)	Inter-Ramp Angle (°)
West Upper 40 m	10	70	7	43.2
East Upper 20 m	10	70	7	43.2
South Upper 20 m	10	70	7	43.2
West Lower	20	70	10	49.2
East Lower	20	70	10	49.2
South Lower	20	70	10	49.2

Table 15-7: Road design parameters

Parameters	Units	Value
Ramp Width Dual Lane	m	20
Ramp Width Single Lane	m	15
Maximum Ramp Gradient	%	10

15.3.2 Pit Design

The staged and ultimate pit designs are shown in Figure 15-5. The ramps exit towards the east where the waste rock storage facility (WRSF) and crusher will be located. The ramps between Stage 3 and 4 are connected at the 151 m level, at which point either ramp exit can be used. The connected ramps will also be used to allow waste from Stage 4 to be backfilled in the Stage 3 pit. Single lane ramps have been used in the lower 50 m of the pits.

A comparison to the selected optimized shell is shown in Figure 15-6 and Table 15-8. The difference in quantities less than 2% which SRK believes is within acceptable tolerances.

The pit inventories have been compared between the resource block model and the regularized mining model. The comparison shows that within the pit design the average loss is 6.7% and the average dilution is 5.8%.

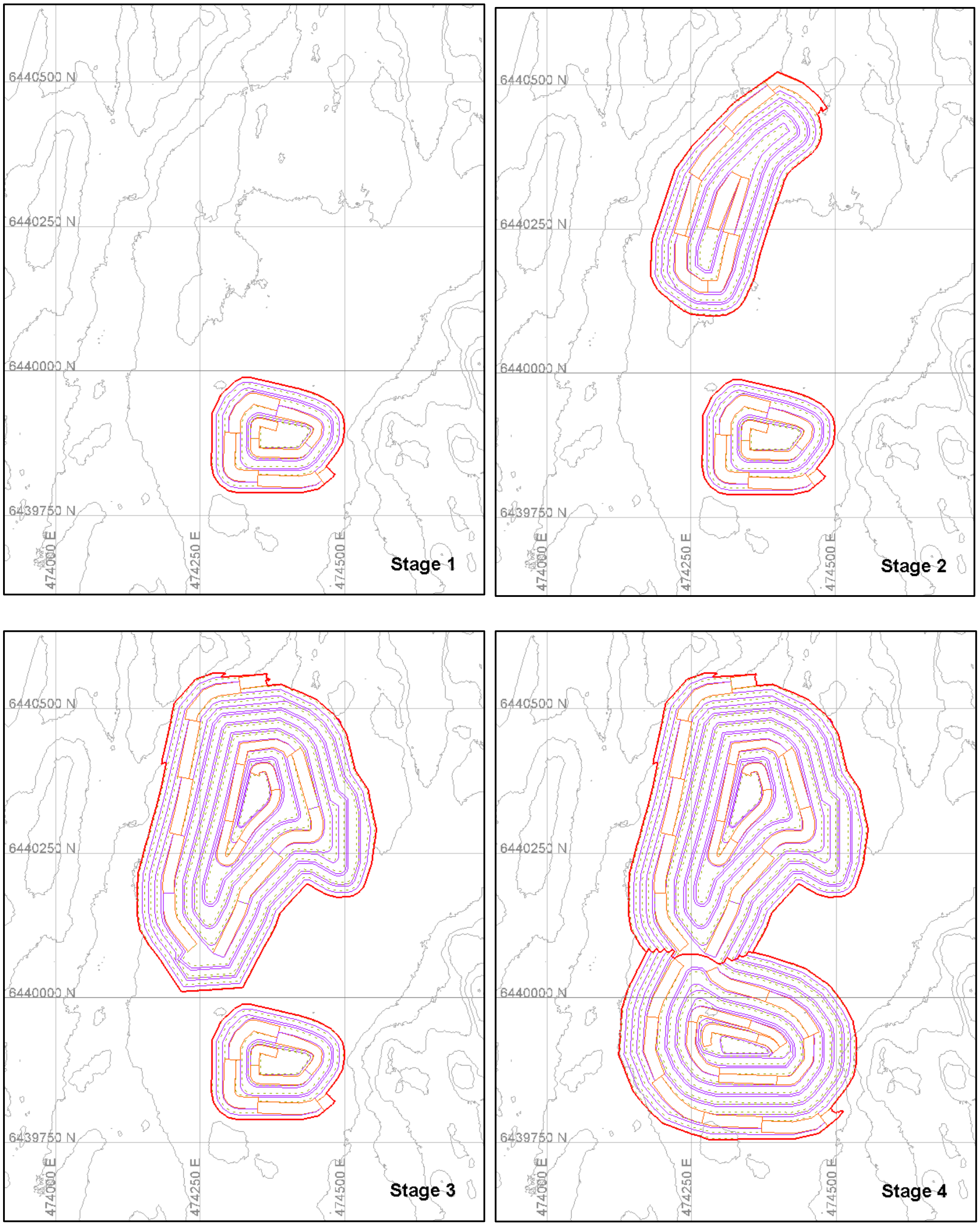


Figure 15-5: Pit design stages

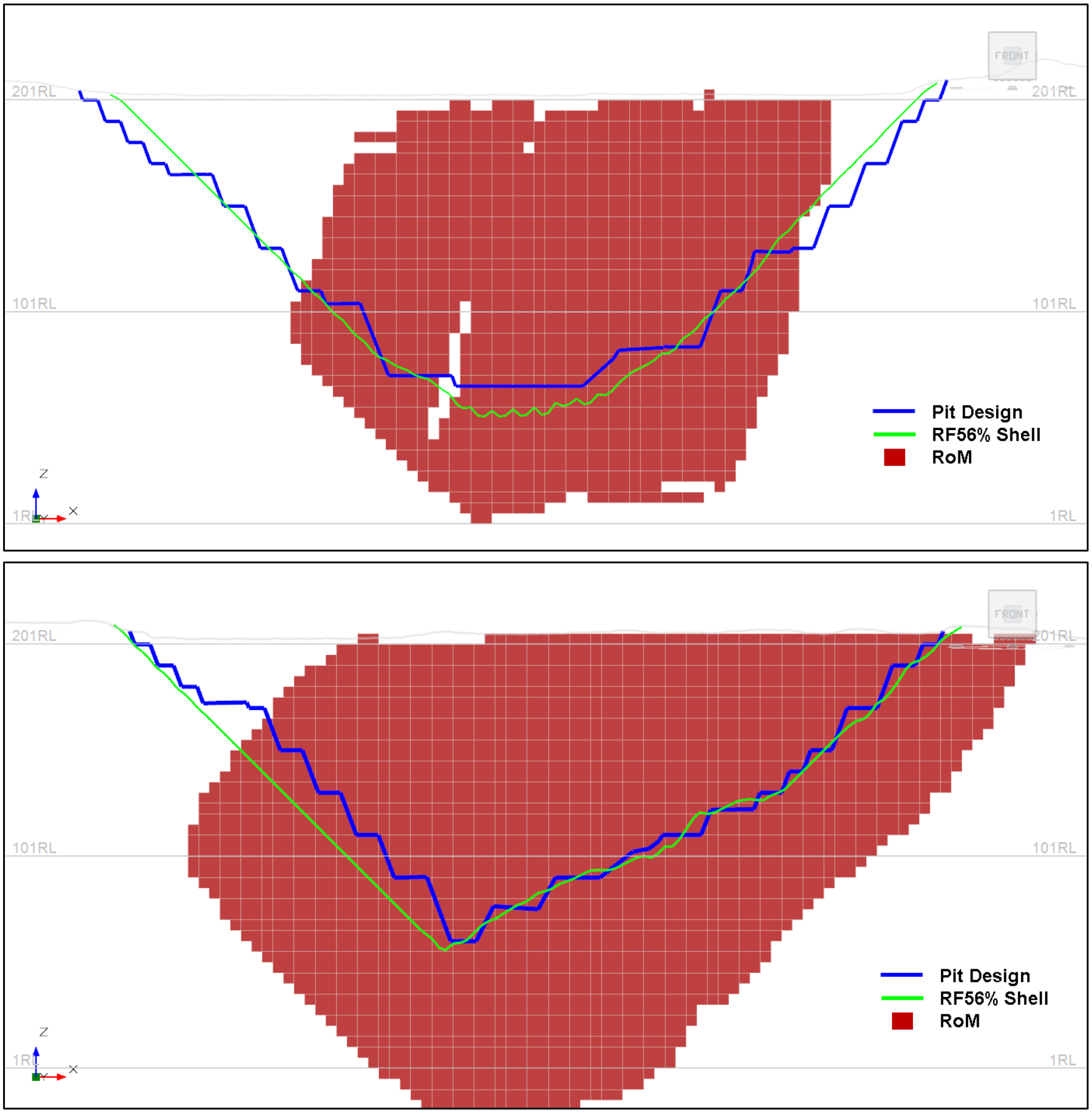


Figure 15-6: Comparison between pit designs and shell looking south (top) and north (bottom)

Table 15-8: Inventory comparison between pit design & shell

Inventory	Units	RF56% Shell	Pit Design	Difference	Difference
Total	kt	38,872	38,687	-185	-0.5%
Waste	kt	9,211	9,372	161	1.7%
Strip Ratio	t:t	0.31	0.32	0.01	2.9%
RoM	kt	29,661	29,315	-345	-1.2%
Ce_2O_3	ppm	1,150	1,155	5	0.4%
Dy_2O_3	ppm	260	261	1	0.3%
Er_2O_3	ppm	180	180	0	0.1%
Eu_2O_3	ppm	21	21	0	0.7%
Gd_2O_3	ppm	187	188	1	0.6%
Ho_2O_3	ppm	58	58	0	0.1%
La_2O_3	ppm	513	514	1	0.2%
Lu_2O_3	ppm	23	23	0	-0.1%
Nd_2O_3	ppm	606	610	4	0.6%
Pr_2O_3	ppm	150	151	1	0.6%
Sm_2O_3	ppm	167	169	1	0.7%
Tb_2O_3	ppm	38	38	0	0.4%
Tm_2O_3	ppm	28	28	0	0.0%
Y_2O_3	ppm	2,057	2,066	9	0.4%
Yb_2O_3	ppm	170	170	0	-0.1%
ZrO_2	ppm	18,668	18,602	-66	-0.4%
Nb_2O_3	ppm	571	571	0	0.0%

The initial staged pit designs were based on the smaller revenue factor shells (Figure 15-4) and are shown in Figure 15-7. The quantities by stage are shown in Table 15-9.

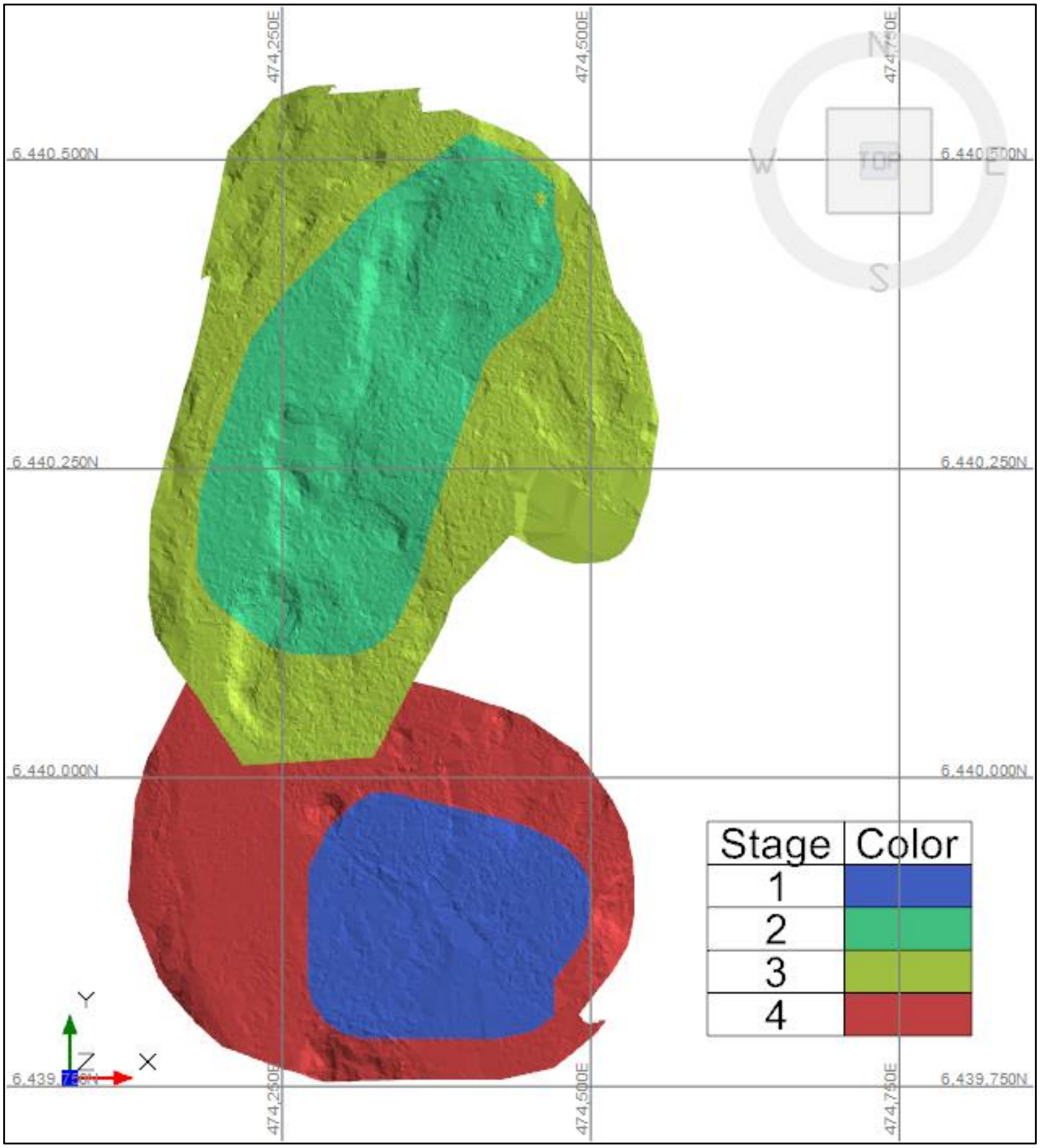


Figure 15-7: Staged pit designs

Table 15-9: Pit design inventory by stage

Inventory	Units	Total	Stage 1	Stage 2	Stage 3	Stage 4
Total	kt	**38,687**	2,371	6,217	17,141	12,958
Waste	kt	**9,372**	443	974	3,355	4,599
RoM	kt	**29,315**	1,928	5,242	13,786	8,359
Strip Ratio	t:t	**0.32**	0.23	0.19	0.24	0.55

15.3.3 Waste Rock Storage Facility

The waste will be stored in an external Waste Rock Storage Facility (WRSF) and backfilled in the northern part of the pit once that area has been mined. The external WRSF design criteria are shown in Section 23.3 and has a capacity of 8.8 M loose cubic meters. The backfill storage facility (BSF)has been designed from the 131 m level to expand radially from the ramp at 35°. The BSF has a capacity of 1.35 m loose cubic meters. The external WRSF and BSF are shown in Figure 15-8.

SRK expects that some of the waste mined in the earlier years of the operation will be used for construction purposes as required.

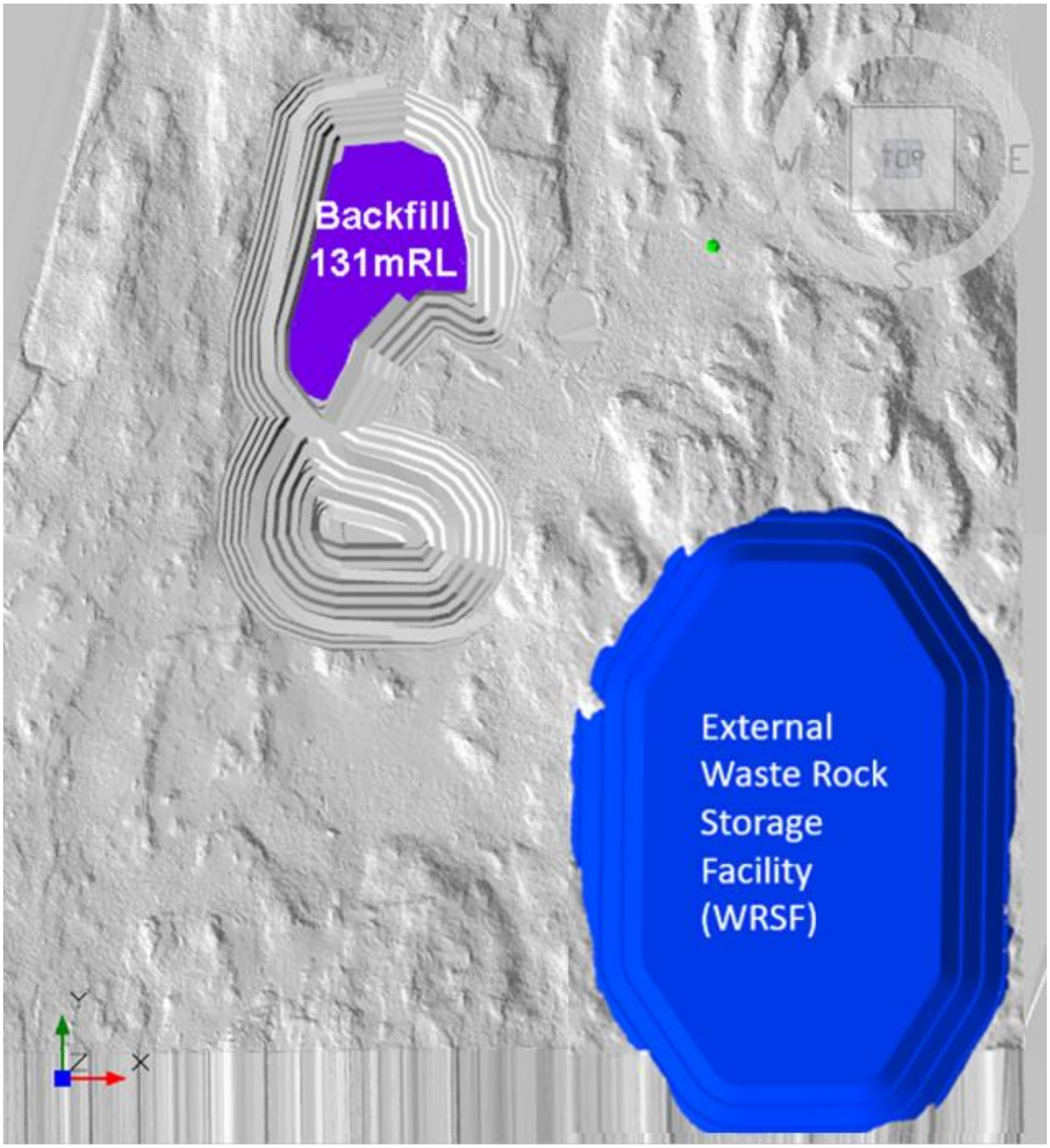


Figure 15-8: External waste & backfill storage locations

15.4 Mine schedule

The mine schedule has been based on the following criteria:

- Crusher feed target: 1.15 Mtpa
- Maximize waste backfill quantities

The mine schedule is shown by material type in Figure 15-9 and by stage in Figure 15-10. The sequence starts in Stage 1, with Stage 2 commencing in Year 2. Stage 3 begins in Year 3, while Stage 4 is delayed until Year 16 to maximize backfill options. Total production averages 1,625 ktpa from Year 3 to 9, after which total material movement decreases as the strip ratio in Stage 3 decreases. Waste stripping requirements increase starting in Year 16 as Stage 4 begins, averaging 1.8 Mtpa until Year 20.

The crusher feed schedule by stage is shown in Figure 15-11. Crusher feed targets are met starting in Year 1 due to the limited waste stripping requirements.

The mine schedule and RoM grades are shown in Table 15-10 and Table 15-11. The mine schedule plots are shown in Figure 15-12 and Figure 15-13, with active mining and dumping coloured in dark gray and active haulage routes in red.

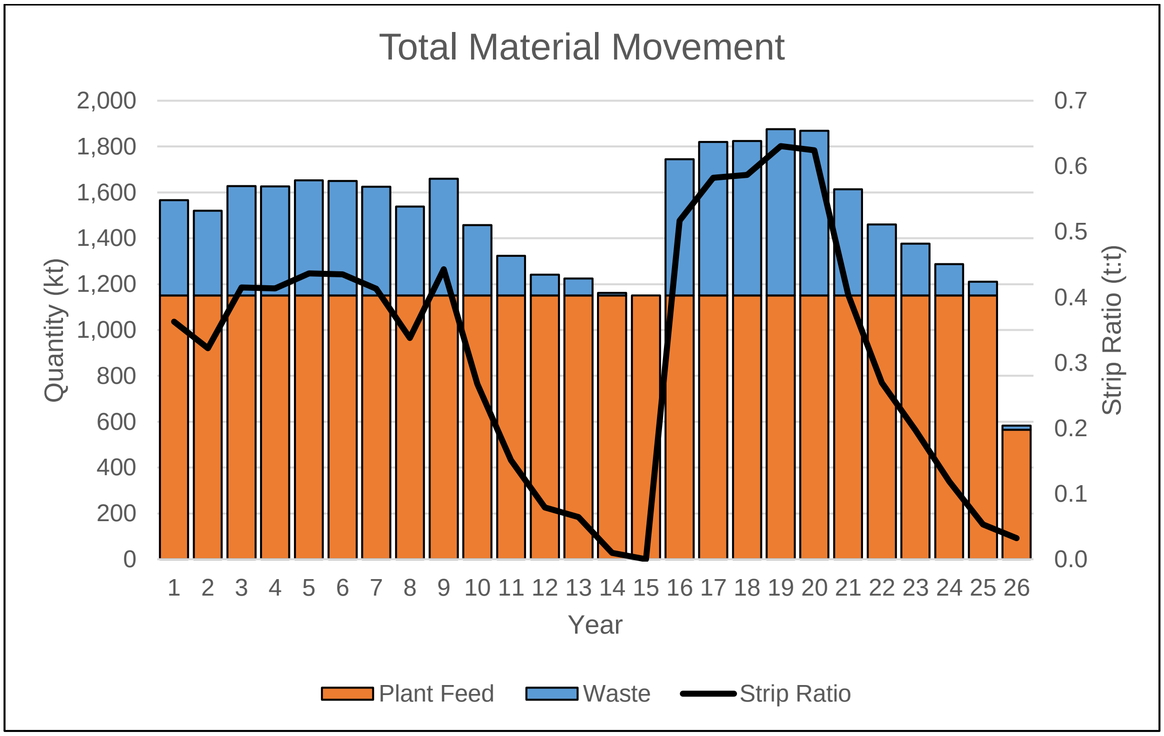


Figure 15-9: Total Material Movement

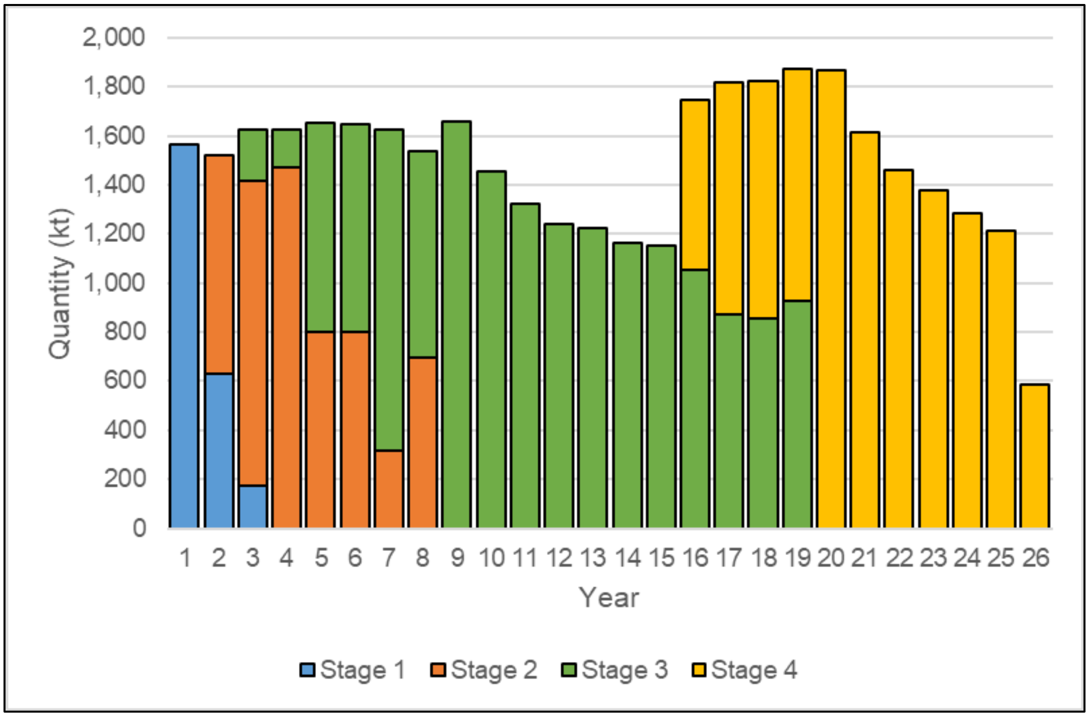


Figure 15-10: Total Mineral Movement by Stage

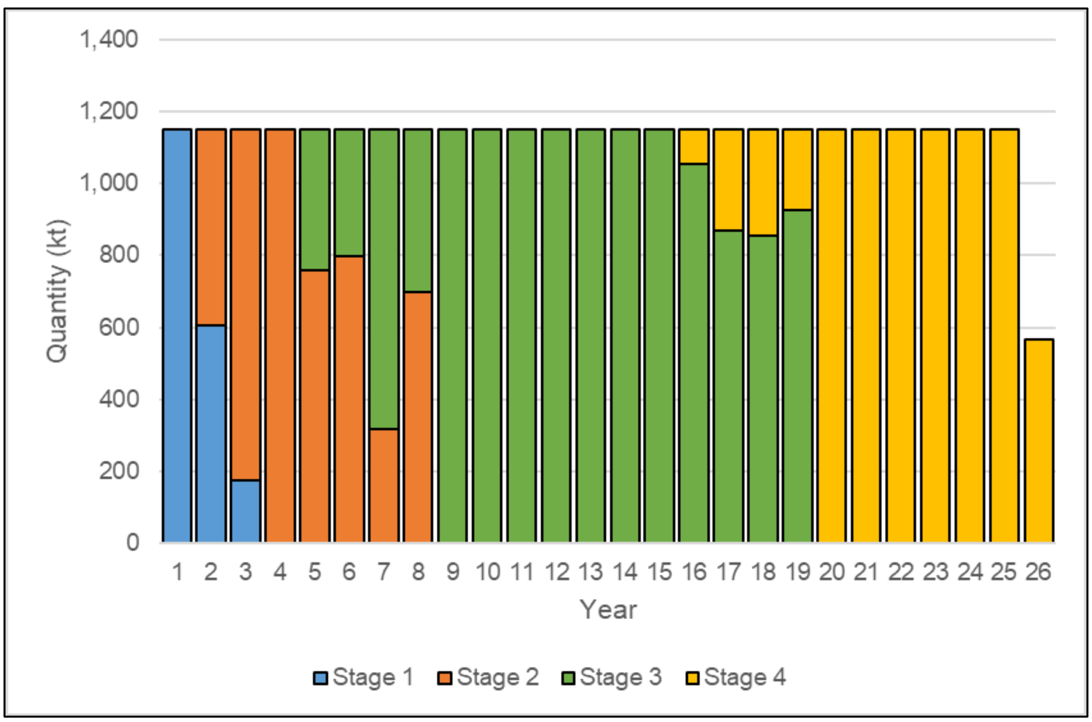


Figure 15-11: Crusher Feed by Stage

Table 15-10: Mine Schedule Year 1 to 13

Mine Schedule	Units	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Total	kt	**29,315**	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150
Waste	kt	**9,372**	417	370	477	476	502	500	475	389	509	308	174	91	74
Strip Ratio	t:t	**0.32**	0.36	0.32	0.41	0.41	0.44	0.43	0.41	0.34	0.44	0.27	0.15	0.08	0.06
RoM	kt	**29,315**	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150
Ce_2O_3	ppm	**1,155**	1,099	1,126	1,098	976	1,031	1,140	1,254	1,196	1,200	1,141	1,262	1,143	1,155
Dy_2O_3	ppm	**261**	245	276	278	253	255	275	247	272	237	232	246	247	259
Er_2O_3	ppm	**180**	165	188	193	176	175	189	169	187	163	161	168	170	179
Eu_2O_3	ppm	**21**	19	22	22	20	20	22	21	23	19	19	21	20	21
Gd_2O_3	ppm	**188**	176	197	198	178	179	197	185	200	173	167	182	173	185
Ho_2O_3	ppm	**58**	54	62	62	57	57	61	55	60	54	53	55	56	58
La_2O_3	ppm	**514**	491	498	477	418	461	492	565	518	540	517	564	518	518
Lu_2O_3	ppm	**23**	21	24	25	23	23	24	22	24	21	21	22	22	23
Nd_2O_3	ppm	**610**	571	611	601	536	564	626	642	648	614	584	648	591	614
Pr_2O_3	ppm	**151**	142	149	147	130	138	153	162	160	156	149	165	150	154
Sm_2O_3	ppm	**169**	162	176	174	158	159	176	167	179	159	153	170	159	168
Tb_2O_3	ppm	**38**	36	40	41	37	37	40	37	40	35	34	36	36	38
Tm_2O_3	ppm	**28**	25	29	30	28	27	29	26	28	25	25	26	26	27
Y_2O_3	ppm	**2,066**	1,945	2,191	2,191	1,987	1,982	2,180	1,964	2,197	1,840	1,794	1,945	1,911	2,032
Yb_2O_3	ppm	**170**	157	177	182	169	166	177	158	175	155	154	159	163	169
ZrO_2	ppm	**18,602**	15,206	18,111	20,134	20,096	19,327	20,207	17,431	19,812	17,824	17,672	18,328	18,912	19,974
Nb_2O_3	ppm	**571**	582	625	598	530	532	574	535	582	525	508	548	536	552

Table 15-11: Mine Schedule Year 14 to 26

Mine Schedule	Units	Total	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24	Y25	Y26
Total	kt	**29,315**	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	565
Waste	kt	**9,372**	11	0	595	670	675	725	718	464	310	227	137	61	18
Strip Ratio	t:t	**0.32**	0.01	0.00	0.52	0.58	0.59	0.63	0.62	0.40	0.27	0.20	0.12	0.05	0.03
RoM	kt	**29,315**	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	565
Ce_2O_3	ppm	**1,155**	1,191	1,092	1,123	1,187	1,231	1,070	993	1,077	1,200	1,214	1,232	1,327	1,395
Dy_2O_3	ppm	**261**	268	281	268	265	273	264	206	221	241	277	286	319	341
Er_2O_3	ppm	**180**	184	196	187	184	188	185	146	154	165	190	198	221	240
Eu_2O_3	ppm	**21**	21	22	22	21	23	20	16	18	20	22	23	25	26
Gd_2O_3	ppm	**188**	193	196	191	191	202	188	147	158	176	200	207	231	243
Ho_2O_3	ppm	**58**	60	62	59	58	60	59	46	49	53	61	63	70	76
La_2O_3	ppm	**514**	530	483	495	522	530	461	463	495	547	549	546	589	629
Lu_2O_3	ppm	**23**	24	25	24	23	24	24	20	20	22	25	25	28	30
Nd_2O_3	ppm	**610**	631	600	604	616	653	562	491	544	611	646	655	712	759
Pr_2O_3	ppm	**151**	157	146	147	152	160	137	126	139	154	159	161	174	184
Sm_2O_3	ppm	**169**	172	172	170	171	181	161	132	146	163	179	184	202	215
Tb_2O_3	ppm	**38**	39	41	39	38	40	38	30	32	35	41	42	47	51
Tm_2O_3	ppm	**28**	28	30	28	28	28	28	23	24	25	29	30	34	37
Y_2O_3	ppm	**2,066**	2,090	2,211	2,108	2,106	2,197	2,055	1,609	1,740	1,930	2,229	2,302	2,581	2,738
Yb_2O_3	ppm	**170**	174	185	176	172	176	177	142	147	157	179	183	202	217
ZrO_2	ppm	**18,602**	19,754	20,886	19,618	19,018	18,861	20,050	15,261	15,939	16,630	17,927	18,157	19,420	19,616
Nb_2O_3	ppm	**571**	565	591	566	556	588	555	451	503	543	654	650	731	791

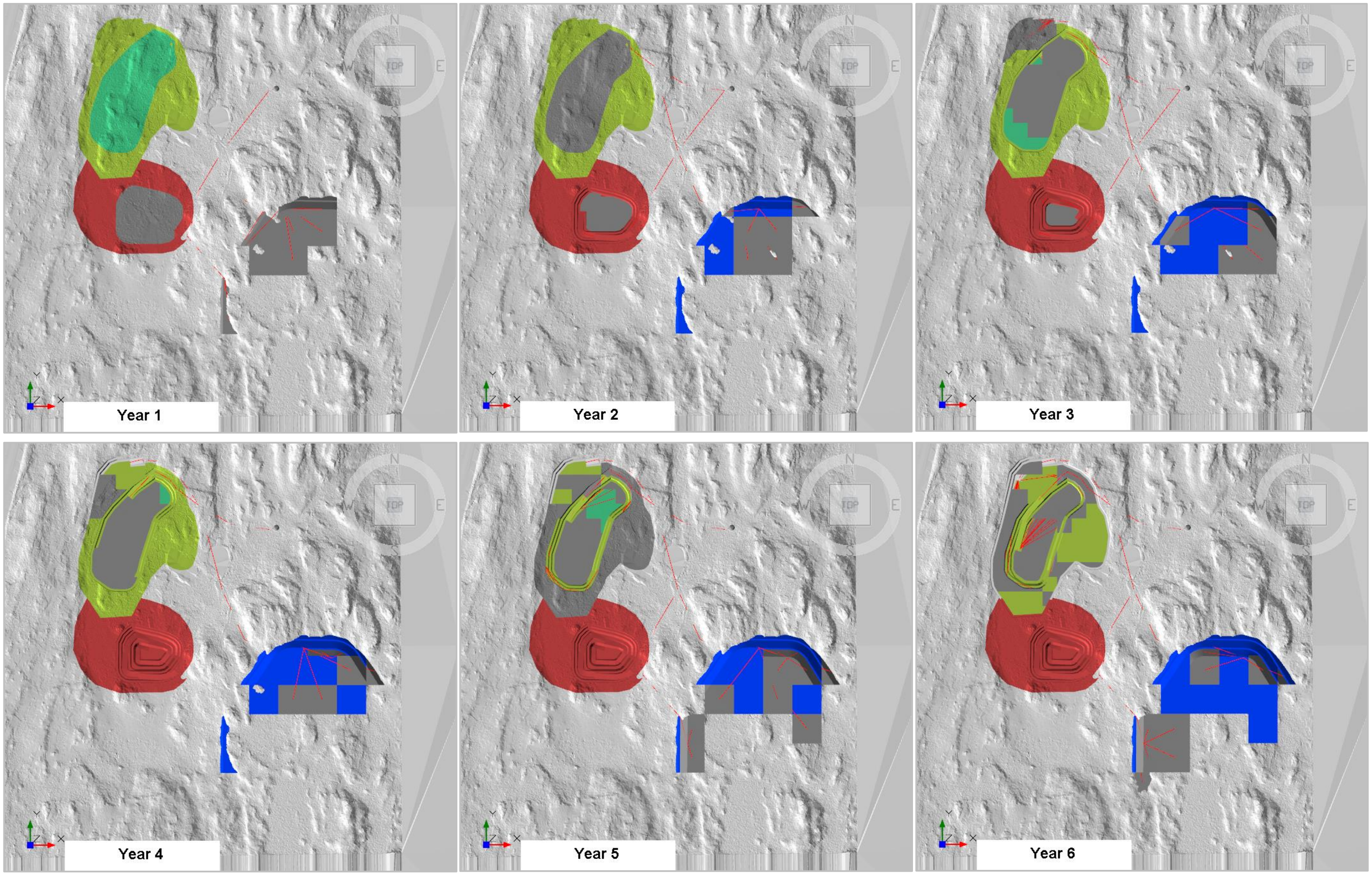


Figure 15-12: Mine schedule plots year 1 to 6

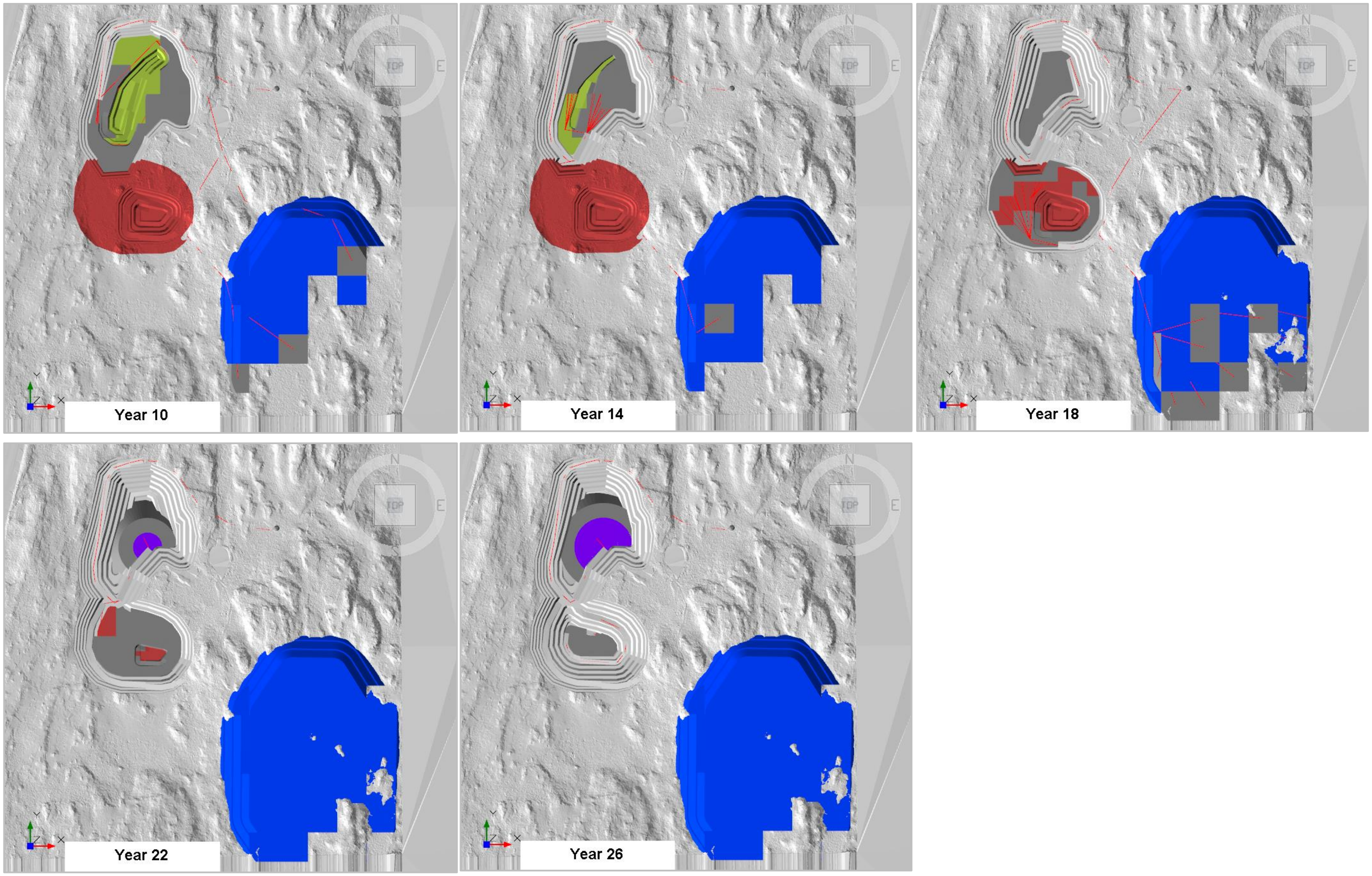


Figure 15-13: Mine schedule plots year 10 to 26

The annual waste destination schedule is shown in Figure 15-14. Waste material will be sent to the external waste dump from Year 1 to 19 (Stage 1 to 3), except for Year 15 where there is no waste mined. In Year 20 mining in Stage 3 has been completed and waste from Stage 4 is directed to backfill the void in Stage 3. Backfilling accounts for 21% of the waste mined (1.9 Mt).

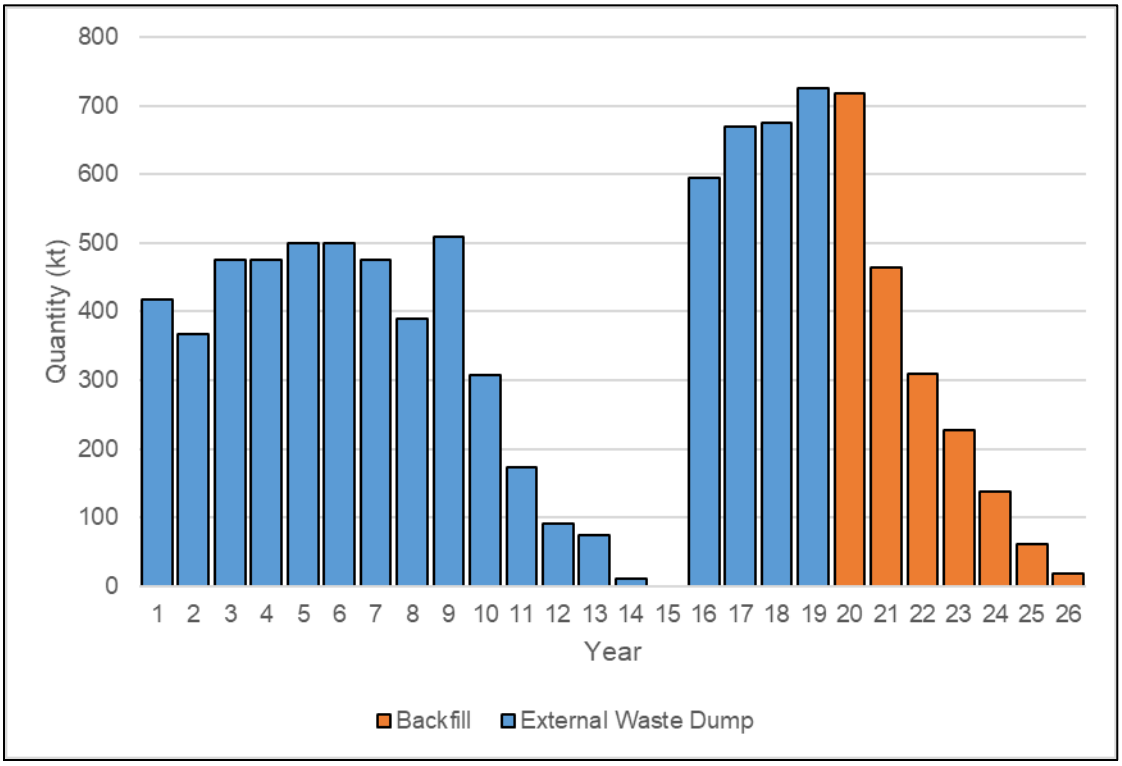


Figure 15-14: Annual Waste Schedule by Destination

15.5 Mining Equipment & Labour

The mine operation is expected to operate Monday to Friday with one 8-hr shift for a total of 241 days. SRK has assumed 5 weather days which will stop production.

The drill and blast patterns have been based on the PFS Report (GBM Minerals Engineering Consultants Limited, 2015) for the top 80 m to limit the potential for fly rock. The lower benches have assumed a larger pattern. A 110 mm blasthole drill has been selected.

The loading units have been sized based on the block size and the mine schedule quantities. A 5.5 m^3 excavator has been selected with 46.8 t payload haul trucks. The loading productivity has been estimated at 558 tph (1.1 Mtpa) in the RoM and 474 tph (0.9 Mtpa) in the waste.

A 4.8 m^3 front-end loader will be used at the RoM stockpile and for development work. It has been assumed that 5% of the crusher feed will be rehandled by the loader.

The haulage cycle times have been estimated in Deswik's Landform & Haulage module and are shown in Figure 15-15. Cycle times increase with time as the pits get deeper. The average waste cycle time decreases significantly in Year 16 with the start of Stage 4 and drop again in Year 20 when the backfill dump becomes available. The RoM cycle times decrease in Year 20 as the majority of the crusher feed shifts from Stage 3 to Stage 4. Trucking requirements range from four to six throughout the mine life.

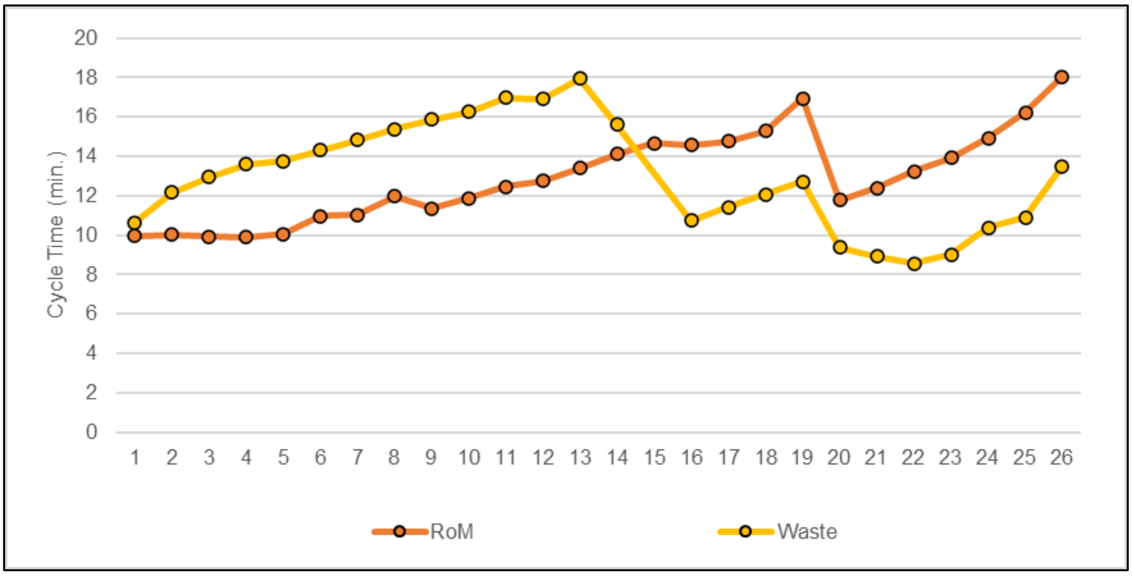


Figure 15-15: Haulage cycle times by material

The maximum required mining equipment is shown in Table 15-12. Ancillary equipment has been included to compliment the primary mine fleet.

Table 15-12: Mining equipment

Equipment	Description	Maximum (#)
Excavator	5.5 m^3 excavator	2
Loader	4.8 m^3 loader	1
Truck	46.8 t payload truck	6
Drill	110 mm drill	2
Track Dozer	260 hp	1
Motor Grader	205 hp	1
Water Truck	26,000 l	1
Fuel/Lube Truck	Mobile fuel and lube truck	1
Lighting Plant		7
Light Vehicle		7

The maximum labour requirements are shown in Table 15-12 and area based on the mine equipment requirements and the expected operation.

Table 15-13: Labour Requirements

Personnel	Maximum (#)
Mine Operations	**20**
Mine Manager	1
Mine Operations Superintendent	1
Mine Operations Supervisor	1
Truck Operators	6
Shovel Operators	2
Loader Operators	1
Ancillary Operators	2
Drillers	2
Blast Crew	3
Administrator	1
Mine Maintenance	**10**
Maintenance Superintendent	1
Maintenance Supervisor	1
Maintenance Planner	1
Maintainer	7
Technical Services	**7**
Geology Superintendent	1
Engineering Superintendent	1
Senior Geologists	1
Senior Mine Engineers	1
Mine Surveyors	2
Geology Technicians	1
Total Labour Requirements	**37**

15.6 Mining Cost Estimate

The mining capital and operating cost estimates and the basis of these is presented in section 20.2.1and 20.3.2 respectively.

16 RECOVERY METHODS

In the 2015 PFS study, all processing was envisioned to occur at the Norra Kärr site. This would require a lined wet tailings impoundment and comprehensive water treatment to ensure environmental protection. Even with this, the processing operation in close proximity to Natura2000 sites caused concern (see section 19).

The critical difference here is now only comminution testwork will be undertaken on site and all chemical reagent processing at a site more suitable for this type of operations.

16.1 Historical Process Method

The PFS (Davidson *et al*, 2015) metallurgical process proposed required comminution, magnetic separation (Figure 16-1) and leaching to produce a mixed REE oxide (REO) concentrate with an average efficiency of 76.4 % (Figure 16-2). Only extraction and recovery of REOs was considered in the PFS flowsheet.

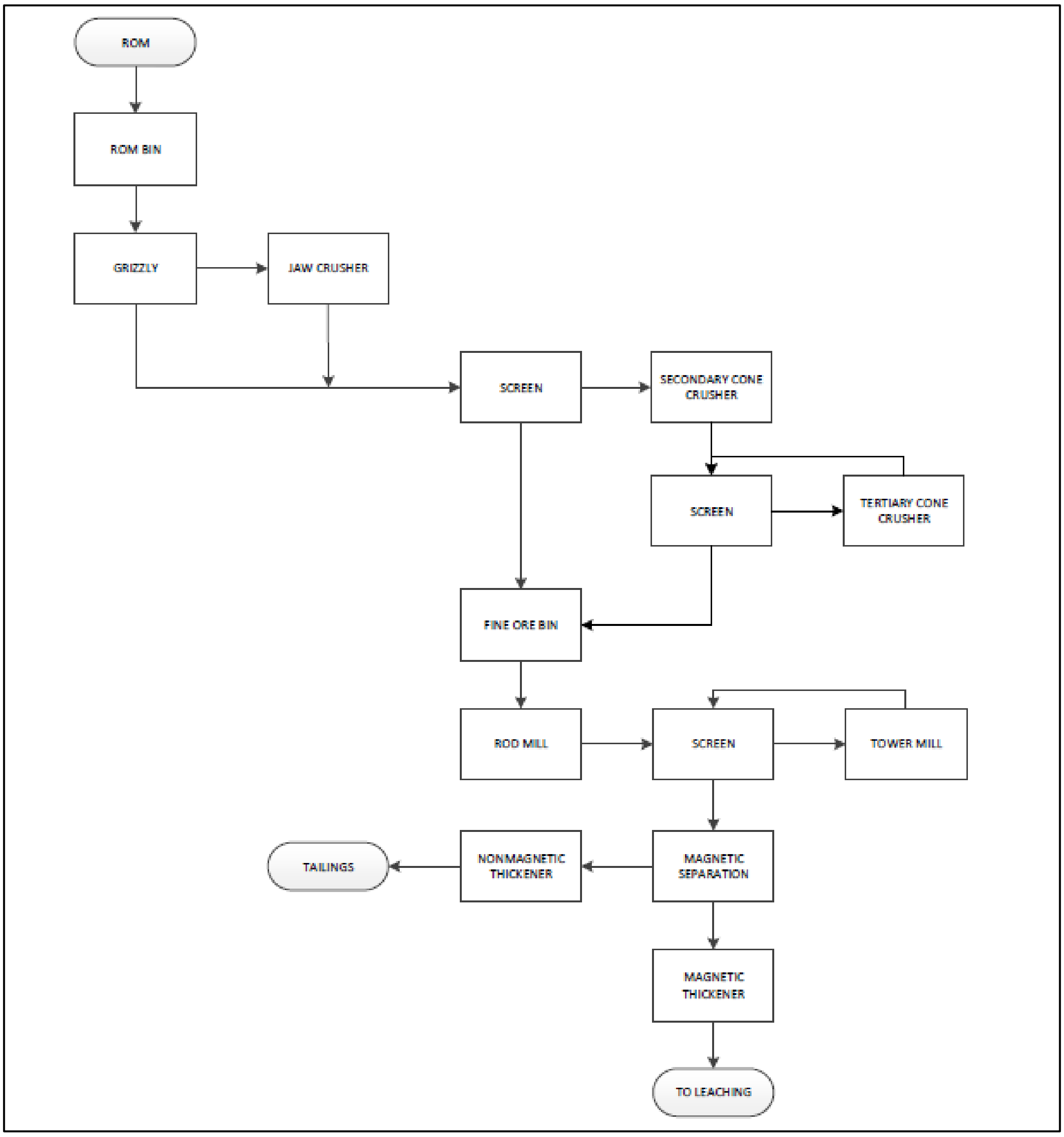


Figure 16-1: Proposed 2015 Flowsheet (Davidson et al 2015)

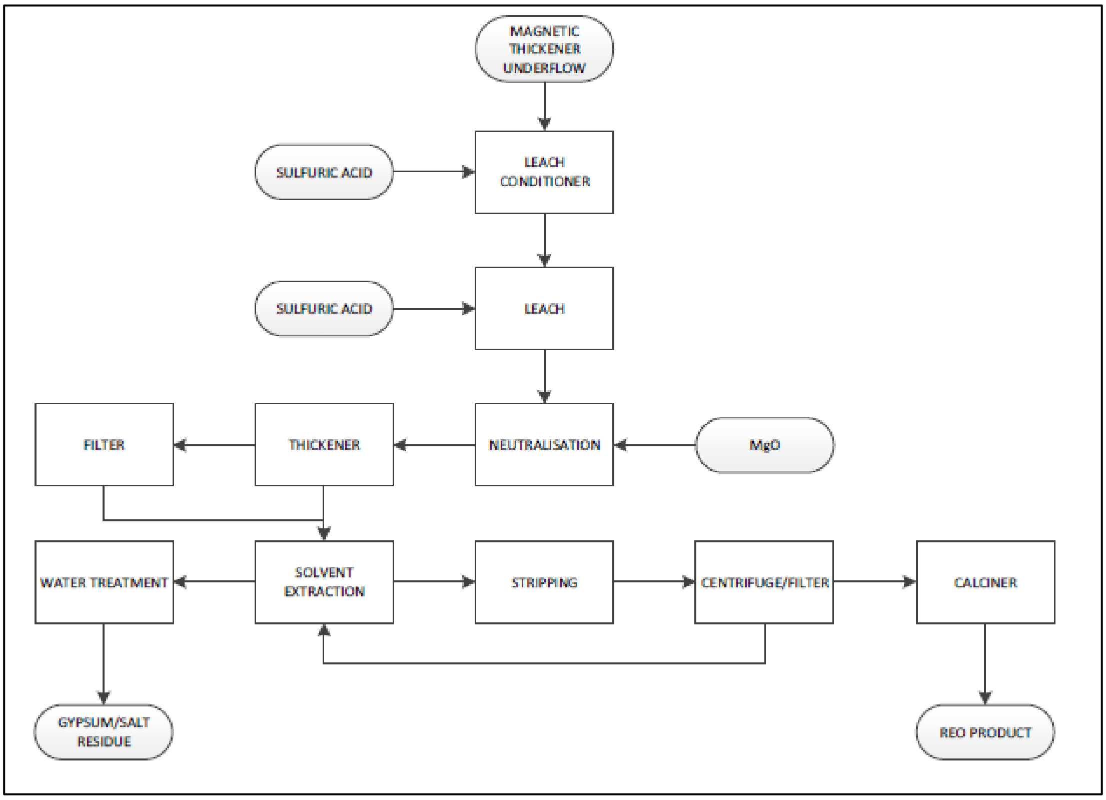


Figure 16-2: Hydrometallurgical Flowsheet (Davidson et al 2015)

16.2 Process Description

Utilizing the additional work completed between 2015 and 2019 modifications have been made to the process route (Figure 16-5). The process design criteria in Table 16-1 and Table 16-2 formed the operational basis for the process flowsheet design.

Table 16-1: Process design criteria

Description	Magnitude	Unit
On site process plant throughput	1150	000 t/a
ROM TREO grade	0.56	%
ROM Zr grade	1.86	%
ROM Nb grade	0.06	%
Contained TREO	6946	t/a
Contained Zr	21,394	t/a
Contained Nb	657	t/a
Process plant operation	24/7/365	-
Crushing mechanical availability	80	%
Grinding and beneficiation availability	91	%
Hydrometallurgy plant availability	91	%

Table 16-2: Overall processing recovery

Mass Balance	Overall Magnetic Separation	Leach Recovery	Solution Separation	Overall Recovery
Ce_2O_3	93%	91%	99%	84.1%
Dy_2O_3	93%	91%	99%	84.1%
Er_2O_3	93%	91%	99%	84.1%
Eu_2O_3	93%	91%	99%	84.1%
Gd_2O_3	93%	91%	99%	84.1%
Ho_2O_3	93%	91%	99%	84.1%
La_2O_3	93%	91%	99%	84.1%
Lu_2O_3	93%	91%	99%	84.1%
Nd_2O_3	93%	91%	99%	84.1%
Pr_2O_3	93%	91%	99%	84.1%
Sm_2O_3	93%	91%	99%	84.1%
Tb_2O_3	93%	91%	99%	84.1%
Tm_2O_3	93%	91%	99%	84.1%
Y_2O_3	93%	91%	99%	84.1%
Yb_2O_3	93%	91%	99%	84.1%
ZrO_2	86%	65%	87%	48.6%
HfO_2	86%	65%	87%	48.6%
Nb_2O_5	93%	91%	96%	81.6%

16.2.1 Comminution

At the Norra Kärr site comminution will take place – to achieve physical separation only (Figure 16-5). All chemical processing will be completed off site. For purposes of this study an industrial complex site in Luleå in northern Sweden has been selected due to available infrastructure and chemical reagent supply but a proper localization study is required in the next phase of work for the hydrometallurgical plant (Figure 16-6).

The beneficiation process is shown schematically in Figure 16-5. An apron feeder will reclaim Run of Mine (ROM) material from the ROM bin onto a vibrating grizzly screen. Oversize material reports to a jaw crusher where the material is comminuted before recombining with the grizzly screen undersize and fed to the secondary crushing screen. Similarly, oversize from the secondary crushing screen will report to the secondary cone crusher where the material will be comminuted before reporting to the tertiary crushing screen. Screen undersize from the secondary and tertiary screens will then be transferred to the fine mineralized rock bin.

Oversize material from the tertiary crushing screen will report to a tertiary crusher where it will be reduced in size before being recycled to the tertiary crushing screen for classification. All material transfers between process units will be by conveyors.

Discharge from the fine mineralized rock bin will be delivered for primary grinding, via a conveyor, to an open circuit rod mill. Rod mill discharge will be screened with the oversize reporting to a tower mill for secondary grinding. Tower mill discharge is combined with rod mill discharge and classified over the same screen in a closed circuit.

Undersize from the mill screen will be fed to a low intensity magnetic separator to remove any residual grinding media, before reporting to a wet high gradient magnetic separator. The process proposed would require crushing and milling to a p_{80} ~ 145 µm. In the first step a mixed eudialyte-aegirine concentrate would be produced at high gauss separation (1.9-2 T) with XRO matrix (GTK, 2014).

This aegirine dominated concentrate would then pass through a second re-grind to a p_{80} ~ 75 µm followed by second magnetic separation by XMO matrix. This would produce a cleaner concentration of eudialyte.

Essentially using a cleaner and rougher magnetic separation in the process. Each concentrate will report to their respective thickeners to recover water. Thickener underflow from the non-magnetic thickener reports to the residue tank for storage and thickener overflow returns to the process water tank. Thickener underflow from the magnetic concentrate reports to the shipped concentrate pile and the thickener overflow is returned to the magnetic separator.

The non-magnetic material from the first separation will report to the nepheline syenite circuit for additional processing prior to packing and sale. The aegirine waste from the second stage magnetic separation will report to a lined impoundment located within the footprint of the waste management facility on site.

Recovered eudialyte concentrate would be shipped to Luleå.

16.2.2 Nepheline Feldspar (Syenite) Production

According to SRK's interpretation of historical testwork, the reject non-magnetic material from the first separation would be 65% of the feed by mass, it would be stored as separate material with nepheline syenite to be prepared by hydrocyclone separation into different size products. Screening of the products and thickening would occur prior to drying and bagging of the material.

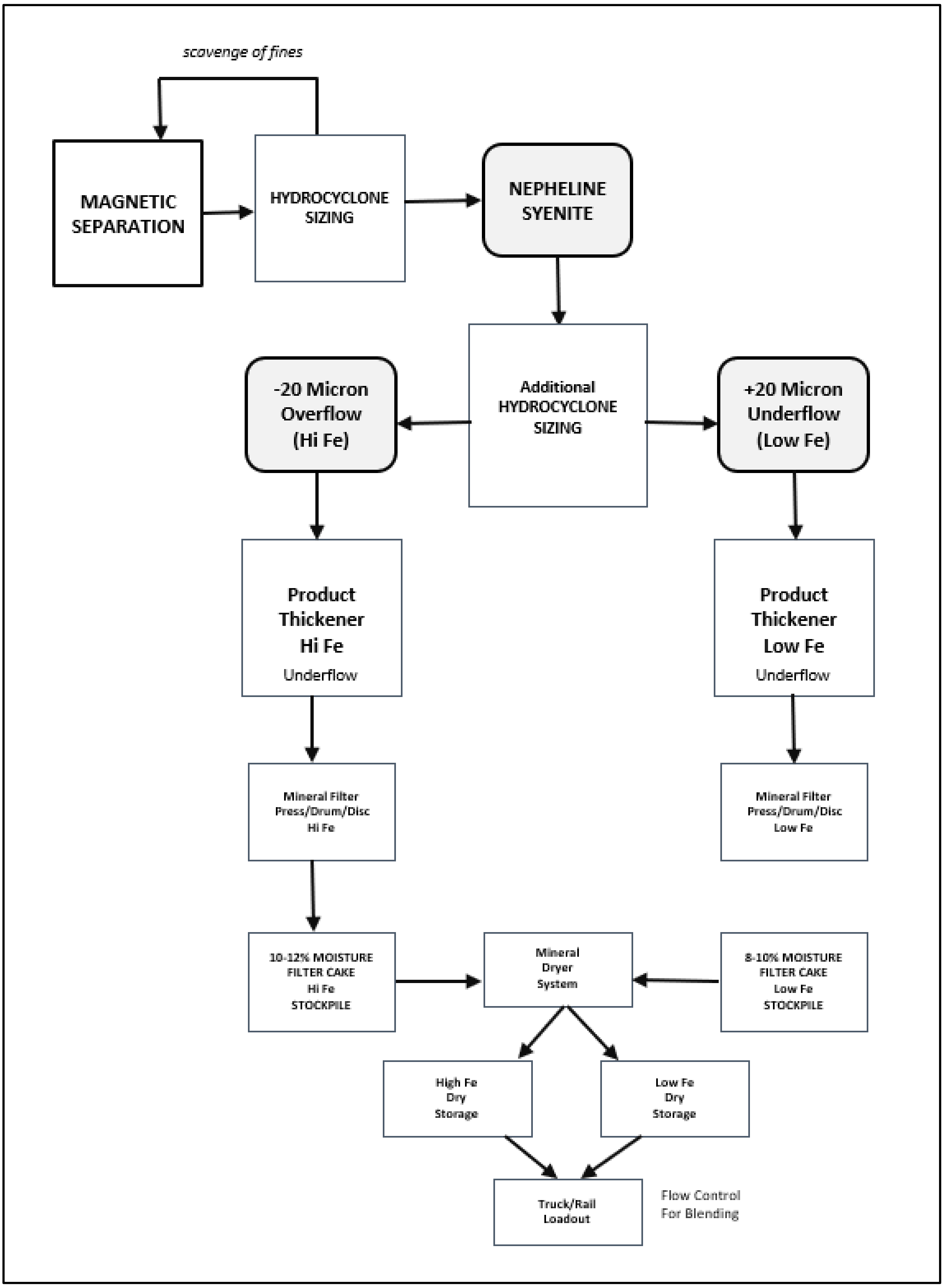


Figure 16-3: Conceptual Nepheline Syenite Flowsheet (source: LEM)

In order to do this, a low iron product will be generated at different size options and a fine product (that will contain trace galena). Dewatering and drying will be required along with a ceramic grinding mill, classification system and packing plant (25 kg paper bags, 1 t super sacks and bulk holding silo for bulk truck supply).

16.2.3 Aegirine storage

Aegirine would be stored on site (see section 23.1) and at present is considered waste. However, aegirine is sold as an alternative material for use in pigments, colouration of ceramics and bricks and as a flux in copper smelting (LEM, 2018). In the next phase of work this option will be evaluated. In order to allow this the residue will be stored separately to allow for reacquiring the material.

16.3 Luleå Processing Facility

16.3.1 Leach Step

To reduce pressure on a sensitive environment the eudialyte concentrate will be shipped to an off-site chemical processing facility elsewhere in Sweden, close to a well-established chemical industry allowing reagents to be readily supplied, reducing the carbon footprint of the reagents and any transport risks and costs associated with this. In this report this is demonstrated through the option to ship the eudialyte concentrate (approximately 105ktpa) approximately 950 km north to Luleå. The rationales behind this are:

- Large chemical industry in Northern Sweden so reagents can be readily supplied, reducing carbon footprint of the reagents, reducing any transport risks and costs associated with the reagents
- Widespread availability of cost competitive and low carbon footprint electricity will help considerably in reducing operating costs and climate impact for the power intensive process.
- Sulfuric acid for the project would most likely be sourced close to Luleå from the Boliden Rönnskarsverken smelter at Skellefteå. Boliden produces 1.7 Mtpa of sulfuric acid at this location, approximately 145 km south of Luleå.

In the current proposed circuit, it is proposed to modify the leach circuit to that proposed by Davris et al (2017) and apply a two-stage acid leach process to reduce impact of silica gel formation (Figure 16-4). This avoids the addition of magnesium oxide and improves REE leaching to over 90%. By this process, sulfuric acid is added to a heated concentrate at 110ºC, S/L ratio of 1/4 followed by water leaching of the treated concentrate at ambient temperature, S/L ratio 1/20 for 30 minutes, resulting in 91% REE recovery.

Fresh sulfuric acid will be added to the concentrate in the leach conditioning tank. Additional sulfuric acid will be recycled from the solvent extraction plant. A two-stage acid extraction, one concentrated, the other a weak or diluted leach will be undertaken with pH set-point controlling sulfuric acid addition.

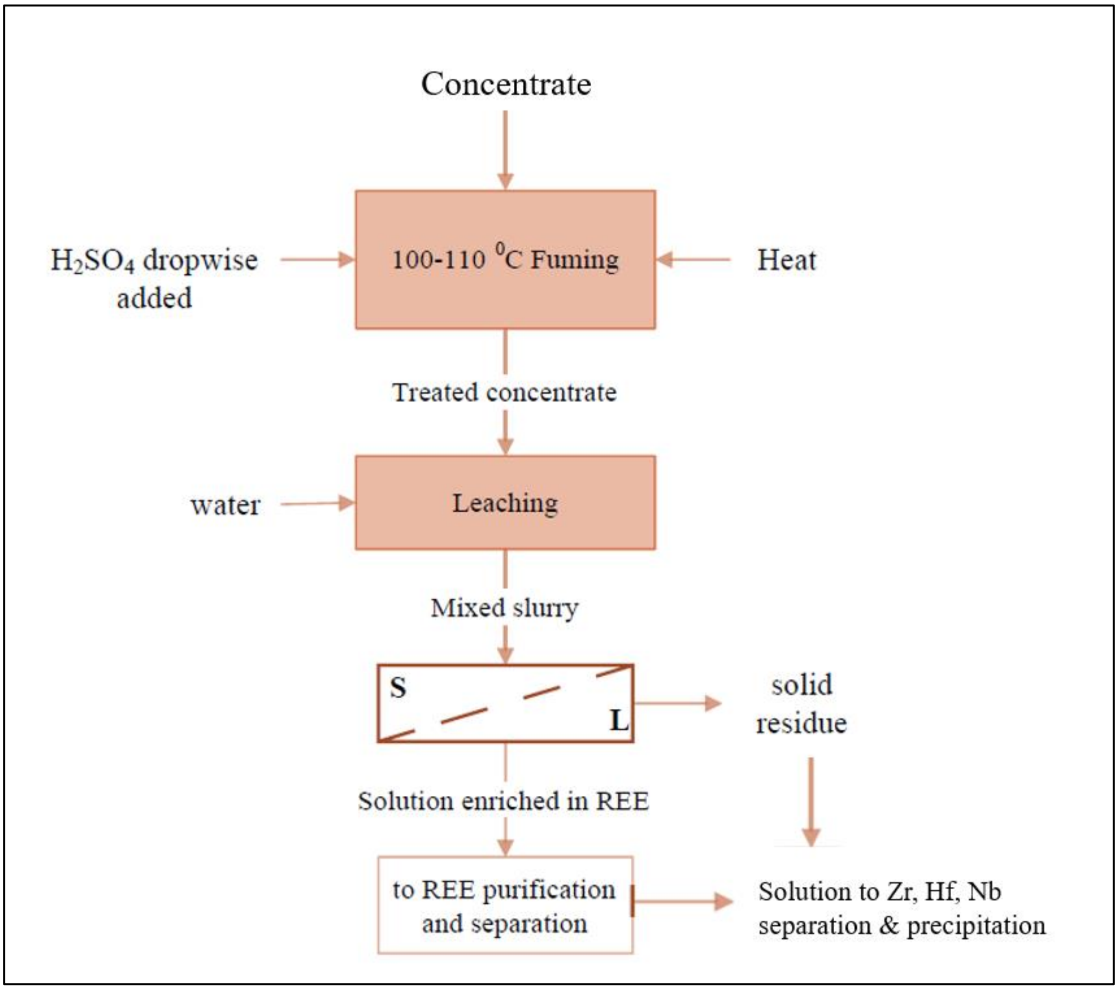


Figure 16-4: Proposed Sulfuric Acid leach circuit for two stage acid leaching (based on work by Davris et al 2017)

Utilization of HCl acid could result in >97% recovery of REE but requires additional work to confirm. Initial leaching of Zr and Hf will result in approximately 60% leaching during the two-stage process described above and approximately 80% Nb (Davris *et al* 2016, 2017). An additional leaching step would recover further metal content with sulfuric acid and possibly addition of fluoride ion (Litvinova and Chirkist, 2013). The additional step will increase liberation of Nb, Zr and Hf from secondary oxides formed to liberate a total of 98-99% of these metals. In the next phase of work this scheme will be verified as part of the overall testwork program.

16.3.2 Recovery Step

After leaching, the pH of the solution will be raised through the addition of sodium hydroxide to pH 2. A pH value is established in order to reject various impurities, which will be precipitated, allowing their removal from the solution via thickening and filtration (filter cake reports to tailings tank). Following impurity removal, the temperature of the pregnant leach solution ("PLS") is raised and the pH reduced, using sulfuric acid, to enhance solvent extraction characteristics.

PLS will report to solvent extraction to recover the REE. Solution will be contacted with a suitable solvent, into which the rare earths will preferentially exchange. The loaded organic solvent will then be scrubbed with dilute sulfuric acid to remove most of the loaded impurities. The REE will be stripped from the pregnant solvent using oxalic acid. The precipitate will then be removed by a combination of centrifugation and filtration of the insoluble rare earth oxalates.

The REE oxalate filter cake is then dried and calcined in a calciner to produce a REE rich mixed oxide. The oxide will then be cooled and packed in preparation for dispatch. The concentrate will have high REE content with low concentrations of impurities that are potentially saleable as a mixed REE intermediate product.

Zirconium separation will involve the use of Alamine 336 in isdecanol/shellsol and differential stripping with HCl acid. Currently due to lack of testing it is assumed Hafnium will be separated and either lost to tailings but potentially this could also be recovered.

For niobium recovery from sulfuric acid, solvent extraction will be used with separation by Alamine 336 followed by EDTA separation and thermal precipitation (Lerner, 1962; Walter, 1963; Shigematsu et al 1964; Ribagnc et al 2017; Hong & Lee, 2018; Tkaczyk et al 2018). EDTA is a chemical that binds and holds on to (chelates) minerals and metals. As no testwork has been undertaken on Norra Kärr mineralized rock to SRK's knowledge a two-step separation is assumed at present to ensure a saleable niobium oxide product. This will require verification in the next phase of work.

Raffinate from solvent extraction will contain various dissolved metals, the concentration of which will build up in the system when recycled. Ultimately, these dissolved species will interfere with the recovery of rare earths if their concentration gets too high. At present the removal of these metals will occur through the addition of lime, nanofiltration, reverse osmosis (RO) and brine evaporation. The gypsum sludge resulting from the addition of lime reports to the tailings discharge tank for disposal.

The neutralised effluent will then undergo a series of nanofiltration and reverse osmosis stages producing a permeate (water) and a brine. The RO water reports back to the raw water tank and the brine will then be further subjected to thermal evaporation to remove the contained water to produce a solid salt residue (predominately sulfates) and a water vapour condensate. The condensate reports back to the raw water tank and the RO residue will be disposed of. Waste produced at Luleå will include the leached silicate waste and gypsum (from spent sulfuric acid). In the next phase of work the application of nanofiltration and the alternative reagent use of hydrochloric acid will be tested to reduce waste and improve recoveries.

16.4 Reagent management

16.4.1 Norra Kärr Mine Site

Principal reagent on site will be water only. If there is any need to discharge site water (storm water, pit water or seepage from the WRSF) then this will

water treatment is needed to remove fine solids from any recycled water from the WRSF to the plant then it is possible some flocculant will be required, most likely an environmentally acceptable one such as magflocc10 for treatment of water to remove suspended solids.

Site water demand is now considerably less (see section 19.5.2). This will be a combination of collected storm water and seepage from the waste facility, dewatering water from the open pit and possibly a small demand from Lake Vättern that will vary over time (discussed in more detail in section 19.5.2.

Diesel fuel oil is to be stored on site for mine infrastructure facility which will service the mining fleet and mobile equipment

16.4.2 Metal Leaching and Recovery Plant

In Luleå, sulfuric acid, lime, oxalic acid, flocculant, solvent Alamine 336 or CY572, iso-decanol (diluent), EDTA, TOPO and Shellsol 2046 will be used.

Delivery of reagents will be every month for solids and every week for liquids. Sulfuric acid will be received twice a week. With varying physical and chemicals of the mineralized rock material, the reagent consumption may change through the life of mine.

The largest volumes of reagents consumed at the plant are sulfuric acid and lime. Sulfuric acid for the project would most likely be sourced close to Luleå from the Boliden Rönnskarsverken smelter at Skellefteå. Boliden produces 1.7 Mtpa of sulfuric acid at this location, approximately 145km south of Luleå. This will be supplied by rail to the process facility at which point it will be transferred to tanks on site. The lime will be supplied domestically in Luleå.

The process plant boiler plant will produce 10 bar steam and distribute at 2 bar for use in the process (solvent extraction) and brine evaporation. The steam boiler plant will be powered by electricity.

16.5 Processing Cost Estimate

The processing capital and operating cost estimates and the basis of these is presented in Section 20.2.3 and 20.3.3 respectively.

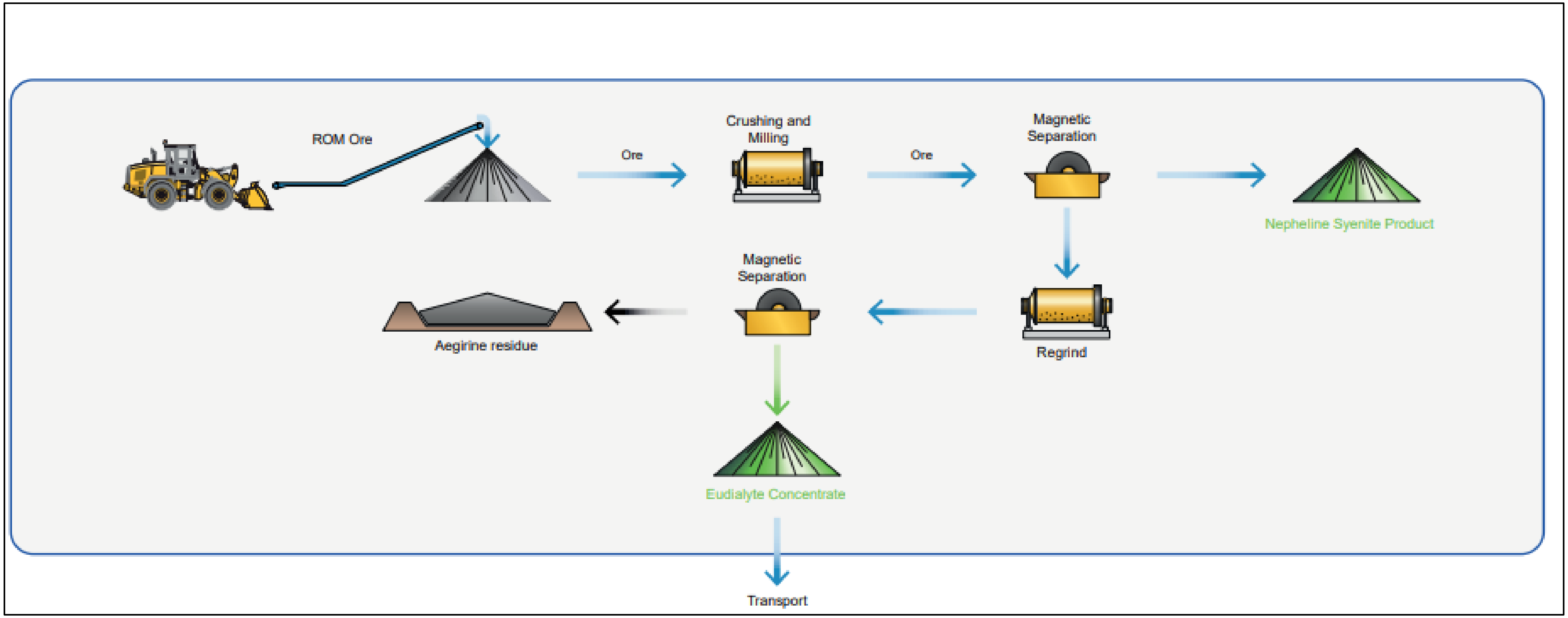


Figure 16-5: Norra Kärr process schematic diagram, Norra Kärr Mine Site (2021)

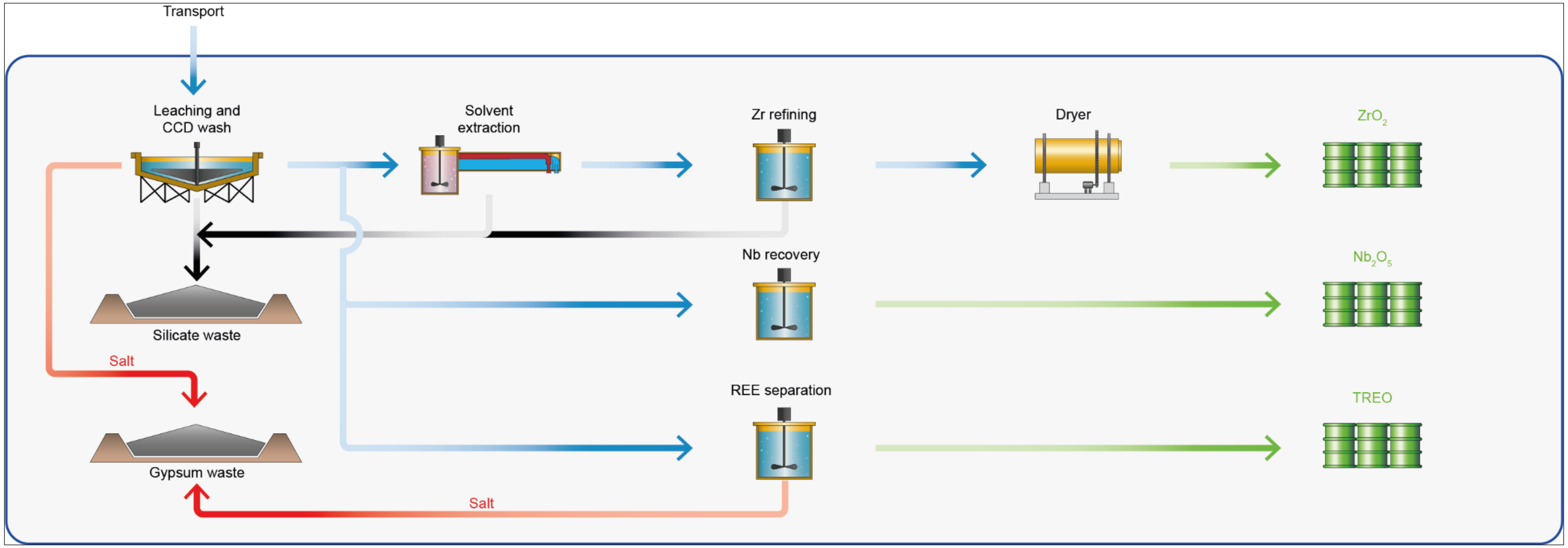


Figure 16-6: Norra Kärr process schematic diagram, Luleå Hydrometallurgical Facility Site (2021)

17 PROJECT INFRASTRUCTURE

17.1 Introduction

The Norra Kärr Project has three project areas:

- The Norra Kärr site where there is open pit mining, comminution plant and waste storage;
- Off-Site Hydrometallurgical Facility ("LPF") where further processing is undertaken; and
- The rail load-out facility, where Eudialyte concentrate is received from Norra Kärr by truck and loaded to rail hopper wagons for transport to the LPF.

All sites require buildings and installations, utilities connections and power in order to support the mining and processing operations. This infrastructure is described in this section. Site wide water management is covered within the Water management section.

The logistics systems for transport of final and intermediate products to the point of sale is discussed in Section 17.5.

17.2 Norra Kärr Project Site

17.2.1 Regional Infrastructure

Location

The Norra Kärr project is located adjacent to Lake Vattern in south central Sweden and some 45 km north-northeast of Jonkoping and 45km south-southwest of Mjölby. The project site is around 240 km south-west of Stockholm. There are a number of small hamlets in the vicinity and a small village of Gränna to the south. Sweden's second largest lake, Vattern, is approximately 1.5km west of the project.

Roads

A main north-south highway, the E4, runs along the western edge of the project, and is a two-way dual carriageway. Current access to site is from the smaller roads (Gränna to Tranås Road 1008) which is reached from the E4 via the junctions at Stava to the north (approx. 4 km) and Gränna to the south-west (approx. 11 km). Gravel roads lead from the Road 1008 to and around the Norra Kärr property.

Power

The 2015 PFS noted a 40 kV power line to the east of the project site, and it was recommended up to 10 MW was to be sourced. Available aerial imagery shows a number of local powerlines crossing the area but would appear to be local supply to farms and hamlets. The nearest major substation is found at Tranås.

Railway

The nearest access to national rail is at Tranås, which is 25 km to the east via a local road named the "Road 1008". Other access points to national rail, via busier highways are Jonkoping and Mjölby, as well as Nassjö and Linköping, which are further distance (circa 80 km by road).

17.2.2 Layout

The Norra Kärr is presented below in Figure 17-1.

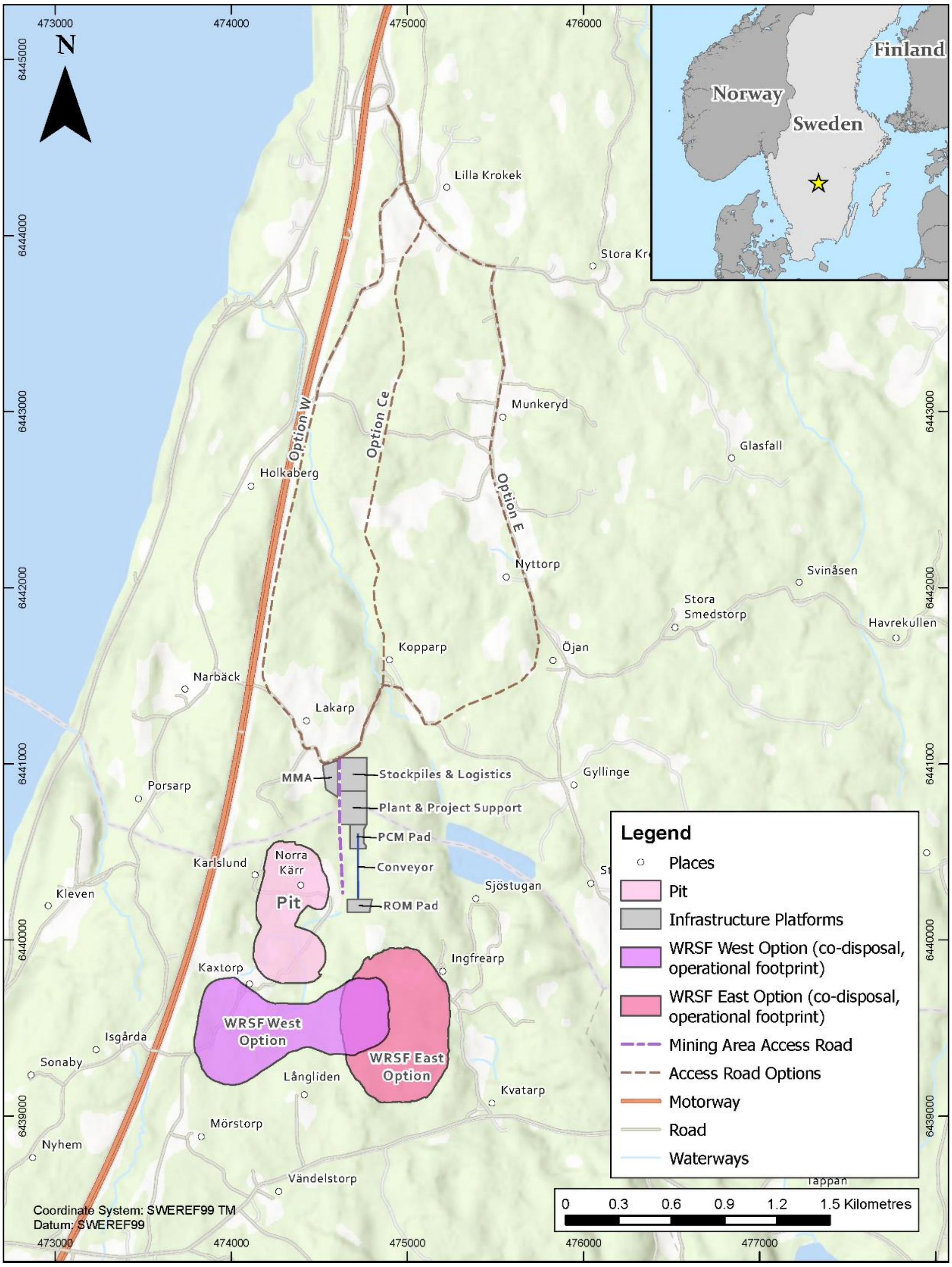


Figure 17-1: Preliminary layout showing WRSF and access road options

Two options for WRSF are shown in Figure 17-1. In the next phase both

17.2.3 Access Road

A new access road to the project site will be required. The local "Gränna to Tranås Road 1008" isn't suitable in its current specification or condition. It is also considered unlikely (or to be cost effective) to seek permission to build a dedicated junction on the E4 immediately at the Project site. Therefore, it is proposed to utilise the existing Stava junction (E4 / Road 500 intersection) and construct a 4.0 km access road to the site. Three preliminary options have been considered and will be assessed further at the next stage of study. These are shown in

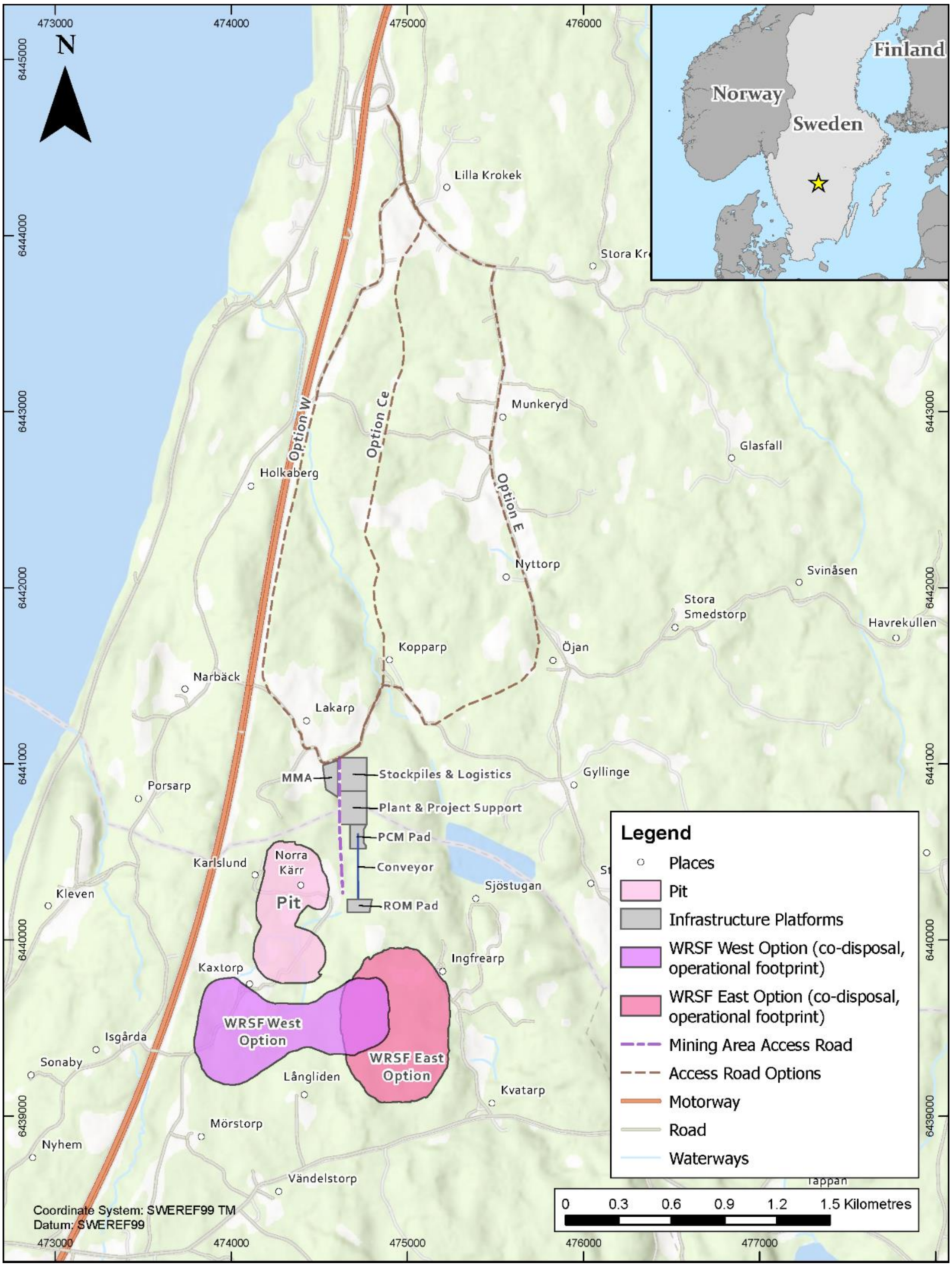


Figure 17-1. The road will be 7.5 m wide with 1.5 m verge on either side. The road surface will be unbound gravel during construction, but a bitumen base and surface course added prior to operations. The exact alignment will be studied at Pre-feasibility stage.

17.2.4 Bulk Power Supply

Strategy

The project will be connected to the national grid via new grid connection to the Swedish National grid system.

Power Demand

Total consumption at Norra Kärr is estimated at 97,000 MWh/annum inclusive of a 10% uplift on the processing plant consumption estimates to cover consumption auxiliary infrastructure. Average demand is estimated to be in the order of 8-9 MW.

Power Infrastructure

The following is envisaged:

- Connection works at the connection substation (e.g. Tranås, Ödeshög, or Gränna);, Ödeshög, or Gränna);
- Overhead transmission lines (e.g. double circuit, circa 10-25 km, at between 20 and 35 kV dependant on requirements and future project engineering);
- Project Main Substation (e.g. 20 or 35kV stepdown to 11/6/0.4 kV dependant on requirements and future project engineering).

From the “Project Main Substation”, located adjacent to the plant, power will be distribution at medium and low voltage to the consumer substations.

Design and construction work will be completed by a Power Engineering Company certified to work on the Swedish National grid system at the cost of the Company. Once constructed, ownership of the new grid infrastructure will be transferred to the Svenska kraftnat for operation and maintenance,

Cost of Power

Primary sources of electricity generation in Sweden include hydropower, nuclear power and wind power. Svenska kraftnat” is the national electricity transmission grid operator. Sweden is part of the Nordic electricity market, which is a common market for electricity in the Nordic countries, where energy is traded on a number of trading venues (e.g. NASDAQ). Sweden is divided into four bidding areas and Norra Kärr is within bidding area SE3. Prices were very low during 2020, however, higher pricing was seen in January 2021. The average wholesale pricing from the last 5 years (including 2020) has been taken as basis for the price assumed for supply, which is SEK 0.38/kWh or EUR 0.037/kWh (or USD 0.044 /KWh).

In Sweden, wholesale electricity prices are the significant contributor to the overall tariff paid by an industrial consumer. Added to wholesale electricity prices are transmission and distribution tariffs which are estimated based on recent published data by the European Union[1] to add around USD 0.026 /KWh bringing the total cost of power to USD 0.07 /KWh.

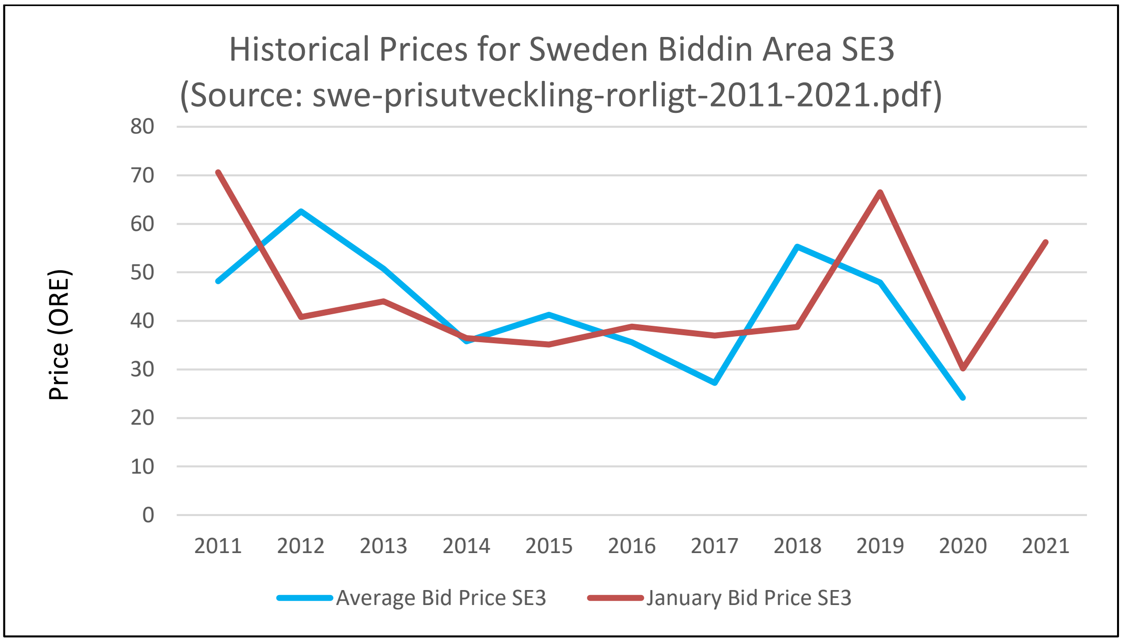


Figure 17-2: Sweden historical average energy prices (source: EON)

17.2.5 Bulk Water Supply

At Norra Kärr, the make-up water to the plant will be via a surface water abstraction pump and pipeline. Once operational, much of the make-up water to the plant will be sourced from dewatering boreholes at the open pit. This is discussed further in the Water Management Section. Potable and drinking water are expected to be via a connection to the local utility or from boreholes.

17.2.6 Buildings and Installation

Buildings and installations are designated as supporting either mining, processing, product logistics, or the project overall. These facilities are positioned into compounds or areas for the purposes of management and control. Each area will be independently fenced.

In general buildings will be either pre-engineered steel portal framed or column and beam style buildings with insulated panel roofs and cladding and with all necessary internal electrical, piping, fixtures and fittings, and architectural details. It's likely that some auxiliary buildings will be prefabricated and pre-fitted, modular, or converted container style buildings.

Project Facilities

Project facilities area provide services to all mining, processing, and auxiliary operations. They consist of the following:

[1] Electricity price statistics - Statistics Explained (europa.eu)

- Administration building, including offices (for administrative functions), technical services offices, stores, training, security offices, first aid;
- Change-house and ablutions;
- Boiler house providing heating and hot water;
- Facilities maintenance building with garage for the emergency services first responder vehicles (ultimately the project will rely on the Swedish emergency services for overall emergency response); and
- Canteen and dining area.

Mine Maintenance Area

The Mine Maintenance Area ("MMA") is a separate compound to service and maintain the open pit mining and waste dump operations. The surface area is around 125 x 125m. The MMA will be constructed by the Project and can be used either under an owner operator strategy or for Contractor Mining. The following buildings and installations will be constructed:

- Mining office;
- Mining Heavy Equipment ("HV") and Light Vehicles Workshop with integrated Tyre Change / Tyre Storage area;
- Mining warehouse;
- Vehicle wash including Raw Water Tanks & Dispensing;
- Diesel Fuel Storage & Dispensing (self-bunded containerised tanks);
- External laydown and parking;
- A waste collection and recycling area (e.g. scrap, oils, tyres etc); and
- Surface water management and collection for contact run-off.

The Mining Equipment Workshop will have multiple bays and be a pre-engineered structural steel-clad building and include machine shop and welding bay. All required maintenance, including major overhaul, is anticipated to be undertaken on site. Other buildings will be prefabricated, or container dome shelters as required. The area will be independently fenced and secured to manage ingress / egress.

Plant

The plant requires the following infrastructure, which is integrated within the plant compound:

- Building to house the processing equipment and materials handling;
- Control rooms;
- Consumables storage
- Plant workshop, warehousing, and laydown area;
- Laboratory;
- Various water, power, compressed air reticulations and installations; and

- Sewerage System & Treatment Plant (receiving from all infrastructure areas).

Logistics & Warehousing / Product Load-out

This area facilitates storage and load-out of the following:

- Eudialyte concentrate for transport to the Company's Luleå Processing Facility; and
- Nepheline Syenite products to customers.

The stockyard area is around 500 x 300 m and will contain:

- Gatehouse and weighbridge;
- Empty truck waiting area, split into Nepheline Syenite and Eudialyte areas;
- Eudialyte storage covered warehouse;
- Open area Nepheline Syenite storage (modelled as 150 m x 150 m);
- Covered open sided warehouse area for bagged Nepheline Syenite products;
- Nepheline Syenite product loading area, with forklifts and front-end loader;
- Ablutions and waiting area (e.g. converted containers).
- Surface water management and collection for contact run-off.

17.2.7 Utilities

Allowances within the capital cost are made for the following site wide utilities and services, which will be required:

- MV / LV electrical distribution (power plant to compound fence lines and consumer substations);
- Area lighting: roads and general areas;
- Potable water storage and reticulation (e.g. for ablutions, kitchen, dining);
- Raw water storage and reticulation from the main raw water tank (fire water, vehicle washing, dust suppression);
- Fire water reticulation to fire water tanks in plant, compounds, and areas;
- Stormwater / surface water management and pollution control;
- Sewerage and wastewater reticulation and treatment;
- IT, communications, telephone;
- Security systems (including front gate), alarm, CCTV and movement detection systems;

Within the infrastructure areas and compounds, the utilities connections from the fenceline to individual installations and buildings are assumed within the individual costs.

17.3 Off-Site (Luleå) Processing Facility

17.3.1 Overview

Eudialyte concentrate will arrive at the off-site processing facility (“LPF”) for refinement to three products, REOs, Zr and Niobium. The LPF is expected to be situated on an industrial park whose exact location is yet to be confirmed, however, a location at Luleå on an existing industrial area within easy reach of rail facilities is assumed as a viable option for consideration in this report. This section summarises the required support infrastructure around the processing plant and waste storage facilities, which are considered in separate sections of this report.

17.3.2 Industrial Park

Options & Assumptions

The selected industrial park is anticipated to be a multi-occupancy site designed for potentially energy intensive industrial usage such as chemical and process plants. It is envisaged that the LPF will be constructed on a parcel of “brownfield” land and for the purposes of this study a bulk power and water connection is required from local connection points as well as for security and other utilities functions. It has been assumed that an existing railway line is located adjacent to the site.

The exact future location for the off-site processing facility will have to be evaluated through a detailed localization study but for the purpose of the PEA report, a number of potentially viable options have been identified by the Company for the purpose of the PEA for situating the LPF including the current and proposed areas shown in Figure 17-3:

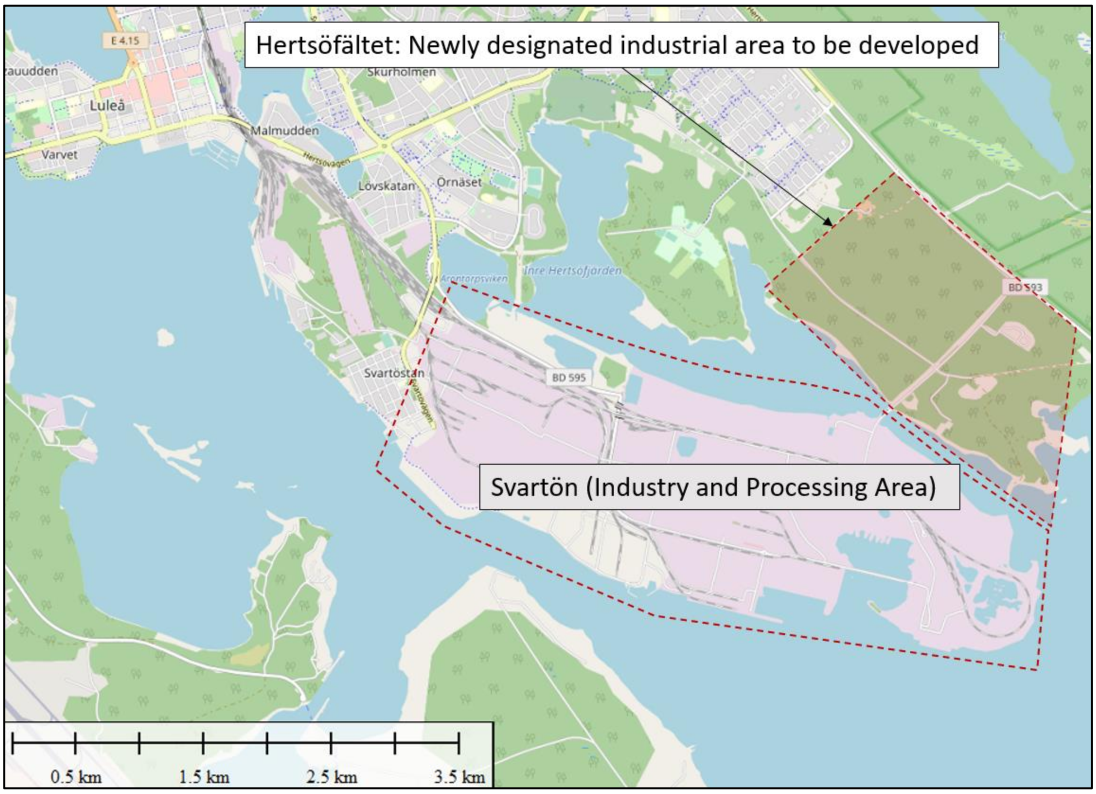


Figure 17-3: Possible options for the Luleå Processing Facility (compiled from several sources)

Preferred Option for the PEA

The preferred option has yet to be determined and will depend on a number of factors which cannot be confirmed at this stage in project development. Therefore, a schematic layout has been developed for the purposes of conceptualisation. Around 20 Ha is required for storage of gypsum and neutralized residues (12 Ha in total) and around 5-6 Ha for plant and other infrastructure.

17.3.3 Layout

Because an exact parcel of land is yet to be determined, a schematic layout was prepared as a basis for the PEA to ensure all technical and cost aspects are captured and the likely footprint is understood. There are four main asset areas:

- Processing facility building, equipment and materials handling;
- Waste storage facility;
- Support Infrastructure
- Logistics and warehousing.

It is noted the arrangement of assets can be changed to suit the available options at the time.

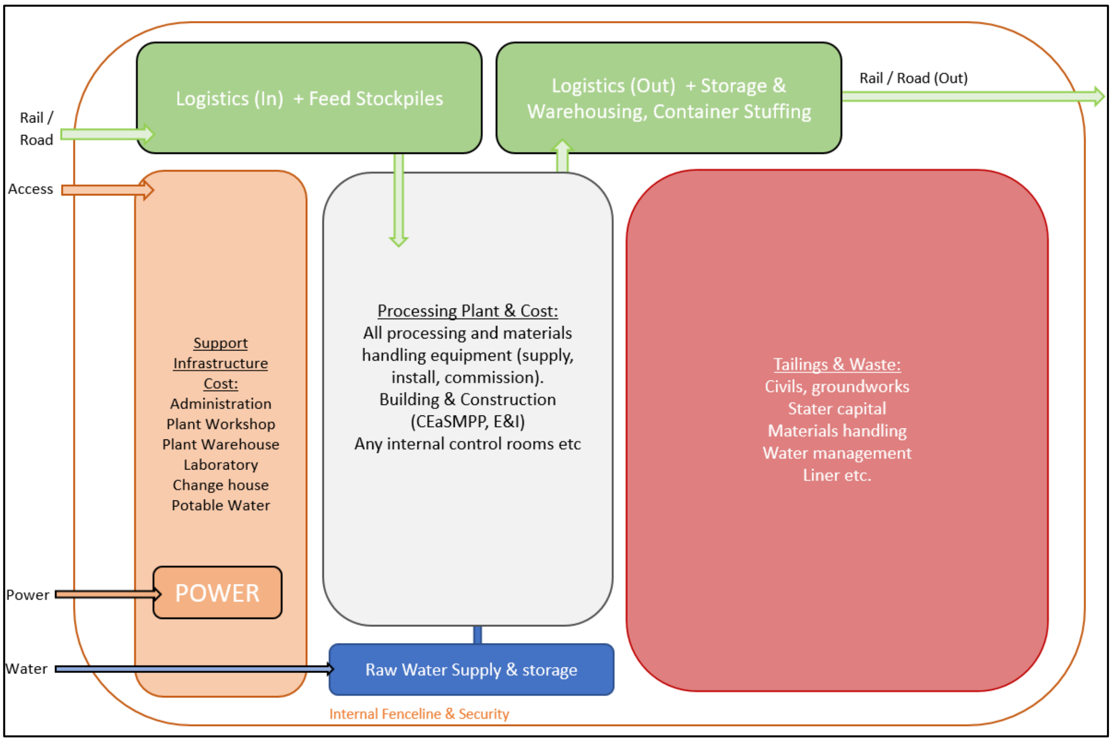


Figure 17-4: Schematic Layout for the Luleå Processing Facility

17.3.4 Regional Infrastructure

There is very good regional infrastructure and connections in the Luleå area:

- Luleå Port
- Connections to the Sweden regional road and rail networks.

- Luleå Airport.

Luleå is the capital of Norrbotten County. Luleå has a population of circa 49,000 and is Sweden's 25th largest city. Luleå has the seventh biggest harbour in Sweden for shipping goods, a large steel industry and is a centre for extensive research and has Luleå University of Technology.

The Port of Luleå is a public harbour which, using locally operated icebreakers, is open for shipping all the year round. There a number of berths allowing for container, bulk, breakbulk and liquid cargo loading and unloading.

17.3.5 Access

Access to the LPF will be via the main entrance and internal roads therein, of the Industrial Park / Area.

17.3.6 Bulk Power Supply

Strategy

The project will be connected to the national grid via new grid connection to the Swedish National grid system. The Swedish national grid high voltage transmission is typical 220 kV, 275 kV or 400 kV. Luleå has a 400 kV connection from where power is stepped down and then distributed at around the city and to industrial facilities.

With regards to the LPF, the following are anticipated, which need to be better defined at pre-feasibility study level:

- A connection bay at a main HV substation is provided by the Industrial Park with line bays available for users / tenants to connect to (nominally, within 3,000 m);
- The Company brings power to the land parcel where it is stepped down to site distribution and equipment voltages;
- It is envisaged that the Project must construct medium voltage (“MV”) line to the LPF compound and an MV / LV substation at the site;
- There will be a number of main consumer substations within the plant (nominally, 10 no.) as well as the waste facility, support infrastructure area and the logistics area.

The Luleå area falls under power pricing area SE1, where power is mainly generated by hydro powerplants.

Power Demand

Average demand is estimated to be in the order of 4-5 MW to cover plant consumption and auxiliary infrastructure.

Power Infrastructure

The following is envisaged:

- Connection works at the connection substation;
- Buried transmission lines (double circuit, circa 3.0 km, at between 20 and 35 kV);

- Project Main Substation (e.g. 20 or 35 kV stepdown to 11/6/0.4 kV dependant on requirements).

From the “Project Main Substation”, located adjacent to the plant, power will be distribution at medium and low voltage to the consumer substations.

Design and construction work will be completed by a Power Engineering Company certified to work on the Swedish National grid system at the cost of the Company. Once constructed, ownership of the new grid infrastructure will be transferred to the Svenska kraftnat for operation and maintenance,

Cost of Power

The Luleå area falls under power pricing area SE1. Similarly, to SE3, prices were very low during 2020, however, higher pricing was seen in January 2021, which is similar to the proceeding years. The average pricing from the last 5 years (including 2020) has been taken as basis for the price assumed for supply, which is SEK 0.37/kWh or EUR 0.036/kWh.

In Sweden, wholesale electricity prices are the significant contributor to the overall tariff paid by an industrial consumer. Added to wholesale electricity prices are transmission and distribution tariffs which are estimated based on recent published data by the European Union (*ec.europa.eu /eurostat/statistics-explained*) to add around USD 0.026 /KWh bringing the total cost of power to USD 0.069 /KWh.

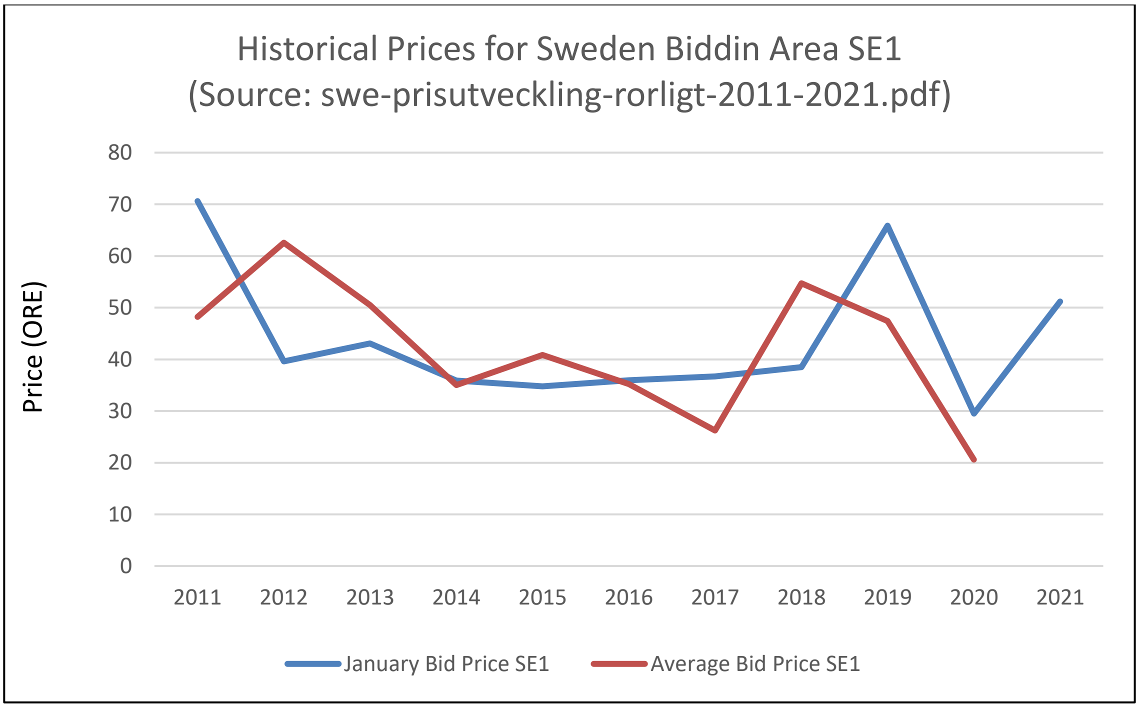


Figure 17-5: Sweden historical average energy prices (SE1) (wholesales market prices)

17.3.7 Bulk Water Supply

The Company anticipates potable water connections to the LPF compound via the industrial park and a water pumping abstraction point at a local water source.

17.3.8 Buildings and Installations

General

All buildings and installations will support the processing and logistics operations These facilities are positioned within the general plant compound. In general buildings will be pre-engineered steel portal framed or column and beam style buildings with insulated panel roofs and cladding and with all necessary internal electrical, piping, fixtures and fittings, and architectural details. It's likely that some auxiliary buildings will be prefabricated and pre-fitted, modular, or converted container style buildings.

Project Facilities

Project facilities area provide services to all processing, and auxiliary operations. They consist of the following:

- Gatehouse and weighbridge;
- Administration building, including offices (for administrative functions), technical services offices, stores, training, security offices, first aid;
- Change-house and ablutions;
- Facilities maintenance building;
- Canteen and dining area.

Processing Plant

The plant requires the following infrastructure, which is integrated within the plant compound and considered within the plant cost:

- Building that house the processing equipment and materials handling;
- Control rooms;
- Consumables and reagents day storage;
- Plant workshop (for fixed plant), spares warehousing, and laydown area;
- Laboratory;
- Various water, power, compressed air reticulation infrastructure and installations; and
- Raw and return water ponds and water treatment (receiving from all infrastructure areas).

17.3.9 Utilities

Allowances within the capital cost are made for the following site wide utilities and services, which will be required:

- MV / LV electrical distribution (power plant to compound fencelines and consumer substations);
- Area lighting: roads and general areas;
- Potable water storage and reticulation (e.g. for ablutions, kitchen, dining);

- Fire water reticulation to fire water tanks in plant, compounds, and areas;
- Stormwater / surface water management and pollution control;
- Sewerage and wastewater reticulation and treatment;
- IT, communications, telephone;
- Security systems (including front gate), alarm, CCTV, and movement detection systems;

Within the infrastructure areas and compounds, the utilities connections from the fenceline to individual installations and buildings are assumed within the individual costs.

17.3.10 Logistics / Warehousing

Overview

Logistics and warehousing at the LPF will be important. There will be a number of different consumables being imported by road and rail as well as the Eudialyte concentrate. The products will be exported in twenty-foot containers ("TEUs") by road or rail. The consumables and products and the anticipated regularity of deliveries and storage capacities are presented in Table 17-1.

Table 17-1: LPF Consumables and Feed Product Imports

Feed Material	TPA	Cargo Type	Storage Capacity (Weeks)	Storage Capacity (t)	Envisaged Regularity of Deliveries
Eudialyte	104,650	Dry Bulk - Rail	1	2,900	5-6 trains per week
Sulfuric acid	52,325	Liquid Bulk - Rail	1	1,006	2 trains per week
Hydrochloric Acid	3,298	Liquid	2	84	2 HGV per Week
Shellsol	23,393	Liquid Bulk - Rail	2	436	7 HGV per Week
Alamine 336	1080	Liquid (heavy)	4	20	1 HGV per Month
Organic (CY923)	280	Liquid (heavy)	2	48	1 HGV per Week
TOPO	3,298	Powder-Bags/Pallet	2	84	2 HGV per Week
EDTA	53	Powder-Bags/Pallet	2	5	1 HGV per Week
Oxalic	69	Powder-Bags/Pallet	2	5	1 HGV per Week
Lime	91,916	Dry Bulk - Rail	1	7,031	12 trains per week

Deliveries

Deliveries by rail will be directed to one of three sidings for offloading of lime, liquid bulk or Eudialyte concentrate (from Norra Kärr), as well as other consumables delivered by rail or road. Adjacent to these sidings will be the storage building, storage tanks and covered warehouses for small liquid and palletised, as well as the warehouse for Eudialyte concentrate. Lime, sulfuric acid (and other liquid bulks) and Eudialyte will require unloading through fixed mechanical or fixed pumping / pneumatic systems. Other palletised and break-bulk cargo will be offloaded by forklift.

Exports

At the south end of the site, near to the road exit / entrance will be the product warehouse and storage areas. Within this area will be container loading operations, empty and loaded container storage and train / truck loading using a reach stacker / crane.

Rail Infrastructure & Equipment

Three siding areas and a run-around loop are proposed at the LPF. The sidings are dedicated to dry bulk deliveries (400m), liquid bulk deliveries (250m) and export of containers (250m). A dedicated shunting locomotive will likely be required at the LPF. Use of the external industrial area railway sidings may be required to marshal wagons and organise deliveries prior to final delivery for unloading at the site.

Table 17-2: Rail Infrastructure (appropriate for shunting up to 8-10 wagons lengths)

Details	Quantity	Units
Railway Sidings	2,500m	Includes tracks and formation
Switches / Turnouts	9 no.	Prefabricated turnouts complete with closures, rails, fixing etc.
Buffer Stops	4 no.	For siding ends.
Mainline Connection	1 no.	Allowance (includes approvals process)
Signalling	1 no.	Allowance (internal / external)

17.4 Rail Loading Facility (Off-Site Infrastructure)

17.4.1 Description

Eudialyte concentrate will be shipped from the Norra Kärr site to the Luleå Processing Facility ("LPF") through a combination of road haulage and rail freight. The project envisages a warehouse and rail siding for wagon loading in the vicinity of existing railway mainline within around 45–80 km of the mine site. Given the well-developed transport infrastructure in Sweden, the project anticipates only small quantities stored at site. The following basis of design for the site has been developed:

Arrival of Eudialyte concentrate from Norra Kärr:

- An independently fenced compound, owned and operated by the Company;
- Working hours for unloading and loading will be 10 hours a day, 5 days a week and 4 hours at the weekend (54 hours per week);
- The site will receive Eudialyte concentrate by road;
- Eudialyte concentrate arriving in 20 tonne or 25 tonne payloads by articulated and / or rigid tipper trucks with covered trailers; and
- Around 100 trucks a week will arrive at the site (2 per hour).

Storage of Eudialyte concentrate:

- Unloading of trucks will carried out in a covered secured portal frame warehouse;
- Given the well-developed transport infrastructure in Sweden, the project anticipates only small quantities stored at site (e.g. 1,000 tonnes).

Loading to Rail Wagons:

- Empty wagons will be supplied at the site by an approved and licenced Rail Freight Operating Company (e.g. the state enterprise, "Green Cargo", or similar); and

- Shunting operations will be carried out at the site by a dedicated shunting locomotive;
- Within the warehouse, Eudialyte concentrate will be rehandled by front end loader to a loading hopper from where the material will be conveyed (by an enclosed belt conveyor) to a loading silo and chute above the rail wagon loading point; and
- The Company envisages around 8 no. hopper wagons (around 550 tonnes per train) are loaded and dispatched in accordance with an agreed rail timetable. Each hopper will measure between 15 and 20m from buffer to buffer. The exact tonnage and length will depend on the type of hopper selected.

17.4.2 Operations

Between five to six trains per week are envisaged to be despatched from the Rail Load-out Facility. Loading will take around 3-4 hours. Empty wagon sets will be supplied by the Rail Freight Operating Company ("FOC") and marshalled to the holding siding by the mainline locomotive. The FOC will supply a shunting locomotive to facilitate loading of wagons and return to the loaded wagon siding. In accordance with the freight schedule, the FOC will dispatch a mainline locomotive to collect the loaded wagons for transport to Luleå. The Company will employ 10 no. staff at the site.

17.4.3 Location

There are multiple potential options for a location for the rail load-out, which would be subject to land negotiations and purchase. Ideally, the parcel of land will be adjacent to the mainline and within or adjacent to existing industrial and manufacturing units on the periphery of a town such as Mjölby, Nassjö or Linköping or other appropriately zoned area of land for development. Mjölby is assumed as basis for the study. The exact location will be determined at Pre-Feasibility stage.

17.4.4 Infrastructure & Equipment

The rail infrastructure and fixed and mobile equipment requirements are presented in and below. Civil infrastructure and building associated to the facility are as follows:

- Access road;
- Site earthworks and levelling;
- Placement of hard standing for laydown, parking and loading;
- Security Fencing, Systems and Front Gate
- Warehouse Building (portal frame, pre-engineered steel building);
- Hopper Loading House (situated above the railway siding);
- Facility Offices (modularised, prefabricated);
- Area Lighting;
- Substation and distribution; and
- Site Wide Utilities (Heating, water, sewerage).

Table 17-3: Rail Infrastructure (appropriate for shunting of 8-10 rail hopper lengths)

Details	Quantity	Units
Railway Sidings	620m	Includes tracks and formation
Switches / Turnouts	7 no.	Prefabricated turnouts complete with closures, rails, fixing etc.
Buffer Stops	2 no.	For siding ends.
Mainline Connection	1 no.	Allowance (includes approvals process)
Signalling	1 no.	Allowance (internal / external)

Table 17-4: Fixed and Mobile Equipment

Details	Units
Front End Loader	Wheel loader unit
Fixed Conveyor and Loading Bin	200 tph, 50m long, Covered / enclosed
Silo and Loading System	Positioned overhead; chute for loading, 120 tonnes
Dual Cab 4x4	Site operations
Covered Hopper Wagons	Provided and operated by the FOC
Mainline Locomotive	Provided and operated by the FOC
Shunting Locomotive	Provided and operated by the FOC

17.4.5 Layout (Conceptual)

The conceptual layout used for establishing scope and costs for the costs is presented in Figure 17-6.

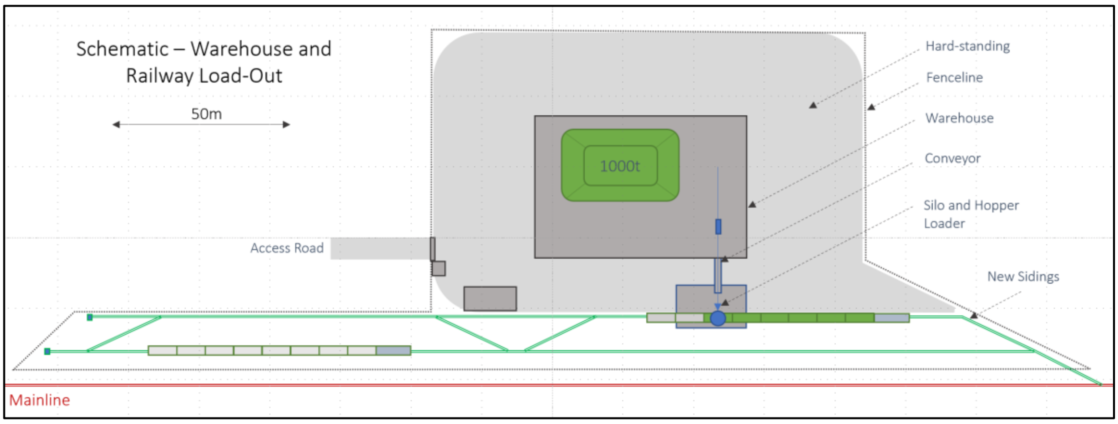


Figure 17-6: Schematic layout of the concept for the Railway Load-out for Eudialyte concentrate

17.5 Product Logistics

17.5.1 Introduction

For the TREO, Zr and Niobium products of the only processing undertaken at the Norra Kärr mine site is comminution and magnetic separation. A concentrate containing all of these metals will be shipped to an offsite facility.

This results in a requirement to transport a “Eudialyte concentrate” to the proposed processing facility (currently conceptually proposed to be at Luleå). Separate TREO, Zr and Niobium products are then generated and transported from the Luleå facility to points of sale anticipated to be destination ports under a CFR freight contract. This section summarises a suitable logistics system and associated costs for the following:

- The “Eudialyte concentrate from mine site to off site process facility”; and
- The REO, Zr and niobium products from the offsite process facility to point of sale.

The Nepheline (Feldspar) Syenite will be sold at “mine gate” and therefore this section only presents a summary of the possible transport options which were explored by the Company prior to agreeing the point of sale at mine gate.

17.5.2 Cargo Types

A summary of expected cargo types is presented in Table 17-5.

Table 17-5: Products by Cargo Type

Product	Description	Cargo Type	Transportation
Nepheline Syenite Product 1	Fine to coarse sand sized granular material (“fine aggregate”)	Bulk dry cargo	Bulk trailers / rail wagons / bulk bags
Nepheline Syenite Products 2 and 3	Coarse granular (“coarse aggregate”)	Bulk dry cargo	Bulk trailers / rail wagons / bulk bags
Eudialyte mineral concentrate	Fine to coarse sand sized granular material. Dense, quite dense: 5 – 6.5 SG).	Assumed as a bulk dry cargo.	Bulk trailers / rail wagons / bulk bags
TREO, Hf-Zr and Niobium products	fine to coarse sand sized granular material (“fine aggregate”)	Bulk dry cargo	Containerised (TEUs)

17.5.3 Access and Infrastructure

The regional road, rail and port infrastructure, the access road to the Norra Kärr site, and any intermediate handling stockyards are described in Section 17.4.

17.5.4 Nepheline Syenite Products

Overview

This product is anticipated to be sold at “mine gate” as a bulk cargo or potentially in bulk bags (1-2 tonnes). Conceptually, the following is anticipated:

- Nepheline Syenite Product 1 (averaging 275ktpa over the LoM) to be purchased in the local market in Sweden.
- The Nepheline Syenite Products 2 and 3 (averaging 403ktpa and 55 ktpa respectively) to be purchased by off-takers in the US and Northern Europe with a roughly 50:50 split per product per destination. It is anticipated these markets are accessed via East Coast US ports and in the case of Europe, a large port such as Rotterdam with road, rail, and inland-waterway connections.

A stockyard area is required at Norra Kärr where road haulage trucks supplied by an off-taker are loaded. Assuming a 28-30 tonne payloads are achieved by rigid 4-axle or articulated 5 / 6 axle highway trucks, and assuming 286 working days per year (12 hours per day except for Saturdays when 6 hours will be operated), around 87 trucks per day are likely to leave the stockyard area carrying Nepheline Syenite.

Logistics System

Nepheline Syenite A is a lower value product produced along with the Nepheline Syenite B / C. The sales price at the mine gate and coupled with road haulage transport costs might likely limit the range of viable destinations to within circa 150-200 km travel distance from Norra Kärr.

Nepheline Syenite B / C attracts a higher sales price to enter the US East Coast and Northern Europe markets the following logistics system is envisaged but would be at the buyer's cost (Table 17-6)

Table 17-6: Possible Logistics System for Nepheline Syenite B / C (at buyers cost)

Stage	Type	Distance	From	To	Description
1	Road	227 km	Norra Kärr stockyard	Mjölby Rail Terminal	Loading to trucks using front end loader. Road Haulage in on-highway 5 / 6 axle haulage trucks with bulk trailers
2	Port	~	Varberg Port	Varberg Port	Unloading / storage and ship loading at Varberg Port on the southwest coast of Sweden. Varberg Port is understood to be well set up for handling of bulk cargo and loading to Handymax bulkers.
3A	Sea Freight	~	Varberg Port	Port of Destination	Ocean freight - for example: Handymax part load direct to East Coast US (e.g. Ports of New York, New Jersey, or Savanah) Coaster vessel full load direct to Port of Rotterdam (or other Northern Europe Port)
4	~	Unknown	Onward journey		Road, rail, inland waterway to destination factory

Alternative Options

As an alternative to road haulage to Varberg, it may be possible to part transport by road to a rail loading siding near to Norra Kärr (e.g. Nassjö for 86 km) and then transport by rail (250 km), however, a suitable rail loading siding and wagon offloading facilities at Varberg would need to be identified.

17.5.5 Eudialyte Mineral Concentrate

Overview

The Eudialyte concentrate (approximately 105ktpa) needs to be transported to the Company's processing facility, which is planned for the Luleå area. The following logistics system is proposed:

- Loading of road haulage trucks at Norra Kärr;
- Road haulage via the National Road E4 to a rail siding using a road haulage Contractor;

- Unloading, storage and loading of rail wagons at the rail siding, which is likely to be located at Mjölby or Nassjö, located on the national railway system and owned and operated by the Company (owner operated or operated by a Contractor);
- Transport via national rail system using a rail freight company who provide wagons, locomotives and arrange access to the rail system; and
- Unloading and handling at siding at the Company's processing facility in Luleå.

Assuming 28-30 tonne payloads can be achieved by rigid 4-axle or articulated 5 / 6 axle highway trucks, and assuming 286 working days per year, around 12 trucks per day are likely to leave the stockyard area carrying eudialyte concentrate. The concentrate contains largely Eudialyte (around 80%) with minor Aegirineand some extremely minor silicates (see section 12.6).

Preferred Logistics System – Road & Rail

The Eudialyte concentrate requires transport from Norra Kärr to Luleå. Sweden has a well-developed national railway system, and this is the preferred option for the PEA. There are a number of locations for potentially accessing rail system, however, Mjölby is selected as the option in the PEA. A stockyard area with road access near to the existing railway system is required and dedicated loading sidings are required. The logistics system is summarised in Table 17-7.

Table 17-7: Preferred Logistics System for Eudialyte concentrate

Stage	Type	Distance	From	To	Description
1	Road	45 km	Norra Kärr stockyard	Mjölby Rail Loading Sidings	Loading to trucks using front end loader. Road Haulage in on-highway 5 / 6 axle haulage trucks with bulk trailers to a dedicated Rail Loading Siding at Mjölby.
2	Rail Sidings	~	Mjölby Rail Loading Sidings	Mjölby Rail Loading Sidings	Unloading / storage and wagon loading at "Mjölby Rail Loading Sidings".
3A	Rail Freight	Circa 1170 km	Mjölby Rail Loading Sidings	Luleå processing facility	Rail haulage via a contract with a rail freight company supplying wagons for loading, and organising locomotives and rail transit on the network.
4	Rail Sidings	~	off-site processing facility (Luleå)		Unloading at off-site processing facility requiring a unloading siding for bulk wagons.

Alternative Options – Rail Siding Location(s)

If a good option for constructing or acquiring a dedicated rail terminal at Mjölby cannot be found, then Nassjö to the southeast is an alternative option but would result in marginally longer rail freight costs (circa 1250 km rail and 80 km road haulage). Equally, the railway and road networks and extend eastward in parallel from Mjölby and so potential locations for sidings could be sough to the east of Mjölby.

Alternative Options –Sea Freight to Luleå

As an alternative, and because Luleå is on the Baltic seacoast, the option to use sea freight was explored. There remains an option to utilise a nearby east coast port suitable for bulk materials (for example Norrköping Port) where product is unloaded, stored, and loaded to coaster vessels (e.g. 5,000 – 10,000 DWT size vessels) for sea freight to Luleå. This option would require suitable unloading, handling, and transfer facilities at Luleå.

17.5.6 MREO, Zr and Niobium Products

Logistics Concept

These products will be containerised and transported by rail or sea-freight from the Company's processing facility at Luleå to destination points. Assuming an average 25 t payload per TEU is achieved, around circa 806 TEUs will be exported per year to a number of destinations conceptually identified as:

- Southern Norway
- East Coast of the UK
- East Coast US Ports
- Rotterdam port.

For the purposes of the PEA economic analysis presented in Section 21 it assumed that the TREO product will be toll processed elsewhere to final products whereas Nb and Zr will be sold as products and exported to Rotterdam for sale.

Once the TEUs are dispatched from the processing facility at Luleå, they will enter the international TEU freight under a long-term contract with an international shipping company.

Sulfuric Acid

Sulfuric Acid is required for processing at the Company's processing facility planned for the Luleå area. For this study, the source of Sulfuric Acid is anticipated to be sourced from a chemical plant in Skellefteå and will arrive to the Company's processing facility by rail in specialised corrosion resistant rail tank-cars.

17.5.7 Project Logistic Concept

The Project's logistics concept for intermediate and final products is presented below in Figure 17-7.

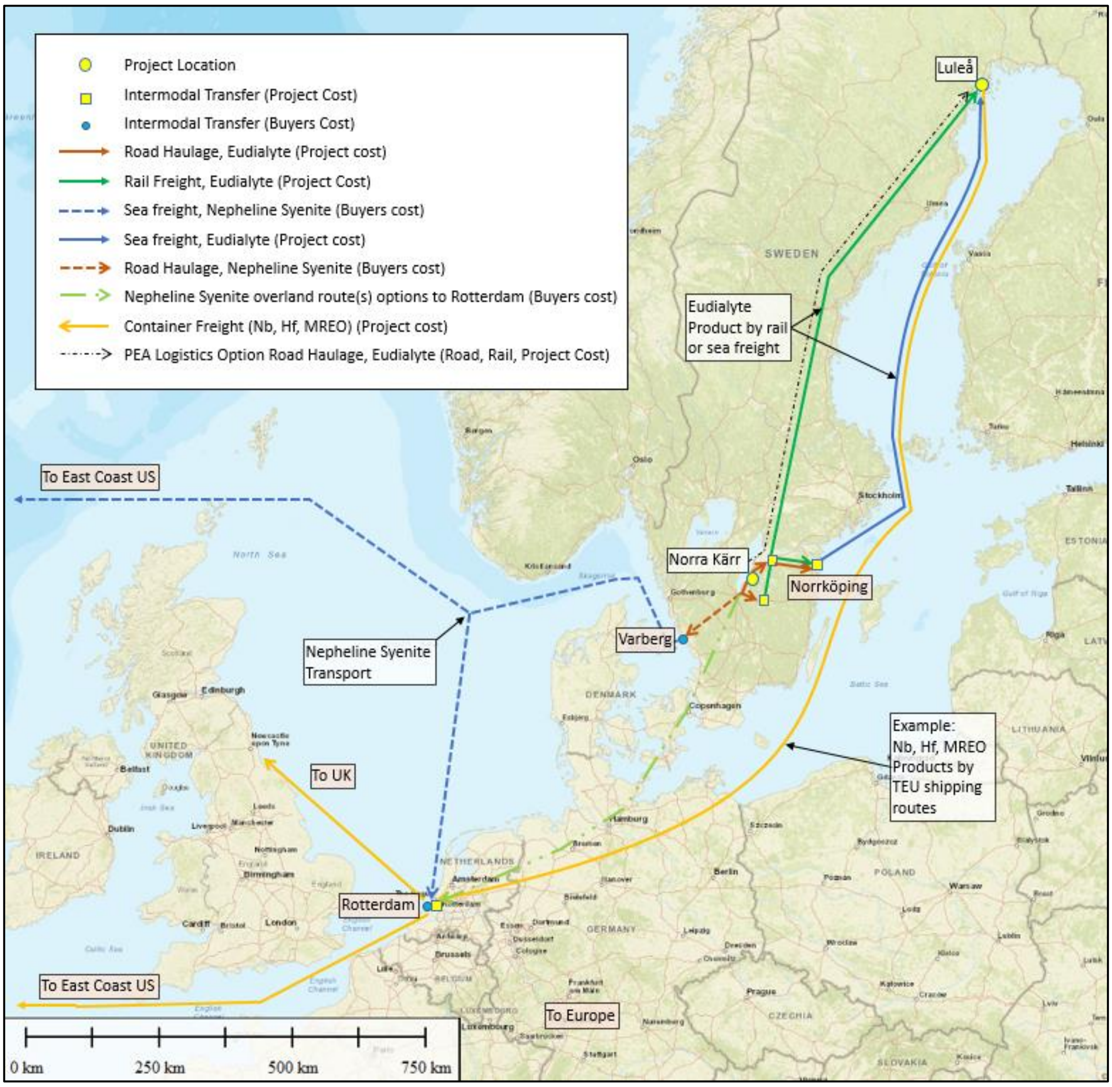


Figure 17-7: Project Logistics Concept for Intermediate and Final Products (TREO product from the processing facility will be shipped to a toll processing facility prior to sale).

17.6 Estimated Costs

17.6.1 CAPEX Summary

The cost estimate for infrastructure is considered overall to have a achieved a Scoping Study / PEA level of accuracy ±40-50% and to fall within the range of an AACE International "Class 5" estimate and is presented in Table 20-5 (Section 20.2).

17.6.2 Basis of Estimate

General

Infrastructure quantities have been derived according to preliminary scope description, capacity requirements, and indicative layouts. Costs are taken from in-house databases and recent budget quotes or benchmarks. The direct costs include fabrication, supply, install, erection works / construction works including any "Contractor indirects".

Earthworks and Civils

Schematic areas have been measured in accordance with function and cost per m2 for bulk earthworks and placement of 0.3m of suitable granular hard standing material. No bulk earthworks estimation has been carried out at this stage. An allowance has been added for internal drainage to a common collection point for treatment of contact water.

Buildings

In general buildings will be pre-engineered steel portal framed or column and beam style buildings with insulated panel roofs and cladding and with all necessary internal electrical, piping, fixtures and fittings, and architectural details. It's likely that some auxiliary buildings will be prefabricated and pre-fitted, modular, or converted container style buildings. A cost per m^2 has been applied dependant on the usage and anticipated style of construction.

Utilities

Lump sum cost allowances have been included for each infrastructure area; design, measurement and specification will be carried out at

Power Supply

Based on the assumed description of the infrastructure required, benchmark costs have been applied.

Explosives Magazine

Explosives supply and any storage requirements will be organised, operated, and maintained by the Explosives Contractor.

Fuel Farm and Dispensing

Assumes self-bunded containerised or prefabricated tanks with concrete bunding in accordance with regulations.

Support Vehicles

Recent quotes (or published prices) have been used on a per item basis.

Off-Site Processing Facility

The costs for the buildings, installations and utilities / services were developed with the same methodology as the Norra Kärr site. Lump sum allowances for site wide earthworks and civils were added. The cost for railway infrastructure was based on measurements from the schematic layout with pricing suing the CESSMM4 price book for civil engineering works, crossed reference against recent benchmark costs, for railway infrastructure. Mobile equipment was costed on a per unit basis using recent benchmarks.

Rail Load-Out Facility

The costs for the buildings and installations are as per the Norra Kärr site. Lump sum allowances for site wide earthworks and civils were added. The cost for railway infrastructure was based on measurements from the schematic layout with pricing as per the Luleå Processing Facility. Railway wagons and locomotives will be supplied by the FOC under the haulage contract.

Road Haulage of Product

This is anticipated to be undertaken by a haulage contractor. All capital costs associated to equipment and maintenance facilities are assumed to be included within the until cost of transport.

Rail Haulage of Product

This is anticipated to be undertaken by a haulage contractor. All capital costs associated to equipment and maintenance facilities are assumed to be included within the until cost of transport.

Indirect Costs

Applied as a factored cost (% of Total Direct Costs) and cross checked against likely actual spend (Table 17-8).

Table 17-8: Indirect Costs

Item	Includes:
Surveys & Detailed Engineering Design	Topographical Geotechnical & Testing Weather station Detailed Design / Tendering for all facilities
Construction Management & Commissioning	TBC
Owners Costs	Project team and office costs. Training of operating personnel;
Contingency	Typically, 20%

17.6.3 Exclusions

The following are excluded from the infrastructure capital cost:

- Pre-feasibility and feasibility studies;
- Processing plant equipment and buildings and other items within the fenceline;
- Land acquisition, planning and permitting costs;

- Changes to the current scope assumptions and other items not stated in the scope;
- Definition drilling, assaying and related reports and models; and
- All taxes and duties.

Note: the estimated costs for the processing plant and waste / tailings dumps as are defined in separate sections of the PEA report are inclusive of all construction and installation work (including but not limited to structural, mechanical, electrical, piping, instrumentation etc.) within the fenceline including groundworks and foundations.

17.6.4 Infrastructure Operating Costs

Norra Kärr

The costs for general operation and maintenance of the site support facilities are covered by G&A costs in the financial model. Loading of product to haul trucks is included in the logistics costs as defined in the financial model.

Off-Site Processing Facility

The costs for general operation and maintenance of the site support facilities and logistics area are covered by G&A and processing costs in the financial model.

Rail Load-Out Facility

The costs for general operation and maintenance of the site, including loading and rail operations within the railway siding are included in the logistics costs as defined in the financial model.

17.6.5 Logistics Operating Costs

Nepheline Syenite Products

The sales prices assumed in the financial model are "FOB Mine Gate" and thus only the stockyard operating costs and truck loading cost by Front End Loader has been estimated. The Company assumes that as part of a long term off-take agreement, the buyer subcontracts transport to a third party. The costs for stockyard operating and truck loading at site are estimated at USD0.5 /tonne product.

Eudialyte mineral concentrate

The logistics costs for Eudialyte mineral concentrate are presented in Table 17-9.

Table 17-9: Preferred Logistics System for Eudialyte mineral concentrate

Stage	Units	Cost (EUR)	Cost (USD)	Description
Norra Kärr Stockyard & Truck Loading	EUR/tonne	0.4	0.5	Front end loader
Road Haulage	EUR/tonne	6.0	7.2	45 km: Haulage Contractor
Rail Siding Operations and Wagon Loading	EUR/tonne	1.5	1.8	Stockyard management, storage, wagon martialling and loading
Rail Freight	EUR/tonne	27.0	23.5	Rail Freight company
Total	EUR/tonne	35.0	42.0	Transport cost

MREO, Zr and Niobium Products

On the basis these are loaded at the processing facility and the international shipping company arranges pickup and transport to the port of destination, the potential costs used in the financial model are as follows[2] (Table 17-10). It is to be noted that the PEA financial model presented in this report assumes that 100% of these products are shipped to Rotterdam with this being the assumed Point of Sale (Pos).

Table 17-10: Container Freight Rates (assuming 25 tonnes per TEU)

Destination	Cost per Container (EUR)	Cost per tonne (EUR)	Cost per tonne (USD)	Source
ROTTERDAM	1,000	40	48	Sirelo.co.uk
EAST COAST US	3,500	140	168	Freightos.com

Sulfuric Acid

During processing of the Eudialyte mineral concentrate Sulfuric Acid is required. On the basis that Sulfuric Acid is obtained from a production facility at Skellefteå and transported by rail in appropriate tank cars, a cost of 14 EUR/t (or 16.8 USD/t) is estimated for rail freight cost including a hazardous material surcharge of per tank car and excluding loading / unloading charges, which are within the supply and processing plant operating cost respectively.

Other Consumables

The cost for supply of other consumables are assumed to include delivery.

18 ENVIRONMENT, SOCIAL AND GOVERNANCE (ESG)

The environmental, social and governance ("ESG") input to this PEA has been developed based on a desktop review of data provided by the client and publicly available information. The information is reported in accordance with NI 43-101 Standards of Disclosure for Mineral Projects.

18.1 Environment and Social Context

The Norra Kärr Project is in south-central Sweden, approximately 300 km south-west of Stockholm. It is located 1.5 km east of Lake Vättern - one of the largest lakes in Sweden and a Natura 2000 site, an ecologically sensitive area designated at European level to safeguard Europe's major habitat types and endangered species. There are two other Natura 2000 protected sites on the shores of Lake Vättern – Holkaberg and Narbäck. The Project site itself does not overlap any Natura 2000 areas. The Project site and the surrounding area is characterised by alternating agricultural land, scattered homesteads and forests. The main north-south E4 highway runs approximately 500 m to the west of the project area with the site itself accessed by rural roads.

[2] Note that container freight market is in a state of flux at the time of writing (since Q2 2020) due to the global COVID-19 pandemic.

The Project occurs along the border of two counties, Jönköping in the south and Östergötland in the north. According to 2019 data the County of Jönköping has a population of 363,599. The Municipality of Jönköping has a population of over 141,081 (City Population 2020).

The closest town to the project is Gränna which is approximately 10 km south-west and in 2018 had a population of 4,124 (Jönköping County 2021). Tourism has a significant economic importance in Gränna particularly during the summer months. In addition to permanent residents, the wider project area is used for outdoor recreational activities such as hiking and fishing.

The nearest city is Jönköping approximately 30 km south of Gränna and has a population in excess of 93,000. The city is accessible by either the E4 highway, rail or through the local airport. A number of stately offices and departments are based in the urban area of Jönköping and it has a university with over 12,000 registered students (Jönköping University). Future employees would likely choose the surrounding towns to reside including Gränna or Jonkoping.

The Project is approximately 200 m above mean sea level (mamsl) with regional topography surrounding the project site consisting of undulating hills ranging from 150 to 300 mamsl.

The climate in the region is considered humid continental with well-defined seasons. Temperature averages vary between 16°C in July to -3°C in February. Precipitation in the region averages approximately 630 mm per year with July and August as the wettest months.

The deposit sits within the immediate catchment of a small lake named Gyllingesjön that is within the larger catchment of Lake Vättern (Figure 18-1). The outflow from this lake flows north north-west into a series of ponds within gardens of homesteads. It is evident from satellite imagery that these ponds are used for recreation, agricultural or aesthetic features within the properties. The stream continues north north-west some 3 km discharging into Lake Vättern.

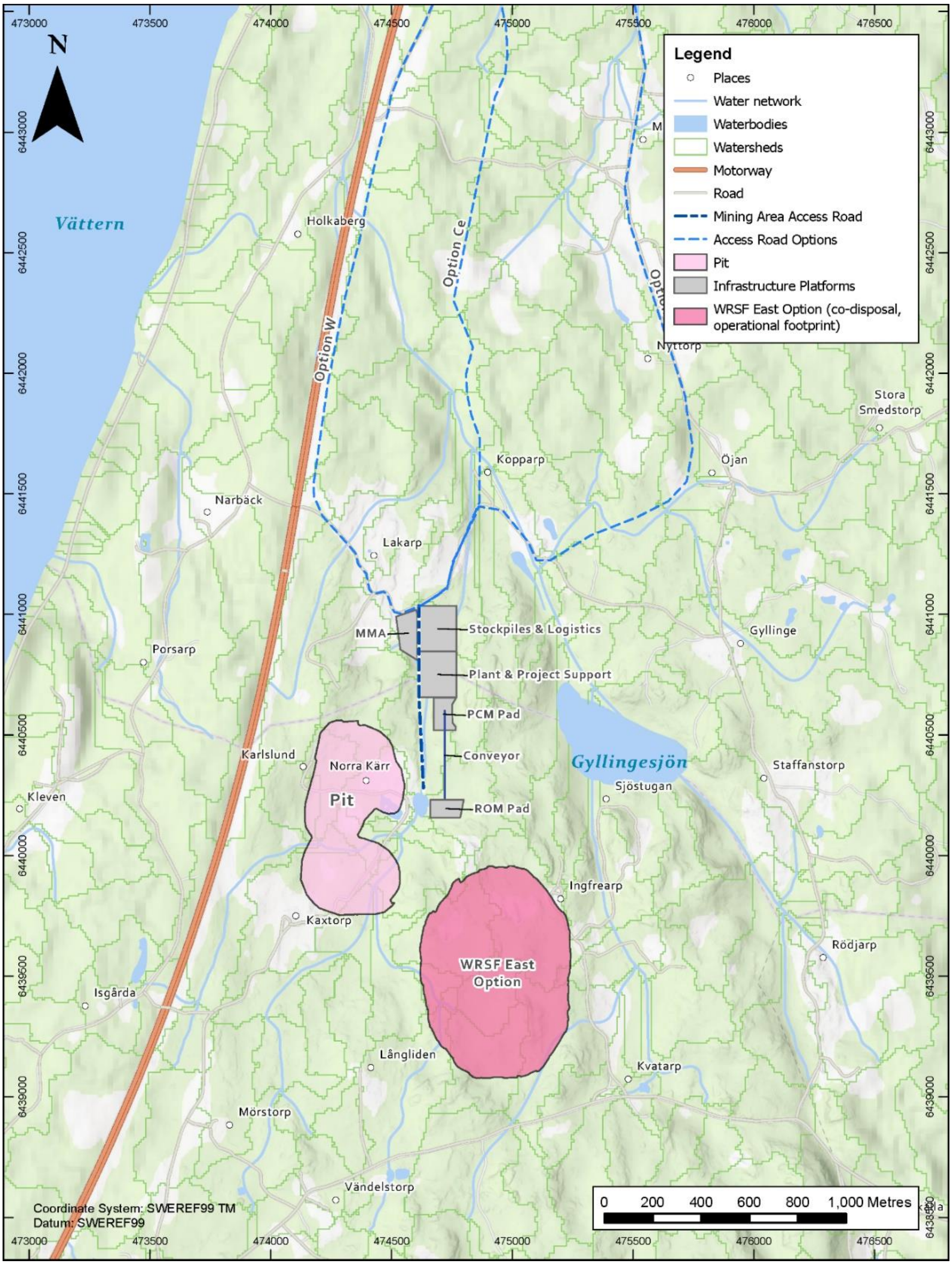


Figure 18-1: Surface water features and watersheds around the project

Groundwater in the area is generally high quality and known to be used by several properties around the project area (Golder 2014). The groundwater in the immediate area around the proposed project site is believed to flow generally north east in the direction of Gyllingesjön.

The Project site itself does not overlap any European designated nature protection sites or Swedish National Parks but there are several protected sites in the immediate vicinity. Lake Vättern has a variety of different environmental designations. The entire lake is protected under the European Habitats Directive and the north eastern portion is also designated as a Special Protection Area, a European protection designation specific to birds under Directive 2009/147/EC, referred to as the Birds Directive. In addition to the Natura 2000 sites, a number of streams draining to the lake are deemed Water Protection Zones. The water protection zone extends up from the lake; these are separate from the Natura 2000 protected areas but are connected. The Water Protection Zones are protected under Swedish legislation but do not have any significance at the EU level. The various protected areas are shown on Figure 18-2 along with the main Project components.

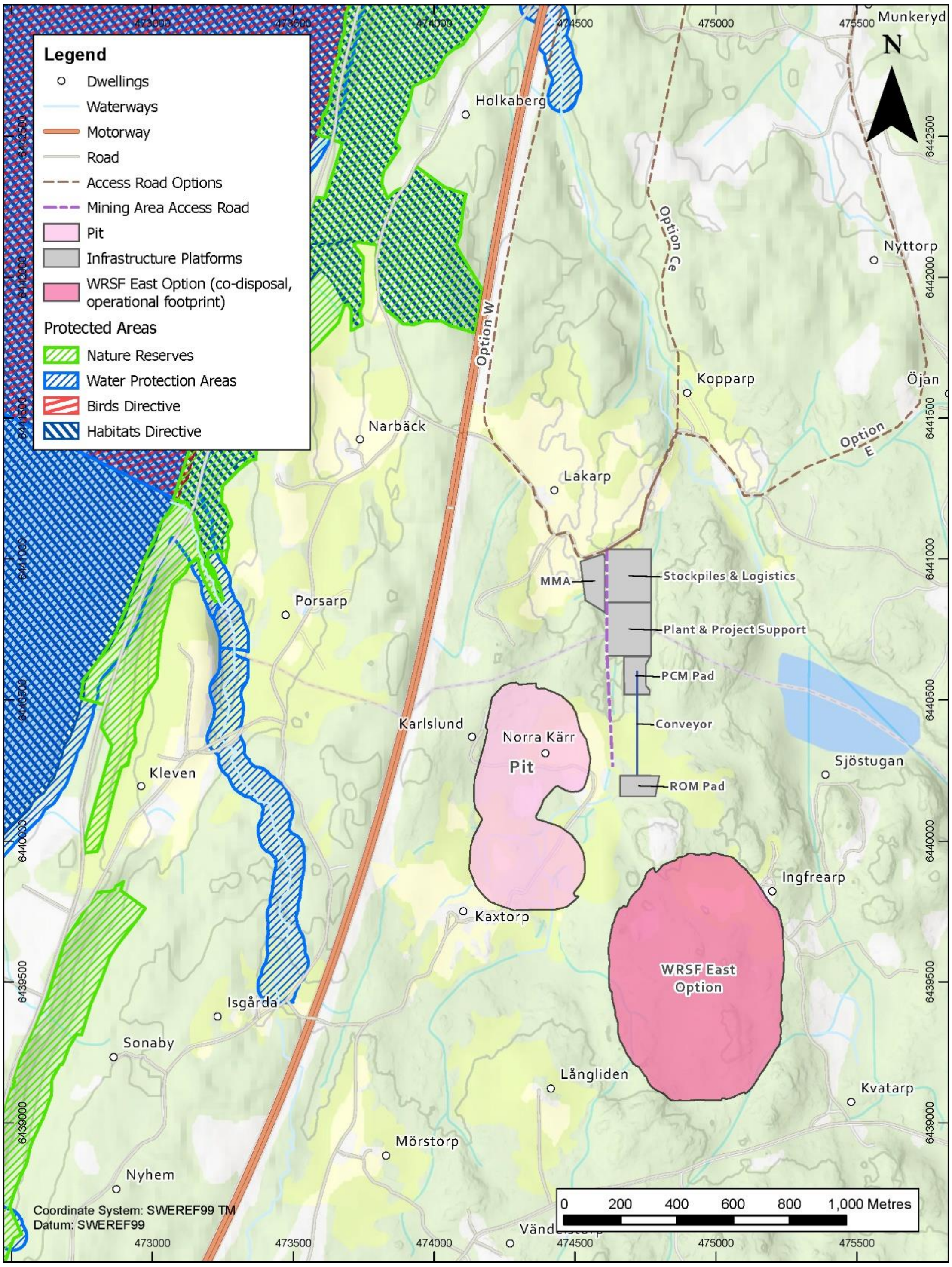


Figure 18-2: Site location in relation to adjacent national protected areas

A 2012 Golder Environmental Impact Assessment notes that 231 threatened species have been identified in the Eastern Vättern Slopes area. The Eastern Vättern Slopes is a long narrow area which lies between the shoreline of the lake and the E4 highway, from Getingaryd in the north to the town of Gränna in the south. The Eastern Vättern Slopes have been identified by the Fund for Nature (WWF) as one of 100 'hotspots' for diverse forestry in Europe A number of the threatened species include those with conservation protection within the EU, such as chiroptera (bats), mustilidae (badgers and otters), falconidae (falcons) and salamandridae (newts).

In addition to the European protected sites, there are areas of natural value in the surrounding landscape managed at the county level and not designated as protected under European Law.

The eastern Vattern shores are part of a UNESCO Man and Biosphere Reserve (East Vattern Scarp Landscape) that was declared in 2012. The status as a biosphere reserve does not convey any legal protection nor impose any limitations of or increase requirements upon the current protected areas. The biosphere area is split into three zones:

- Core Area - comprises a strictly protected zone that contributes to the conservation of landscapes, ecosystems, species and genetic variation
- Buffer Zone - surrounds or adjoins the core area, and is used for activities compatible with sound ecological practices that can reinforce scientific research, monitoring, training and education.
- Transition Area - where communities foster socio-culturally and ecologically sustainable economic and human activities.

The Norra Kärr deposit lies approximately 1 km from the core zone. The buffer zone has the same boundary as the national conservation areas, while the wider ‘development zone’ covers the whole Project.

The nearest known cultural heritage site of significant value is located approximately one km west of Norra Kärr along the shoreline of Lake Vattern. The site contains the ruins of a manor house from the 1600s. This is outside the zone of influence of the project for this type of receptor. There is also a neo-classical 19th century manor in the village of Uppgränna and the town of Gränna has a variety of preserved buildings and houses. The coastal plain of Lake Vättern within the municipality of Jönköping is designated as a Natural Interest Area for protection of the cultural environment (Swedish: *Riksintresse för Kulturmiljövård*).

As a member of the European Union Sweden is subject to the bloc’s submissions for nationally determined contributions (“NDC”s) to the Paris Agreement tracking of carbon mitigation. Through this NDC Sweden is committed to a 40% greenhouse gas reduction on 1990 emissions by 2030. In July 2021 the European Commission adopted a series of proposals targeting the net reduction of greenhouse gases by 55% by 2030 relative to 1990. Sweden has voluntarily commitment to a 63% reduction by 2030 and net neutrality by 2045 through the Swedish Policy Framework (Ministry of Environment and Energy – undated). Many Swedish minerals businesses have committed to decarbonisation in line with this national commitment.

18.2 Project Design Context

The current work being undertaken as part of this PEA has resulted in a substantive update to the design of the project. This includes transport of a concentrate by road to a railway siding and then taken to Luleå by train before further processing at a brownfield industrial site near Luleå. This is a new development relative to the work carried out as part of the previous PFS in 2015 (see GBM footprint in figure below) and Golder EIA produced in 2012. The layout of various project components at the proposed mine site has also evolved with the consolidation of the footprint of the infrastructure and waste management facilities. Changes to the waste management include the use of filtered tailings disposal which is in line with EU recommended Best Practice. These changes have reduced the project footprint at the Norra Kärr site and have also reduced the number of sub-catchments potentially impacted by the infrastructure (Figure 18-3).

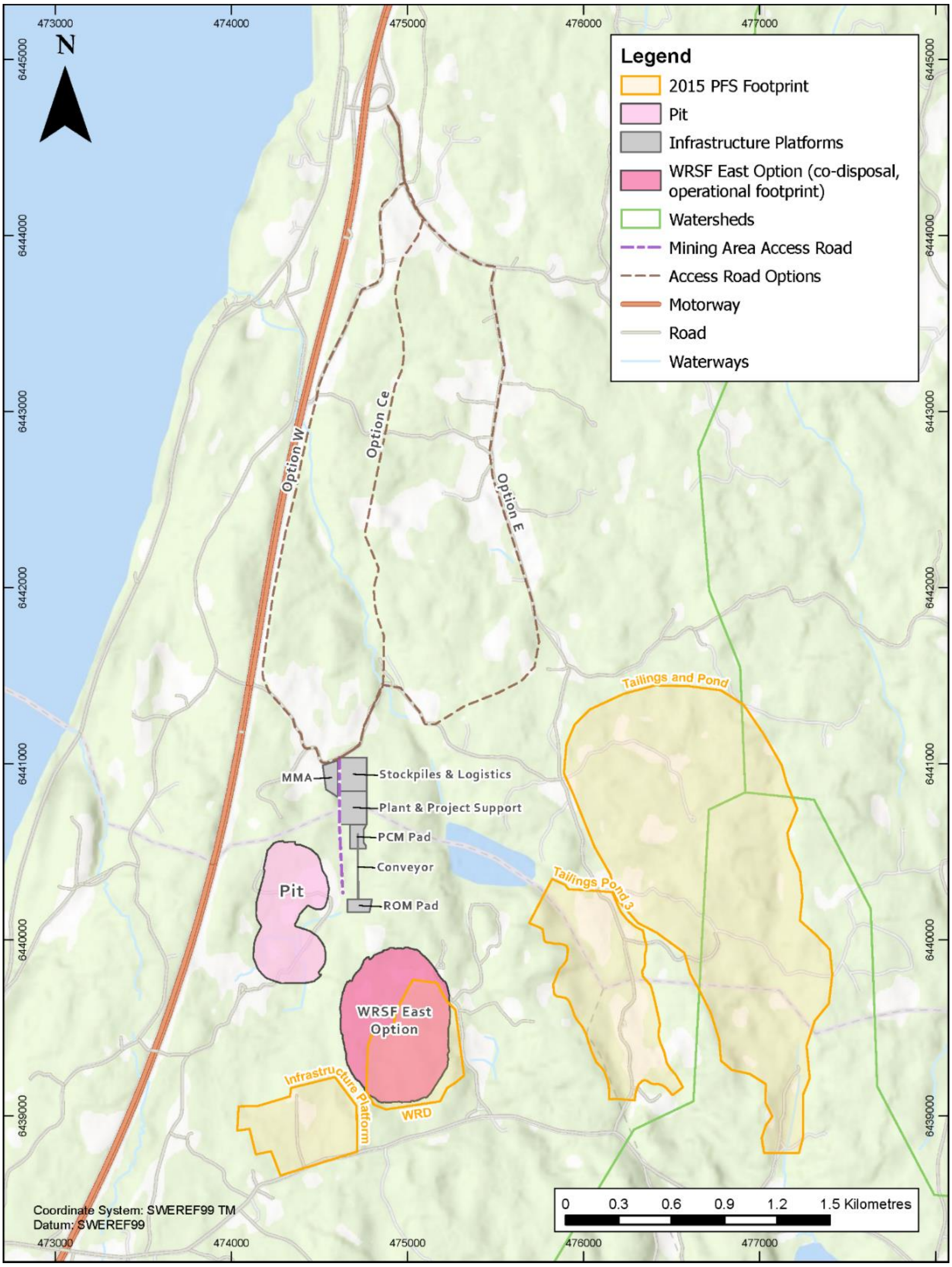


Figure 18-3: Current infrastructure footprint option in relation to the 2015 PFS GBM design

The Norra Kärr deposit is listed as a mineral deposit of national interest by the Geological Survey of Sweden. This designation is linked to the potential for the deposit to provide a supply of rare earth elements to Sweden and Europe.

18.3 Legislation and Permitting

Sweden is a member of the European Union and as such is subject to the Directives and Regulations of the European Parliament and its Commission. European Directives must be transposed into member states legislation that often merely reference the text of the Directives. Key directives applicable to the project and details of their requirements are outlined in Table 18-1.

Table 18-1: Key EU Directives applicable to ESG in mining

Directive	
Short name	**Full name and notes**
EIA Directive	Directive 2011/92/EU of the European Parliament and of the Council of 13 December 2011 on the assessment of the effects of certain public and private projects on the environment. This was amended by Directive 2014/52/EU on 16 April 2014. Transposition of the Directive into national law was required by 16 May 2017. The developer may request to competent authority to say what should be covered by the EIA information to be provided by the developer (scoping), the developer must provide information on the environmental impact (report), the Environmental Authorities and the public (and affected Member States) must be informed and consulted, and the Competent Authority decides, taking into consideration the results of the consultations. The public must be informed of the decision and can challenge the decision before the courts.
Public Participation Directive	Directive 2003/35/EC of the European Parliament and of the Council of 26 May 2003 providing for public participation in respect of the drawing up of certain plans and programmes relating to the environment and amending with regard to public participation and access to justice Council Directives 85/337/EEC and 96/61/EC - Statement by the Commission
Habitats Directive	Council Directive 92/43/EEC on the Conservation of natural habitats and of wild fauna and flora. The Habitats Directive alongside the Birds Directive establishes the Natura 2000 Network across Europe. The network consists of protected areas across the continent, and ensures the conservation of rare, threatened or endemic animal and plant species. Over 200 habitat types are also targeted for conservation in their own right, and over 1,000 species.
Birds Directive	Council Directive 2009/147/EC on the conservation of wild birds – replaces Council Directive 79/409/EEC of 2 April 1979 on the conservation of wild birds.
Water Framework Directive (WFD) Daughter directives: Environmental Quality Standards Directive (also referred to as the “Priority Substances Directive”); and Groundwater Directive	Directive 2000/60/EC of the European Parliament and of the Council establishing a framework for community action in the field of water policy) The WFD pulls together a number different legacy pieces of legislation. The multinational nature of the European Union enables a river basin approach to be instigated in instances where a river system originates in one Member State and passes through one or more other Member States before its estuary. The Directive requires the development of River Basin Management Plans (RBMPs) for each river basin district. It requires surface waters be managed or improved to good ecological and chemical status, and that groundwater should not be polluted. Priority Substances

Directive	
Short name	**Full name and notes**
	The directive on discharge of dangerous substances (Directive 76/464/EEC) was fully repealed in 2013. The requirements of this directive have been integrated into the Water Framework Directive. The Water Framework Directive provides for a list of Priority Substances (in Annex X). The history of this list is as follows: Decision 2455/2001/EC established the First list. **The Environmental Quality Standards (EQS) Directive**, a daughter directive of the Water Framework Directive (officially named “Directive 2008/105/EC of the European Parliament and of the Council of 16 December 2008 on environmental quality standards in the field of water policy”) set the quality standards as required by Article 16(8) of the Water Framework Directive. Annex II to the Environmental Quality Standards Directive replaced Annex X of the Water Framework Directive. The list of priority substances was updated in 2013 by the Directive 2013/39/EU (Directive 2013/39/EU of the European Parliament and of the Council of 12 August 2013 amending Directives 2000/60/EC and 2008/105/EC as regards priority substances in the field of water policy). This directive both revised some EQS for priority substances and added 12 substances to the list of priority substances. The **Groundwater Directive** (Directive 2006/118/EC of the European Parliament and of the Council of 12 December 2006 on the protection of groundwater against pollution and deterioration) is the other daughter directive of the Water Framework Directive. Annex II sets forth threshold values for groundwater pollutants and indicators of pollution. Annexes I and II of the Groundwater Directive 2006/118/EC were reviewed in 2013. Annex II was amended by Directive 2014/80/EU of 20 June 2014.
Floods Directive	Directive 2007/60/EC of the European Parliament and of the Council of 23 October 2007 on the assessment and management of flood risks. The Directive requires governments to assess flood risk, to produce flood risk maps and instigate management plans.
Drinking Water Directive	Council Directive 98/83/EC of 3 November 1998 on the quality of water intended for human consumption. The Directive sets minimum drinking water quality standards based on World Health Organisation (WHO) guidelines, measured at the tap.

Directive	
Short name	**Full name and notes**
Mine Waste Directive	Directive 2006/21/EC of the European Parliament and of the Council of 15 March 2006 on the management of waste from extractive industries. (Note this directive also amends Directive 2004/35/EC – the Environmental Liability Directive.) Several decisions have also been published implementing the requirements of the Mine Waste Directive, including - 2009/337/EC on Criteria for the classification of waste facilities in accordance with Annex III - 2009/335/EC on Technical guidelines for the establishment of the financial guarantee - 2009/360/EC on technical requirements for waste classification - 2009/359/EC on Definition of inert waste in implementation of Article 22 - 2009/358/EC on the Harmonisation, the regular transmission of the information and the questionnaire referred to in Articles 22(1)(a) and 18. Mining Waste Facilities are those in which extractive wastes are stored for a time period (a time period is not applicable to higher risk facilities) and are required to apply for and maintain a permit. Material destined for such a facility must be adequately characterised prior to deposition.
Waste Framework Directive	Directive 2008/98/EC of the European Parliament and of the Council of 19 November 2008 on waste and repealing certain Directives
Industrial Emissions Directive	Directive 2010/75/EU of the European Parliament and the Council on industrial emissions is the main EU instrument regulating pollutant emissions from industrial installations. It recasts seven previously existing directives, including the Integrated Pollution Prevention and Control (IPPC) Directive and directives concerning large combustion plants, waste incineration, solvent emissions and waste from the titanium dioxide industry. The Directive requires Operators apply Best Available Techniques, including technology, management systems and emission limits decided at a European community level.
Ambient Air Quality Directive Daughter directive: Directive 2004/107/EC	Directive 2008/50/EC of the European Parliament and of the Council of 21 May 2008 on ambient air quality and cleaner air for Europe. Directive 2004/107/EC of the European Parliament and of the Council of 15 December 2004 relating to arsenic, cadmium, mercury, nickel and polycyclic aromatic hydrocarbons in ambient air.
Environmental Noise Directive	Directive 2002/49/EC of the European Parliament and of the Council of 25 June 2002 relating to the assessment and management of environmental noise.
Major Accidents (Seveso Directive III)	Directive 2012/18/EU of the European Parliament and of the Council of 4 July 2012 on the control of major-accident hazards involving dangerous substances, amending and subsequently repealing the Seveso-II Directive (Council Directive 96/82/EC).
Environmental Liability Directive	Directive 2004/35/EC of the European Parliament and of the Council of 21 April 2004 on environmental liability with regard to the prevention and remedying of environmental damage. This applies to serious environmental damage to land, water and to species and habitats.
EU Emission Trading Scheme Directive	Directive 87/2002/EC of the European Parliament establishes a trading scheme for greenhouse gas emissions across the EU. The flagship carbon directive for the EU it was the first of its kind internationally. Phase IV begins January 2021.

Directive	
Short name	**Full name and notes**
Energy Efficiency Directive	Directive 2012/27/EU is an EU directive which mandates energy efficiency in the EU and includes energy efficiency targets, building renovation, energy efficiency obligation schemes, energy audits, promotion of energy efficiency in heating and cooling and other rights
REACH	Registration, Evaluation, Authorisation of chemicals came into force in 2007 replacing a number of systems in the EU for chemicals management. The Regulations require essentially all products coming into the EU to be registered and is the most comprehensive and wide-reaching supplier requirement ever constructed by the EU.

The key Swedish legislation relevant to the environmental and social aspects of the Norra Kärr Project is outlined in Table 18-2.

Table 18-2: Key Swedish law applicable to mining

Title	Summary
The Minerals Act, *Minerallag* (1991:45)	This is applicable to the exploration and exploitation stages of mine development
The Environmental Code, *Miljöbalk* (1998:808)	The purpose of this Code is to promote sustainable development which will assure a healthy and sound environment for present and future generations. The procedure and requirements for environmental impact assessments, plans and planning documents should follow this Code. The applicant is obliged to consult the County Administrative Board ("CAB") or the local Environmental and Public Health Committee before submitting an application for a permit and a public hearing is often held.
The Planning and Building Act, *Plan- och bygglag* (2010:900)	Once the Land and Environmental Court has granted permission to begin operations, a construction permit is required by the local municipality. A construction permit normally takes between four and eight weeks to process and covers buildings and other facilities that need to be constructed in connection with the mining project.

18.3.1 Mineral authorisations

Primary authorisations in Sweden are separated according to the development phase of the project. For both exploration and exploitation authorisations the following restrictions apply:

- must be more than 30 m from transport infrastructure such as roads and railways;
- must be more than 200 m from an inhabited building;
- cannot be on electrical infrastructure sites;
- must be more than 200 m from churches, assembly halls, hotels, hospitals or anywhere accommodating more than 50 people;
- must not be in areas of fortification;
- must not be in churchyards or burial grounds;
- must not be in certain specified mountain areas in Sweden; and

- must not be in National Parks.

Exploration Licences (Permits)

Exploration Permits (Swedish: *undersökningstillstånd*) are granted for a period of three years. They may be extended by application to 11 years and can be further extended to a maximum period of 15 years but only in exceptional circumstances.

According to Section 3 of the Minerals Act a holder of an Exploration Permit may have priority in applying for an exploitation concession. A minimal financial assurance must be provided and guaranteed to provide for any damage and restoration. Should exploration terminate and the project not progress to mining, the Exploration Permit holder may have to provide a report to the Government on the minerals explored and results.

Exploration Permits cannot be granted for land within a protected zone including a buffer of 1,000 m.

Mining Lease (Exploitation Concessions)

Mining Leases, also known as Exploitation Concessions (Swedish: b*earbetningskoncession*), are granted for a period of 25 years. If exploitation is ongoing the Mining Lease may roll-over without the need to submit additional applications.

A pre-requisite for the granting of a Mining Lease is that Chapters 3 and 4 of the Environmental Code relating to suitability of land use versus other interests (1998:808; basic and special provisions respectively for the management of land and water) are complied with. Applications for a Mining Lease must be accompanied by an Environmental Impact Assessment ("EIA", Swedish: *miljökonsekvensbeskrivning,* or *MKB)* and must be made to the Mining Inspectorate to be evaluated for approval by the local County Administrative Board ("CAB", Swedish: *Lansstyrelsen*).

Status of authorisation

There is a single Exploration Permit (Norra Kärr No.1) covering the area of the proposed pit (see Figure-3-3). The Exploration Permit, and additional perimeters now lapsed, were first granted to Tasman Metals AB in 2009 and was valid for three years. The Exploration Permit was renewed twice, and a request for a further five-year extension was submitted to the Mining Inspectorate in August 2019. The extension was granted in June 2020 with the Exploration Permit being valid until August 2024. Subsequently the Swedish parliament passed legislation to mitigate the impacts of COVID-19 by giving exploration companies an additional year to carry out their work which extends the Norra Karr exploration license to August 31, 2025. The five-year extension of the exploration license was appealed, and the administrative court of Lulea rejected the appeal in March 2021, upon which the case has been appealed to the next instance which is pending decision to grant leave of appeal. The extension of the exploration license remains in force until a final ruling in the case has been made, and remains in force until a final ruling has been made on the mining lease application (see below).

A 25-year Mining Lease was granted to the Company's Swedish subsidiary Tasman Metals AB, recently renamed to GREENNA Mineral AB, covering Norra Karr in 2013 by the Mining Inspectorate with approval of the local CAB. In 2014 the Government of Sweden upheld the granting of the mining lease after an appeal. In 2016, following an appeal to the Supreme Administrative Court (SAC) in Sweden regarding the decision-making process of the Bergsstaten under the Minerals Act, the Norra Karr mining lease reverted from granted to application status. On May 5, 2021, The Mining Inspectorate of Sweden ("Bergsstaten") rejected the mining lease application with the motivation that since the Company had not acquired a Natura 2000 permit for the Project, they were not able to rule on the mining lease application.

The Company subsequently lodged an appeal to the Government to cancel Bergsstaten's rejection of the mining lease application and continue the evaluation of the application once the SAC has ruled whether a Natura 2000 permit should be a pre-condition for the granting of a mining lease or not. This is based on the fact that this is not an isolated incident and similar case outcomes are still pending for other mining companies in Sweden too.Most importantly, the Company is looking to use the redesigned scope of the Project as presented in this report to form the basis for additional environmental and hydrological studies as a basis for an amended or new mining lease application.

18.3.2 Environmental permitting

Primary environmental approvals in Sweden are separated according to the development phase of the project. There are no specific environmental approvals required to obtain exploration permits; however, an Exploration Permit will not be granted for land within a protected zone including a buffer of 1,000 m (including Natura 2000 sites).

As described above, a pre-requisite for the granting of a Mining Lease is that the application must be accompanied by an EIA/MKB. The emphasis of this initial EIA is on showing there is no obvious conflict with the surrounding land uses and the assessment is based on early project design information. Stakeholder consultation is required, and the local CAB will provide comment on the EIA.

In addition to a Mining Lease, commencement of mining activities require an Environmental Permit (Swedish: *miljötillstånd*) under the Swedish Environmental Code, which is issued by the Environmental Court (Swedish: *Miljödomstolen*). The Environmental Permit will define the conditions for the design, building, operation and closure of a mining installation. The permit application must be supported by a second (more comprehensive) EIA/MKB that includes formal consultations with stakeholders, including local communities, indigenous people and the local CAB. Decisions by the Environmental Court may (with leave to appeal) be appealed to the Environmental Court of Appeal and further to the Supreme Court. Construction activities within water areas (such as tailings dam, clarification pond), requires special considerations in the application for an environmental permit. One such consideration is the right of disposition of the water, which the Company must have before the application is submitted. Right of disposition of the water is normally obtained by acquisition of the land where the water works will take place or through an easement granted either by the landowner or by an authority.

The Environmental Act designates special protected areas. Any operations or activities that risk significant impact on such areas need to apply for a permit to commence operations (Natura 2000 permit). In April 2021, the Swedish Parliament voted for the Government to propose legislative change so that a Natura 2000 permit should not be required at the Mining Lease stage of permitting but rather at the Environmental Permitting stage.

In addition to the abovementioned permits, mining activities require an agreement with the landowner(s) or a decision by the Mining Inspectorate regarding designation of land above ground to be used for the activities.

A building permit is also needed under the terms of the Planning and Building Act (1987:10); this permit is issued by the local authority.

Status of authorisation

An Environmental Permit (under the Environmental Code) has not yet been applied for. An application for this permit will be submitted subsequent to the approval of a Mining Lease (as described above).

18.3.3 Surface Rights

As long as a project proponent holds either an exploration or exploitation authorisation, they are permitted entry over that land for the purposes of the activities outlined in their authorisation. However, all activities that cause damage to that property must be paid for; either in terms of payment for damage to the landowner, or outright purchase of the property if the damage is extensive. Surface rights and rights of access to the property and other required land must be purchased or leased. Although the landowner is not considered to have a right to the sub-soil of their land, the Minerals Act makes it clear that 0.15% of the value of the mineralized rock must be paid to the landowner in compensation. In the event there is more than one landowner this must be shared amongst them.

Land designation

The designation of land and water in the project area is regulated by the Planning and Building Act (2010) and based on the planning of the Municipality of Jönköping.

When it comes to land designation according to the Minerals Act, this is not necessary prior to submitting an application according to the Environmental Code. Land designation matters are dealt with after the decision from the Land and Environmental Court has been made.

Status of surface rights

The land around the deposit and the areas for the facilities are held by a number of landowners. SRK understands the project proponents have some land holding in the proposed Project area, the details of which are still to be confirmed in relation to the revised Project footprint (see Section 18.5.6).

Surface rights and rights of access to the property and other required land must be purchased, including a premium as per Swedish mining act, or leased. All purchases of land need to be registered with the Swedish Land Survey authority to register the title to the property.

Alternatively, it is possible to get a designation of land decision from the Mining Inspectorate granting access to land needed for areas outside of the Mining Lease. If landowners refuse to sell the land, the designation may still be granted, and the landowners must be compensated as per Swedish practice.

18.4 Status of ESG activities

Baseline studies have been undertaken encompassing the area around the Mining Lease. However further studies are expected to be needed for environmental permitting. SRK is not aware of any ongoing environment or social studies intending to inform an environmental and social impact assessment as this will only be relevant using the new design of the project reported in this report.

It is understood that the baseline data on ecology and cultural heritage as per the 2015 PFS does not cover the entire project area and needs to be further studied. This is also the case for groundwater, noise, air quality, visual, traffic and greenhouse gases. This is particularly relevant given the substantive updates to the project design currently being considered. The 2012 Golder EIA is preliminary in nature and includes recommendations to 'further identify and explore the environmental conditions in the surroundings and the expected environmental impact of mining activities in their entirety'. This is in keeping with the two-stage approach to the full EIA process required.

A preliminary water management strategy has been developed as part of the 2015 PFS (Davidson *et* al, 2015b) and a site wide water balance modelling exercise undertaken. This will require review in light of the project updates.

18.4.1 Stakeholder Engagement

Review of previous studies indicate that early stakeholder engagement activities were undertaken in 2012 with meetings held with the Jönköping CAB, the municipality of Jönköping, landowners and the general public. SRK understands that these activities included a public hearing process as a part of the permitting requirement for the Mining Lease.

Perception surveys were conducted in 2013 and 2014. The 2013 survey was carried out by an external company to gather an understanding of local attitudes towards the project. The 2014 survey results were reported in the local newspaper.

Documents reviewed by SRK suggest that the perception surveys indicated the respondents' attitudes towards the project as more positive than the activist-led discussion. There are however ongoing community concerns around loss of peace, increased noise and dust, and loss of livelihoods including farming, forestry and the possibility to rent summer houses to tourists.

In the beginning of 2020, a stakeholder consultation meeting was held with the Östergötland CAB as part of a future Natura 2000 permit application. SRK understands that more recent and ongoing engagement with the local communities to communicate Project related information has been limited.

Potential shortfalls in the current stakeholder system include:

- lack of clarity whether stakeholders have been identified and mapped according to their level of interest and influence over the Project; and
- lack of engagement records for meetings with any stakeholder group.

Based on media research by SRK it is clear there has been vocal opposition towards the project in the local area and in Stockholm. The opposition is being led by anti-mining activists and local residents. There have been protests and considerable public discussion around the toxicity of rare earth mineral mining and of radioactivity. Concerns around the perceived impacts on Lake Vattern, drinking water and plants and animals in the area have also been expressed.

There is a possibility that, with the limited information on the potential social and environmental impacts, limited engagement by the company and an increase in speculation and rumours, more local residents might be influenced by those already in opposition to the project. Also of importance is the SAC's ruling in 2016 in favour of the 'mining-sceptical movement' in relation to the Mining Lease.

18.5 Key ESG matters

The following points summarise the key environmental and social matters that could pose a risk to the project. These are issues that require proactive management to facilitate the obtaining of approvals, mitigate local communities concerns and reduce long-term management costs that could affect the value of the asset either in terms of direct costs or through delays in being able to develop and implement the Project.

18.5.1 Mine Waste

Geochemical analysis of the waste rock reported in the 2015 PFS showed low levels of alpha and beta radiological activity in most samples with only one leach sample from humidity cell tests on kaxtorpite material showing results that exceed World Health Organisation ("WHO") guidelines for drinking water quality. The report recommended further investigations.

With respect to acid drainage potential, the analysis suggests a neutralisation potential ratio above 3 and a sulfide-sulfur content less than 0.1% suggesting the waste will not be acid generating or generate acid rock drainage. It was noted that arsenic and zinc concentrations in the kaxtorpite rock samples are above values common to 99 Swedish quarries (Golders) .

The nature of the waste rock and any tailings expected to be generated are discussed further in section 23.3. It is predicted that the waste rock stored at Norra Kärr will be non-hazardous and non-inert.

The transport of the concentrate off site for final processing and the sale of the nepheline syenite (originally considered a waste product) will reduce both the footprint and pollution risks at the Norra Kärr site.

SRK understands the Luleå facility will host the acid leaching stage of the mineral recovery process and will produce tailings which will need to be neutralised with lime. The wastes from the Luleå processing facility are yet to be fully quantified and have not been characterised or classified. Concentrations of thorium and uranium will require further assessment when the process circuit for this facility is confirmed.

The processing of the concentrate material will also generate gypsum. The intention is to sell the gypsum as a by-product and there are a number of precedents for this. If this is not possible it would be classified as a waste under the EU Waste Framework Directive. Gypsum waste is subject to control under European Council Decision 2003/33/EC and must be managed within waste disposal facilities to prevent the generation of hydrogen sulfide gas.

18.5.2 Water management

Surface water

As part of the project design in 2014 the waste facilities extended over two river catchments with water from the tailings facility draining into a catchment to the southeast[3].

The 2021 project update has resulted in a reduced project footprint with no large scale slurry tailings facility required. This consolidation places all the infrastructure at Norra Kärr within a single watershed draining to the northwest (Figure 18-1). The highway to the west of the project area is a high point within the local topography and as such forms a watershed for the sub catchment in which the project area lies.

Water management infrastructure will be designed to collect runoff from the waste rock storage areas and any excess water from the pit dewatering activities. Process water will be recycled within the process plant. The infrastructure at the Norra Karr site is being designed to eliminate the need for discharge of any mine or process contact water. This will need to be confirmed as part of the next stage of the study when the water balance is further refined. Given the proximity of the Natura 2000 sites and other areas of conservation significance, management of surface water and the associated water quality will be a key feature of the final design.

Groundwater

Hydrogeological modelling conducted by Wardell Armstrong International in 2014 indicated the groundwater gradient prior to construction to be in the direction of Lake Gyllingesjön. Water from the Lake would then follow the drainage pathways north north-west to Lake Vattern. Over the 20-year LOM modelled by Wardell Armstrong the pit would act as a groundwater sink within five years of start-up and would remain so for the LOM. Gyllingesjön may act as a constant head of water unless the sediments below the lake become dewatered by the mine draw-down. No mention is made in the report regarding water levels in the lake in this event.

Although Sweden is not considered a water stressed region, SRK understands that dwellings in the area abstract groundwater for domestic use. Golder Associates in 2014 met with a stakeholder with two properties in the area that use boreholes to provide potable water for the dwellings. One of these is a summer house on the east shore of Gyllingesjön that may be impacted by mine dewatering, although Golder (2014) reports that drawdown is most likely to impact the west of the Project more than the east[4].

[3] LEM PFS – Supplemental site visit, Environment and Social Scope (2014) Golder Associates
[4] Technical Memorandum – Hydrogeological Description (translated from June 2012) (2014)

18.5.3 Climate change and greenhouse gas emissions

All new industrial developments are being scrutinised in relation to their carbon footprint and also in relation to their climate change resilience. The Norra Kärr project will need to demonstrate that all aspects of the project have been made as energy efficient as practicable such as through the use of electrified equipment (operated using renewable energy from the national grid). Consideration should also be given to the use of biodiesel where practical. There is the potential for the Project to be expected to offset its carbon footprint. The infrastructure design criteria should also show how climate change predictions have been taken into consideration.

18.5.4 Landscape and Visual

The Project is in an area that is important for tourism and recreation. As well as minimising the project footprint and protecting the water quality of the surrounding water bodies which are important fishing and recreational resources, the project will need to be sensitively designed to blend with the landscape. This will require consideration to be given to the location, height, shape and colour of key infrastructure and waste facilities. One option is to make use of waste rock to develop visual and noise berms to shield key infrastructure. However, planting of vegetation screens coupled with progressive rehabilitation of the WRSF may prove less intrusive and equally effective depending on the topography.

18.5.5 Biodiversity and Habitats

The importance of different habitats is described in Section 18.1. Lake Vättern has three overlapping protection designations, two of which (under the EU Birds Directive and EU Habitats Directive) are established at a European scale and designated by the national government. In addition, there are several water protection areas west of the E4 highway and nationally designated nature reserves along the coast of the lake. These include the Holkaberg[5] and Narbäck[6] Natura 2000 sites that are designated protected areas under the EU Habitat Directive as 'Sites of Community Importance' ("SCI"). The sites are down-stream of the Project site and on the western side of the E4 highway (the Project is on the eastern side). These sites are protected due to the quality of the deciduous forest that provides "*great number of species a unique and prosperous place to live in*" according to the EU description. The two areas combined form the Holkaberg and Narbäck Nature Reserve (Swedish: *naturreservat*) that has protected status under the International Union for the Conservation of Nature ("IUCN") category IV (Habitat/Species Management Area)[7]. There are also areas within the project footprint that were identified as having ecological value to the Jönköping Municipality in 2012.

[5]Natura 2000 Holkaberg: https://natura2000.eea.europa.eu/?query=Natura2000Sites_9883_0,SITECODE,SE0230331
[6]Natura 2000 Narback: https://natura2000.eea.europa.eu/?query=Natura2000Sites_9883_0,SITECODE,SE0230184
[7]IUCN protected area: https://www.protectedplanet.net/182786

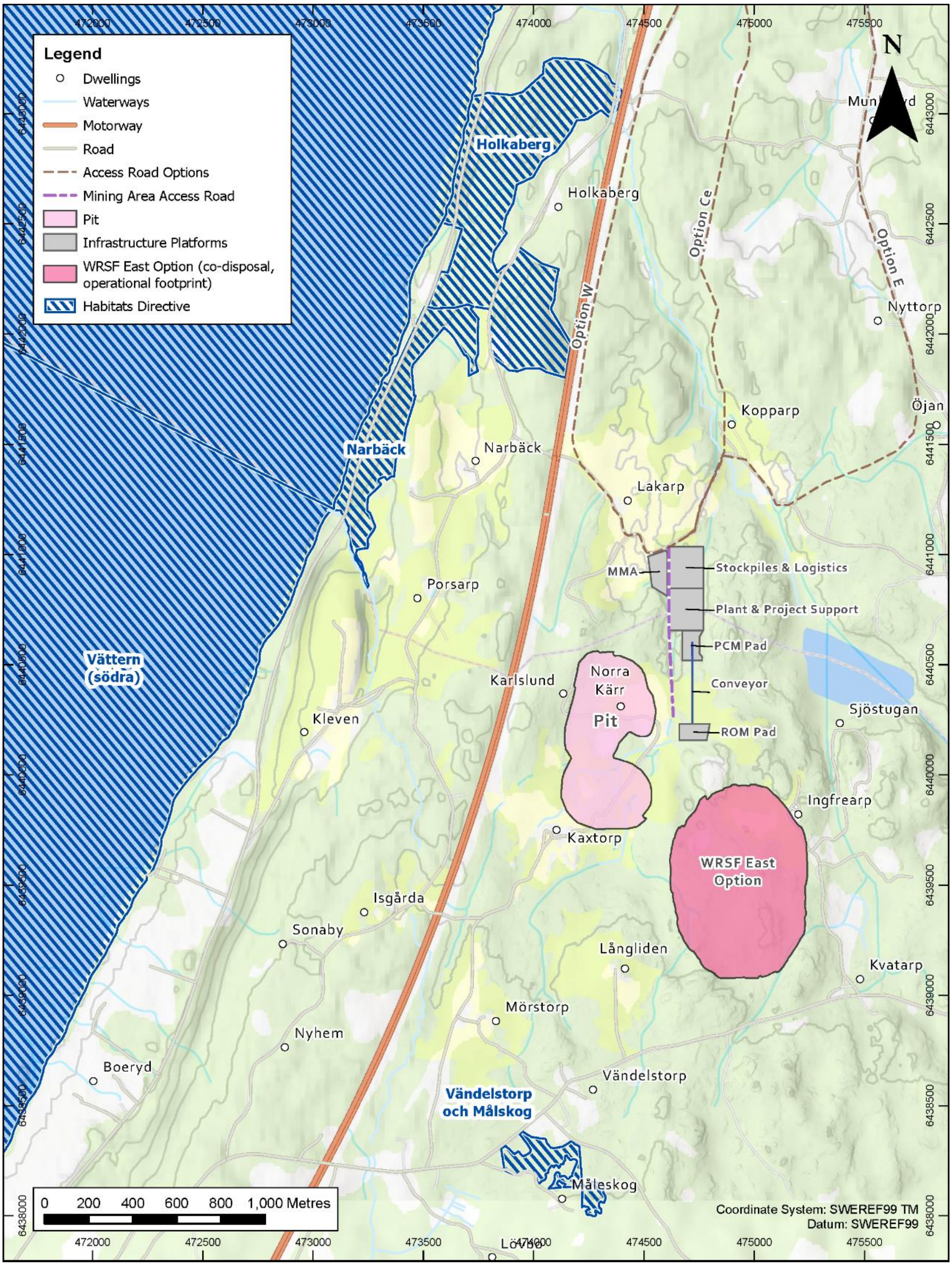


Figure 18-4: Site location in relation to adjacent Natura 2000 designated protected areas

Various ecological baseline studies have been conducted as part of the earlier project work but these will require updating. It is likely the scope of the studies will require review and updating in light of the increased scrutiny on biodiversity management, the Man and Biosphere Reserve boundary areas and Natura 2000 areas.

18.5.6 Current land use and resettlement

Tourism has a significant economic importance in Gränna particularly during the summer months. The surrounding area is used for outdoor recreational activities such as hiking and fishing. The area is also used as farm and pastureland, grazing meadows and for commercial forestry.

It is understood that several dwellings and summer homes belonging to several landowners will be affected in the project area. The new mine site layout aims to reduce this Project footprint and therefore the number of houses directly impacted should reduce. The number of project affected persons ("PAP"s) that may need to be considered for physical or socioeconomic resettlement will be determined as part of the next stage of the Project.

The time to negotiate the lease or purchase of the required land will need to be factored into the overall project schedule.

18.6 Closure requirements and costs

The project has a conceptual mine closure plan prepared for LEM Metals Ltd by Golder Associates in 2014. However, the project design has changed to the extent that this closure plan has limited value.

SRK has not carried out a closure planning exercise as part of this PEA. An updated closure plan including the additional infrastructure areas (rail and Luleå processing sites) will be required. The plan will need to identify closure objectives, closure risks and closure actions on which an update closure cost estimate will be based.

The cost provided in the 2014 report provides closure costs broken down into a life of mine cost ("LOM") and a bond cost, referred to in other jurisdictions as financial assurance ("FA"). The 2014 LOM closure cost is estimated as SEK 101 million (USD 12.1 million). Golder also calculated an initial bond for the project of SEK 38.5 million (USD 4.6 million) and a total annual bond of SEK 63 million (USD 7.5 million) to be accrued over the 20-year life of mine.

SRK considers USD 12.1 million to be low for a project of this size and type. Closure activities that would influence the overall costs include reprofiling benches of waste rock dumps, adding topsoil, planting appropriate vegetation, management of waste that cannot be deposited in a mine waste facility including contaminated waste, ripping and burying pads, demolition of buildings, long-term water management processes and retrenchment and socio-economic transitioning.

Based on an initial PEA schedule it is anticipated that closure will more likely cost up to USD 25 million for the mine site and USD 10 million for the process facility. However, given the project is at the PEA stage, the development of a more detailed closure cost is not currently practical.

19 MARKET STUDIES AND CONTRACTS

This chapter has been compiled by the Company and reviewed by the QP to support the commodity price assumptions used in this technical report. The Company has relied on its knowledge of the rare earth markets combined with information from the report “Rare Earth Magnet Market Outlook to 2030” published in 2020 and updated in 2021, by Adamas Intelligence (Adamas). In addition, the Company has relied on the following sources for the other relevant markets for the by-product revenue streams:

- Norra Kärr Nepheline Syenite – General Market Summary report, IMMC 2021
- Summary of the potential for a new source of Zr chemicals from Sweden, MinChem Ltd. 2021
- Niobium Industry Annual Report 2020 and historical price series, Asian Metal Ltd. 2021

19.1 Rare Earth Elements and Products

19.1.1 Introduction

The 15 chemical elements of the lanthanoid series, together with yttrium and scandium, are commonly referred to as rare earth elements (“REE”). Yttrium is added to the group due to similar chemical properties as the lanthanoids, whereas scandium is included due to often occurring in REE-bearing mineralizations, the latter not being the case for Norra Kärr.

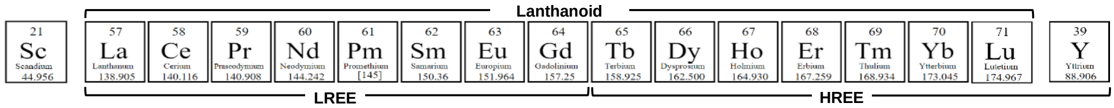


Figure 19-1: Rare Earth Elements of the Periodic Table

Rare earth elements are fairly abundant in the Earth’s crust, however, due to their geochemical properties they are typically dispersed and as such what is ‘rare’ is to find them concentrated enough in a deposit where they become potentially economically viable to exploit.

The first discovery of a rare earth mineral is credited to Swedish army lieutenant Carl Axel Arrhenius who in 1787 who found an unusual black mineral (gadolinite) in a small quarry near the village of Ytterby on an island 20 kilometres north-east of Stockholm, Sweden. Four of the elements are now named after Ytterby; yttrium, terbium, erbium and ytterbium. A number of the other rare earth elements were also discovered in Sweden. Due to the chemical similarities between the individual rare earth elements, scientists have spent significant effort since the first discovery and are still continuing today to research efficient methods to separate and purify them.

The value chains for rare earth elements can be simplified as shown in Figure 19-2 with their respective products that feed end-markets.

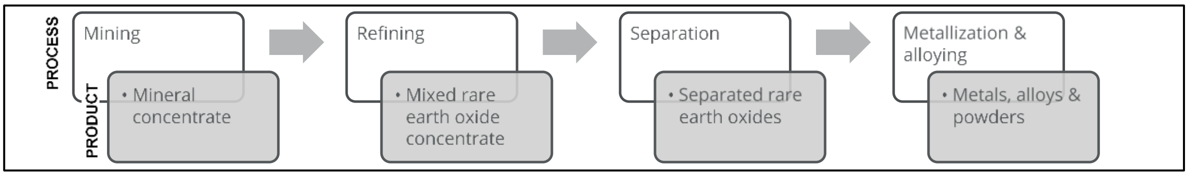


Figure 19-2: Value Chain for Rare Earth Elements

The processing methods proposed in this report produces a mixed rare earth oxide concentrate. However, for the economic analysis the toll separation charge of USD19/kg rare earth oxide ("REO") from the 2015 PFS has been assumed to account for revenue to sell separated rare earth oxides. In further studies, a trade-off study should be conducted to evaluate whether to sell a mixed rare earth oxide concentrate, use a toll separator or to develop internal separation capacity.

Due to their unique chemical properties rare earth elements have found their way into many modern technologies used today. The application of rare earth elements can be grouped into eight end-use categories:

- Battery alloys
- Catalysts
- Ceramics, pigments and glazes
- Glass polishing powders and additives
- Metallurgy and alloys
- Permanent magnets
- Phosphors
- Other

In 2019, demand for permanent magnets represented 38% of REO volumes, but in terms of value they represent 91% according to Adamas Intelligence. The magnet rare earth oxides (neodymium, praseodymium, dysprosium and terbium) will hence be the main economic driver behind most rare earth projects which is also the case for Norra Kärr where these four elements represent some 85% of the rare earth gross revenue. Thus, marketing studies for this report have been focussed on the magnet rare earth oxide ("magREO") products.

The main permanent magnet type of relevance is neodymium-iron-boron ("NdFeB") magnets. This type of permanent magnet alloy has been commercialised since the 1980s and is the strongest available today. The advantage of magnetic strength vs volume makes these magnets the preferred choice in many growth technologies such as electric motors for electromobility and generators for wind turbines.

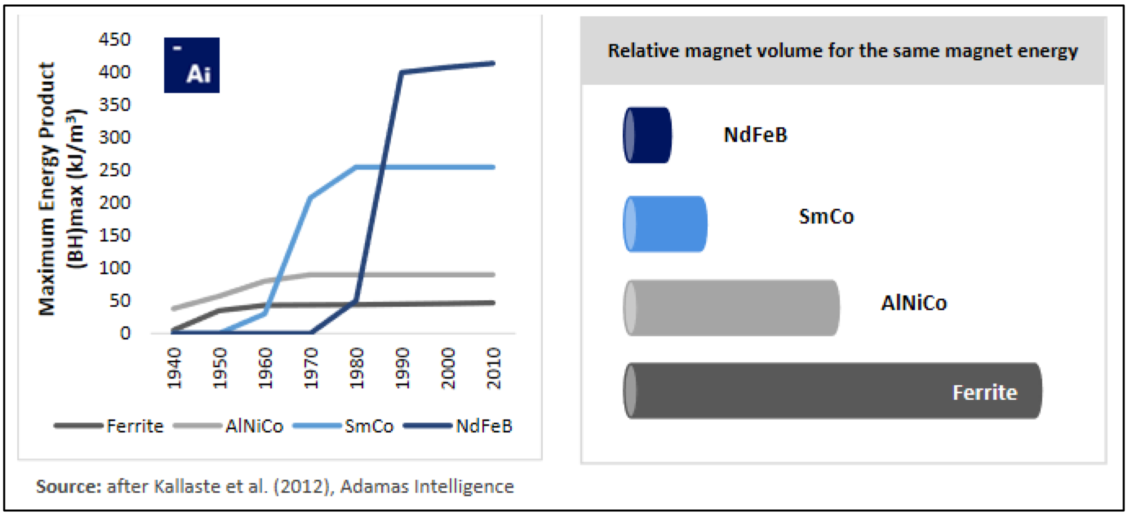


Figure 19-3: Maximum Energy Products Between 1940-2010 (left), Relative Magnetic Volume for the Same Magnet Energy (right) (Adamas Intelligence, 2021)

19.1.2 MagREO Supply and Forecasts

World combined mine and secondary magREO supply is estimated to grow from 65,900 tonnes in 2020 to 130,949 tonnes by 2030 at a CAGR of 7.1%. China is the dominant source of supply although Australia and the United States have emerged as significant suppliers in recent years due to the success of Lynas Corporation and MP Materials. Myanmar is a significant supplier when it comes to HREE mine supply with all of that supply going to China for further processing. It is estimated that China relies on Myanmar for more than half of its HREO feedstock. In 2020 there was no magREO mine supply in Europe. In addition to mine supply there is secondary supply of magREO of which 95% is estimated to be from recycling of magnet production waste with the balance coming from end-of-life recycling.

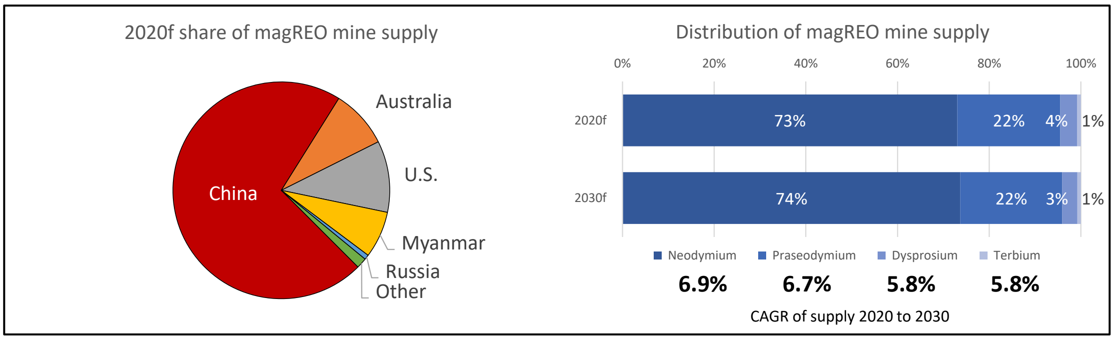


Figure 19-4: 2020f Share of MagREO Mine Supply (left), Distribution of MagREO Mine Supply (right) (Adamas Intelligence, 2020)

In addition to mine supply there is secondary supply of magREO of which 95% is estimated to be from recycling of magnet production waste with the balance coming from end-of-life recycling.

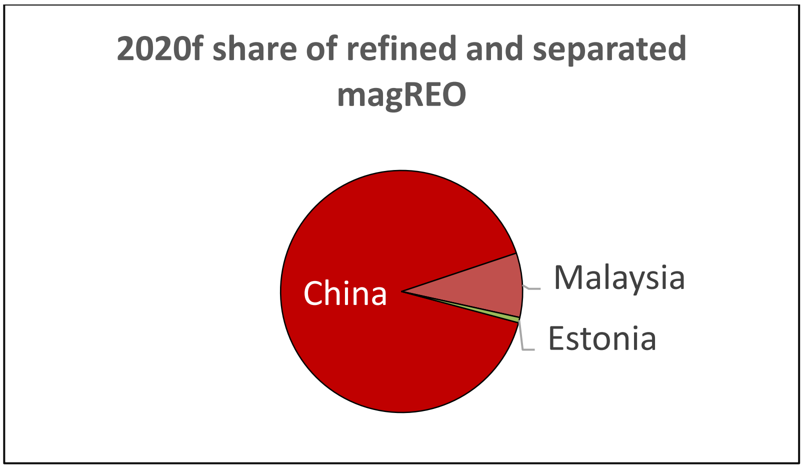


Figure 19-5: 2020f Share of Refined and Separated MagREO (Adamas Intelligence, 2020)

Refining and separation of magREO concentrates does not necessarily occur in the same country as it is mined. When it comes to production of separate magREOs, China's dominance becomes even further pronounced. MP Materials in the United States are exporting their mineral concentrate to China which only leaves Lynas and its processing plant in Malaysia and Neo Performance Materials and its SIlmet operations in Estonia as the only significant downstream processors of magREOs outside of China.

New LREO and HREO mines are estimated to contribute 11% and 3% of total magREO mine supply by 2030.

19.1.3 MagREO demand and forecasts

World magREO demand in 2020 is estimated at 59,195 tonnes and expected to grow to 148,847 tonnes by 2030 at a total CAGR of 9.7%. China is the main destination for magREO due to China's dominance of downstream processes from metal, alloys and powders to NdFeB magnet production.

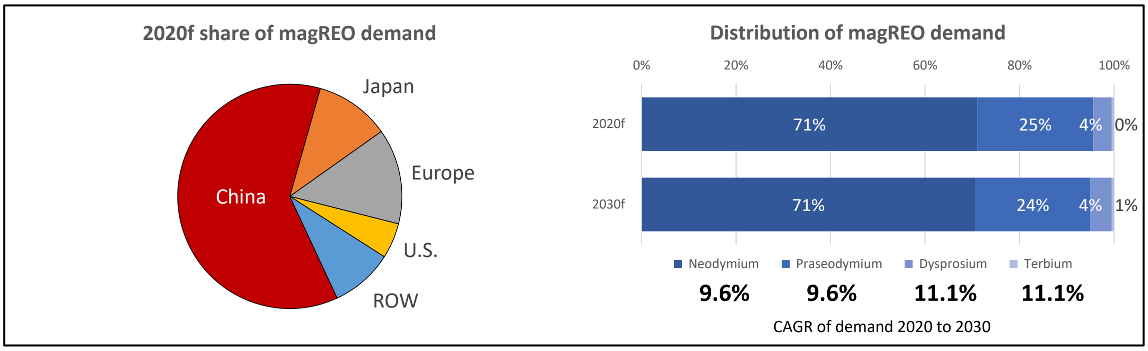


Figure 19-6: 2020f Share of MagREO Mine Demand (left), Distribution of MagREO Mine Demand (right) (Adamas Intelligence, 2020)

The higher growth rates expected for the HREOs until 2030 is due to the expected strong demand growth for higher-performance NdFeB magnets that contain elevated concentrations of dysprosium and terbium.

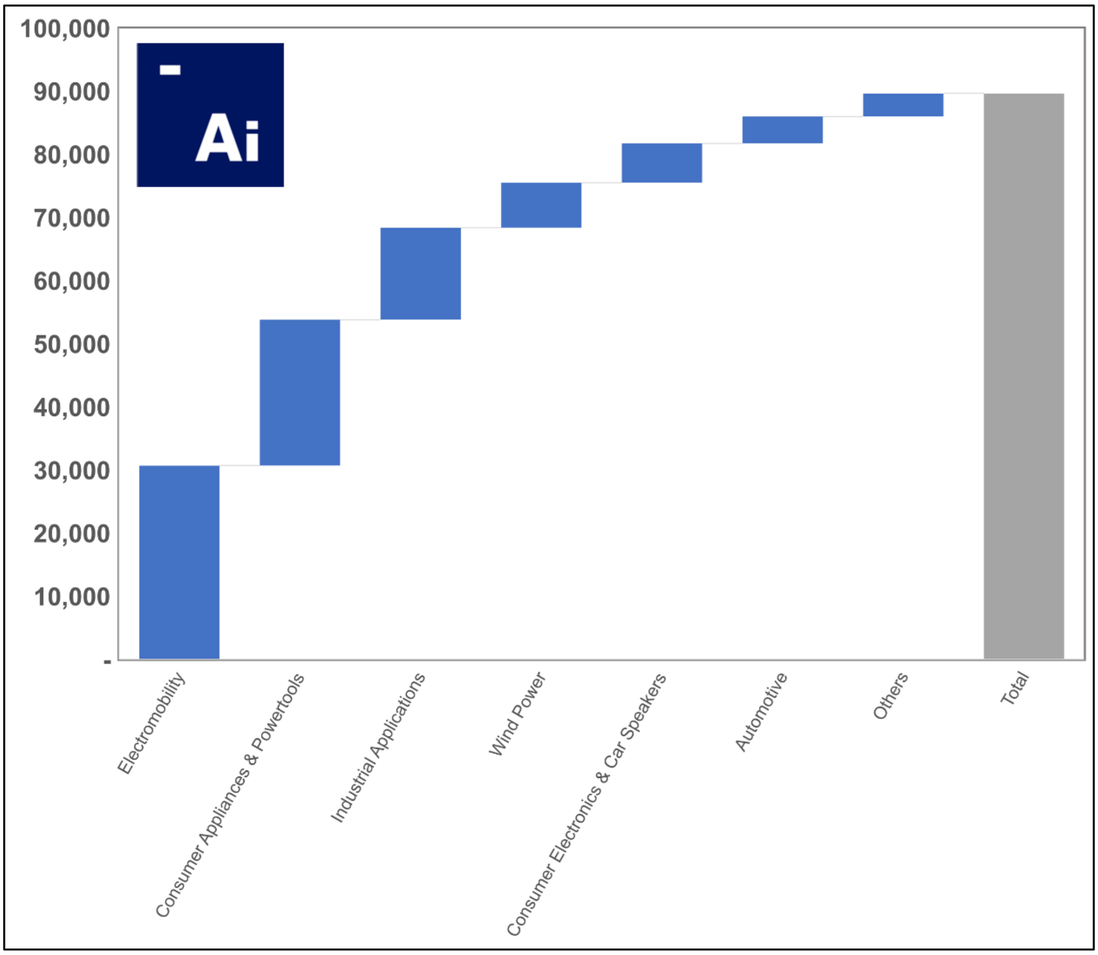


Figure 19-7: Increase in Forecasted Demand for Respective End-Use Categories Between 2020 and 2030 (Adamas Intelligence, 2020)

19.1.4 MagREO Supply and Demand Balance

Combined mine and secondary supply of magREO is expected to grow at a CAGR of 7.1% between 2020 and 2030, while demand is expected to grow at a CAGR of 9.7% over the same period, the market is therefore expected to develop significant supply/demand deficits over the next decade.

For neodymium and praseodymium oxide, production is expected to lag demand from 2022 onward and shortages due to depleted inventories emerging from 2023 onward without additional supply being added.

For dysprosium oxide, production is expected to lag demand from 2021 onward and shortages due to depleted inventories emerging from 2023 onward without additional supply being added.

For terbium oxide, production is expected to lag demand from 2021 onward and shortages ongoing and increasing due to already depleted inventories without additional supply being added.

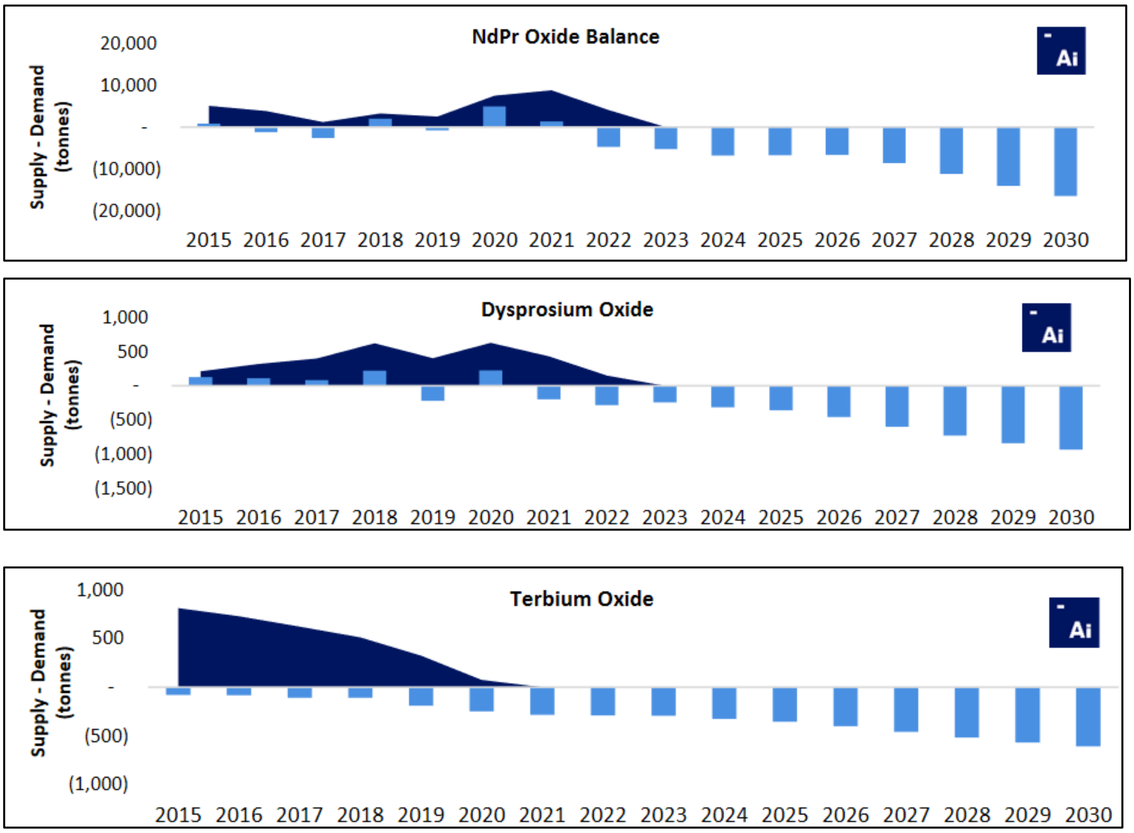


Figure 19-8: NdPr, Dysprosium and Terbium Oxide Supply and Demand Balance from 2015 and Onward to 2030 (Adamas Intelligence, 2021)

19.1.5 MagREO Pricing and Forecasts

Rare earth pricing is more complex compared to other commodities and is opaque. Historically, there has been no open market for rare earths (which are primarily 'traded' individually – all 17 of them) and most of the sales up to this point have been in the form of long term supply agreements, which are subject to a number of variables including volume discounts, cost structure of the buyer, potential cost subsidies and effect of by-products at the producing mine. Furthermore, there is a range of rare earth products sold, which has an impact on price. In this report the focus is on REOs (typically 99.9 %), which is the common product reported.

Since mid-2020 many of the REOs have risen significantly in price, especially the magREOs. This has mainly been driven by increased demand for NdFeB magnets for the EV industry combined with supply chain disruptions like the ongoing COVID pandemic and lately the military coup in Myanmar. Near-term volatility, due to event risks combined with estimated significant demand growth until 2030 from widespread EV adoption, makes price forecasting challenging. For this report, the REO price forecast is relying on a forecast provided by Adamas who provided three alternative pricing scenarios;

1. High price scenario: "Ongoing sentiment and speculation driven high prices"
2. Medium price scenario: "Near-term supply increase drags prices down moderately, price drops minimised by ongoing Myanmar-related uncertainty and speculation"
3. Low price scenario: "Near-term supply increase drags prices down significantly, Myanmar-related uncertainty and speculation are diminished"

The “Low price scenario” has been adopted for the purposes of this PEA assessment using forecast prices for each year from 2025 (construction is assumed during 2023 and 2034 with production and revenue generation commencing from 2025) until 2030 for the first 5 years of production and then using the 2030 forecasted price for the remainder of the life of the project.

The average weighted REO prices over the life of the project are shown below in Table 19-1. Further details on the commodity prices used for the PEA assessment are presented in Section 21.3.2 of this report.

Table 19-1: Average Weighted REO Prices Used Over the Life of the Project

REO	USD/kg
Ce	2.25
Dy	486.33
Eu	54.20
Gd	39.66
La	3.19
Lu	800.00
Nd	103.36
Pr	108.38
Sm	2.71
Tb	1215.83
Y	6.75

Figure 19-9 provides and illustration of main magREO products and pricing applied from the Adamas price forecasts. Diamond markers are included for actual Q1 2021 prices for April 2021, the grey channel gives the high- and low-price scenarios and the dotted line the medium-price scenario. The thick black line illustrates the prices used for first 5 years (2025 to 2030) of the PEA assessment for the project. All prices are rebased to April 2021 prices at 100.

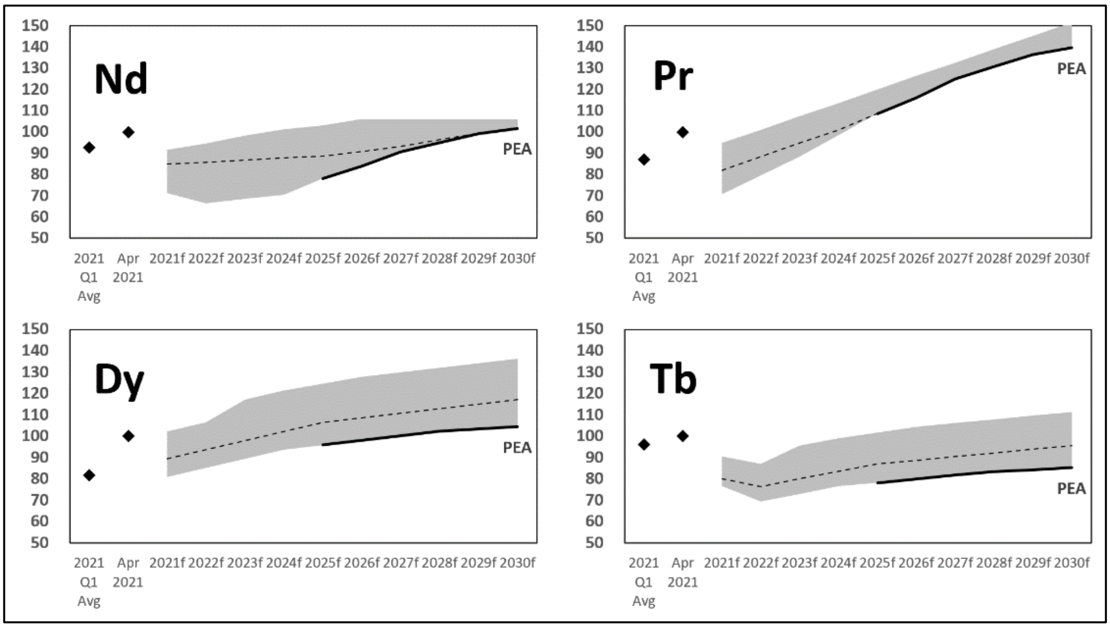


Figure 19-9: MagREO Forecasted, recent Pricing and applied pricing in the PEA rebased to April 2021 prices at 100 (Adamas Intelligence, 2021)

19.1.6 Norra Kärr magREO Products Target Markets

The most natural target market for the Norra Kärr products will be future downstream customers in Europe. For neodymium and praseodymium, the volumes from Norra Kärr are a small fraction of future expected demand in Europe. Dysprosium and terbium from Norra Kärr is expected to be around 30-50% of forecasted European demand by 2030. Even though this is a significant portion of total European demand, there should be no issues with placing this material on the European market considering the lack of other European future supply alternatives for these high-demand REOs.

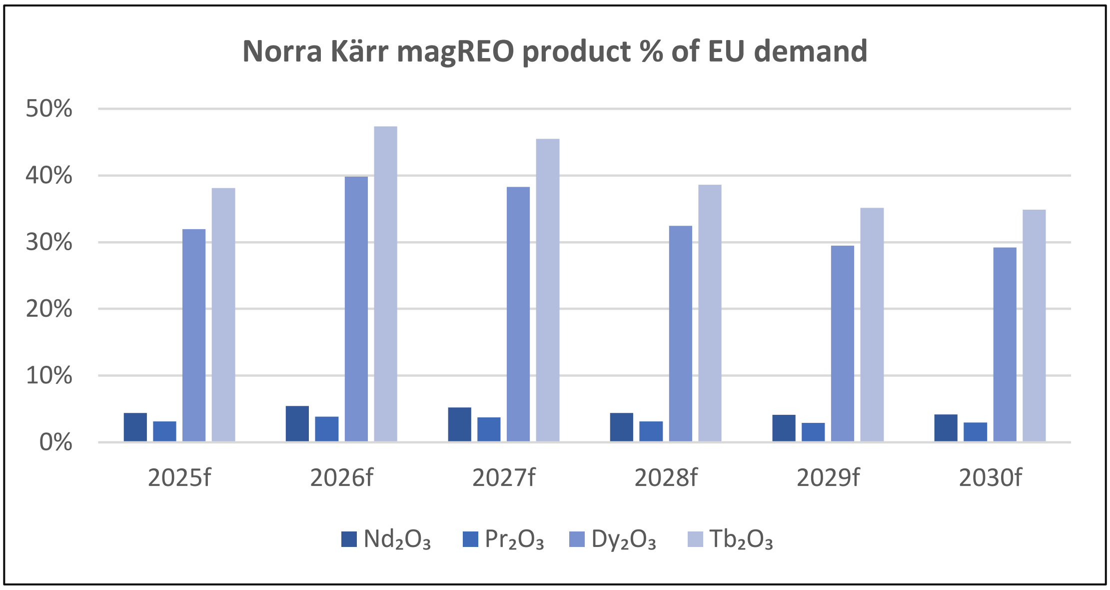


Figure 19-10: Norra Kärr MagREO Product % of EU Demand Between 2025f and 2030f (Company calculations based on Adamas Intelligence, 2021)

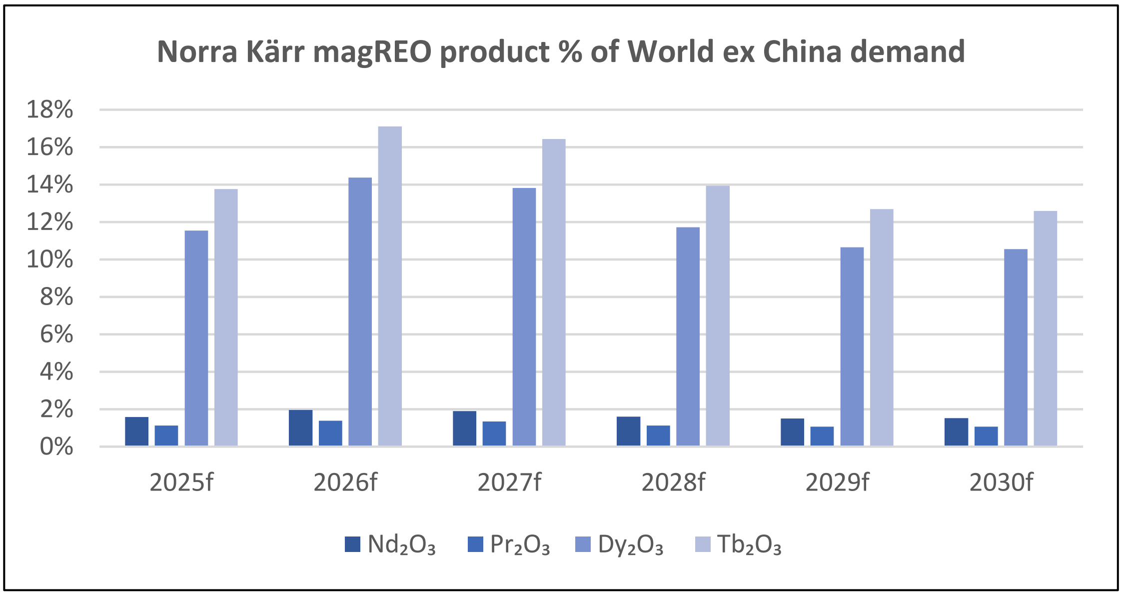


Figure 19-11: Norra Kärr MagREO Product % of World ex China Demand Between 2025f and 2030f (Company calculations based on Adamas Intelligence, 2021)

19.2 Nepheline Syenite Market Overview

Nepheline syenite (NS) is an aluminium silicate consisting of the minerals nepheline, microcline and albite. A quartz-free feldspathoid, nepheline syenite is commonly reported together with feldspars but has a number of advantages in industrial use. The high alumina content combined with a low melting point makes the material attractive for ceramic flux, glass, coatings, paint, functional fillers and cement fillers. Nepheline syenite has also been extensively looked at as an alternative feedstock for production of aluminium instead of from bauxite.

Nepheline syenite products are often incorrectly classified as Feldspar, undermining the greater performance benefits of a higher quality NS with a higher market price than Feldspar. Therefore for the purposes of this PEA it is planned to target the well-established and traditional feldspar market by introducing the compositionally superior and non-toxic NS products as a replacement option for feldspar products.

Due to its unique chemical properties, nepheline syenite has found its way into many modern industrial functions used today. The application of NS can be grouped into six main end-use categories:

- Paints - White pigment filler enhancing brightness
- Fiber glass insulation - Provides strength to the glass and lowers melting point
- Ceramics - Enhancing high gloss and white glaze
- Ambering - Colouring of glass preventing liquids to deteriorate in the presence of light
- Flux - Acts to lower the melting points of SiO2 to make glass containers and flat glass such as windshields for vehicles.
- Cement filler - Increases the density of concrete materials while lowering the amount of cement used reducing the carbon footprint

The current industrial applications for nepheline syenite is made achievable by utilising six main unique characteristics, providing properties that are lacking in traditionally used feldspar.

- Opacity - Bright white colour characteristics and free of quartz impurities
- Strengthening of glass - Provided by the alumina in the NS
- Colouring and strengthening elements - For less pure products due to the Fe-oxide composition
- Refractive properties - Provide matting textures while maintaining its transparency and anti-settling characteristics. The refractive index is identical to most used clear binders, including polyurethanes, epoxies, acrylic, alkyds, and nitrocellulose
- Hardness - 6 on the Mohs hardness scale means a significant abrasion improvement and scratch resistance in comparison to other minerals providing less weathering in paints and improved coatings
- Non-toxic - Quartz free eliminating the need for any respiratory or carcinogenic warning labelling

Currently global high-quality nepheline syenite supply is dominated by two main operations in Canada and one in Norway. Industrial grade nepheline syenite is also produced in the United States. Ownership of production is concentrated to a small number of companies which control supply and the market.

On a global scale, the world market for feldspar was on average 26.3Mt during the period 2012-2016 and by 2018 had grown to 28.4 Mt and worth €2,000m (European Commission, Study on the EU's list of Critical Raw Materials (2020)). Global feldspar use is primarily towards ceramics and glassware, in 2017 was approximately 24.7 Mtpa in 70 countries and 40 companies. The growth for feldspar focusing on ceramics and glassware is already estimated to see 5% compounded annually through to 2027. The pricing of feldspar is relatively low but stable and seems to have flat-lined at approximately USD60 US/tonne over the last 15 years as the traditional markets have not changed (USGS Mineral Yearbook - Feldspar and Nepheline Syenite Statistics, 2017).

An EU focused study by the European Commission, indicates the EU consumption of feldspar was on average 7.5Mt per year during the period 2012-2016 and by 2018 had grown to 10.9 Mt. The EU import reliance during the same period was 34% and by 2018 increased to 53%. Demand in the EU is experiencing constant growth and saw approximately 93% increase between 2010 and 2018. The EU production of feldspar has been rather stable over the last decade fluctuating around 5 Mt per year. Within the European context, there is moderate concern about the toxicity of respirable crystalline silica (quartz) towards workers in mining and manufacturing industries which are strictly regulated by the EU Directive 2017/2398, CLP Regulation 1278/2008 and Regulation (EC) 1272/2008. This provides a unique advantage for NS deposits to be supplied as a quartz-free non-toxic replacement with EU free trade agreements, as the EU currently depends on around 90% imports of NS. The average pricing for feldspar seen in the EU over the last decade ranged from €30-200 per tonne depending on Feldspar type and content. In contrast Nepheline Syenite saw a upward trend ranging from €105-135 per tonne (2013-2016) and stabilized at €120 per tonne (European Commission, Study on the EU's list of Critical Raw Materials (2020)).

Turning attention to the North American markets of feldspar and nepheline syenite. NS production in Canada for 2019 was reported to total 523,000 tonnes with a value of USD118.7m, averaging a NS price of USD 227/tonne (Canada's Mineral Production, Preliminary estimates 2019).

In the US the feldspar production was estimated to be 440,000 tonnes valued at USD 27.8m in 2017 and the price for imports around USD 24 per tonne. The US imports of NS in 2017 was 1.46m tonnes valued at USD 88.4m, an increase of NS to meet their growing demands with average NS prices between USD 220-227/tonne (USGS Mineral Yearbook - Feldspar and Nepheline Syenite Statistics, 2017).

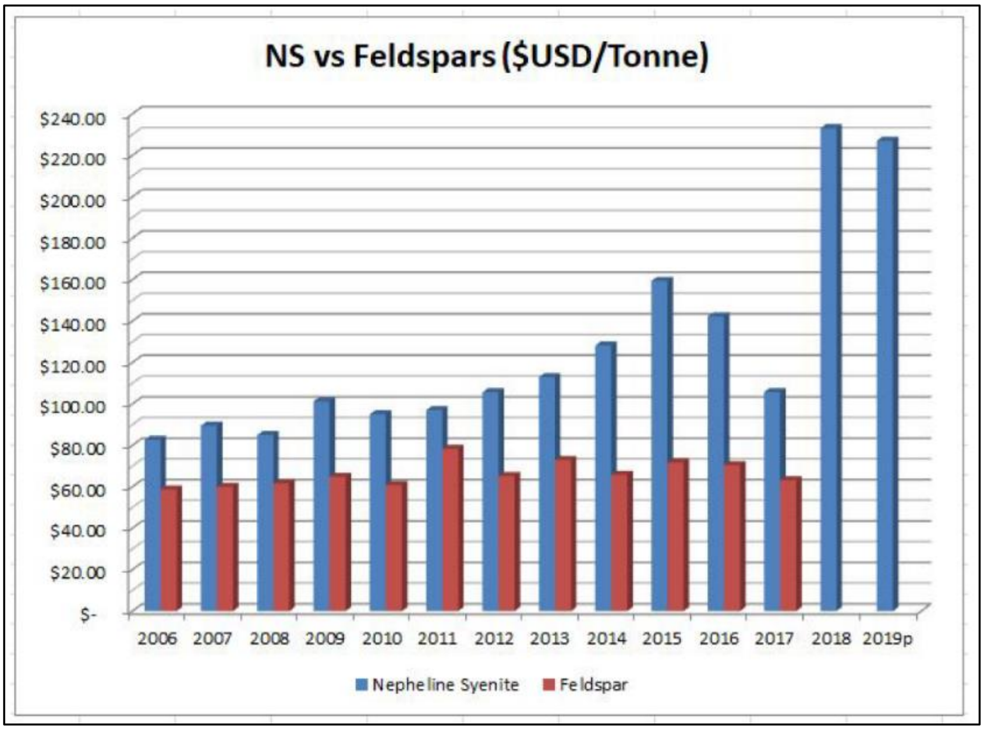


Figure 19-12: Nepheline Syenite and Feldspar historical pricing from 2006-2019, USD/t (Source: IMMC, 2021)

Three different NS products are assumed to be produced from the Norra Kärr project with forecast prices ranging from USD 12-65/tonne assumed for this PEA assessment (IMMC, 2021). Ceramic tile bodies at USD 50/tonne, fibre glass flux at USD 65/tonne and cement fillers at USD 12/tonne. These 3 potential markets have been chosen since they require the least amount of additional processing and product qualification. The fibre glass flux EU market alone was over USD 720m in 2019 and is forecasted to be USD 913m by 2027. Pricing is conservative considering the much higher prices displayed in trade statistics. In the future the Company should target further processing of the nepheline syenite to address higher value markets such as the functional filler products where prices between USD 100-500/tonne are demonstrated ((IMMC, 2021).

For this PEA assessment the nepheline syenite products chosen does contribute a by-product revenue, however, the main advantage is that a significant volume of the material mined can be potentially sold as products and by doing so not requiring significant waste storage facilities reducing the required land and water use for the project.

19.3 Zirconium Market Overview

Zirconium (Zr) is a metallic element which in its pure form as zirconium metal, or in its various compound forms like zircon and zirconia, has a number of physical, mechanical and nuclear properties making it attractive for a large number of industrial, commercial and scientific applications. Zirconium is not naturally occurring and is mainly found in its silicate form in the mineral zircon (ZrSiO4). Zirconia (ZrO2) is the oxide form and can be found in the mineral baddeleyite, then called natural zirconia. However, zirconia is mainly produced synthetically through various production routes. Approximately 97% of zirconium compounds and metal is produced using zircon recovered from heavy-mineral sands deposits as a feedstock. The main starting point for most of the Zr chemicals is the production of ZOC from zircon. (ZIA, 2019).

Zirconia demonstrates a very high hardness, high melting point, chemical stability at high temperatures, high oxide ion-conductivity and abrasion and corrosion resistivity (Imerys, 2021). These characteristics makes the various zirconium compounds, attractive for use in ceramics, chemicals, refractories, foundry and other end-markets, with process route and product quality varying for specific end-product use cases. The ion-conductive properties have resulted in various zirconium ceramic materials being considered promising materials for fuel cells and solid-state batteries.

A non-exhaustive list of zirconium compounds that are currently being produced are;

- ZOC – Zirconium Chlorides
- ZOS/ZBS – Zirconium Sulphates
- ZBC/AZC/KZC – Zirconium Carbonates
- ZAC – Zirconium Acetate
- ZP – Zirconium Phosphate
- ZOH – Zirconium Hydroxide
- Chemical Zirconia
- Fused or thermal zirconia
- Stabilized Zirconia
- Zirconium Metal – with/without hafnium

In 2018 the overall global market for zirconium was 1,524k tonnes worth around 2,000 million Euros (European Commission, Study on the EU's list of Critical Raw Materials (2020)). China has become the world's main supplier of ZOC and other Zr chemicals in some cases representing over 90-95% of world supply (Minchem Ltd., 2021,Table 19-2).

Table 19-2: Chinese zircon produce exports 2020 (Minchem Ltd, 2021)

	Chinese exports 2020 (tonnes)	Price USD/kg ZrO2 contained
ZOC	38,500	7.0
ZBS/ZOS	4,218 combined	7.9/8.8
ZBC	15,000	8.0

Source: Minchem Ltd., 2021

Chemical zirconia (ZrO_2), normally produced through calcination or co-precipitation of ZOC, is more reactive than fused/thermal zirconia since the surface area is higher. In 2016 world production capacity of chemical zirconia stood at around 90,000 tonnes with output of 30,000 tonnes, which has since then increased. In 2020, China exported 20,000 tonnes of chemical zirconia. Pricing for chemical zirconia can vary significantly, with lower grades being priced around USD4/kg while top specification grades of products can achieve up to USD50/kg (Minchem Ltd., 2021).

An EU focused study by the European Commission, indicates the EU consumption of zircon between 2012 and 2016 averaged 230,000 tonnes per year and zirconium metal averaged 3,200 tonnes per year during the same period. Since there are currently no registered production sites for zirconium ore within the EU, the reliance of imports is 100%. The main suppliers feeding 97% of the EU demand, are South Africa (~110kt), Australia (~84kt), Mozambique (~27kt), Senegal (~24kt), Kenya (~8kt), Ukraine (~6kt), Madagascar (~4kt) and the US (~3kt). African sourced zirconium serving the EU market alone makes up 30-40% of the African zirconium output and this share is trending further upward. The zirconium metal imported into the EU making up 88% of the share was sourced from three dominating countries, US (~1300t), China (~1278t) and the UK (~534t) (European Commission, Study on the EU's list of Critical Raw Materials (2020)).

Production facilities have been built with little concern for environmental and waste management. Combined with heavy water and electricity use is an emerging problem with potential production impacts. Additional supply constraints are that most of the low uranium and thorium content feedstocks have been depleted and increasing content of radionuclides in the feedstock is becoming a concern, with an increasing focus on finding low radionuclide bearing feedstocks. Lastly, recent supply chain disruptions due to Covid and significant increases in shipping costs from China is driving global buyers to be interested in alternative suppliers of Zr chemicals outside of China (Minchem Ltd., 2021).

Zirconia from Norra Kärr is envisioned to be recovered through solvent extraction after separation from hafnium. This should result in a high-purity chemical zirconia product that could further be processed into any of the Zr chemical compounds subject to availability of chemical reagents. Further, a number of the Zr chemical compounds are produced through energy intensive processes enabling a competitive advantage for these due to the access of low carbon footprint electricity in Sweden. Due to uncertainty around which products to target at this stage of the PEA assessment a conservative price of USD4/kg for the chemical zirconia from Norra Kärr has been applied in the economic model.

19.4 Niobium Market Overview

Niobium is a relatively hard, paramagnetic, refractory transition metal. It has a very high melting point and is highly resistant to chemical attack and behaves as a superconductor at very low temperature. Niobium is not found as a free metal in nature with the main hosting minerals being columbite and pyrochlore.

The main end-use market representing 90% of demand for niobium is when added as ferro-niobium to High Strength Low Alloy Steels (HSLA).

Table 19-3: Forms of Nb According to Global Application, Markets and Appropriate Grade (MSP-REFRAM, 2017)

Form of Nb	Applications	Principal markets	Nb grade
HSLA FeNb	HSLA steels	Automobiles, gas linepipe, construction, heavy engineering	~0.1%
	Stainsless and heat resistant steels	Automobiles, petrochemical and power plants	0.04-0.08%
Vacuum grade FeNb and NiNb	Superalloys	Aircraft engines, electricity generation, petrochemicals	3-5%
Nb metal and alloys	Superconductors	Particle accelerators, magnetic resonance imaging, various small tonnage uses	45-89%
Nb chemicals	Functionnal ceramics and catalysts	Optical, electronics	21-66%

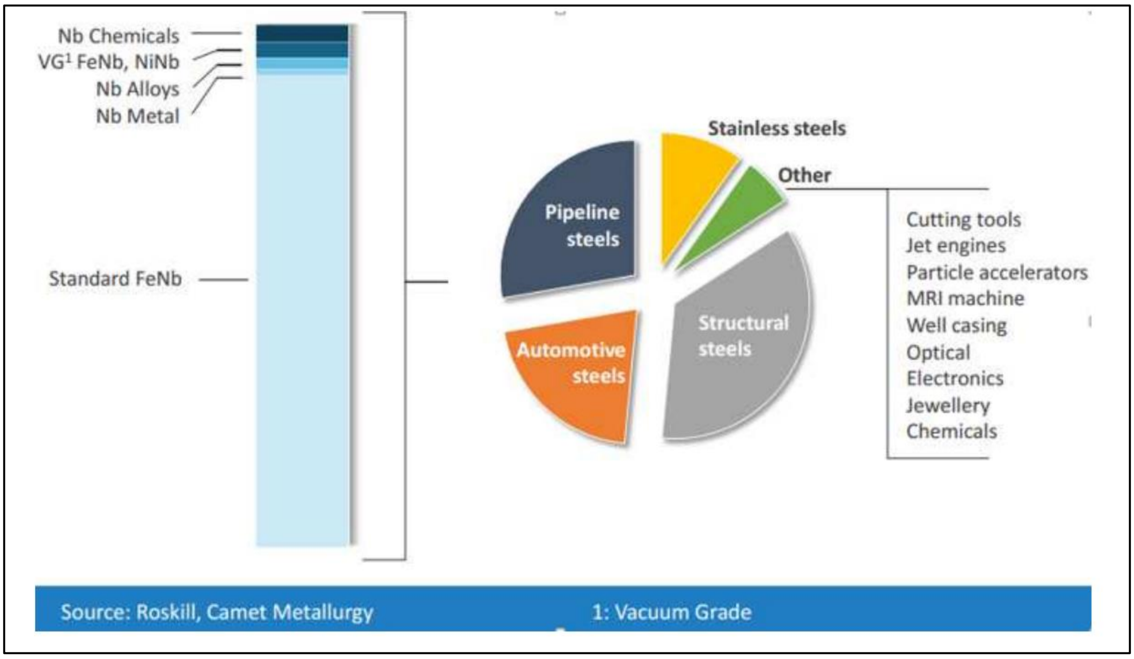


Figure 19-13: Categorised Forms and Global Applications of Nb (MSP-REFRAM, 2017)

Almost all of the world's supply of niobium is produced by three operating mines, with CBMM's Araxa mine in Brazil by itself representing more than 80% of annual sales. The other two mines are CMOC in Brazil which is Chinese owned, and the Niobec mine in Canada. These three mines represent 99% of the market. The production has historically been associated with spare capacity but CBMM in 2019 announced an expansion from 100ktpa of FeNb to 150ktpa by the end of 2020 to meet future demand.

The product proposed to be produced from this Project is niobium pentoxide. Although the historic market for this product has been small, CBMM recently communicated it is expecting to increase sales of niobium oxides from 100tpa to 45,000tpa by 2030. The main driver behind this increase in production is the future use of niobium in high-performance and fast charging electric vehicle batteries.

Niobium is designated as a critical raw material by the European Union with the region being 100% dependent on imports. There is only data available for ferroniobium trade data for Europe. However, with the significant increase in battery production within the EU, and a number of leading niobium battery start-ups located in the region, this is a market that is expected to grow significantly.

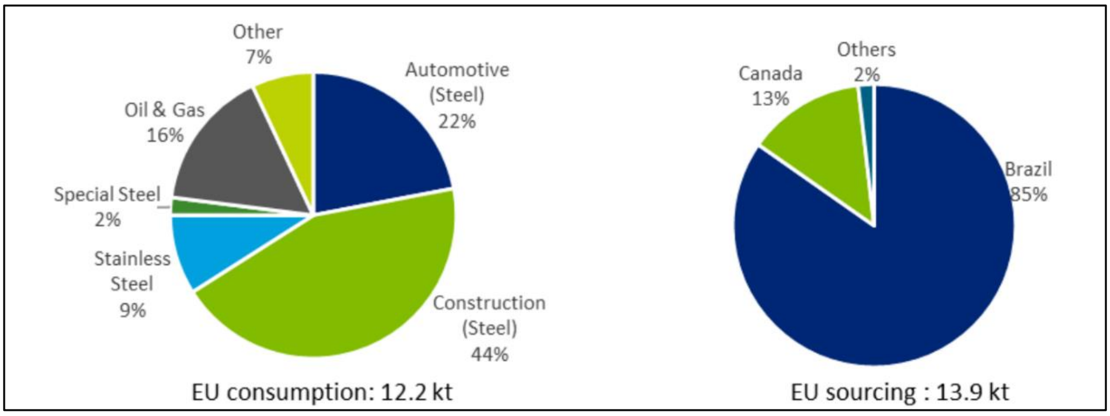


Figure 19-14: EU Consumption End Uses and EU Sourcing of Ferroniobium (Average 2012-16) (European Commission, Study on the EU's list of Critical Raw Materials (2020), Factsheets on Critical Raw Materials)

A 2021 annual report provided by Asian Metal, indicates Chinese niobium oxide production output for 2020 was 3,014t, which is a 41.77% year-on-year increase. This supports the notion for the growing demand from the downstream steel industry and special alloys leaning towards the output in 2021 increasing even further as global economies pick up and overseas consumers remain active in purchasing.

According to Asian Metal, the production capacity of Chinese niobium oxide producers in late 2020 was 5,920t, an increase of 16.31% year-on-year.

Niobium Pentoxide prices for end Q2 2021 provided by Asian Metal, indicate price per Kg of Niobium Pentoxide 99.5%min FOB China approximately USD34-35/kg

For the purposes of this PEA assessment of the Project, a forecast price of USD35/kg niobium pentoxide has been assumed.

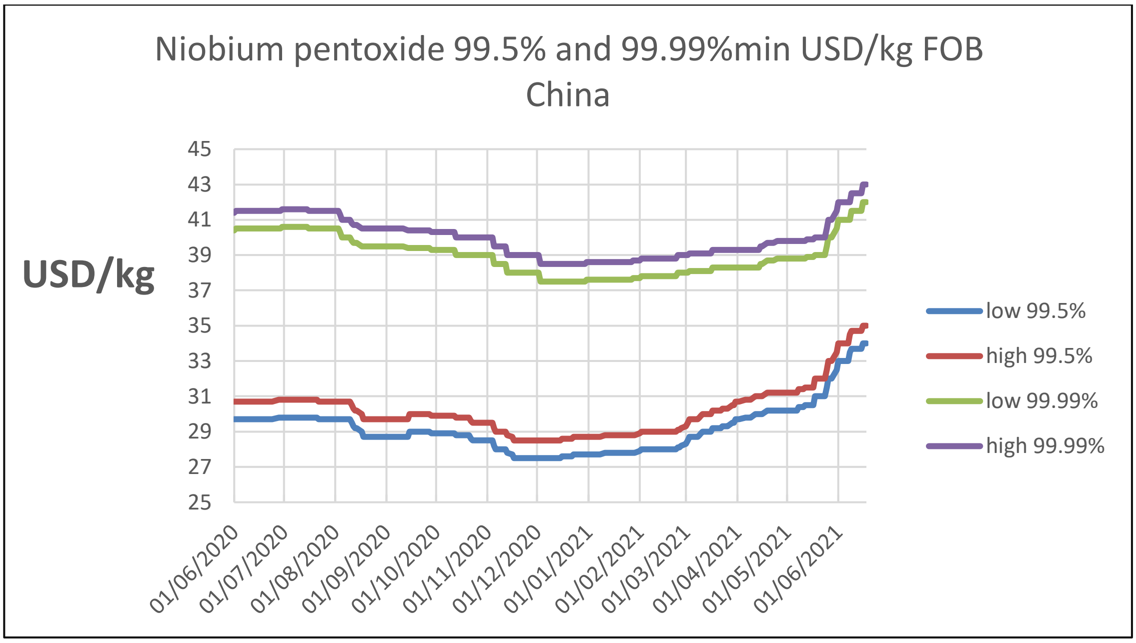


Figure 19-15: Price History for Niobium Pentoxide 99.5% and 99.99%min USD/Kg FOB China 06-2020 to 06-2021, (Asian Metal, 2021)

19.5 Future product development opportunities

Nepheline Syenite as a functional filler:

According to IMMC, if NS products are not used as a replacement of Feldspar or substituting another material, it may be utilised as a standalone product in the form of NS functional fillers.

The physical and chemical properties such us as weatherability, sheen uniformity and silica free NS creates optimal opportunities in the ceramics, paints, coatings, adhesives and sealant markets. The value of potentially directing NS supply towards the latter mentioned markets as a functional filler has the advantage of reaching an average selling price of USD 220-227/tonne.

Although this is part of the market that needs to be better understood in next stage of the Project development.

Nepheline Syenite as a substitute for TiO2:

Due to the similarity of physical and chemical composition between NS and Titanium Oxide (TiO_2), NS is widely known and used as an effective product to extend or replace TiO_2 which sells for USD2,500 per tonne (IMMC, 2021). This allows the replacement of TiO_2 by a significantly lower costing NS product without sacrificing on specialised product integrity. This material substitution does not only create a new market entrance for the potential NS supplier, but also becomes a more cost effective and non-toxic material for paint producers and their end-consumers.

The market share for TiO_2 used in the paint and coatings industry in 2019 was 55% which represents USD 8.75 billion for 3.5 million tonnes (IMMC, 2021).

Aegirine as a pigment filler:

There are historical applications for aegirine which include refractory products, lubricating coatings, fibre glass and mineral wool. However, there is also the potential for aegirine to be applied in the manufacturing of weather-resistant mineral pigment fillers in the construction sector. Aegirine naturally occurs as a dark green mineral and retains the green colour as milled and grinded to a concentrate. During heating aegirine melts at low temperatures of approximately 990 °C resulting in dark green/black glass. Currently magnesium dioxide is the most widely used material for black/darker staining of construction bricks and this is a section of the market the Company would like to further investigate both as an economical addition for the Project and for resource utilization utilising waste rock and reducing the environmental footprint.

20 CAPITAL AND OPERATING COSTS

20.1 Introduction

This section summarises the key capital and operating cost inputs and assumptions for the purposes of the PEA economic assessment of the Project.

20.2 Capital Costs

20.2.1 Introduction

The capital cost estimate is considered overall to have a achieved a Scoping Study / PEA level of accuracy of ±40-50%. Costs are taken from in-house databases and recent budget quotes or benchmarks.

20.2.2 Mining

The mining capital cost estimate has been based on the 2020 Costmine database (InfoMine Inc., 2020). SRK has included an additional 10% to equipment purchase costs to account for freight, insurance, and assembly. SRK has also included a 5% contingency in the mining capital costs. The mining capital and sustaining costs are shown in Table 20-1, excluding water management costs which are included separately. A total of USD12.7M is included in the initial project capital cost with a further USD9.9M included for sustaining capital over the LoM.

Table 20-1: Mining Capital Costs

Mining Capital Cost	Units	Project	Sustaining
Equipment Capital	(USDk)	12,141	2,108
Equipment Replacement	(USDk)	-	7,235
Miscellaneous	(USDk)	-	80
Contingency	(USDk)	607	471
Total	(USDk)	**12,748**	**9,894**

20.2.3 Processing

The costs for the process plant and related infrastructure are summarised in Table 20-2 The costs a presented showing the split between the Norra Kärr and Luleå sites. Costs were taken from factored estimates and from first principles.

The direct capital costs include:

- All labour required for project construction and management activities (excluding EPCM cost).
- All material and equipment required for construction.
- Mechanical, electrical, control, instrumentation, civil works, earthworks and piping installation
- Services;
- Transport and freighting services;
- Insurance and capital spares; and
- Construction and install

Table 20-2: Processing Capital Costs

Processing Capital Cost	Units	Project	Norra Kärr	Luleå
Earthworks	(USDk)	904	226	678
Civil	(USDk)	58,790	14,698	44,093
Structural	(USDk)	35,819	8,955	26,864
Acid plant/NF	(USDk)	-	-	-
Mechanical	(USDk)	134,178	33,544	100,633
Mobile	(USDk)	955	239	717
Electrical	(USDk)	10,510	2,627	7,882
Control	(USDk)	4,777	1,194	3,583
Piping	(USDk)	6,688	1,672	5,016
Platework	(USDk)	8,599	2,150	6,449
Total	(USDk)	**261,220**	**65,305**	**195,915**

20.2.4 Water Supply

Capital costs for water supply have been included as summarised below in Table 20-3. Water supply is assumed to come from a combination of pit dewatering and lake sources transferred to the site via a water pipeline.

Table 20-3: Water Supply Capital Costs

Water Supply Capital Cost		
Function	**Units**	**Project**
Pit Dewatering	(USDk)	277
	(USDk)	83
	(USDk)	37
	(USDk)	120
Water Pipeline	(USDk)	74
	(USDk)	130
Groundwater Monitoring	(USDk)	249
	(USDk)	37
Total	(USDk)	**1,007**

20.2.5 Waste Management

The life of mine capital cost for the aegirine residue, leach waste and gypsum waste storage facilities has been estimated at USD 8.2M which are summarised below in Table 20-4. SRK has included for costs associated with future design and permitting within this amount. Further details on the waste management elements of the Project are discussed in Section 23.2.

The unit capital cost is based on similar sites ('dry' stacked, fully lined storage facilities) in Sweden and the Scandinavian region. It is assumed to include rockfill cell wall construction, ground preparation and lining, monitoring instruments and water management requirements.

Table 20-4: Waste Management Capital Costs

Waste Management Capital Cost	Units	Project
Aegirine Waste (Norra Karr Site)	(USDk)	3,607
Neutralised Leach Waste (Lulea Site)	(USDk)	2,248
Gypsum Waste (Lulea Site)	(USDk)	2,313
Total	(USDk)	**8,168**

20.2.6 Infrastructure

Table 20-5 and Table 20-6 present the site Infrastructure & Utilities and Transport/Handling capital costs respectively. The cost estimate is considered overall to have a achieved a Scoping Study / PEA level of accuracy of ±40-50% and to fall within the range of an AACE International “Class 5” estimate. Further details on the infrastructure elements of the Project, including basis of estimate, are discussed in Section 17.6.

Table 20-5: Infrastructure and Utilities Capital Costs

Site Infrastructure & Utilities Capital Cost	Units	Project
Norra Kärr Site		
Project Facilities	(USDk)	2,976
Roads (Main and Site)	(USDk)	1,680
Warehouse & Logistics (All)	(USDk)	3,192
Mine Maintenance Area	(USDk)	5,532
Site Wide Utilities / Services	(USDk)	1,800
Support Mobile Equipment	(USDk)	780
Bulk Power Supply	(USDk)	3,960
Sub-total	(USDk)	**19,920**
Luleå Processing Facility		
Project Facilities	(USDk)	3,072
Bulk Power Supply	(USDk)	3,000
Rail Infrastructure	(USDk)	8,040
Fixed M&E Equipment	(USDk)	900
Warehousing / Tanks	(USDk)	4,080
Site Wide Earthworks & Civils	(USDk)	2,424
Site Wide Utilities / Services	(USDk)	1,680
Support Mobile Equipment	(USDk)	864
Sub-total	(USDk)	**24,060**
Total	(USDk)	**43,980**

Table 20-6: Transport/Handling Capital Costs

Transport/Handling Capital Cost	Units	Project
Buildings and Installations	(USDk)	2,928
Utilities / Services	(USDk)	672
Rail Infrastructure	(USDk)	3,864
Fixed M&E Equipment	(USDk)	480
Mobile Equipment	(USDk)	408
Total	(USDk)	**8,352**

20.2.7 Owners/General

An allowance of USD15M is included to cover Owners/General capital costs which is some 5% of the overall direct capital costs for the Project.

20.2.8 Other

Other capital costs included in the estimate are:

- EPCM costs based on 9% of the total direct capital cost;
- Indirects based on 10% of the total direct capital cost; and
- Contingency based on 20% of the total direct capital cost.

20.2.9 Total Project Capital

Table 20-7 presents a summary of the total Project Capital costs which equate to some USD487M.

Table 20-7: Total Project Capital Costs

Project Capital Cost Summary	Units	Project	Norra Kärr	Luleå
Mining	(USDk)	12,748	12,748	-
Processing	(USDk)	261,220	65,305	195,915
Water Supply	(USDk)	1,007	1,007	-
TSF/Waste Management	(USDk)	8,168	3,607	4,561
Transport/Handling	(USDk)	8,352	8,352	-
Infrastructure/Utilities	(USDk)	43,980	19,920	24,060
Owners/General	(USDk)	15,000	7,500	7,500
Sub-total Direct	(USDk)	**350,475**	**118,439**	**232,036**
EPCM	(USDk)	31,543	10,659	20,883
Indirect	(USDk)	35,047	11,844	23,204
Contingency	(USDk)	70,095	23,688	46,407
Sub-total Indirect	(USDk)	**136,685**	**46,191**	**90,494**
Total	(USDk)	**487,160**	**164,630**	**322,530**

20.2.10 Sustaining and Closure

A general allowance of USD3.4Mpa (1% of total direct capital cost, excluding mining capital) for sustaining capital has been included and assumed to be incurred from Year 3 (2027) of operations and ceasing 2 years before (2048) the end of the LoM (2050) and totalling USD74.3M over the LoM. In addition to this general allowance a further USD9.9M is included for mining sustaining capital giving a total allowance of USD84.2M over the LoM.

Closure costs of USD35M have been included and are assumed to be incurred over a six-year period – 3 years pre-closure and 3 years post-closure (i.e. between 2048 and 2053).

20.3 Operating Costs

20.3.1 Introduction

The operating cost estimate is considered overall to have a achieved a Scoping Study / PEA level of accuracy of ±40-50%. Costs are taken from in-house databases and recent budget quotes or benchmarks.

Common assumptions across operating cost categories include:

- Diesel: USD1.44/litre
- Norra Kärr site power: USD0.070/kWh (refer to Section 17.2.4)
- Luleå site power: USD0.069/kWh (refer to Section 17.3.6)

20.3.2 Mining

The mining operating cost estimate has been based on the following:

- 2020 Costmine database (InfoMine Inc., 2020)
- Orica pricelist (Orica Limited, 2021)

The mine operating costs are shown in Table 20-8 and excludes water management (i.e. pit dewatering costs which are included in Processing) and does not include any contingency.

Table 20-8: Mining Operating Costs

Mining Operating Costs	Units	LoM	Av Annual	USD/t	USD/kg REO
Labour	(USDk)	77,316	2,974	2.00	0.56
Maintenance	(USDk)	7,643	294	0.20	0.06
Fuel	(USDk)	51,591	1,984	1.33	0.37
Lubricants	(USDk)	6,699	258	0.17	0.05
Tires	(USDk)	1,934	74	0.05	0.01
Wear Parts	(USDk)	1,495	57	0.04	0.01
Explosives	(USDk)	17,944	690	0.46	0.13
Miscellaneous	(USDk)	338	13	0.01	0.00
Contingency	(USDk)	-	-	-	-
Total		**164,960**	**6,345**	**4.26**	**1.19**

20.3.3 Processing

The estimate of processing operating costs includes the processing facilities at the Norra Kärr site and the Luleå site and include the water supply costs at both sites and waste management costs at each site (i.e. aegirine waste at the Norra Kärr Site and leach/neutralised waste and gypsum waste at the Luleå site.

The overall processing operating costs are shown in Table 20-9 which are also shown as split between the Norra Kärr and Luleå sites. The most significant contributors to the processing operating costs are reagents (42%) and labour (23%). This is also show graphically in Figure 20-1.

Table 20-9: Processing Operating Costs

Total Processing	Units	LoM	Av Annual	USD/t	USD/kg REO
Power	(USDk)	167,370	6,437	5.71	1.21
Grinding Media & Liners	(USDk)	45,233	1,740	1.54	0.33
Reagents	(USDk)	680,983	26,192	23.23	4.90
Diesel	(USDk)	52,476	2,018	1.79	0.38
Water Treatment	(USDk)	43,725	1,682	1.49	0.31
Waste Management	(USDk)	26,759	1,029	0.91	0.19
Maintenance & Operating Spares	(USDk)	68,408	2,631	2.33	0.49
Labour	(USDk)	299,000	11,500	10.20	2.15
General	(USDk)	117,262	4,510	4.00	0.84
Total	(USDk)	**1,501,216**	**57,739**	**51.21**	**10.81**
Norra Kärr Processing	**Units**	**LoM**	**Av Annual**	**USD/t**	**USD/kg REO**
Power	(USDk)	159,788	6,146	5.45	1.15
Grinding Media & Liners	(USDk)	45,233	1,740	1.54	0.33
Reagents	(USDk)	-	-	-	-
Diesel	(USDk)	26,238	1,009	0.90	0.19
Water Treatment	(USDk)	35,413	1,362	1.21	0.26
Waste Management	(USDk)	16,609	639	0.57	0.12
Maintenance & Operating Spares	(USDk)	34,204	1,316	1.17	0.25
Labour	(USDk)	149,500	5,750	5.10	1.08
General	(USDk)	58,631	2,255	2.00	0.42
Total	(USDk)	**525,617**	**20,216**	**17.93**	**3.79**
Luleå Processing	**Units**	**LoM**	**Av Annual**	**USD/t**	**USD/kg REO**
Power	(USDk)	7,582	292	0.26	0.05
Grinding Media & Liners	(USDk)	-	-	-	-
Reagents	(USDk)	680,983	26,192	23.23	4.90
Diesel	(USDk)	26,238	1,009	0.90	0.19
Water Treatment	(USDk)	8,311	320	0.28	0.06
Waste Management	(USDk)	10,150	390	0.35	0.07
Maintenance & Operating Spares	(USDk)	34,204	1,316	1.17	0.25
Labour	(USDk)	149,500	5,750	5.10	1.08
General	(USDk)	58,631	2,255	2.00	0.42
Total	(USDk)	**975,599**	**37,523**	**33.28**	**7.03**

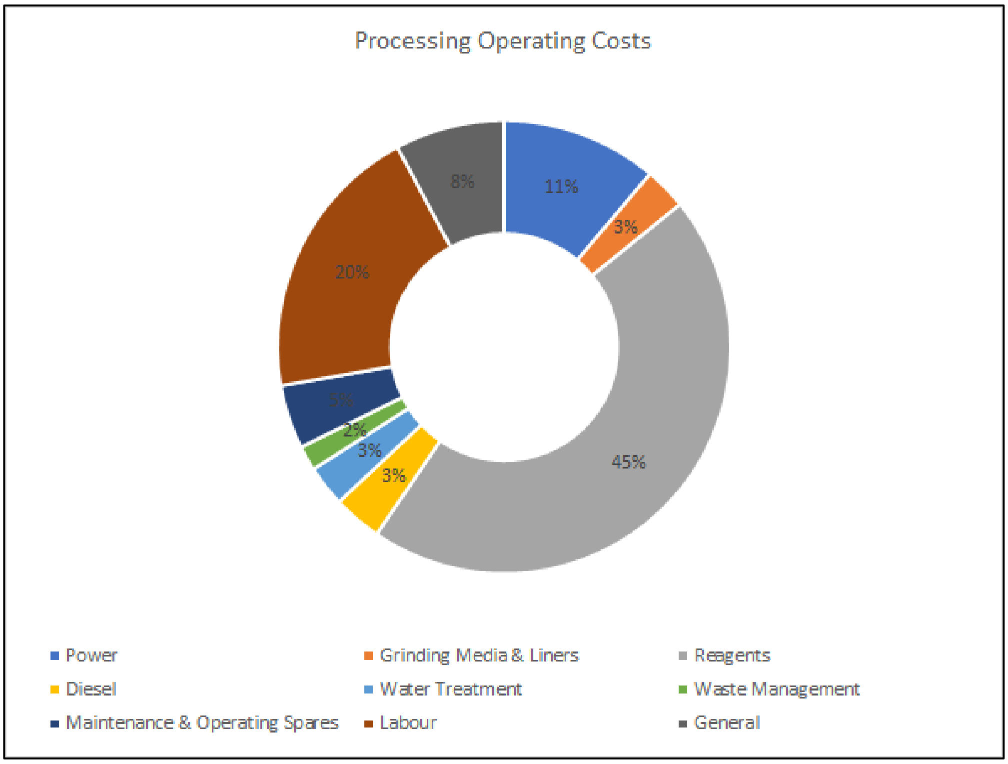


Figure 20-1: Total Processing Operating Costs

Table 20-10 provides a breakdown of the processing reagent operating costs which is also shown graphically in Figure 20-2. Key contributors to the cost include lime, sulfuric acid, organic (CY923) and organic (CY572) which combined contribute some 81% to the total reagent operating costs.

Unit prices assumed for key reagents include:

- Lime: USD 144/t
- Sulfuric acid: USD 66/t
- Organic (CY923): USD 2,470/t
- Organic (CY572): USD 9,200/t

Table 20-10: Processing Reagent Operating Costs

Processing Reagent Costs	Units	LoM	Av Annual	USD/t	USD/kg REO
Water	(USDk)	19,207	739	0.66	0.14
Sulphuric Acid	(USDk)	87,927	3,382	3.00	0.63
Shellsol	(USDk)	36,493	1,404	1.24	0.26
Organic (CY572)	(USDk)	49,824	1,916	1.70	0.36
Oxalic	(USDk)	11,553	444	0.39	0.08
HCl Acid	(USDk)	11,149	429	0.38	0.08
TOPO	(USDk)	42,879	1,649	1.46	0.31
Organic (CY923)	(USDk)	69,350	2,667	2.37	0.50
EDTA	(USDk)	8,467	326	0.29	0.06
Lime	(USDk)	344,133	13,236	11.74	2.48
Total	(USDk)	**680,983**	**26,192**	**23.23**	**4.90**

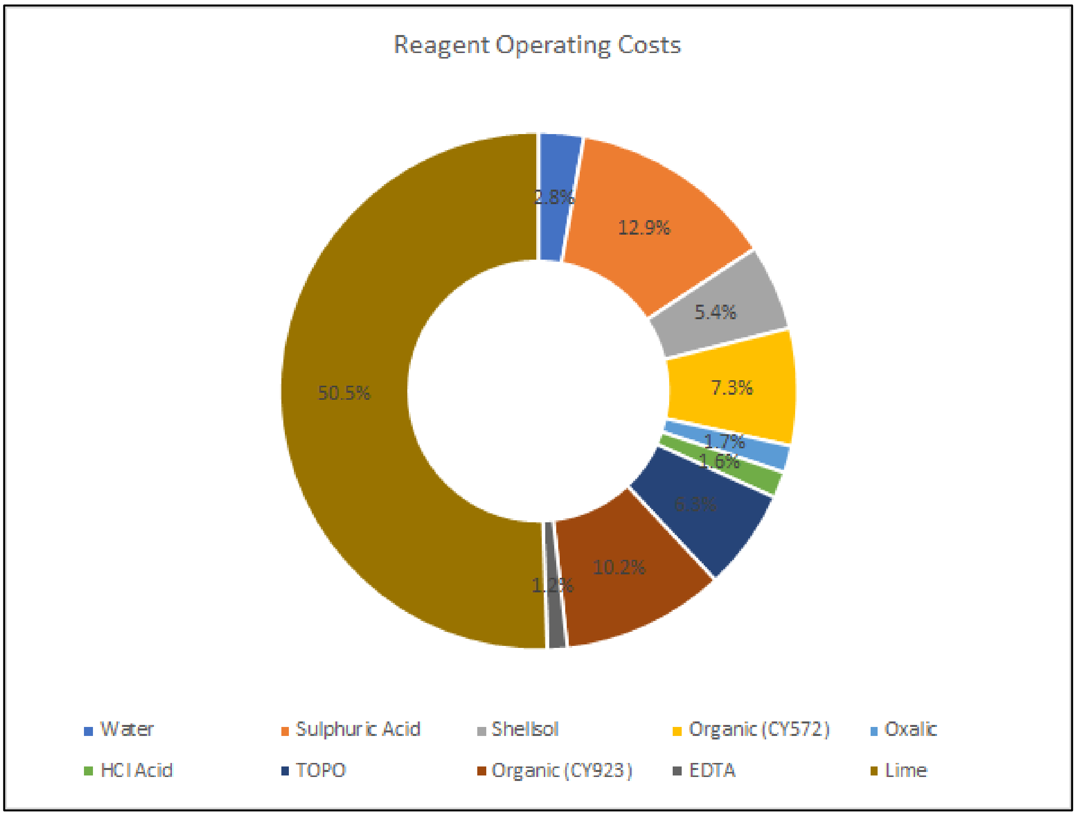


Figure 20-2: Processing Reagent Operating Costs

Other cost assumptions including in the Processing category include:

- Water supply costs (i.e. power costs for pumping) from a combination of pit dewatering and transfer from Lake Vätternvia a pipeline;
- Waste management facilities: USD2.2/t waste material placed;
- Labour costs: USD11.5Mpa;
- Maintenance and operating spares: USD2.7Mpa (based on 2%pa of the plant mechanical equipment capital cost); and
- Processing General costs: USD4/t mineralized rock processed

20.3.4 G&A

An allowance of USD5/t mineralized rock processed has been included to cover Owners G&A costs. This equates to some USD5.75Mpa at steady state.

20.3.5 Transport

- As described in Section 17.6.5 of this report, the following assumptions have been included to cover transport costs:
- USD42/t of concentrate transported from Norra Kärr to the Luleå processing site which equates to some USD4.9Mpa at steady state.
- USD48/t of each of the final products (REO, ZrO_2, Nb_2O_5) transported to the assumed point of sale, this being the port of Rotterdam. At steady state these costs equate to some USD0.75-0.85Mpa.

20.3.6 Toll Treatment

A toll treatment charge of USD19/kg of REO sold is applied as a sales cost and included in the operating costs in the PEA assessment. This is the same figure as applied in the previous PFS study on the Project and SRK considers this assumption remains reasonable for the purposes of this PEA assessment. Updated quotes from potential off takers will need to be obtained for subsequent studies. This is the most significant contributor to the operating costs accounting from some 57% of the total operating costs over the LoM.

21 ECONOMIC ANALYSIS

21.1 Introduction

SRK has constructed an Excel based Technical Economic Model (TEM) to assess the viability of the Norra Kärr project based on assumptions derived by SRK for the technical and economic aspects of the Project. In summary this includes the following:

- A mining/processing schedule based on the MRE.
- Processing mass yields and recoveries to final products.
- Mining operating and capital costs.
- Processing operating and capital costs.
- Waste management capital and operating costs.

- Owners' development and business services costs ('G&A').
- Infrastructure/transport operating and capital costs.
- Closure costs.

SRK has constructed a pre-finance and post-tax TEM on an annual basis and assumed:

- The currency is USD in Q1 2021 real terms.
- Construction starts in 2023 and continues over a 2-year period with processing of mineralized rock commencing in 2025.
- A base case discount rate of 10% to derive a NPV at 1 January 2023 (i.e. the start of construction).
- All numbers presented in this chapter are on a 100% ownership basis.

21.2 Project Financial Summary

- Pre- and post-tax Net Present Value (NPV) of USD1,026M and USD762M respectively using a 10% discount rate;
- Pre- and Post-tax Internal Rate of Return (IRR) of 30.8% and 26.3% respectively;
- Accumulated LoM project gross revenues of USD9,962M;
- Average annual EBITDA of USD206M;
- Initial Capital Expenditures of USD487M;
- Pre-tax Payback Period from first production of 5.1 years;
- Life of mine average gross basket price per kg of separated mixed REO product at USD53/kg;
- Operating cost per kg of separated mixed REO product at USD33/kg including toll separation charges;
- By-product revenue per kg of separated mixed REO product USD19/kg
- Operating cost per kg of separated mixed REO product at USD14.57/kg including toll separation charges and after by-product revenues

21.3 Model Assumptions

21.3.1 Mining/Processing Schedule

Figure 21-1 shows the mining schedule that has been developed by SRK which in summary assumes some 1.15Mt of mineralized rock will be mined each year for a 26-year mine life totalling some 29.3 Mt of mineralized rock. Waste mining varies from zero to some 725 ktpa with a LoM average stripping ratio (waste:mineralized rock) of 0.32. Further details on the mining schedule are discussed in Section 15 of this report.

SRK has incorporated a ramp-up to the mineralized rock feed to the processing plants of some 75% in Year 1 (2025), 95% in Year 2 and capacity of 1.15Mtpa thereafter. Processing parameters used in the TEM are discussed further in Section 16 of this report. Figure 21-2 shows the revenue generating products produced over the LoM. In summary the following revenue generating products are produced on average per annum over the LoM:

- Mixed Rare Earth Oxide (MREO): 5,340tpa
- ZrO_2 Product (fused zirconia): 10,200tpa
- Nb_2O_5 Product: 525tpa
- Nepheline Syenite Product 1: 275ktpa
- Nepheline Syenite Product 2: 403ktpa
- Nepheline Syenite Product 3: 55ktpa

Waste streams generated on average over the LoM are as follows:

- Aegirine waste: 292ktpa (LoM total 7.59Mt) to be stored on the Norra Kärr site
- Neutralised waste: 86ktpa (LoM total 2.25Mt) to be stored on the Luleå site
- Gypsum waste: 92ktpa (LoM total 2.39Mt) to be stored on the Luleå site

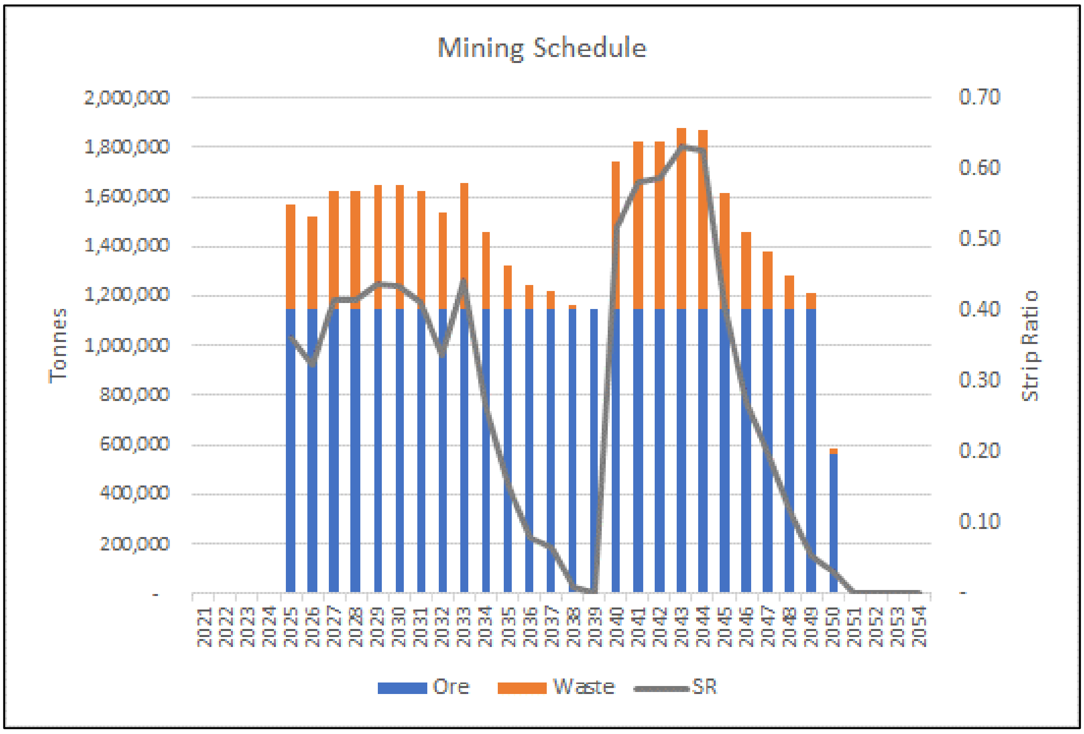
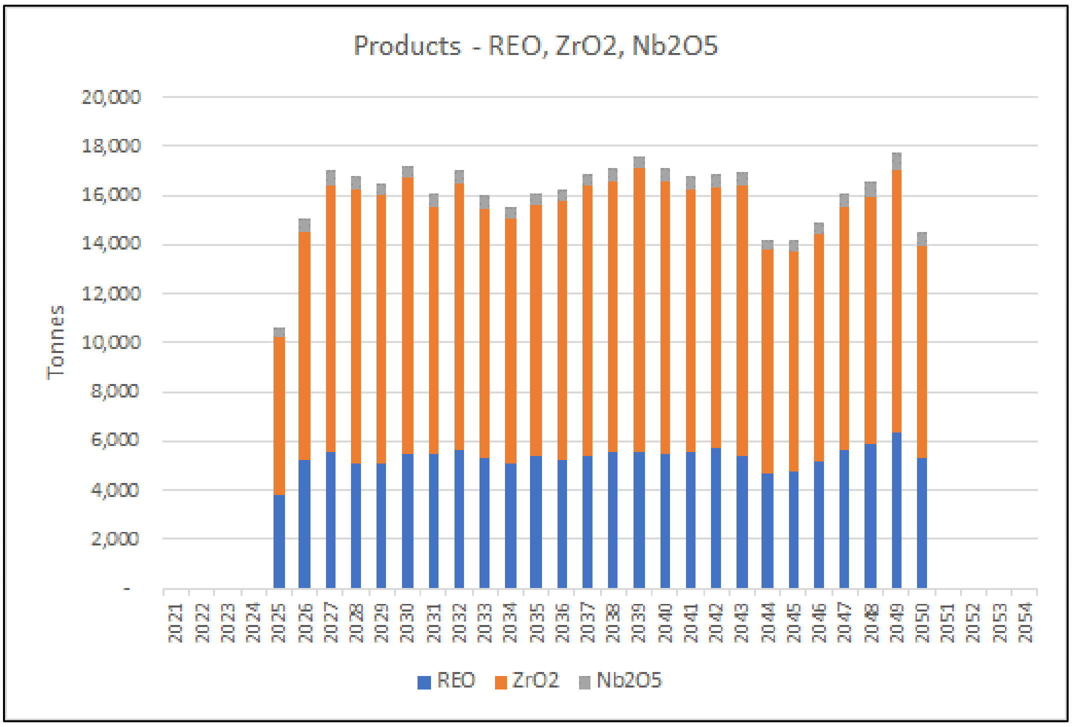


Figure 21-1: Mining Schedule



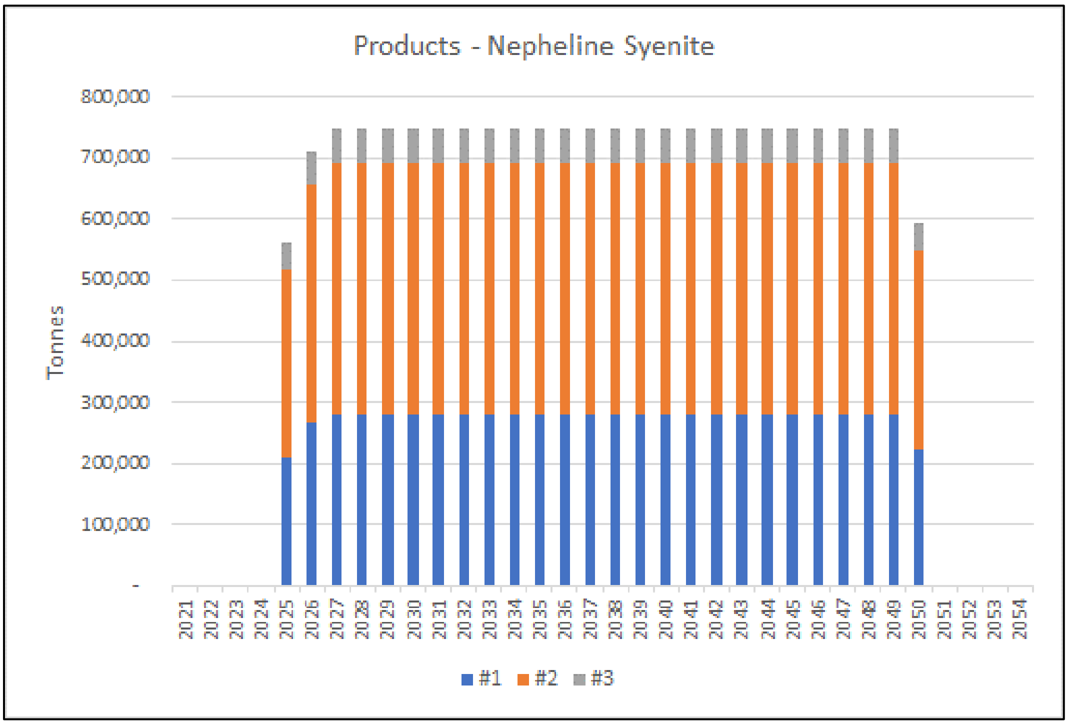


Figure 21-2: Products

21.3.2 Commodity Prices

Table 21-1 presents the assumed commodity prices for the purposes of this PEA assessment between 2025 and 2031. The following is noted:

- Individual REO commodity prices have been supplied by Adamas and are quoted in real USD terms (April 2021). The price profile used for the purposes of this PEA is the most conservative of three profiles supplied by Adamas (see Section 19). The Adamas profile shows real terms increases over the period 2021 to 2030, after which SRK has flat lined the prices at the 2030 values. Table 21-1 shows resulting the weighted average REO Basket Price assumed in the PEA between 2025 and 2031. The LoM average REO Basket Price is USD53.05/kg and varies between USD46.70/kg and USD54.74/kg.
- Given the real terms price profile for REO, SRK has assumed that construction for the project will be over a 2-year period between 2023 and 2024, with production commencing from 2025. As such the commodity prices applicable to the evaluation of the project are those from 2025 onwards.
- No revenue is assumed for some of the 'heavy' rare earth oxides, namely: Er_2O_3, Ho_2O_3, Tm_2O_3, Yb_2O_3.
- A constant price is assumed for a fused zirconia (ZrO_2) product over the LoM (no revenue is assumed for HfO_2) (refer to Section19).
- A constant price is assumed for Nb_2O_5 product over the LoM (refer to Section 19).
- Constant Nepheline Syenite product prices have been assumed over the LoM which have been sourced from an external nepheline syenite industry expert (refer to Section 19).

The point of sale for the various products is assumed to be the following:

- REO, ZrO_2, Nb_2O_5: Port of Rotterdam, therefore transport costs are included to transport the products from Luleå to Rotterdam.
- Nepheline Syenite Products: FOB Norra Kärr site and therefore no transport costs are included for these products.

Table 21-1: Commodity Prices

Commodity Prices	Units	2025	2026	2027	2028	2029	2030	2031
Average REO Basket Price	(USD/kg)	**46.70**	**49.19**	**51.49**	**53.35**	**54.24**	**54.63**	**52.72**
ZrO_2	(USD/kg)	4.00	4.00	4.00	4.00	4.00	4.00	4.00
Nb_2O_5	(USD/kg)	35.00	35.00	35.00	35.00	35.00	35.00	35.00
Nepheline Syenite Product 1	(USD/t)	50.00	50.00	50.00	50.00	50.00	50.00	50.00
Nepheline Syenite Product 2	(USD/t)	65.00	65.00	65.00	65.00	65.00	65.00	65.00
Nepheline Syenite Product 3	(USD/t)	12.00	12.00	12.00	12.00	12.00	12.00	12.00

21.3.3 Royalties

Per the in-country legislation, royalties are calculated based on 0.2% of in-situ value of the various products.

21.3.4 Corporation Tax

SRK has assumed a 20.6% corporation tax rate on operating profits and has assumed the following for the purposes of this calculation:

- Taxable deprecation is calculated using the 'straight line' method and assuming 20 years for all initial capital costs and 10 years for sustaining capital costs.
- Tax losses can be carried forward and offset against future profits.

21.3.5 Working Capital

Changes in working capital movement at the end of each year are modelled as a cash inflow/outflow as appropriate and which net off to zero over the LoM. The following assumptions are made:

- Debtor days:45
- Creditor days: 45
- Stores days: 30
- Stores: 10%

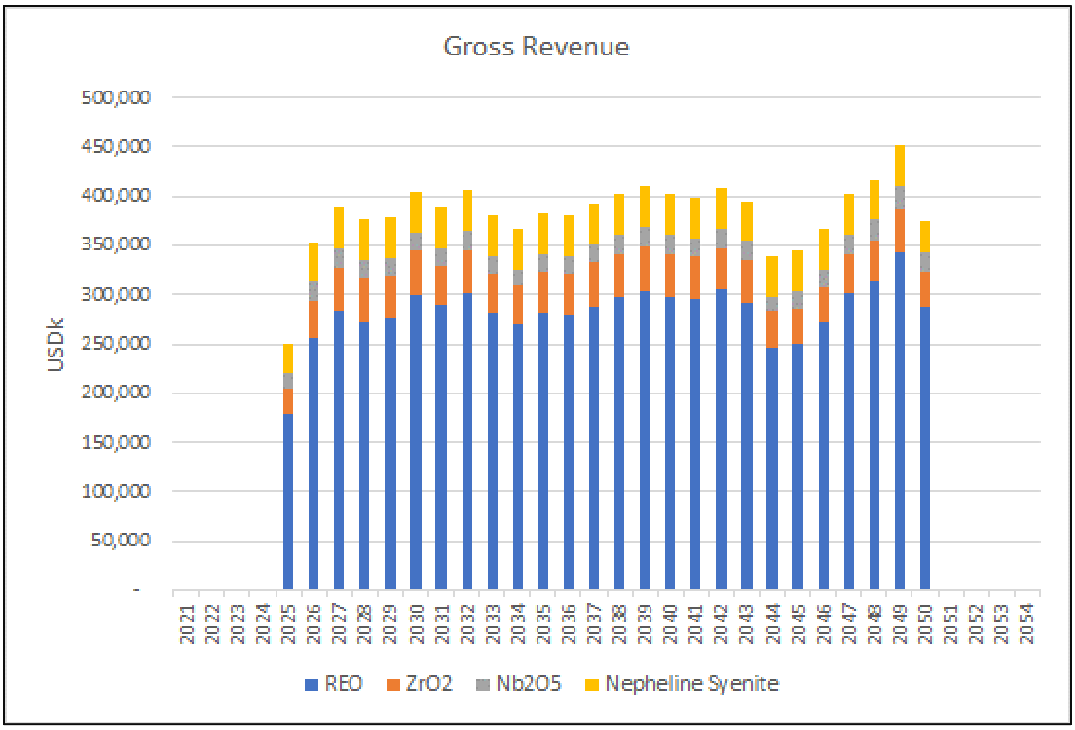
21.4 Cashflow Summary

21.4.1 Revenue

Table 21-2 presents a summary of the LoM gross revenue (before toll treatment charge) and annual average gross revenue split between the various products. This is also shown graphically in Figure 21-3. In summary at steady state the product will generate gross revenue in the region of USD385Mpa on average with some 74% of the revenue coming from the REO product, 10.6% from ZrO_2 product, 11.6% from Nepheline Syenite products and 4.8% from Nb_2O_5 product. The LoM average REO basket selling price equates to some USD53/kg. The most significant contributors to the REO revenue itself are Dy_2O_3 (43%), Nd_2O_3 (21%), Tb_2O_3 (16%) which together account for some 79% of the REO gross revenue.

Table 21-2: Gross Revenue

Gross Revenue Split	Units	LoM	Av Annual	REO %	Total %
MREO					
Ce_2O_3	(USDk)	64,127	2,466	0.9%	0.6%
Dy_2O_3	(USDk)	3,130,566	120,406	42.5%	31.4%
Er_2O_3	(USDk)	-	-	0.0%	0.0%
Eu_2O_3	(USDk)	27,841	1,071	0.4%	0.3%
Gd_2O_3	(USDk)	183,706	7,066	2.5%	1.8%
Ho_2O_3	(USDk)	-	-	0.0%	0.0%
La_2O_3	(USDk)	40,374	1,553	0.5%	0.4%
Lu_2O_3	(USDk)	460,084	17,696	6.2%	4.6%
Nd_2O_3	(USDk)	1,554,191	59,777	21.1%	15.6%
Pr_2O_3	(USDk)	404,200	15,546	5.5%	4.1%
Sm_2O_3	(USDk)	11,261	433	0.2%	0.1%
Tb_2O_3	(USDk)	1,146,951	44,113	15.6%	11.5%
Tm_2O_3	(USDk)	-	-	0.0%	0.0%
Y_2O_3	(USDk)	343,662	13,218	4.7%	3.4%
Yb_2O_3	(USDk)	-	-	0.0%	0.0%
Total	(USDk)	**7,366,963**	**283,345**	**100.0%**	**74.0%**
TREO basket price	(USD/kg)	**53.05**			
Co-Products					
ZrO_2	(USDk)	1,060,842	40,802		10.6%
Nb_2O_5	(USDk)	478,172	18,391		4.8%
Nepheline Syenite Product 1	(USDk)	357,190	13,738		3.6%
Nepheline Syenite Product 2	(USDk)	681,264	26,202		6.8%
Nepheline Syenite Product 3	(USDk)	17,163	660		0.2%
Total Gross Revenue	(USDk)	**9,961,593**	**383,138**		**100.0%**



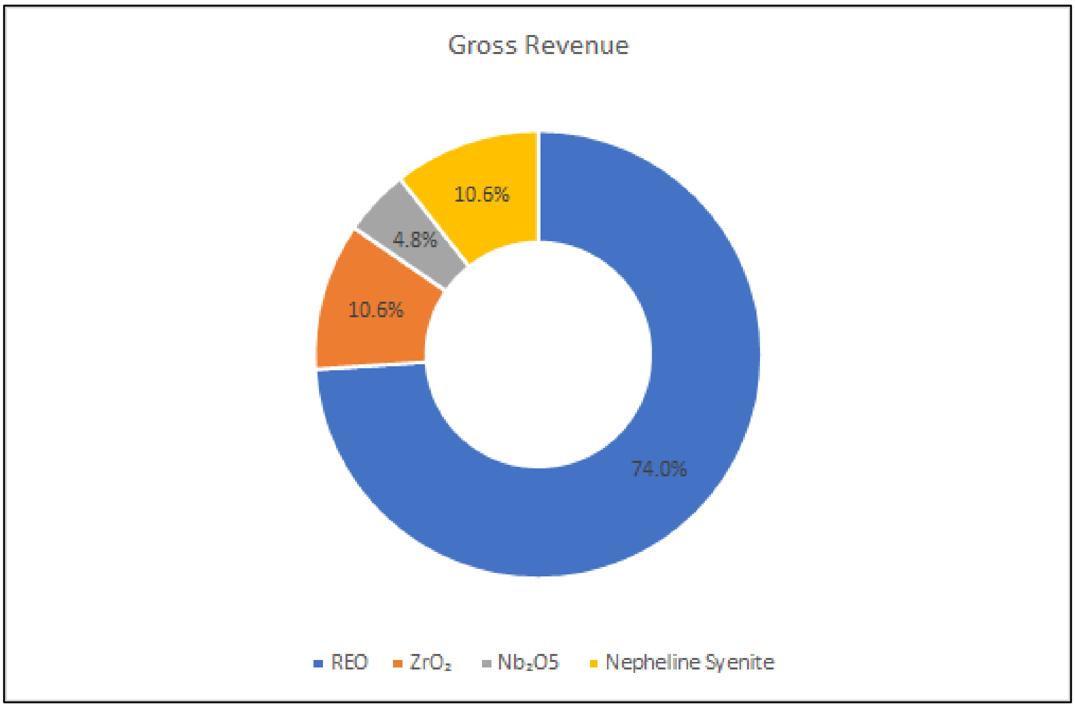


Figure 21-3: Gross Revenue by Product

21.4.2 Operating Costs

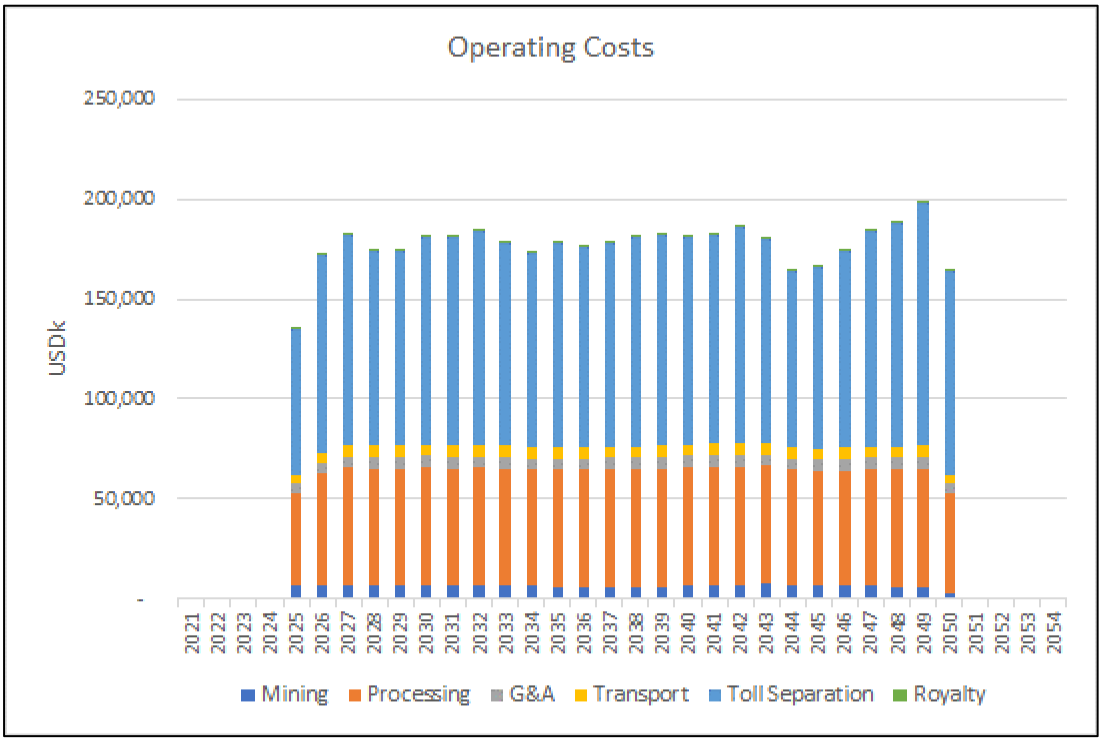
Further details on the operating cost assumptions are included above in Section 20. In summary operating costs include:

- Norra Kärr mining costs for mining of mineralized rock and waste.
- Norra Kärr and Luleå site processing costs for processing of mineralization and production of magnetic concentrate and Nepheline Syenite products at Norra Kärr and REO, ZrO_2 and Nb_2O_5 products at Luleå and including water supply and waste management costs for disposal of aegirine waste (Norra Kärr Site) and leach/neutralised waste and gypsum waste (Luleå site).
- Owners G&A costs.
- Transport costs for transport of magnetic concentrate from Norra Kärr to Luleå and transport of REO, ZrO_2 and Nb_2O_5 products to the Port of Rotterdam.
- Sales costs for third party toll treatment of the MREO (USD19/kg REO).

Table 21-3 presents a summary of the LoM operating costs including the annual average and unit costs expressed in terms of per tonne mineralized rock mined and per Kg of REO produced (excluding and including co-product credits). In summary, the most significant cost items are the sales costs (third party treatment charge) of USD19/kg REO and the mineral processing costs which together account for some 89% of the total operating costs. This is also shown graphically in Figure 21-4(but excluding the impact of co-product credits in this figure).

Table 21-3: Operating Costs

Operating Cost Summary	Units	LoM	Av Annual	USD/t ore	USD/kg REO
Mining	(USDk)	164,960	6,345	5.63	1.19
Processing - Norra Kärr	(USDk)	525,617	20,216	17.93	3.79
Processing - Luleå	(USDk)	975,599	37,523	33.28	7.03
G&A	(USDk)	146,577	5,638	5.00	1.06
Transport	(USDk)	144,544	5,559	4.93	1.04
Sales	(USDk)	2,638,378	101,476	90.00	19.00
Royalty	(USDk)	21,898	842	0.75	0.16
Total	(USDk)	**4,617,572**	**177,599**	**157.51**	**33.25**
By-product revenue credit				-	**18.68**
Total (net of by-product credit)					**14.57**



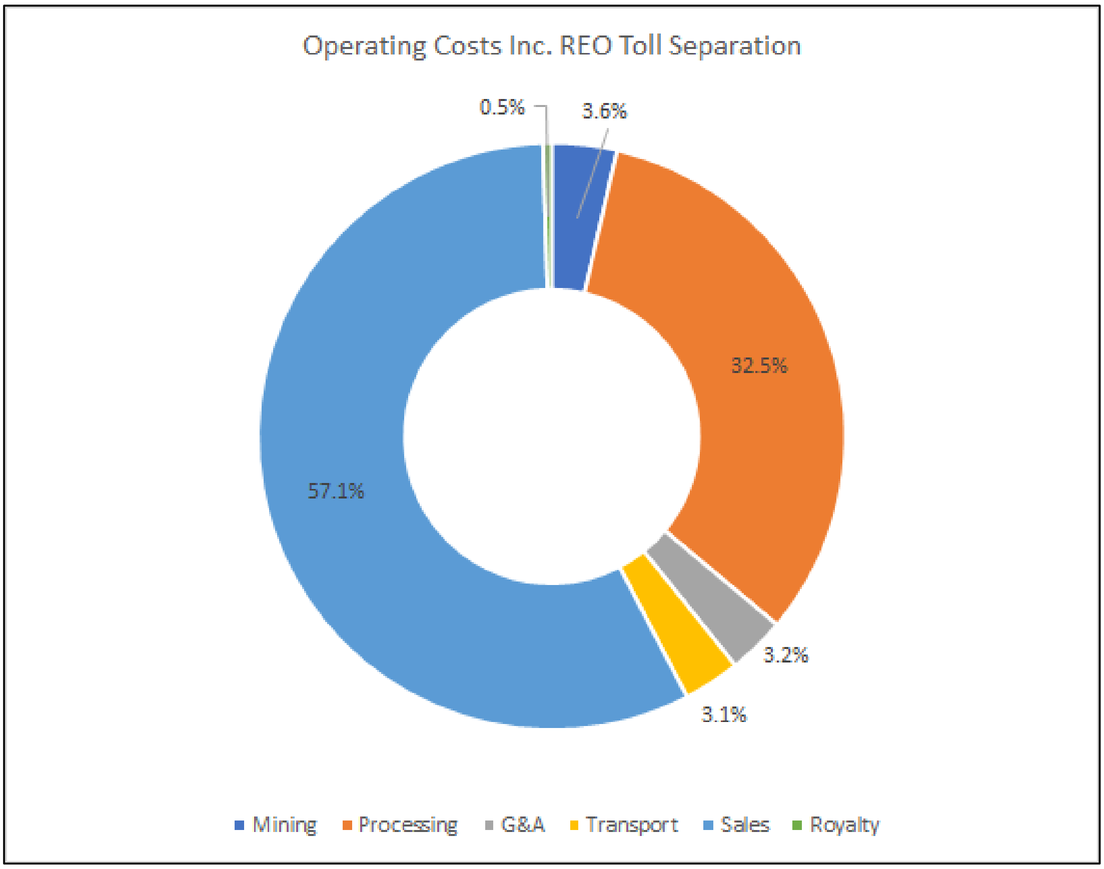


Figure 21-4: Operating Costs

Table 21-4 presents an analysis of various unit operating cost metrics over the LoM. This includes on a per tonne mined and processed basis and per Kg REO product including and excluding both toll separation charges and by-product credits.

Table 21-4: Unit Operating Costs

Unit Operating Costs Analysis	Units	LoM
Total per tonne ore mined (total)	(USD/t)	157.51
Total per tonne ore mined (ex toll separation)	(USD/t)	67.51
Mining - per tonne ore mined	(USD/t)	5.63
Processing - per tonne ore processed	(USD/t)	51.21
G&A - per tonne ore processed	(USD/t)	5.00
Transport - per tonne ore processed	(USD/t)	4.93
Per kg REO (noby-product credit)	(USD/kg)	33.25
Per kg REO (ex toll separation, noby-product credit)	(USD/kg)	14.25
By-product credit per kg REO	(USD/kg)	18.68
Per kg REO (ex toll separation, by-product credit)	(USD/kg)	- 4.43
Per kg REO (incl. toll separation, by-product credit)	(USD/kg)	14.57

Figure 21-5 below illustrates the Project yields an average LoM net operating margin of USD38.48/kg REO after considering credit from by-product revenue and with an operating cost net of by-product credit of USD14.57/kg REO.

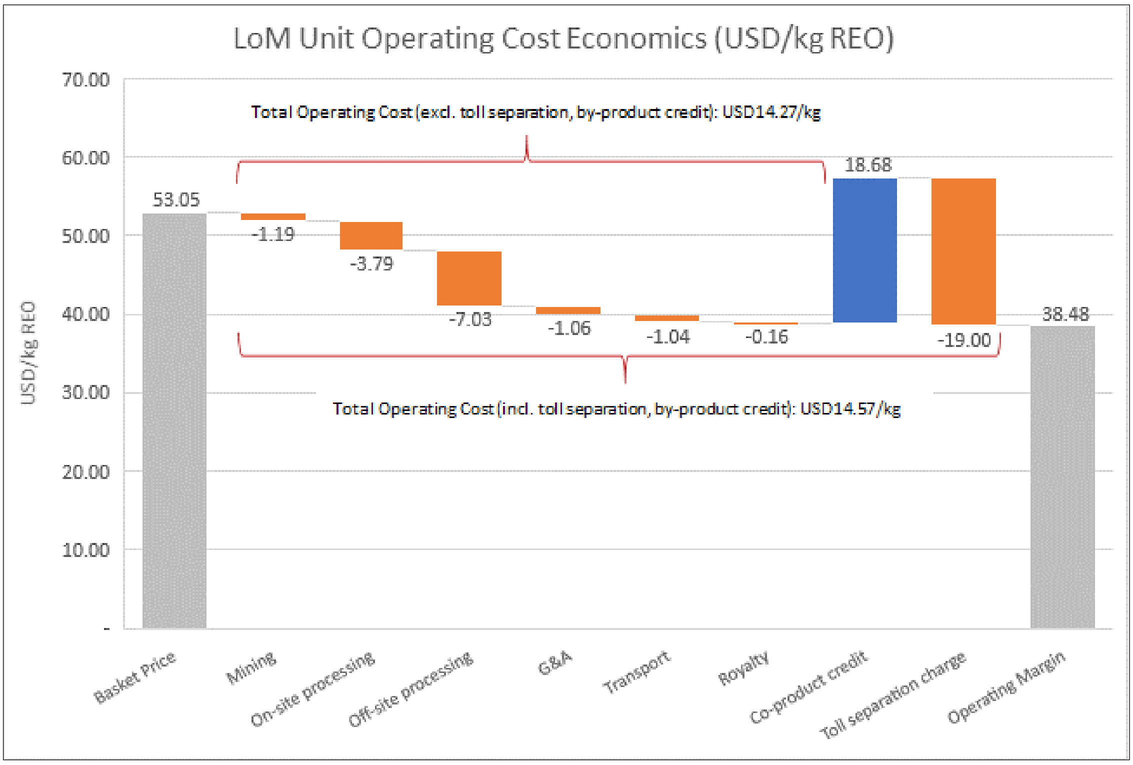


Figure 21-5: LoM Unit Operating Economics

21.4.3 Capital Costs

Further details on the capital cost assumptions are included above in Section 20. In summary capital costs include:

- Project Capital of USD487M assumed to be incurred between 2023 and 2024.

- Sustaining capital including:
  - Mining specific totalling USD9.9M over the LoM
  - General allowance of USD3.4Mpa incurred from Year 3 (2027) of operations and ceasing 2 years before (2048) the end of the LoM (2050) and totalling USD74.3M over the LoM.
- Closure costs of USD35M assumed to be incurred over a six year period – 3 years pre-closure and 3 years post-closure (i.e. between 2048 and 2053).

21.4.4 Project Cashflow

Cashflow model analysis on an annual basis is presented in detail in Appendix A [insert PEA Rep CF Table]. A summary of the LoM project cashflow is presented below in Table 21-5 while the annual pre-tax and post-tax cashflows are presented graphically on an annual and cumulative basis in Figure 21-6 and Figure 21-7 respectively. At a 10% discount rate the project yields a post-tax NPV of some USD762M and a post-tax IRR of 26.3%

Table 21-5: LoM Cashflow Summary

Cashflow Summary	Units	LoM
Gross Revenue	(USDk)	9,961,593
Operating Costs	(USDk)	(4,617,572)
Net Operating Cashflow	(USDk)	**5,344,021**
Capital Costs	(USDk)	(606,354)
Working Capital	(USDk)	0
Net Project Cashflow (Pre-Tax)	(USDk)	**4,737,667**
Corporation Tax	(USDk)	(985,074)
Net Project Cashflow (Post-Tax)	(USDk)	**3,752,593**

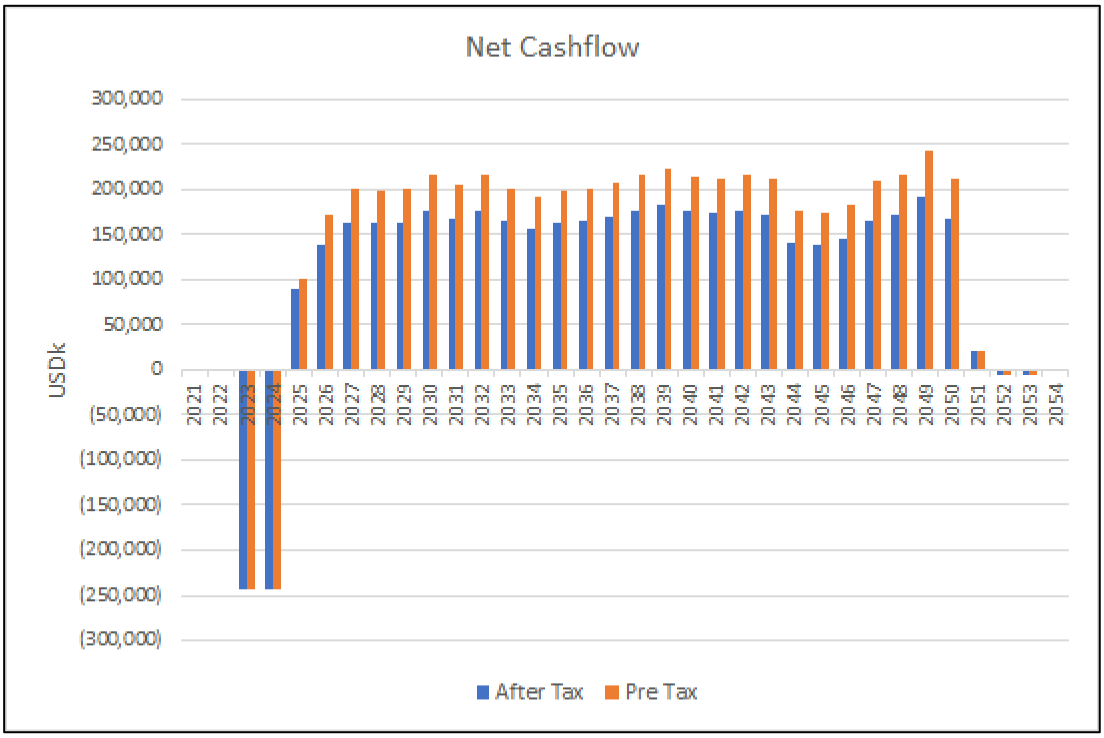


Figure 21-6: Annual Cashflows

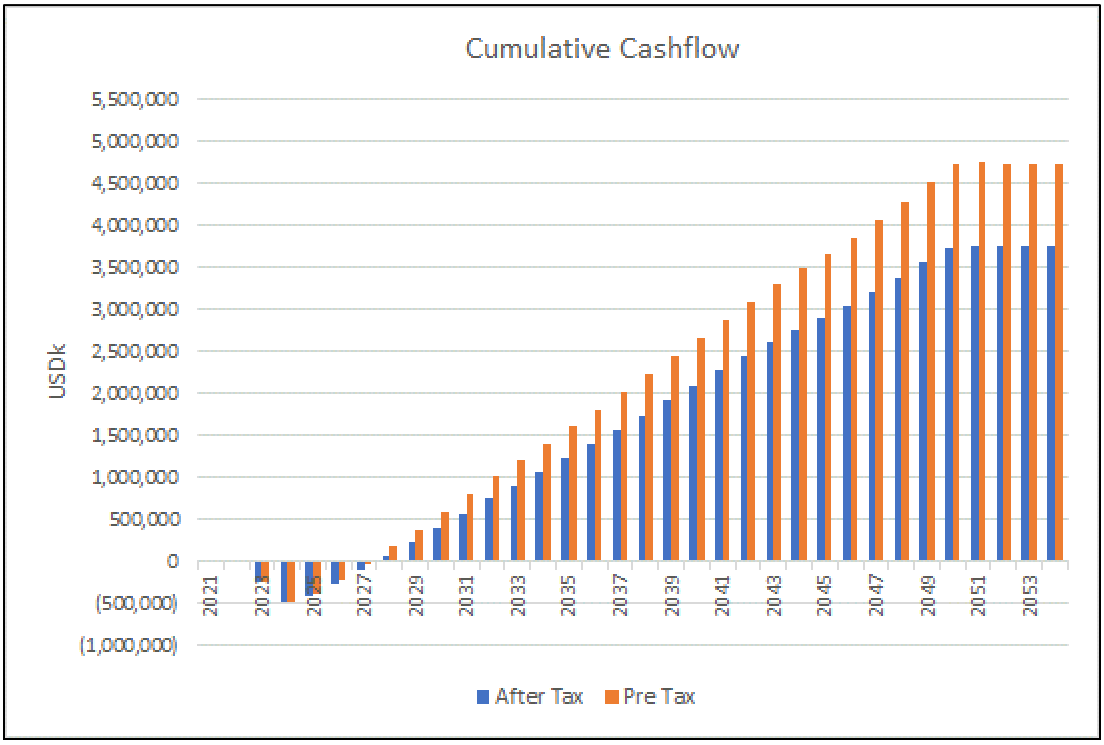


Figure 21-7: Cumulative Cashflows

21.4.5 Sensitivity

Table 21-6 shows the project NPV (pre- and post- tax at varying discount rates while Figure 21-8 shows the varying post-tax NPV at a 10% discount rate for single parameter sensitivities (+/-%) for commodity prices, total operating costs and total capital costs.

Table 21-6: NPV at varying discount rates

Discount Rate	Units	Pre-Tax	Post-Tax
6%	(USDk)	1,814,648	1,397,433
8%	(USDk)	1,358,088	1,029,412
10%	(USDk)	**1,026,372**	**762,160**
12%	(USDk)	780,516	564,277
14%	(USDk)	594,912	415,114

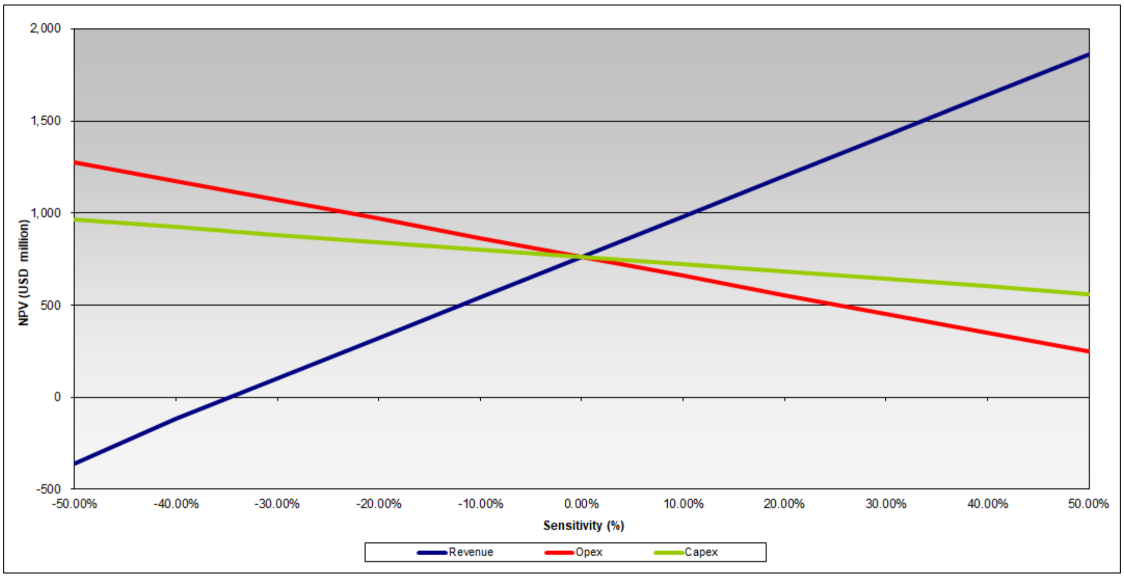


Figure 21-8: Single parameter sensitivity

It is noted that a fall of around 35% in all commodity price assumptions would result in a break-even NPV. Even though the REO pricing forecast used in this PEA is based on the most conservative forecast provided by Adamas (section 19) it is noted that some of the commodity prices are higher than recent market prices. While SRK considers the price profile used in the PEA to be reasonable considering supply and demand market dynamics, SRK has assessed a specific sensitivity to REO prices by using current market prices (BAIINFO, July 14, 2021). While the LoM average REO basket using current prices reduces to USD42.67/kg (USD53.05/kg PEA Base Case), the Project still demonstrates robust economics with a post-tax NPV of USD467m and IRR of 21.1%.

Finally, given the significant contribution of the toll separation charge assumption (USD19/kg MREO) to the overall operating costs for the project, Table 21-7 presents the post-tax NPV using a 10% discount at varying assumptions for this.

Table 21-7: NPV at varying toll separation charges

Toll Charge (USD/kg)	NPV (USDk)
10	1,040,090
12	978,328
14	916,565
16	854,803
18	793,041
19	**762,160**
20	731,278
22	669,516
24	607,754
26	545,991

22 ADJACENT PROPERTIES

The only mineral properties in the immediate area are the exploration permit held by LEM. This covers the potential strike extent of the Norra Kärr intrusive body which is known to be completely encompassed within the mining licence. As such, the additional exploration licences are of limited intrinsic value and have no influence over the economic viability of the Norra Kärr project.

23 OTHER RELEVANT DATA AND INFORMATION

23.1 Waste Management

A concept-level volumetric study for waste management has been carried out for Norra Kärr Rare Earth Elements (REE) Project, Sweden. The study was carried out in support of a Preliminary Economic Assessment (PEA), with the objective of providing a PEA-level cost estimation.

Single-stage models were produced for three waste streams to accommodate the required storage capacity, based on current estimations and examples of similar waste materials. The three waste streams included in this study are silicate waste, gypsum waste and aegirine residue as shown in the schematic diagram Figure 16-5.

In addition to the Rare Earth Element (REE) products Niobium (Nb) and Zirconium (Zr), other materials produced in the process are Nepheline Syenite (sold by-product) and waste rock. The Rare Earth Element (REE) products and Nepheline Syenite by-product will require temporary storage only, accounted for in infrastructure costs under Section 17.6. Waste rock haulage and storage is accounted for in mining costs under Section 15.6.

CAPEX and OPEX cost estimations are provided based on model volumes and item costs from comparable operations in the region. In general terms, the CAPEX costs are based on starter berms and liner installation; the OPEX costs on facility operation costs.

23.1.1 Locations

The Norra Kärr project site is located in the South of Sweden, on the east side of Lake Vattern. Magnetic separation will be carried out onsite to produce Nepheline Syenite (product) and to separate the remaining material into aegirine residue and a mineral concentrate for further processing. Aegirine residue will be stored onsite and the Eudialyte mineral concentrate will be transported offsite to Luleå a large industrial port 920 km north east of the mining facility.

Figure 23-1: Norra Kärr and Luleå locations in Sweden (aerial imagery sourced from Google Earth, 2021)

23.1.2 Mine Plan

The target storage volumes are based on the mass of each waste stream according to the current mine plan and assumed densities for each waste material produced. The mine plan features a mining rate of 1.15 Mtpa, over a 26-year life of mine (LOM). The masses of the waste streams assumed at this stage are:

- Aegirine residue - 292,026 tpa
- Silicate waste - 86,537 tpa
- Gypsum waste - 91,916tpa

There may be opportunity to optimize the waste dry stack designs at a further stage of study when more information is available. Geotechnical and geochemical characterisation of the waste materials will determine the requirements for safe storage of each waste stream.

Design

The aegirine residue, silicate waste and gypsum waste will be stacked as ‘dry’ crushed, granular material. The silicate and gypsum waste will be stacked in engineered waste ‘dry stacks’. The aegirine waste will be stored with waste rock, in an onsite co-disposal waste management facility. The facility will feature an engineered ‘dry stack’ in the centre with waste rock dumped around the perimeter. The co-disposal waste management facility will also have a cover of waste rock and a low-permeability cap over the aegirine residue for closure.

The design of the dry stacks aims to safely store the required volume of material, while minimising the facility footprint area, final surface area for rehabilitation, the closure time and closure cost. In addition, the onsite aegirine waste dry stack design aims to minimise visual impact and catchment area (avoid existing streams and significant drainage channels).

The concept at this stage includes a perimeter bund and a liner system over the footprint of each ‘dry stack’ and the aegirine waste storage area, to minimise risk of receptor contamination. The perimeter bunds will be constructed from engineered fill sourced from the mine (waste rock) or locally derived material. Future stages of design should include suitable mitigation for erosion and dust generation.

A material dry density of 1.6 t/m^3 has been assumed for each dry stack at this stage, based on examples of similar materials.

The dry stacks were sized according to the design aims and the design constraints summarised in Table 23-1.

Table 23-1: Dry stack design summary table

Dry stack component	Constraint	Unit	Value
Dry stack (overall)	Slope inclination	(none)	1V:3H
	Maximum height	m	60
Benches	Slope inclination	(none)	1V:2.5H
	Height	m	10
	Width	m	5
Perimeter bunds	Slope inclination	(none)	1V:2.5H
	Height	m	5
	Crest width	m	5

Figure 23-2 shows an example of a conceptual dry stack design with a similar geometry, in cross-section.

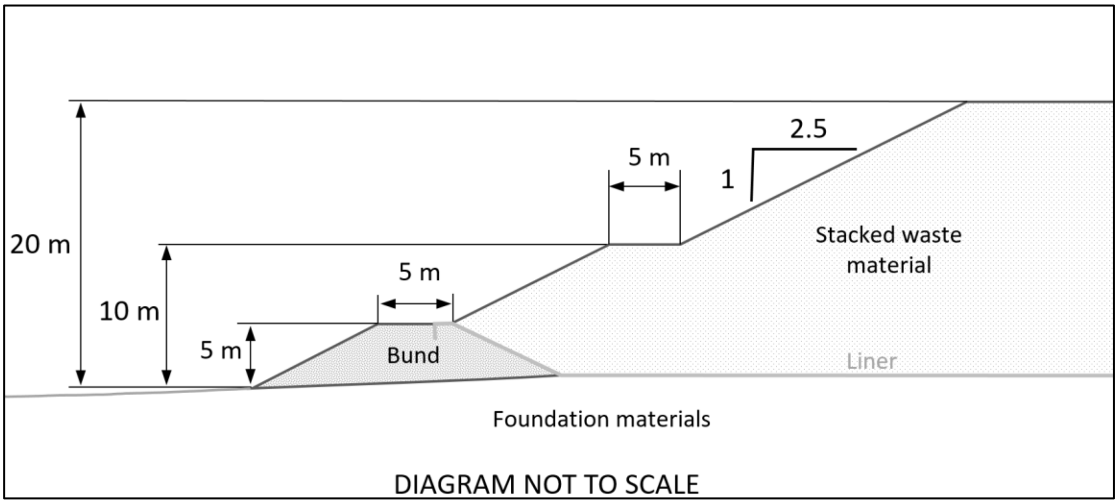


Figure 23-2: Schematic diagram of a dry stack in cross-section

The Luleå industrial zone is the planned location of the silicate waste and gypsum waste dry stacks and has been assumed to be level ground at this conceptual stage of study.

23.1.3 Output

The results of the volumetric models were used to quantify the material and operational requirements before application of unit costs in the waste management cost model.

A summary of the measurements and quantities according to the volumetric model output for each facility is presented in Table 23-2. These results were used to determine the requirements for footprint preparation (earthworks), perimeter bund, liner area and water management in the next stage of cost estimation.

Table 23-2: Model output measurements and quantities

Property	Unit	Silicate waste dry stack	Gypsum waste dry stack	Co-disposal facility	
				Aegirine residue	Waste rock
Total Volume	m3	1,406,228	1,493,634	4,745,428	4,685,967
Crest elevation	m ASL	39	40	247	
Maximum height	m	34	35	36	
Maximum length	m	300	305	880	
Maximum width	m	300	305	625	
Footprint area	m2	90,136	93,163	445,036	
Height of perimeter bunds	m	5	5	5	
Volume of starter berms	m3	60,000	61,000	125,000	

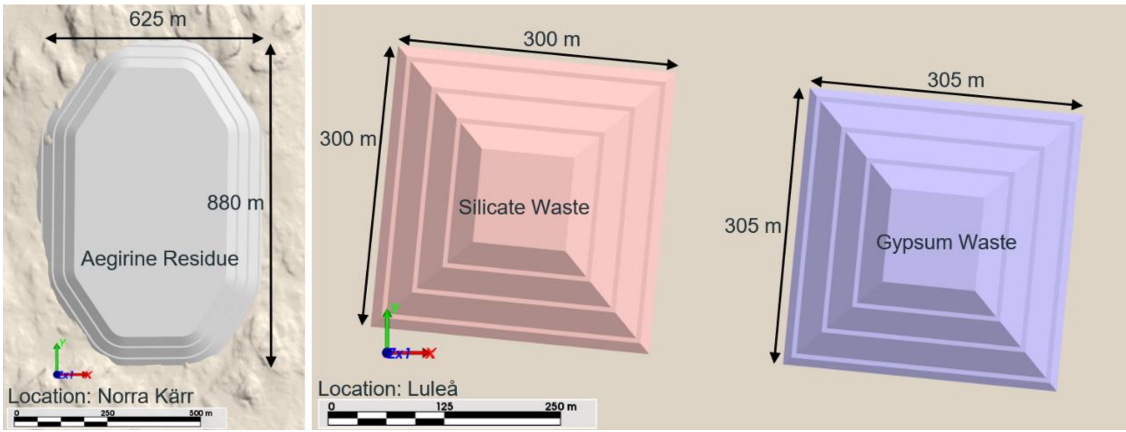


Figure 23-3: Graphical results of the dry stack volumetric models with approximate dimensions in the x, y plane

23.2 Cost estimate

The life of mine capital and operating costs have been estimated for the purposes of a Preliminary Economic Assessment (PEA), based on a PEA-level volumetric study, assumptions defined in Section 15.3.3and a mining rate of 1.15 Mtpa, over a 26-year life of mine (LOM).

The life of mine capital cost for the aegirine residue and waste rock co-disposal waste management facility, silicate waste and gypsum waste storage facilities has been estimated at USD 7.4 M. SRK recommends including an additional 10% allowance, at this preliminary stage for potential costs associated with future design and permitting.

The unit capital cost is based on similar sites (‘dry’ stacked, fully lined storage facilities) in Sweden and the Scandinavian region. It is assumed to include rockfill cell wall construction, ground preparation and lining, monitoring instruments and water management requirements.

The life of mine operating cost for the aegirine residue and waste rock co-disposal waste management facility, silicate waste and gypsum waste storage facilities has been estimated at USD 26.8 M (or some USD2.2/t material placed).

The unit operating cost is based on similar sites and includes the cost of waste material transportation and stacking. It does not include the cost of crushing as this is accounted for elsewhere in the cost model.

At the end of mine life the facilities should be closed according to a suitable closure design in conformance with the Swedish Environmental Social Governance (ESG) regulations and international guidelines for closure. Closure will be carried out as an additional cost which has been included separately under Section 18.6 and is not included in the waste management cost estimation.

A summary of the costs contributing to the total capital and operating costs is presented in Table 23-3.

Table 23-3: Waste Management CAPEX and OPEX summary table

	Task	Sub-task	Unit	Unit rate(USD)	Quantities	Cost (USD)
Capital Costs	**Onsite Aegirine Residue Management Facility**					
	Foundation prep and lining	Clear, grub, shallow excavation (assume 0.2m depth)	m2	0.5	445,036	$ 222,518
		Composite liner (2mm HDPE + 1m compacted clay)	m2	7.5	247,662	$ 1,857,465
	Earthworks	Perimeter bund fill (place, spread, compact)	m3	5	97,500	$ 487,500
		Instruments and monitoring	LS	90000	1	$ 90,000
	Water management	Non-contact surface water diversion channel	m3	10	7,500	$ 75,000
		Contact water collection pond	LS	546497	1	$ 546,497
	Offsite Silicate Waste Management Facility					
	Foundation prep and lining	Clear, grub, shallow excavation (assume 0.2m depth)	m2	0.5	90,136	$ 45,068
		Composite liner (2mm HDPE + 1m compacted clay)	m2	14	90,136	$ 1,261,904
	Earthworks	Perimeter bund fill (place, spread, compact)	m3	5	60,000	$ 300,000
		Instruments and monitoring	LS	60000	1	$ 60,000
	Water management	Non-contact surface water diversion channel	m3	10	3,600	$ 36,000
		Contact water collection pond	LS	340594	1	$ 340,594
	Offsite Gypsum Waste Management Facility					
	Foundation prep and lining	Clear, grub, shallow excavation (assume 0.2m depth)	m2	0.5	93,163	$ 46,582
		Composite liner (2mm HDPE + 1m compacted clay)	m2	14	93,163	$ 1,304,282
	Earthworks	Perimeter bund fill (place, spread, compact)	m3	5	61,000	$ 305,000
		Instruments and monitoring	LS	60000	1	$ 60,000
	Water management	Non-contact surface water diversion channel	m3	10	3,660	$ 36,600
		Contact water collection pond	LS	350493	1	$ 350,493
Operating Costs	**Onsite Aegirine Residue Management Facility**					
	Aegirine residue	Haulage of tailings (assume less than 1 km distance)	m3	2	4,745,428	$ 9,490,855
		Place, spread, compact	m3	1.5	4,745,428	$ 7,118,141
	Offsite Silicate Waste Management Facility					
	Silicate waste	Haulage of tailings (assume less than 1 km distance)	m3	2	1,406,228	$ 2,812,456
		Place, spread, compact	m3	1.5	1,406,228	$ 2,109,342
	Offsite Gypsum Waste Management Facility					
	Gypsum Waste	Haulage of tailings (assume less than 1 km distance)	m3	2	1,493,634	$ 2,987,268
		Place, spread, compact	m3	1.5	1,493,634	$ 2,240,451
Total Costs	**Total CAPEX**					$ 7,425,502
	Opex					$ 26,758,513
	Allowance	For design permitting, etc				$ 742,550
					Total cost	**$ 34,926,565**

23.3 Mine Waste Characterization

As discussed in Section 15 and 16 respectively, the proposed mine operations and processing circuit will generate the following waste streams:

- Mine waste rock (9.4 Mt);
- Tailings on-site from the two stage magnetic separation, dominated by aegirine (7.5 Mt);
- neutralized residue tailings at the Luleå processing site (2.25 Mt);
- gypsum residue at the Luleå processing site (2.39Mt).

23.3.1 Relevant Legislation

The regulation on the extraction of waste (SFS 2013:319) transposes Directive 2006/21/EC of the European Parliament and of the Council of 15 March 2006 on the management of waste from extractive industries) (Extractive Waste Directive).

Regulation SFS 2013:319 specifies the criteria which renders extractive waste as 'inert waste'. Notably, the waste:

1. cannot to any significant degree disintegrate or dissolve or otherwise changes in any significant way that could cause harm to human health or the environment;
2. does not contain more than:
   a. 0.1% sulfide sulfur; or
   b. 1.0% sulfide sulfur, if the neutralization potential ratio (NPR) of the waste is greater than 3, calculated as the ratio between the neutralization potential (NP) and the acid potential (AP), in a static test according to the standard SS-EN 15875: 2011;
3. does not risk self-ignition and cannot burn;
4. neither in its entirety nor in its fine fraction contains a higher content of arsenic, cadmium, cobalt, chromium, copper, mercury, molybdenum, nickel, lead, vanadium, zinc or any other substance that may be harmful to human health or the environment and that the content does not exceed national natural background levels;
5. is essentially free from the substances used in extraction or processing that may harm human health or the environment.

Regulation SFS 2013:319 defines hazardous waste as a substance with a waste code denoted by an asterisk (*) in Annex 4 to the Waste Ordinance (SFS 2011:927) or which according to documents issued pursuant to Section 12[8] of the Waste Ordinance are hazardous waste. Waste Ordinance SFS 2011:927 has been repealed by SFS 2020:614 which will enter into force on 01 January 2022. Both Waste Ordinance SFS 2011:927 and SFS 2020:614 stipulate that they do not apply to extractive wastes, therefore, they are only referred to with regards to the waste codes and hazardous waste classification as specified in SFS 2013:319. SRK uses online software HazWasteOnline™ to classify wastes as non-hazardous / hazardous in accordance with EU technical guidance and legislation.

Waste materials which meet the above scope can be deposited as non-hazardous materials. Whilst materials should not be deliberately diluted to meet these exemption levels, for NORM with unevenly distributed activity, an average value may be calculated per deposit occasion. If the materials are to be used for building, the scope of SFS 2018:396 and 2018:506 still applies

23.3.2 Waste Rock

The total waste rock produced during the life of mine is approximately 9.4 Mt (Section 15). Waste rock will be used for construction of embankments, ROM pad and haul roads around the mine site. Remaining waste rock will be placed into a waste rock dump which is planned for siting to the east of the open pit. Golder (2013) conducted a geochemical characterisation of the waste rock where they collected 65 samples from six different material types:

- 10 samples of Pulaskite (PUL) (reported to comprise ~10% of overall materials)
- 10 samples of Kaxtorpite (KAX) (reported to comprise ~8% of overall materials)
- 15 samples of Grennaite, katapleite bearing (GTC) (reported to comprise ~55% of overall materials)
- 10 samples of Lakarpite (LAK) (reported to comprise ~2% of overall materials)
- 10 samples of Syenite (granite) (SYE) and 10 samples of Granite (GR) (reported to comprise ~25% of overall materials)

Composite samples were generated for each of the material types which comprised 10 samples per material type except for the Norra Kärr Grennaite (GTC) which comprised 15 samples. The key findings of the geochemical characterisation are summarised here in relation to the regulation on extractive waste (SFS 2013:319).

[8] Section 12 is repealed by Ordinance 2018:2003.

Acid Generating Potential

ABA testing was undertaken on the composite samples at ALS Environmental, Vancouver, Canada (Table 23-4). Paste pH in the six different composite samples ranged between 9.7 and 10 s.u. Total sulfur content ranged from 0.02 to 0.07 % and NPR ranged from 10.5 to 155 in each of the material types indicating that they would all be classified as non-acid forming (NAF). Sulfate sulfur was low in all samples (<0.01 to 0.05 % by sodium carbonate digestion and <0.01 to 0.03 % by hydrochloric acid digestion) indicating that what sulfur is present, is predominantly in the form of sulfide sulfur. Based upon these ABA results each of the waste rock material types analysed met the inert criteria of sulfide sulfur content <0.1 % according to SFS 2013:319.

Table 23-4: Summary of Key ABA Results (Golder, 2013)

Material Type	Paste pH	Total S	Sulfate S(*)	Sulfate S(ƚ)	Sulfide S	AP	NP	NNP	NPR
	s.u.	%	%	%	%	tCaCO$_3$/1000t			
PUL	9.9	0.07	0.05	0.02	0.02	2.2	23	21	10.51
KAX	10.2	0.06	<0.01	0.01	0.06	1.9	62	60	33.07
GR	9.7	0.04	<0.01	0.01	0.04	1.3	15	14	12
GTC	10.1	0.02	<0.01	0.03	0.02	0.6	97	96	155.2
LAK	10.1	0.06	<0.01	0.02	0.06	1.9	32	30	17.07
SYE	9.7	0.02	<0.01	<0.01	0.02	0.6	20	19	32

Notes:

(*) Sodium carbonate digestion of sulfates (complete dissolution of $BaSO_4$ and $SrSO_4$)

(ƚ) _Hydrochloric acid digestion of sulfates (little to no dissolution of $BaSO_4$ and $SrSO_4$)

Multi Element Analysis

The 65 waste rock samples underwent a multi-element analysis at ALS Environmental, Vancouver, Canada which involved a lithium borate fusion prior to acid dissolution (hydrochloric, nitric and hydrofluoric acid) and ICP-MS analysis. Golder (2013) confirmed that analysis of base metals cobalt, copper, nickel, lead and zinc was not accredited. It is also noted that arsenic, cadmium and mercury were not analysed. Average results per material type are summarised in Table 23-5 and compared to average results of 189 samples from 99 Swedish quarries (Golder, 2013)[9]. The Swedish quarry samples provide an indication of natural background levels and Golder also refers to the granite (GR) results as surrounding reference material.

The six composite samples also underwent multi-element analysis at ALS, Luleå which involved leaching with nitric acid in a microwave oven, followed by lithium metaborate fusion and further dissolution in diluted nitric acid prior to ICP-MS analysis to determine total elemental concentrations. The results are summarised in Table 23-6 relative to the average Swedish quarry results. SRK has also calculated weighted averages for the multi-element assay data based upon the overall proportions reported in Golder (2013) and Section 23.3.2.

Considering the parameters described within the inert criteria for Regulation SFS 2013:319 (Section 23.3.1), the following key points are noted:

[9] Golder (2013) provides the following reference for the quarry data:
Hallberg, A. 2011. Final report - Material characterization - Positive List for residual material - Part 2. Minbas II, report 3.2a-3, SGU

- The GTC samples, reported to comprise 55% of the overall mined materials, reported enrichment in REE but no appreciable enrichment in key environmentally sensitive parameters except for lead (3 samples above background [max 228 mg/kg relative to 74 mg/kg background]) and uranium (8 samples above background [average 4.4 mg/kg relative to background 4.2 mg/kg]).
- The KAX samples reported to comprise 8% of the total materials, showed the greatest enrichment. Concentrations were elevated relative to the natural background quarry materials for several REE plus: arsenic, cadmium, copper, lead, thorium, uranium and zinc. Golder (2013) states that these materials are more comparable to other granites still used as a building materials throughout Sweden (e.g. Bohus granite).
- Zinc was above background levels in the KAX composite, 100% of the KAX and LAK waste rock samples, 70% of the PUL waste rock samples and 30% of the SYE waste rock samples. Zinc concentrations ranged between 510 and 3,500 mg/kg in the KAX relative to 92 mg/kg background.
- Concentrations of uranium were above background levels in the KAX composite, 100% of the KAX and LAK waste rock samples, 90% of the PUL samples, 53% of the GTC samples and 40% of the GR and SYE samples. Enrichment was highest within the KAX materials (averaging 24 mg/kg in the KAX waste rock samples relative to 4.2 mg/kg background). However, Golder (2013) reports that the Swedish Energy Agency reports average uranium in Swedish granites ranges from 8 to 27 mg/kg.
- Thorium concentrations were above background levels in the PUL, KAX, GR and LAK composites and 100% of the LAK waste rock samples, 90% of the KAX samples, 80% of the PUL samples, 50% of the GR samples and 20% of the SYE samples. Greatest enrichment was within the KAX materials (61 mg/kg relative to natural background 15 mg/kg). However, Golder (2013) reports that the Swedish Energy Agency reports average thorium in Swedish granites ranges from 8 to 90 mg/kg.

Golder (2013) reported further multi-element analysis undertaken on the composite samples at ALS, Luleå; results were compared to Price (1997) for the purpose of screening any further potentially problematic elements in addition to those screened using the Swedish quarry background averages. Golder reported no further elevated elements in the composite samples. SRK notes that silver, gold, beryllium, bismuth and cadmium content was elevated by at least three times average crustal concentrations (Price, 1997) in one or more material types and the calculated weighted average. Concentrations of manganese, lead, thorium, uranium and zinc were also elevated by at least three times average crustal concentrations but not in the calculated weighted average.

SRK has conducted a hazardous waste assessment through HazWasteOnline™ to determine whether the waste materials contain any hazardous properties. The assessment uses the multi-element assays for the composites and average assays per material type for the 65 waste rock samples plus calculated weighted averages. All material types were classified as “non-hazardous”. Elevated concentrations of manganese and zinc in the KAX materials (average 3,800 and 1,800 mg/kg respectively) have the potential to be classified as hazardous parameters depending upon speciation within the waste materials. However, despite this, the KAX materials overall maintain a “non-hazardous” classification.

Based on the geochemistry the waste rock is classified as non-hazardous, non-inert. According to Waste Ordinance SFS 2020:614 the waste can be allocated waste code “01.01.01 wastes from mining of metallic materials”. On this basis, no specific hazardous facility is required to store the waste rock. A Mine Waste Management Plan is required according to Section 23 of the Regulation on extractive waste (SFS 2013:319). An assessment of potential seepage water quality is also recommended to ensure that any risks to the surface and groundwater environment have been considered.

The majority of waste is non-hazardous and as such has potential to be used as road and construction material. This will be assessed more fully in the next phase of work.

Table 23-5: Summary of Average Multi Element Analysis on 65 Samples, Compared to Average Concentration from Swedish Quarry Samples (Golder, 2013)

Parameter	Units	Average from 99 Swedish Quarries (Golder, 2013)	GTC	GR	LAK	SYE	KAX	PUL	Weighted Average (*)
			n = 15	n = 10	n = 10	n = 10	n = 10	n = 10	
Ag	mg/kg	--	0.5	0.8	0.6	-1.0	0.8	-1.0	0.2
Ba	mg/kg	549	79	**1100**	**760**	510	520	**660**	368
Ce	mg/kg	86	**360**	**130**	**440**	**260**	**440**	**300**	**321**
Co	mg/kg	12	0.4	7.8	7.0	5.3	9.2	7.9	3.5
Cr	mg/kg	123	75	100	100	100	65	120	85
Cs	mg/kg	--	4.3	6.4	2.3	4.4	12	5.8	5.3
Cu	mg/kg	17	4.3	9.1	2.4	1.7	**19**	7.7	6.1
Dy	mg/kg	5.3	**130**	**6.9**	**48**	**24**	**33**	**56**	**85**
Er	mg/kg	3.3	**100**	**4.3**	**28**	**16**	**23**	**38**	**64**
Eu	mg/kg	1.1	**9.1**	**1.8**	**5.6**	**3.2**	**4.1**	**5**	**6.6**
Ga	mg/kg	--	78	22	41	37	70	47	61
Gd	mg/kg	6.4	**81**	**7.0**	**41**	**23**	**28**	**40**	**55**
Hf	mg/kg	--	230	8.2	50	35	41	81	144
Ho	mg/kg	1.1	**32**	**1.5**	**9.9**	**5.1**	**7.3**	**12**	**20**
La	mg/kg	42	**150**	**63**	**210**	**120**	**290**	**150**	**148**
Lu	mg/kg	15	15	0.7	3.5	2.4	3.0	5.4	9.5
Mo	mg/kg	4.2	-2.0	0.6	-2.0	**5.0**	-2.0	-2.0	-0.8
Nb	mg/kg	15	**180**	**19**	**93**	**33**	**98**	**110**	**126**
Nd	mg/kg	37	**200**	**53**	**210**	**140**	**150**	**150**	**165**
Ni	mg/kg	16	4.3	0.7	1.3	0.0	6.3	8.7	3.9
Pb	mg/kg	74	53	22	68	34	**110**	56	52
Pr	mg/kg	10	**48**	**15**	**54**	**35**	**46**	**39**	**41**
Rb	mg/kg	--	270	160	270	160	210	230	234
Sm	mg/kg	6.9	**63**	**9.3**	**44**	**27**	**30**	**36**	**46**
Sn	mg/kg	--	48	5.3	26	13	25	28	34
Sr	mg/kg	246	45	**410**	240	150	**380**	230	153
Ta	mg/kg	--	19	1.2	4.4	1.9	4.3	7.3	12
Tb	mg/kg	1.0	**18**	**1.1**	**7.4**	**3.9**	**5.2**	**8.2**	**12**

Parameter	Units	Average from 99 Swedish Quarries (Golder, 2013)	GTC	GR	LAK	SYE	KAX	PUL	Weighted Average (*)
			n = 15	n = 10	n = 10	n = 10	n = 10	n = 10	
Th	mg/kg	15	4.6	**16**	**36**	12	**61**	**24**	14
Tl	mg/kg	--	0.9	1.0	0.6	0.4	1.1	0.8	0.9
Tm	mg/kg	0.5	**17**	**0.7**	**4.2**	**2.4**	**3.4**	**6.1**	**11**
U	mg/kg	4.2	**4.4**	**4.7**	**12**	4.2	**24**	**9.5**	**6.6**
V	mg/kg	63	-5.0	56	49	34	60	47	19
W	mg/kg	20	3.7	0.9	0.4	1.3	0.7	2	2.6
Y	mg/kg	30	**860**	**47**	**320**	**140**	**230**	**400**	**561**
Yb	mg/kg	3.1	**110**	**4.6**	**26**	**15**	**21**	**39**	**69**
Zn	mg/kg	192	110	90	**360**	140	**1800**	**290**	**269**
Zr	mg/kg	208	**9300**	**320**	**2300**	**1500**	**1700**	**3400**	**5865**

Notes:

Results to 2 significant figures

Bold text indicates value exceeds average natural background (Swedish quarries)

(*) Weighted average calculated using approximate proportions reported above (Section 23.3.2) (Golder, 2013)

(green)	Indicates between 3 and 6 times average natural background (Swedish quarries)
(yellow)	Indicates between 6 and 12 times average natural background (Swedish quarries)
(pink)	Indicates greater than 12 times average natural background (Swedish quarries)
-1	Negative numbers indicates results less than detection limit

Table 23-6: Summary of Average Multi Element Analysis on Composite Samples, Compared to Average Concentration from Swedish Quarry Samples (Golder, 2013)

ELEMENT	SAMPLE	Average from 99 Swedish Quarries (Golder, 2013)	GTC	GR	SYE	LAK	KAX	PUL	Weighted Average(*)
As	mg/kg TS	2.8	*0.1*	0.8	0.2	0.7	**29.0**	*0.1*	2.5
Ba	mg/kg TS	549	96	**1200**	**570**	**860**	540	**720**	406
Be	mg/kg TS	--	9.4	8.4	13	19	120	21	20
Cd	mg/kg TS	1.2	0.8	0.1	0.2	0.7	**1.6**	0.8	0.7
Ce	mg/kg TS	86	**340**	**130**	**270**	**450**	**440**	**320**	**313**
Co	mg/kg TS	12	0.2	6.9	4.1	1.7	6.5	4.2	2.4
Cr	mg/kg TS	123	19	17	25	27	33	39	23
Cu	mg/kg TS	17	0.8	13	4.6	2.8	**22**	9.2	5.4
Dy	mg/kg TS	5.3	**140**	**7.4**	**26**	**46**	**36**	**65**	**91**
Er	mg/kg TS	3.3	**110**	**5.0**	**16**	**31**	**26**	**44**	**70**
Eu	mg/kg TS	1.1	**8.6**	**2.1**	**3.2**	**5.6**	**4.4**	**5.2**	**6.4**
Gd	mg/kg TS	6.4	**84**	**8.1**	**27**	**44**	**32**	**48**	**59**
Hg	mg/kg TS	0.26	*0.04*	*0.04*	*0.04*	*0.04*	*0.04*	*0.04*	*0.04*
Ho	mg/kg TS	1.1	**35**	**1.7**	**5.8**	**11**	**8.6**	**15**	**23**
La	mg/kg TS	42	**160**	**64**	**130**	**220**	**290**	**160**	**156**
Lu	mg/kg TS	15	**17**	0.6	2.4	3.6	3.3	6.0	11
Mo	mg/kg TS	4.2	***5.0***	***5.0***	***5.0***	***5.0***	***5.0***	***5.0***	***5.0***
Nb	mg/kg TS	15	**180**	**18**	**32**	**83**	**98**	**100**	**125**
Nd	mg/kg TS	37	**200**	**48**	**130**	**190**	**150**	**160**	**164**
Ni	mg/kg TS	16	0.5	4.8	6.0	2.2	6.2	6.7	2.8
Pb	mg/kg TS	74	25	23	32	66	**110**	**91**	40
Pr	mg/kg TS	10	**47**	**15**	**35**	**54**	**47**	**40**	**41**
S	mg/kg TS	460	*50*	270	79	59	**550**	67	123
Sc	mg/kg TS	--	1.8	9.3	3.2	8.1	11	8.2	4.4
Sm	mg/kg TS	6.9	**63**	**8.7**	**28**	**41**	**30**	**40**	**46**

ELEMENT	SAMPLE	Average from 99 Swedish Quarries (Golder, 2013)	GTC	GR	SYE	LAK	KAX	PUL	Weighted Average(*)
Sr	mg/kg TS	246	49	**450**	180	**250**	**380**	**250**	166
Tb	mg/kg TS	1.0	**19**	**1.2**	**4.1**	**7.6**	**5.9**	**9.2**	**13**
Th	mg/kg TS	15	11	**17**	14	**35**	**68**	**25**	10.9
Tm	mg/kg TS	0.5	**17**	**0.7**	**2.3**	**4.4**	**3.9**	**6.3**	**11**
U	mg/kg TS	4.2	**21**	4.1	**4.6**	**10**	**24**	**8**	**11.4**
V	mg/kg TS	63	2.7	51	33	44	55	44	22
W	mg/kg TS	20	***50***	***50***	***50***	***50***	***50***	***50***	***50***
Y	mg/kg TS	30	**790**	**46**	**130**	**300**	**230**	**380**	**519**
Yb	mg/kg TS	3.1	**110**	**4.7**	**16**	**27**	**24**	**41**	**70**
Zn	mg/kg TS	192	29	71	130	120	**1300**	150	162
Zr	mg/kg TS	208	**9000**	**330**	**1500**	**2300**	**1700**	**3400**	**5701**

Notes:

Results to 2 significant figures

Bold text indicates value exceeds average natural background (Swedish quarries)

Indicates between 3 and 6 times average natural background (Swedish quarries)

Indicates between 6 and 12 times average natural background (Swedish quarries)

Indicates greater than 12 times average natural background (Swedish quarries)

0.04 Indicates results less than detection limit

(*) Weighted average calculated using approximate proportions reported above (Section 23.3.2) (Golder, 2013)

Metal Leaching Potential

Shake Flask Extraction

During the 2013 geochemical characterisation (Golder, 2013), a two stage shake flask leach test was undertaken on the six composite samples (L/S 2 and 10). pH ranged between 8.6 and 10 in the leachates and Golder reported a low potential for short-term leaching based upon the results. The KAX composite sample, which was most enriched in the multi element analysis, did not show any increased tendency to leach relative to other material types, expect for uranium (0.14 mg/kg reported during the L/S 10 stage).

Kinetic Humidity Cell Tests

Humidity cell testing (HCT) was undertaken on the composite samples at ALS, Vancouver, Canada for 52 weeks. There was an initial flush of metal(loids) during the first few weeks of testing, as is typical for HCTs, but the majority of parameters reported a relatively steady state and low release rates from around week 20 onwards. pH was consistently between 7.6 and 9.8 and sulfate was consistently below 1.8 mg/L in all samples from week 7 onwards apart from the KAX composite. The KAX composite, which reported the greatest enrichment in the multi element analysis, typically showed the greatest level of leaching. Sulfate concentrations in the KAX composite reached a maximum of 28 mg/L at week 10 then showed a decreasing trend to 11 mg/L at week 52. Lead fluctuated throughout the duration of the test (and was highest in the KAX composite).

Golder (2013) estimated annual weathering from the different material types by applying a number of estimated scaling factors to the HCT results (including temperature, oxygen saturation, grain size, water supply) and concluded that overall leaching potential per year is likely to be limited. This should be further investigated with numerical modelling predictions as mine planning proceeds.

Radionuclide Testing Waste Rock

Gross alpha and beta was analysed on the week 25 HCT leachate at ALS, Fort Collins, USA. Golder (2013) report that the results show low activity concentrations and activity indexes (AI) equal to the activity index reported within data from 99 granite, diabase and limestone quarries in Sweden. Only one leachate from the KAX composite reported alpha activity exceeding the drinking water quality guideline from WHO (>0.5 Bq/l). Therefore, additional investigations and analysis should be performed if the water is to be used as a drinking water source.

Golder (2013) reports the uranium and thorium activity levels ranging between 0.076 and 0.53 kBq/kg and the potassium activity levels ranging between 0.86 and 1.1 kBq/kg for the different material types. These activity levels are below the thresholds specified in SSMFS 2018:4 for NORM which is exempt from Radiation Protection Act (SFS 2018:396) and Radiation Protection Ordinance (2018:506). Only the KAX materials are reported to have an activity index which is equivalent to the European Commission thresholds (EC,1999[10]) for building in small quantities (e.g. tiles, boards), GTC, SYE and GR materials have activity indexes equivalent to the thresholds for building in larger quantities (e.g. concrete) and PUL and LAK materials fall between the two categories (Golder, 2013).

23.3.3 Metallurgical Residues

Aegrine waste, Onsite

The on-site processing residue is dominated by aegirine which has the potential for dissolution and release of ferrous iron that can be oxidized to ferric iron that will hydrolyse releasing protons. However, within the mineralogy of the aegirine residue, feldspar will react with generated protons and neutralize this. In addition, the material will be encapsulated within an excess of neutralising waste rock and placed on a liner. The metal content of the residue is extremely low and thus no metals are expected to be mobilized at any concentration of concern (Table 23-7). However, representative magnetic separation residues should undergo confirmatory geochemical characterisation to verify these conclusions and enable this waste stream to be classified according to the regulation on the extraction of waste (SFS 2013:319). An assessment of the risks to the surface and groundwater environment from the TSF should also still be undertaken.

Table 23-7: Composition of On-Site Residue (GTK, 2012)

SiO2 %	Al2O3 %	Fe2O3 %	CaO %	MgO %	Na2O %	K2O %	Cr2O3 %	TiO2 %	MnO %	P2O5 %	SrO %	BaO %	LOI %	Total %	Zr ppm
51,6	6,45	23,7	1,95	0,38	12,5	0,75	0,05	0,25	0,31	0,11	0,01	<0.01	1,66	99,72	6390
51,2	6,49	22,4	1,73	0,37	12	0,8	0,03	0,24	0,32	0,01	0,01	<0.01	1,71	97,31	6760

During the 2013 study, one sample of tailings underwent ABA testing plus multi-element analysis. The sample predicted non-acid forming characteristics with total sulfur at 0.01% and an NPR of 70. When compared to average natural background levels from Swedish quarries, the sample reported an enrichment of several parameters including cadmium (3.8 mg/kg relative to background 0.2 mg/kg), lead (550 mg/kg relative to background 13 mg/kg) and uranium (12 mg/kg relative to background 1.8 mg/kg).

SRK conducted a hazardous waste assessment on available data for the 2013 nephelite / feldspar tailings using HazWasteOnline^{TM}. The sample is classified as non-hazardous, therefore based on the cadmium, lead and uranium concentrations this material would be classified as non-hazardous, non-inert and would be allocated waste code “01 03 06 Tailings other than those mentioned in 01 03 04 and 01 03 05”.

[10] European Commission Radiation Protection 112 (1999), Radiological Protection Principles concerning the Natural Radioactivity of Building Materials, Directorate-General Environment, Nuclear Safety and Civil Protection.

Neutralised Residue, Luleå

There is insufficient information available to classify the neutralised residue which will be generated at the Luleå processing plant. These will need to undergo geochemical characterisation when suitable material is available. It is anticipated that they will be acid generating and likely classified as hazardous waste requiring a Category A hazardous waste storage facility.

In addition to the neutralised residue tailings, 2.7 Mt of gypsum will be generated at the Luleå processing plant. There is currently insufficient information available to classify this material and characterisation work should be undertaken when suitable material is available during future project phases. It is anticipated that at least some of this gypsum will be sold as a product, in which case it will not require waste classification.

Radionuclide Testing

Testing of deportment of radionuclides was also conducted by ANSTO Minerals in 2013-2014. While the testing focused on product and was undertaken for the purposes of process development, the related memorandum (ANSTO, 2014) states that the neutralised waste (corresponding to the leached and neutralised tailings) contains only very small amounts of uranium and thorium and is not radioactive as per the definition of a radioactive substance by the International Atomic Energy Agency.

23.4 Open Pit Geotechnics

The open pit geotechnics input into this PEA is developed by a desktop method. SRK has not conducted any additional data collection, data verification, material testing, or revised stability analysis. The majority of this section is extracted from the Geotechnical Study report completed by WAI in November 2014. SRK has completed a review of this report and utilised information supporting the PEA assessment where applicable. Some content has been modified by SRK to ensure it reflects the alternative interpretation of geotechnically applicable aspects.

Further appraisal of the inputs to slope stability analysis and a check on the analysis results by WAI has been performed. This has resulted in an adjustment to the WAI slope design parameters to supply realistic and practical bench-berm configuration typical in Nordic hardrock environments. This is the initial input to geometrically derive inter-ramp (IRA) and overall slope angles (OSA). SRK have low confidence in the aggressive IRA angles proposed by WAI, as the intrinsic bench configuration to achieve these angles is considered unpractical and contains a level of risk to slope stability and rockfall risk.

The updated bench-berm configuration has informed the SRK pit optimisation and pit design process completed for the PEA. A review of the resulting design and pit development schedule is included to highlight the aspects that require assessment in further study stages.

A list of specific recommendations for study elements to be completed in further project stages is included.

23.4.1 Data Availability

Tasman Metals Limited (Tasman) have conducted exploration and metallurgical drilling on the Norra Kärr project since 2009, and the drill holes have been geologically logged with some structural data recorded. Also, rock mass classification has been recorded as both Q and RMR values to a representative quality along with RQD measurements. These earlier geotechnical investigations were undertaken with assistance from Itasca Consultants providing logging templates and guidance.

SRK was only provided with drilling up until the NKA12 series. The data supplied did not contain the NKA14 series of 5 geotechnical holes drilled in 2014.

Specific to the geotechnical assessment a summary of the main rock types and their characteristics are listed in section 6.3.2.

23.4.2 Rock Mass Characterisation

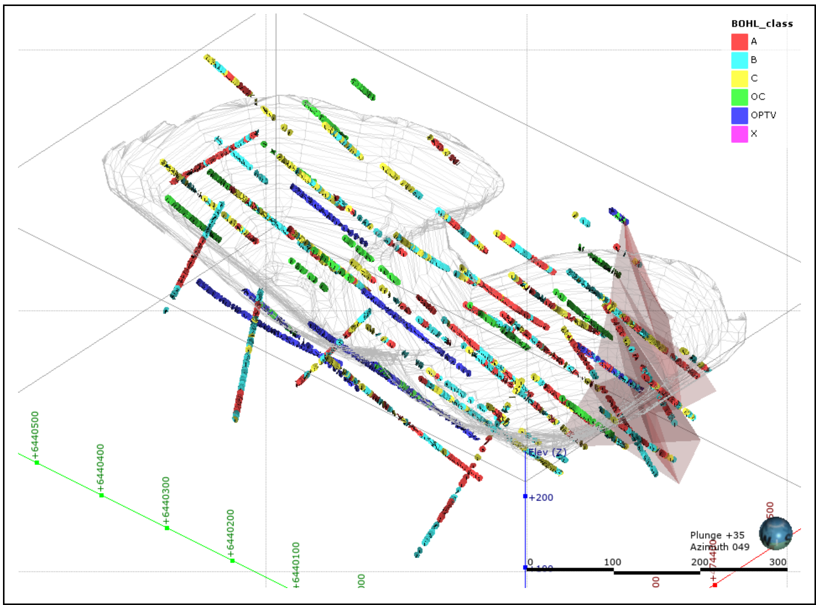
Rock mass logging from 41 drill holes, received from Tasman has been used in the WAI study. The logging has been based on intervals based on a similar geotechnical and lithological domain for a total of 7,382m. This format of logging was introduced to Tasman by Itasca Consultants. Although the method is sound and acceptable by identifying domains from logging, it is noted that it can give rise to issues, where due to the interval length; the influence of poor ground can be masked by a large interval length.

RQD data have been received from 104 drill holes for a total of 17,492.2m recorded. The RQD format includes a simplified approach to RQD logging by metre intervals mainly.

Structural Information

Structural logging and interpretation are completed on 62 drillholes deriving 7001 points in both core logging (6254 count) and optical televiewer logging (747 count). The quality confidence in core orientation was recorded and this has yielded about 55% of the recordings as high (A) to very high (B) confidence (see Figure 23-4). The OPTV data has been combined with the higher confidence logging to provide 4468 points in total for analysis.

Structural logging from 51 drill holes has been received by WAI from Tasman, the data totals 4,357 structural records where depth, type, alpha, beta, orientation confidence, and separation, roughness and infilling have been recorded in RMR format. SRK has seen a total of 61 drillholes and this is considered a large, informative and high quality dataset for suitable joint characterisation applied to slope analysis.



Confidence	Count	% of Dataset
OPTV	747	11%
A	1768	25%
B	2053	29%
C	1286	18%
OC	1147	16%
Total	7001	

BOHL Class is a confidence level in structural measurements:

OPTV: is borehole televiewer

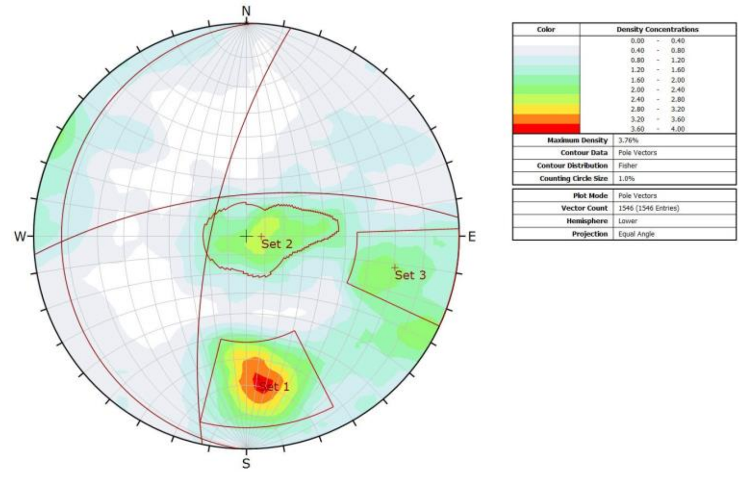
Core logging: A highest, B moderate, C lowest, and OC very low

Figure 23-4: Extent of structural logging data coloured by core orientation confidence

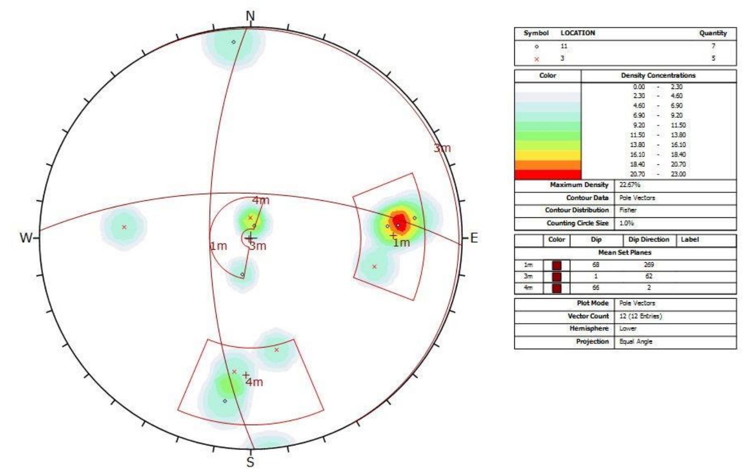
Surface trial pits and outcrop exposures were mapped by WAI which indicates consistency with the joint sets from logging. Spacing and persistence exposed in these pits are used to estimate these parameters, however, this was not calibrated against true spacing from drilling data.

The results from field mapping (example is shown in Figure 23-5) and core logging have been combined and the data show a clear orthogonal joint pattern (Figure 23-6). WAI plots do indicate multiple minor concentrations around the plot but only the three dominant concentrations were be used for further analysis as they represent 42% of all data and are the major orientations found in the investigation. This is considered a limitation that requires further localised joint pattern and foliation trend changes to increase confidence in the kinematic stability assessment.

Figure 23-5: Extent of structural logging data coloured by core orientation confidence



Unweighted contour stereonet plot of all data with high confidence orientation.



Stereonet of surface trial pit mapping by WAI

Figure 23-6: Stereonets of logging data (top) and surface mapping data (bottom)

Table 23-8: Summary structural sets (adapted from WAI, 2014)

Parameter	Set Number		
	1	2	3
Count	229	295	126
Dip	68	8	71
Dip Direction	355	271	282
Variability	13.9	20.3	13.2
Roughness	Sl Rough, pl-rough to undulating. smooth	Sl Rough, stepped smooth	Sl Rough, smooth stepped
Infill	Sl Weath?	Sl Weath?	Sl Weath?
Spacing (m)	(0.15-0.5) 0.2	0.3	0.15
Persistence (m) (from trial pits)	0.3 - 5	(1 – 10)	(1 - 10) 5

Major Geological Structures

A single major fault is identified intersecting the SE area of the deposit. The South-Eastern fault acts as a boundary to mineralisation of the deposit and is identified by intense fracturing of drill core and localised alteration of the rock mass nearby. The fault zone itself is approximately a zone of 1.5-3.0m; however, the material within the fault zone is often heavily crushed with clay evident (as recovered by drilling) in places. Figure 23-7 is an example of drilling recovery of the fault zone representing the degree of fracturing and intensely crushed and clay material.

The geological and magnetic maps are indicative of additional parallel structures to the interpreted South-Eastern fault. There are intersections of core loss and low RQD (RQD<40) that are indicative of other features. This is not currently investigated or modelled and is a trigger for further structural geology assessment to support geotechnical and hydrogeology components of later studies.

Figure 23-7: BHID NKA11054 Tray#15: Example of SE Fault intersection (redbox)

The fault zone has been located by interception of the drilled core; the fault is easily identified as poor to no RQD as opposed to generally very competent ground. The intercepted angle does appear to be less than perpendicular, and the actual width of the zone may be less than recorded.

Rock Strength

Uniaxial (Unconfined Compressive testing) and triaxial testing was completed on the available core of major lithologies and the results are summarised in Table 23-9. Triaxial testing results are shown in

Table 23-9: Uniaxial Lab Testing Results (WAI, 2014)

Lithology	Sample	Bulk Density	UCS (MPa)
Kax	A	2.65	72.9
PGT - C	A	2.55	128.6
PGT - M	A	2.68	112.8
PGT - F	A	3.01	268.5
GTM	A	2.58	277.8
GTM	B	2.74	152.3
GTC	A	2.72	229.8

Table 23-10: Traiaxial Testing Results (WAI, 2014)

Lithology	Sample	Bulk density (t/m3)	Internal Friction Angle (°)	Confining Stress (MPa)	UCS (MPa)
Kax	B	2.72	44.2	2.5	40
				15	113
				35	223
	C	2.64	28.0	7.5	76
				25	181
				50	199
PGT - C	B	2.61	48.9	5	30
				15	74
				37.5	215
				60	420
PGT - M	B	2.74	41.2	2	49
				35	193
				55	310
	C	2.74	47.1	10	47
				45	153
				70	451
PGT - F	B	2.78	53.6	2.5	28
				12.5	77
				35	253
				50	403
				65	616
GTM	A	2.48	44.3	5	50
				25	145
				50	234
				70	437
	C	2.51	42.9	10	73
				40	225
				60	337
GTC	B	2.71	48.9	2.5	43
				25	198
				55	416
	C	2.73	49.2	10	77
				40	215
				70	511

Shear strength results are shown in Table 23-11. Friction testing was completed on sawn surfaces, normal loads of 1.25, 2.5, and 5.0 MPa were applied to the samples. The PGT-F sample was tested on a failed, pre-existing shear surface.

Table 23-11: Direct Shear Test Results (WAI, 2014)

Lithology	Residual Results		Peak Results	
	Cohesion (kPa)	Friction Angle (°)	Cohesion (kPa)	Friction Angle (°)
PGT-F	257	11.1	0	32.8
PGT-M	423	4.5	596	25.8
KAX-A	235	26.2	486	27.3
PGT-C	4	12.5	588	13.5
GTM-B	164	14.0	164	27.5
Average	**217**	**13.7**	**367**	**25.4**

Rock Quality Classification

The characterisation was made from drill logging and surface mapping, supplemented by rock laboratory testing. For the purpose of the assessment, a Grännaite suite central to the mineral deposit has been modelled as the main units have very similar properties along with the Granitoid material. These rock types can be considered as RMR89 "Good". The western area has Fenite which is strongly altered and has weaker strength areas and a higher prevalence of fracturing. The SE Fault zone has poorer rock quality ratings confined to the SE FW location of the deposit.

The basic RMR and Q values have been calculated for rock mass classification. The categories and ratings have been adjusted depending on the range of the value to the category. The summary statistical results for RMR are presented in Table 23-12 and for Q in Table 23-13.

Table 23-12: Rock Mass Rating Basic RMR89 Classification System (WAI, 2014)

(RMR)89		Fenite	SE Fault[1]	Kax	GTC	GTM	PGT	Granitoid
Strength	Mean	9.7	4	5.8	13.3	13.4	11.1	10.3
	SD[2]	0.9		3.2	5.2	3.7	4.5	3.8
	Min	9.2		5.6	4.9	8.7	6.9	8.2
	Max	11.9		16.2	21.5	18.8	19.0	20.1
RQD	Mean	17.3	3	17.7	18.7	19.0	18.8	16.2
	SD	4.8		4.6	4.0	3.5	4.1	6.2
	Min	3.0		3.0	3.0	3.0	3.0	3.0
	Max	20		20	20	20	20	20
Spacing	Mean	7.8	5	9.4	10.8	10.0	11.1	9.2
	SD	3.1		3.6	3.4	2.7	3.5	2.8
	Min	5		0	0	5	0	1
	Max	20		15	20	20	20	15
Condition	Mean	15.7	2	19.6	19.3	19.0	20.6	17.7
	SD	4.9		5.0	4.7	3.9	4.3	7.0
	Min	12		3	0	0	0	0
	Max	20		27	30	30	30	30
Water	Mean	15	15	15	15	15	15	15
Basic RMR89 Rating	Mean	**58**	**29**	**68**	**77**	**76**	**77**	**68**
	SD	9.9		12.2	8.1	8.7	8.1	25.2
	Min	15		11	15	19	8	0
	Max	78		70	67	70	61	67
RMR89 Classification		**Fair**	**Poor**	**Good**	**Good**	**Good**	**Good**	**Good**

Notes: 1) The SE Fault does not have sufficient intercepts to determine statistical analysis; therefore, expected conditions have been used. 2) The statistical analysis has been based on Schmidt Hammer readings

Table 23-13: Rock Mass Rating Basic Q Classification System (WAI, 2014)

Q System		Fenite	SE Fault[1]	Kax	GTC	GTM	PGT	Granitoi
RQD	Mean	83	10	85	90	92	91	75
	SD	24		23	21	19	22	34
	Min	0		0	0	0	0	0
	Max	100		100	100	100	100	100
Jn[2]	Mean	9	15	9	9	9	9	9
Jr	Mean	1.7	1.0	1.9	1.7	1.7	1.7	1.8
	SD	0.7		0.8	0.8	0.7	0.9	0.7
	Min	1.0		0.5	0.0	0.5	0.5	1.0
	Max	4.0		3.0	4.0	4.0	4.0	3.0
Ja	Mean	2.3	6.0	1.4	1.6	1.7	1.7	2.6
	SD	1.2		1.0	1.0	1.1	1.0	2.1
	Min	0.75		0.75	0.75	0.75	0.75	1.0
	Max	4.0		8.0	8.0	8.0	10	8.0
Jw[3]	Mean	1.0	1.0	1.0	1.0	1.0	1.0	1.0
SRF[4]	Mean	1.0	5.0	1.0	1.0	1.0	1.0	1.0
Q' System Rating	Mean	**6.8**	**0.1**	**12.8**	**10.6**	**10.2**	**10.1**	**5.8**
	SD	10.7		10.8	10.9	9.1	12.2	10.1
	Min	0.0		0.0	0	0	0	0
	Max	44.4		33.3	59.3	59.3	59.3	31.7
Q Classification		**6.8**	**0.02**	**12.8**	**10.6**	**10.2**	**10.1**	**5.8**
		Fair	**Extremely Poor**	**Good**	**Good**	**Good**	**Good**	**Fair**

23.4.3 Simplified Domains

The results of the rock mass classification by WAI show good correlation between separate analysis methods and similarities between some of the units. The GTC, GTM, PGT, and Granitoid units all report similar rock mass classification results and strengths; therefore, they are consolidated as one geotechnical domain for the slope design analysis. The domaining summary is:

- Domain 1 – Overburden and– All surface soil/transition materials and highly fractured rock;
- Domain 2 – Fenite / Western Alteration Zone - Western margin areas and altered granites;
- Domain 3 – Grännaite Suite – GTC, GTM, PGT, KAX, and Granitoid units have similar geotechnical properties they have been grouped together; and
- Domain 4 – South East Fault – Due to the significance of this structure intersecting the planned pit and influence to slope stability, the SE fault has been set as a unique domain (there may be other parallel faults that may exhibit similar properties, but this is not well defined yet)

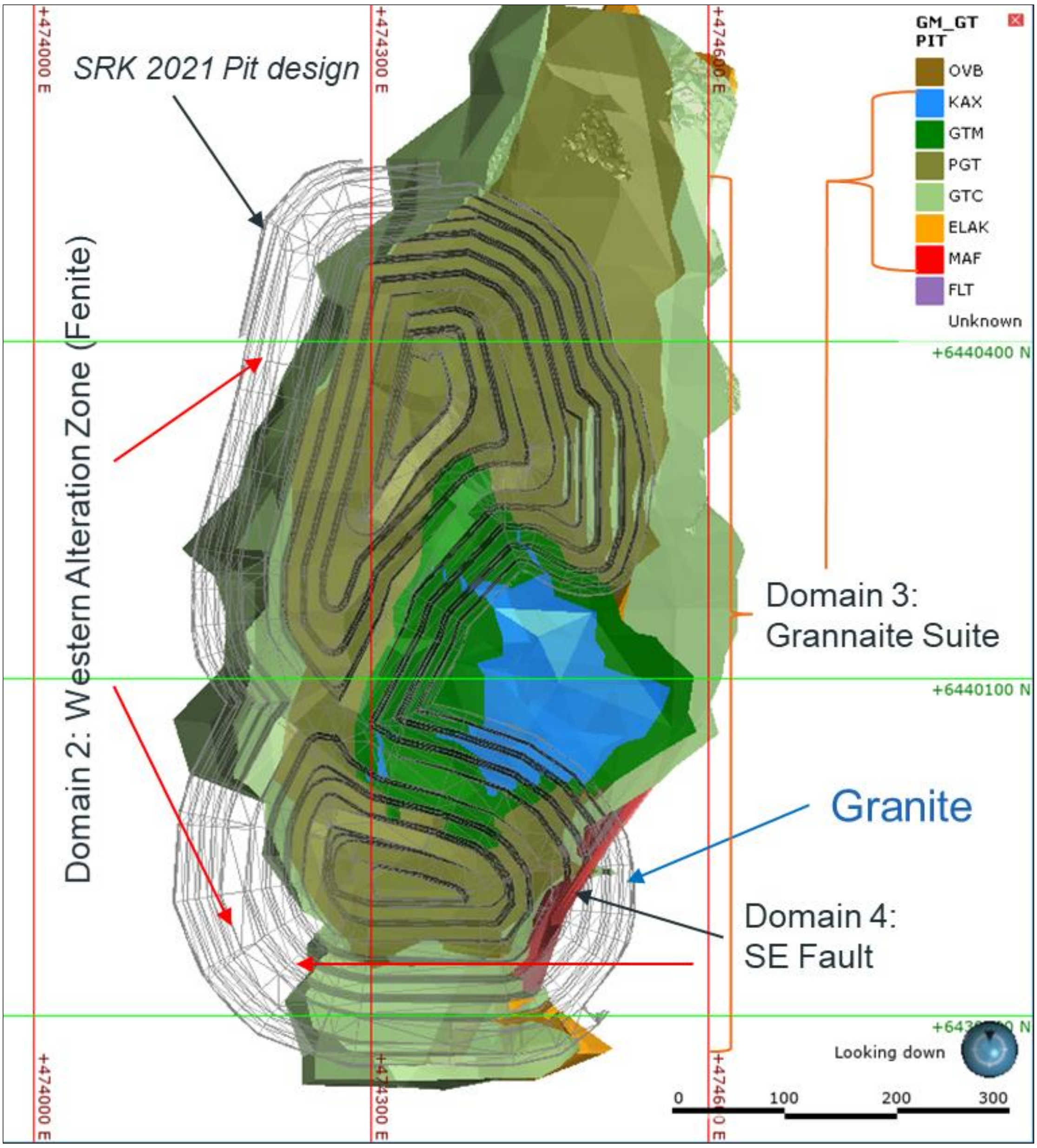


Figure 23-8: Main Geotechnical Domains intersected with the SRK 2021 pit design

23.4.4 Rock Characterisation Comments

The 2014 geotechnical drilling of 5 holes need to be complied with the database to update the analysis.

The Q logging and RMR logging is suitable for early-stage studies, however, the implementation of these systems has not been to industry best practice. The Q logging discounts the Jn number and is fixed to 3 with an oversimplification to the number of joint sets. The RMR logging does not define joint set groups into 3 alpha angle categories. As a result, the spacing of specific sets is not determinable. The practice of picking the lowest strength set in logging characterisation limits (and biases) the data collected as the controlling joint sets depends on the orientation of the slope in that area.

WAI mention that the South West area has significant intercepts of over a few metres of "highly fractured" and jointed core in NKA14098G and NKA14099G (WAI, 2014). This increased jointing could be part of weaker fault systems and needs to be further investigated for influence to slopes and ramp access in this area.

The geotechnical investigation noted some poor areas in the western flank of the deposit which is intersected by the top of the western slopes. The investigation has identified the area as a poor domain and analysed within the overall slopes. However, the drill results could indicate a larger scale system, and it is recommended this region should be investigated in more detail using the existing drilling initially. The effect of minor faulting on bench / inter-ramp scale is not considered, and efforts to locate and manage these structures with additional drilling is recommended.

Slope Parameter Terminology

The adopted naming of the components in slope design which are applied in this report are introduced and shown in Figure 23-9. These are accepted as industry standards with abbreviations that are referred to in this study:

- Bench Components: Bench Height (BH), Bench Face Angle (BFA), Catch Berm (CB, or berm)
- Inter-ramp angle (IRA) is the angle formed by a consistent bench configuration, usually separated by a wider ramp or Geotechnical Berm. This forms a bench stack height (BSH).
- The overall slope is from the uppermost crest to the lowest toe and the overall slope angle (OSA) is measured from a line joining these points

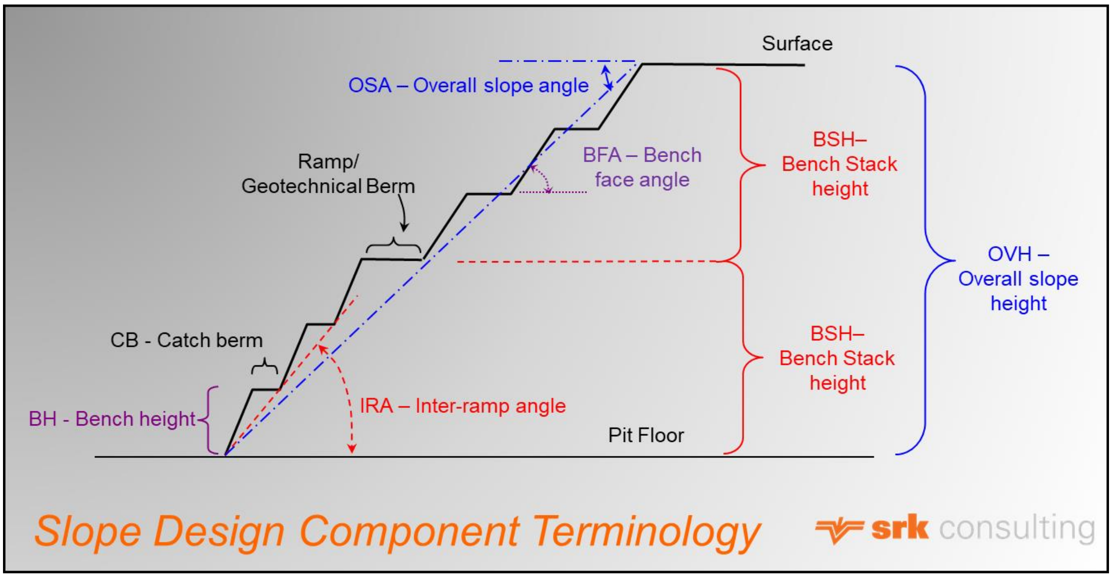


Figure 23-9: Slope design component terminology

Slope Stability Analysis

Analysis for Scoping level has been conducted by WAI with the available geological, geotechnical and hydrogeological information. The assessment is thorough and follows an industry standard structured approach that is auditable. SRK have assessed the study and the provided analysis results and stated where in agreement. No revised analysis has been completed at this stage. Suggestions are provided to improve the analysis, with the existing data set, in order to increase confidence in the slope stability design, as well as identify risks and opportunities.

A fundamental principle for determining a potential failure mechanism is that stronger rocks are likely to have discontinuities as the primary control, whereas in weak rocks, the controlling feature is likely to be the strength. Therefore, there are three possible modes of rock mass failure as described in Karzulovic & Read, (2009) These are applied to Norra Kärr. These are typical conditions in the Nordic rock mass and existing pit operations in the region

- Structurally Controlled Failure: Failure occurs only along the joints, bedding, or fault planes. This is the case for planar or wedge slides, which are most likely to occur at bench and inter-ramp scale. In this case, the shear strength and orientation of the structure are the most important parameters in assessing slope stability.
- Failure with partial structural control: Failure occurs partly through the rock mass and partly through structures. Usually at the inter- ramp and overall scale. In this case, both the strength of the rock mass and strength and orientation of structures are equally important in assessing slope stability.

WAI performed a series of analysis in Bench Scale (Planar, wedge, and toppling failure potential) and overall Scale (Whole Slope Analysis) and referenced industry standard pit design acceptance criteria as documented in Read & Stacey, 2009).

Bench Scale Stability Analysis

Stereonet kinematic analysis is a good first pass assessment of the kinematic potential of planar, wedge and toppling failure. This is limited to the 3 joint sets (see Section 0) applied to the entire deposit and does not assess the stability based on localised jointing pattern variation influencing slope orientations. The large data set of joint condition and joint shear strength testing was used for the kinematic inputs.

The stereonet analysis results for the three failure types are listed in Table 23-14. This is considered overly generalised and has likely discounted significant fracture sets that are influencing slopes in specific areas. This is particular to the west, south and south-west walls where there is also a reduction in rock strength due to the fenitised granites. Specific areas of planned slopes should be sub-domained to further refine the jointing fabric specific to the main slope locations. The kinematic analysis will be better defined and localised. The aim is to better determine the wedge forming potential.

Table 23-14: Slope Sector Stereographic (kinematic potential) analysis (WAI, 2014)

Slope Sector	Planar Failure	Wedge Failure	Toppling Failure
North (000°)	None	None	Yes (Set 1)
N East (045°)	None	None	None
East (090°)	Yes (Set 3)	None	Risk (Set 3)
S East (135°)	None	Yes (Set 1 + 3)	None
South (180°)	Yes (Set 1)	Risk (Set 1 + 3)	None
S West (225°)	None	None	None
West (270°)	None	None	Yes (Set 3)
N West (315°)	None	None	Yes (Intersection Set 1+3)

Summary points of the failure potential results directly from WAI (2014) and SRK comments of suitability are provided below:

- Planar Failure: Stereographic analysis highlighted planar failure on the eastern and southern pit sectors. Owing to the steepness of the failure plane (Eastern Sector = Set 3 (71°), Southern Sector = Set 1 (67°)). This is limited to the average dip angle and not assessed in terms of the probabilistic variability of dip.
- Wedge failure: The Southeast pit sector is at risk from wedge failures formed by Set 1 and Set 3. The program Swedge by Rocscience was used for analysis, using the mean dip and dip direction of Set 1 and 3, the plunge and trend of the wedge formed is (326°/64°). This is considered limited as it is expected that with variation in Set3 (foliation) dip azimuth that there will be more wedges formed and of variable size and crest back break. Wedge failure potential needs to be investigated further.
- Toppling failure: Toppling failure is not normally expected in blocky hard rock conditions with orthogonal jointing. WAI has assessed this and also indicate the application of RocTopple to the geometry and geology of Norra Kärr is considered inappropriate owing to the strength of the rock and joint opening lengths viewed in surface trial pits. SRK agree that this failure type is unlikely, however, targeted studies are required if weak foliation fabric is identified in the western walls

Suggested Kinematic analysis

SRK strongly suggest that probabilistic block size and block forming potential is conducted. This is possible with the current data set having spacing data available per joint set from the logged structures. A tool like SBLOCK and further application of SWEDGE can be implemented to further assess the bench-scale stability probabilistically. Parametric and sensitivity analysis of the inputs for joint set orientation and shear strength can be run over many models quickly to derive the statistical distribution of stable angles and berm widths.

This analysis is considered the minimum in competent and joint controlled rock masses approaching PFS level of study. Bench face angles and berm widths are better defined with block stability analysis as failure volumes and potential crest loss are more accurately determined.

The bench configuration will then geometrically define inter-ramp angles possible and with ramp systems, the overall angles to initiate optimisation through to practical slope and pit design.

Overall Slope Analysis

Overall slope stability is generated using 2D limit equilibrium methods yielding Factor of Safety results (FoS) was applied by WAI. Stability results for the maximum IRA determined from WAI are listed in Table 23-15. The results confirm that the WAI are stable with FoS greater than common design recommendations 1.3 (Read & Stacey, 2009). The resulting factors of safety are well above the design limits in circular and non-circular analysis through a single strength rating of the rock mass at IRA and overall slope scale. WAI state, compared to acceptance criteria, the main East and West slopes with the modelled parameters can be considered stable, although there is some potential poor ground in the western slope that was highlighted in the 2014 investigation which would need further investigation.

Table 23-15: WAI Overall slope stability analysis results in SLIDE (WAI, 2014)

Slope	D Value	Circular FoS	Non-Circular FoS
		Bishop	*Janbu*
Eastern Slope	0.7	6.23	5.71
	1.0	5.03	4.62
Western Slope	0.7	4.16	4.20
	1.0	3.12	3.36

The current application of Slide (limiting equilibrium) is considered indicative only for stability modelling as applied to the overburden material. This method is not applicable in high strength and joint controlled rock masses. The downgrading of the strength criteria by damage and stress unloading (D Factor) is overly assumed in the models as the whole material is downgraded and not only the excavation surface to 5-20m depth.

The analysis should consider the jointing fabric to model anisotropy and step-path failure potential as the jointing fabric controls stability. Further application of FEM modelling using RS2 is suggested which can incorporate a representation of the jointing fabric controlling stability. Spacing data and joint strength characteristics data are available for modelling inputs.

Rockfall Risk

Rockfall analysis is not completed yet and is required to assess this risk and catch the effectiveness of the slope design. This can be done in 3D using Trajec3D software (Basrock.net) which imports the designs to run fall analysis of multiple shapes from various heights.

23.4.5 Slope Design Parameters

2014 Design

Slope parameters adopted were largely based on the average foliation angle and issued at 67° with a range from 65° to 71° (Table 23-16). Included were bench heights between 15 and 20 m and the resulting IRAs which varies. The berm widths were determined from empirical catch bench widths to be 7.5m and 8.4m, as well as a calculation using toppling potential to derive 7.4m berm for 15m height and 8.9 for 20m height. These WAI defined berm widths are not further simplified along with the design parameters in one table, so the text is used for this review.

Table 23-16: WAI recommended bench design parameters (WAI, 2014)

<table>
<tr><th colspan="7">Table 7.6: Sector Slope Angles</th></tr>
<tr><th>Slope Sector</th><th>Bench Height (m)</th><th>BFA (°)</th><th>IRA (°)</th><th>Bench Height (m)</th><th>BFA (°)</th><th>IRA (°)</th></tr>
<tr><td>North (000°)</td><td rowspan="8">15</td><td>67</td><td rowspan="2">50</td><td rowspan="8">20</td><td>67</td><td rowspan="2">53</td></tr>
<tr><td>N East (045°)</td><td>*67</td><td>*67</td></tr>
<tr><td>East (090°)</td><td>71</td><td>53</td><td>71</td><td>56</td></tr>
<tr><td>S East (135°)</td><td>65</td><td>49</td><td>65</td><td>51</td></tr>
<tr><td>South (180°)</td><td>67</td><td rowspan="4">50</td><td>67</td><td rowspan="4">53</td></tr>
<tr><td>S West (225°)</td><td>*67</td><td>*67</td></tr>
<tr><td>West (270°)</td><td>67</td><td>67</td></tr>
<tr><td>N West (315°)</td><td>67</td><td>67</td></tr>
</table>

** Indicates no specific failure mechanism, therefore 67° has been used.*

SRK Updated Slope Design

Hardrock drill and blast bench construction is not exact and resulting face angles will not have a tolerance <5° from design. It is difficult to develop slopes less than this angle tolerance and for simplicity and practicality, it is rounded to the nearest 5° increment. Similarly, drill and blast slope construction will not have a <1m tolerance in berm formation due to blast damage and blocky crest loss etc. A simplified recommendation to realistic berm widths is to 7m for 10m bench height and 10m for 20m bench height.

Slope design parameters are usually defined with an allowance of likely blast damage to crests and also expected wedge backbreak into the berms. The berms are increased to 10m for double benching in all domains as it is expected on average there will be a 2-3m loss of the catch width. This is highly likely in the fenitised areas of the western slopes.

The updated design parameters for bench configuration are listed in xx which includes:

- Bench heights (BH) of 10 and 20m depending on depth from the surface
- Bench face angle (BFA) normalised at 70 degrees based on stability analysis and practical blast hole drilling
- Berm width (BW) of 7m for 10m high benches and 10m for 20m high benches
- These parameters geometrically derive the IRA independent of bench stack height.

These angles are combined for slope heights to indicative reserve pit depths to 160m derived from the mining optimisation studies. Overall slope height (OSH) controls the overall slope angle (OSA) and this is the primary slope angle input into the pit shell optimisation process. If the mineralisation depth or surface crest positions change, the overall slope angle will change, but the IRA is fixed for the bench stacks between ramps.

Table 23-17: SRK revised bench design parameters

Parameter		West - Walls		East - Walls		South - Walls	
Depth Range		0 - 40m	40 -160m	0 - 20m	20-160m	0 - 20m	20-160m
Bench Face Angle (°)	BFA	70	70	70	70	70	70
Berm Width (m)	BW	7	10	7	10	7	10
Bench Height (m)	BH	10	20	10	20	10	20
Inter-Ramp Angle (°)	IRA	43.2	49.2	43.2	49.2	43.2	49.2
Ramp Width (m)	RW	20	20	20	20	20	20
Overall Slope Height (m)	OSH	-	160	-	160	-	160
Overall Slope Angle (°)	OSA	-	47.6	-	48.4	-	48.4

The proposed slope design parameters presented in Table 23-17 is based on the bench design configuration approach applied. These are drawn in Figure 23-10 with annotation. Summarised key points for the design rationale are:

- West wall allowance for overburden, weathering, lower strength and frequent fracturing in the fenite altered granite (40m deep restricted to 10m high benches)
- East and south walls have the top 2 benches at 10m high maximum due to overburden and upper blocky weathered rock mass.
- Berm widths are marginally aggressive and simplified to 1m intervals for practicality and batch width loss by blasting and failure. These are to be revised if bench heights are changed.
- Allowance for one ramp at 20m wide in each slope as defined in the Mining Section 15 and adopted here.
- Overall slope angles derived from bench configuration and allowance for one ramp in each 160m high slope. This is the minimum requirement, and the resulting OVA and pit design may increase the number of ramps.
- Resulting overall angles at 160m depth vary from 47-49° to accommodate the bench and ramp-controlled slope angle.

SLOPE DESIGN CONFIGURATION

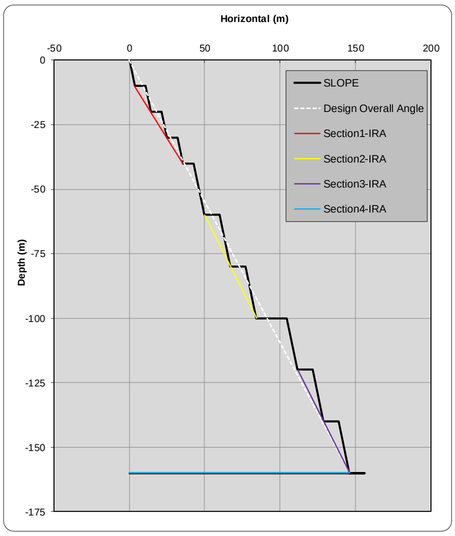
srk consulting

Client:	Leading Edge	Profile	
Project:	Norra Kärr	West - SRK 20m bench	
Date:	31 March 2021	Slope Azimuth Range:	45 to 135

Depth (m)	Bench Height (m)	Bench Angle (°)	Berm Width (m)	IRA (°)	Benches	Stack Angle (°)	Stack Height (m)	RAMP / GT Berm
Section 1								
0.0 -40.0	10	70	7	43.2	4	48.4	**40**	
Section 2								
-40.0 -100.0	20	70	10	49.2	3	55.1	**60**	20
Section 3								
-100.0 -160.0	20	70	10	49.2	3	55.1	**60**	
Section 4								
-160.0 -160.0	20	70	10		7	0.0	**0**	
Section 5								
-160.0 -160.0	-	-	-	-	-	0.0	**0**	
Design Overall Angle								
0.0 -160.0	47.6			**Slope Height (m)**			**160**	

COMMENTS:

Upper 40m expected weaker Fenite and weathered rock mass. Overall Slope height indicative to 160m



SLOPE DESIGN CONFIGURATION

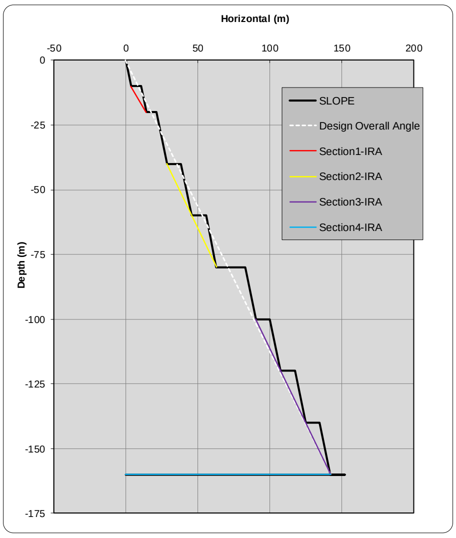
srk consulting

Client:	Leading Edge	Profile	
Project:	Norra Kärr	East - SRK 20m bench	
Date:	31 March 2021	Slope Azimuth Range:	225 to 300

Depth (m)	Bench Height (m)	Bench Angle (°)	Berm Width (m)	IRA (°)	Benches	Stack Angle (°)	Stack Height (m)	RAMP / GT Berm
Section 1								
0.0 -20.0	10	70	7	43.2	2	54.5	**20**	
Section 2								
-20.0 -80.0	20	70	10	49.2	3	55.1	**60**	20
Section 3								
-80.0 -160.0	20	70	10	49.2	4	53.5	**80**	
Section 4								
-160.0 -160.0	20	70	10		8	0.0	**0**	
Section 5								
-160.0 -160.0	-	-	-	-	-	0.0	**0**	
Design Overall Angle								
0.0 -160.0	48.4			**Slope Height (m)**			**160**	

COMMENTS:

Upper 20m expected blocky/weathered rock mass. Overall Slope height indicative to 160m



SLOPE DESIGN CONFIGURATION

srk consulting

Client:	Leading Edge	Profile	
Project:	Norra Kärr	South - SRK 20m bench	
Date:	1 April 2021	Slope Azimuth Range:	225 to 300

Depth (m)	Bench Height (m)	Bench Angle (°)	Berm Width (m)	IRA (°)	Benches	Stack Angle (°)	Stack Height (m)	RAMP / GT Berm
Section 1								
0.0 -20.0	10	70	7	43.2	2	54.5	**20**	
Section 2								
-20.0 -100.0	20	70	10	49.2	4	53.5	**80**	20
Section 3								
-100.0 -160.0	20	70	10	49.2	3	55.1	**60**	
Section 4								
-160.0 -160.0	20	70	10		8	0.0	**0**	
Section 5								
-160.0 -160.0	-	-	-	-	-	0.0	**0**	
Design Overall Angle								
0.0 -160.0	48.4			**Slope Height (m)**			**160**	

COMMENTS:

Upper 20m expected blocky/weathered rock mass. Overall Slope height indicative to 160m

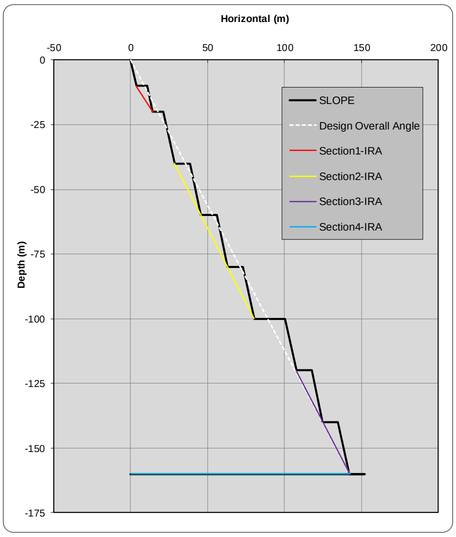
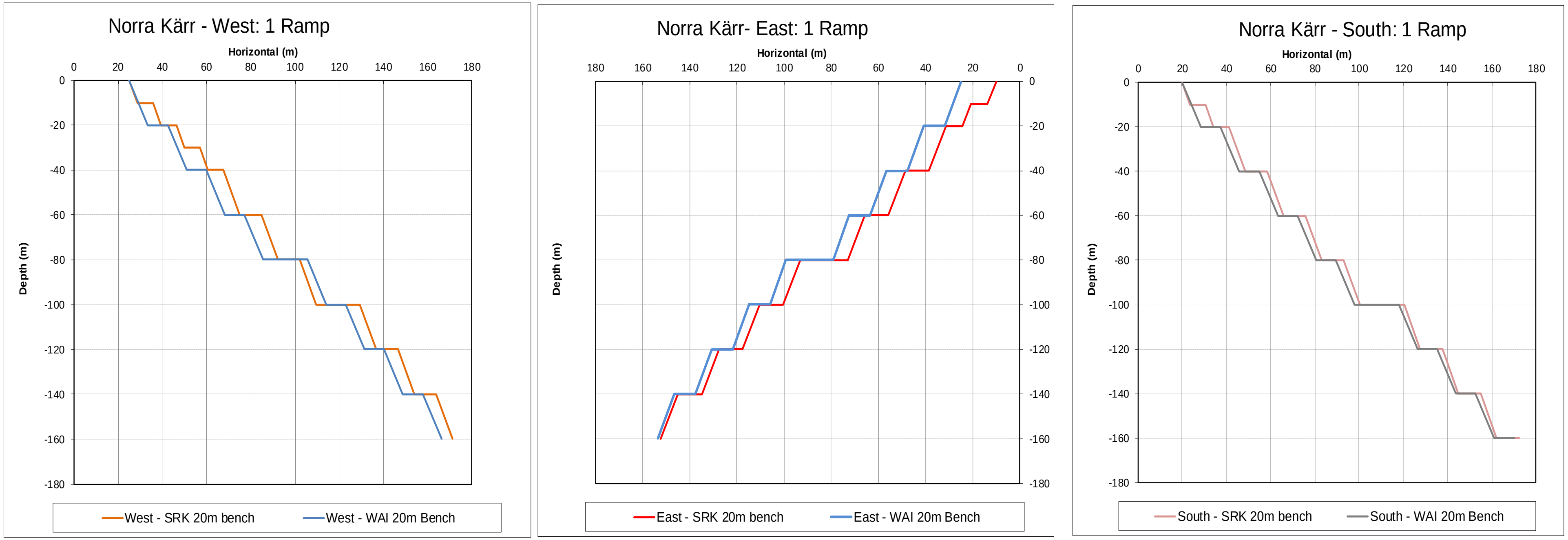


Figure 23-10: SRK detailed slope design components for the 3 main design sectors

A comparison between the WAI issued parameters and updated SRK parameters are displayed in idealised design cross-sections shown in Figure 23-11 for the 3 main design sectors.



	West - SRK 20m bench		West - WAI 20m Bench	
Depth	Upper 40m	40-160m	Upper	0-160m
Bench Face Angle	70	70	-	67
Berm Width	7	10	-	8.9
Bench Height	10	20	-	20
Inter-Ramp Angle	43.2	49.2	-	49.0
Ramp Width		20		19.9
Slope Height		160		160
Overall Slope Angle		**47.6**		**48.6**

	East - SRK 20m bench		East - WAI 20m Bench	
Depth	Upper 20m	20-160m	Upper	0-160m
Bench Face Angle	70	70	-	71
Berm Width	7	10	-	8.9
Bench Height	10	20	-	20
Inter-Ramp Angle	43.2	49.2	-	51.7
Ramp Width		20		19.9
Slope Height		160		160
Overall Slope Angle		**48.4**		**51.3**

	South - SRK 20m bench		South - WAI 20m Bench	
Depth	Upper 20m	20-160m	Upper	0-160m
Bench Face Angle	70	70	-	67
Berm Width	7	10	-	8.9
Bench Height	10	20	-	20
Inter-Ramp Angle	43.2	49.2	-	49.0
Ramp Width		20		19.9
Slope Height		160		160
Overall Slope Angle		**48.4**		**48.6**

Figure 23-11: Slope design parameter comparison between WAI 2014 and updated SRK 2021

23.4.6 Pit Design Review Comments

Pit optimisation is completed with overall slope angles around the deposit as a major input. The SRK approach is to derive these angles based on what practical bench-berm configurations are possible, and consideration of the access ramp system. This ensures that the pit design that fits into the shell meets as close as possible to the shell overall angles. SRK's design as documented in Section 15 displays that the slope positions closely match the optimisation shell. Example of this are shown in Figure 23-12.

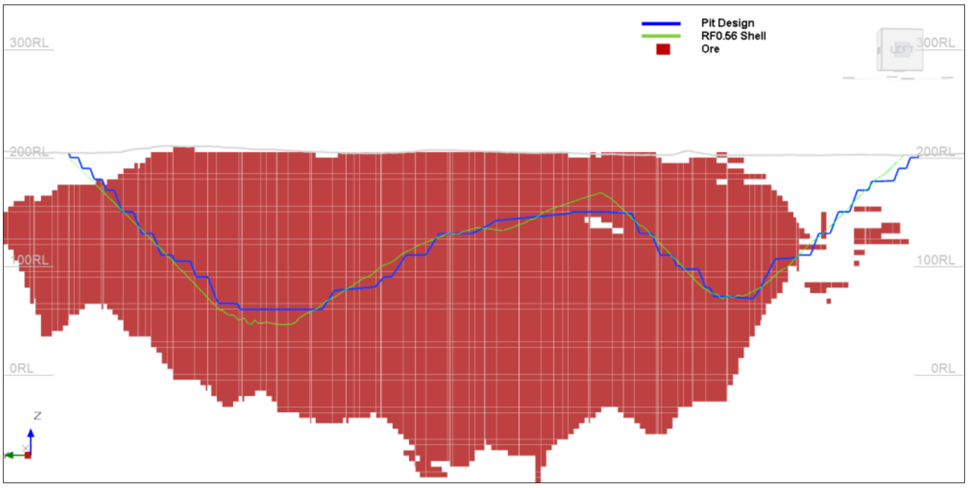


Figure 23-12: Slope design comparison with the pit optimisation shell

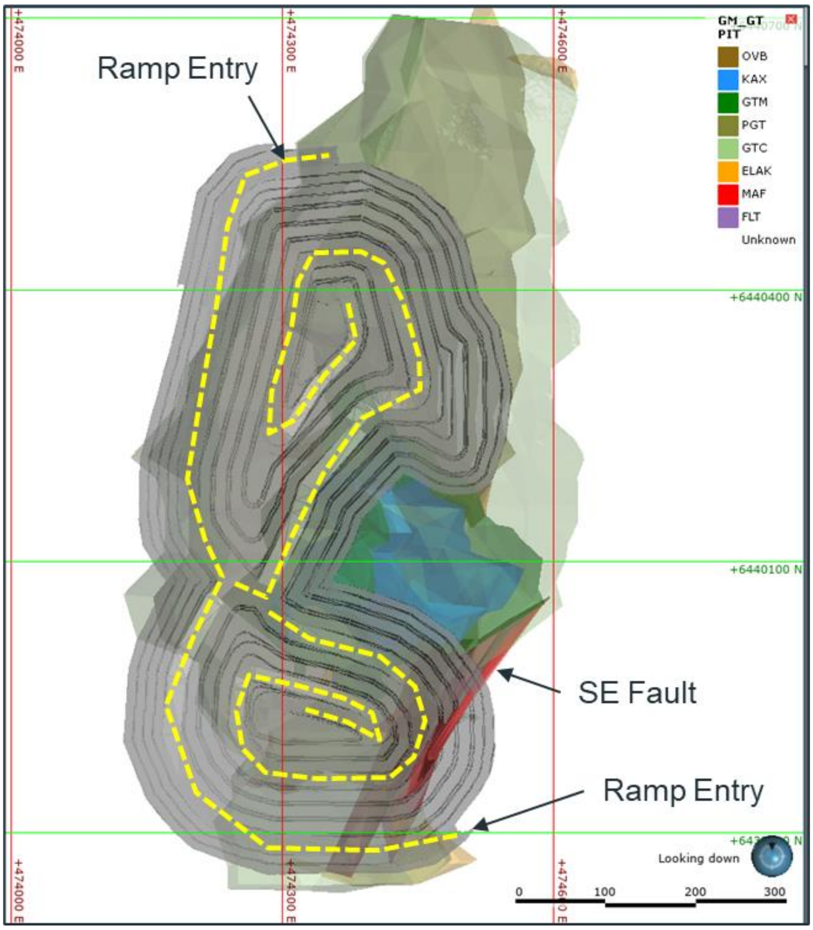


Figure 23-13: Slope design and ramp systems with main geology

This pit design is considered preliminary and will require geotechnical refinements in further studies. Points of note are:

- The Western wall has a 10m bench height restriction related to the expected weaker Fenite and this has been designed to the top 40m. Further analysis of the fracturing and weathering creating weaker rock mass needs to be done and this bench height restriction may be higher or lower than 40m. The ramp access is along these walls and risk to the ramp width reduction by wedge failure or multi-bench failure is to be assessed. The SW wall may have additional parallel fault zones to the SE fault (trending SW-NE), which could form multi-bench failure. This needs further assessment.
- The eastern slopes are all on the more competent Grennaite suite rock types. Kinematic analysis indicates some planar failure potential on the foliation. This needs to be further asses based on localised foliation orientation and dip changes. The wedge forming potential of localised structure patterns and not only on the average 3 sets identified, is required to be assessed. There is a high likelihood that wedge type blocky failure is possible as these conditions are common in the FW of foliated rock masses in Nordic operations.
- The Southern and Northern slopes may have increased wedge failure potential. The ramp systems enter the pit stages from both of these walls, so the risk to the ramp width reduction by wedge failure and multi-bench failure is to be assessed.
- The SE slope and ramp access will be influenced by the South East fault and has not been exclusively considered in this pit design. This is flagged as an area of specific detailed investigation in further pit design updates. Additional FEM numerical modelling is required testing scenarios of the location and distance to the slope that the fault is positioned. A conservative option is to cut back fully the SE wall to not have any portion of the fault intersecting the slope.
- The bullnose formation between the pit stages is a potential stability risk. These will need further investigation in later studies to assess if a larger catch width is required at the sharp points of these protrusions. Usually in blocky rock, the bullnose areas are unconfined on two sides and free to allow block movement compared to the confinement in long slopes.

23.5 Hydrogeology

23.5.1 Introduction

This part of the study addresses the water management aspects of the Norra Kärr project. Management of water at the mine is important both to meet make-up requirements, which include the process, dust control, wash down of plant and domestic use, and to limit the potentially adverse effects of run-off, groundwater flow and elevated pore pressures in the pit and its environs on the day-to-day operations of the mine.

23.5.2 Information Sources

The project benefits from earlier evaluations of the site dating back to the technical studies undertaken between 2012 and 2015. Many of the concepts described here draw heavily from this past work. The key documents from which the bulk of hydrological information has been obtained is listed below together with very brief summaries of relevant content:

- **Golder Associates AB, May 2014. Norra Kärr Hydrogeological Description (Translated Memorandum Hydrogeologisk Beskrivning):**

The memorandum contains a hydrogeological description of the project area for the preliminary economic assessment (PEA) (Runge Pincock Minarco, 2013). The memo describes the climate/meteorology and hydrogeology and, using analytical techniques predicts groundwater seepage to the pit and potential environmental impacts. The local formation hydrogeological properties were derived from literature sources and from pumping tests performed in existing drill holes at the project site. Surface run-off was calculated using the national hydrometric model (HYPE), but recharge was derived from [11]SKB's study site at Laxemar in Eastern Småland, which was considered analogous.

- **Golder Associates AB, August 2014. Well Installation for Aquifer Tests and Water Sampling at the Norra Kärr Prospect, Sweden:**

The memorandum documents a series of hydrogeological tests, using airlift, in boreholes targeted at fracture zones in the western and south-eastern areas of the project. The flow rates produced by the airlift indicated that the holes were suitable for step drawdown and constant rate pumping tests.

- **Wardell Armstrong International (WAI); Nov. 2014; Norra Kärr Preliminary Feasibility Hydrogeological Study:**

The report describes the construction and calibration of a preliminary, steady state numerical model for predicting groundwater inflows to the Norra Kärr open pit. These inflows were simulated for each of the designed pit pushback years and the results were then included in the site water balance.

SRK also note that WAI completed and reported on a preliminary geotechnical stability assessment (Nov. 2014) to define appropriate open pit slope angles for the 20 years mine life. This used the theoretical drawdown data from the 2012 Golder hydrogeological study to inform the stability modelling but was prepared before the results of the WAI groundwater modelling were available, so is quite heavily caveated.

- **GBM; July 2015; Amended & Restated Prefeasibility Study - NI 43-101 - Technical report for the Norra Kärr Rare Earth Element Deposit:**

23.5.3 Climatic and Geohydrological Setting

The climate local to Norra Kärr is defined as humid continental and is typified by large seasonal temperature differences, with warm to hot summers and cold winters.

According to earlier studies, there is no weather station on site and hence reliance is placed on data sourced from nearby meteorological stations located between 5km and 30km from the site. These reveal that average monthly temperatures peak around 16°C in July and attain a minimum of around -3°C in February. Mean annual precipitation (MAP) at the nearest station, Högemålen, some 5km south of the project site, is 632mm with the wettest months being July and August.

[11] Svensk Karnbranslehantering Aktiebolag (Swedish Nuclear Fuel and Waste Management Company)

The project area is located at approximately 210m above sea level (mASL) some 1.5km to the east of Lake Vattern. The ground elevation declines from the north towards the south, and from west to east towards Lake Gyllinge, which is approximately 0.85km east of the site. The pattern of stream channels also suggests that run-off from the site drains in these directions. Ultimately, Lake Gyllinge drains in a northwesterly direction towards Lake Vattern, north of the project area.

The deposit is overlain by a discontinuous layer of overburden, occasionally interspersed with rocky outcrops. The overburden thickness ranges between 0 and 11m across the site and consists of sandy soil, glacial till and peat in the marshy hollows. The sporadic nature of the cover and its composition probably mean that recharge to the bedrock on the upper ground is unhindered; only in the topographic depressions where there are accumulations of finer soil and peat is it likely there will be reduced recharge and ponding. Recharge was estimated by WAI to be 10-15% of MAP (63-95mm) and was based on knowledge from similar hydrogeological environments.

Norra Kärr is a lopolithic nepheline syenite intrusion. The intrusion has a concentric layering pattern comprising, from the centre outwards kaxtorpite, migmatitic grennaite, pegmatitic grennaite and fine grained grennaite. The deposit is structurally controlled with three phases of deformation with shallow dipping foliations striking E-W (D1), a synformal depression striking N-S (D2) and E-W or NE-SW trending kink folds (D3). The deposit has two large-scale structural features:

i). a fault to the southeast that strikes SW-NE, dipping 70 degrees to the west, and which comprises a 1.5 – 3m zone of heavily crushed rock that acts as a contact to the intrusion. This fault trends towards to Lake Gyllinge and may therefore provide a hydraulic link between the two; and

ii). an alteration zone to the west that is fractured and, in places completely decomposed; this structure is influenced by the larger regional shear zone.

The heterogeneity in rock type and geological structures is likely to impart a similar variation in hydrogeological characteristics at the site. The extent to which the system is hydraulically connected, or compartmentalised is poorly understood. However, the current thinking is that the hydraulic conductivity of the rock mass is very low with most of the flow confined to major structures such as the fault in the SE corner of the deposit.

Golder evaluated the hydraulic properties of the igneous rock at the site, drawing information from literature sources and by conducting hydrogeological testing. Broadly, the results point to a fractured crystalline rock with K values ranging between 0.2 m/d and 0.5 m/d respectively on the fringes and in the core of the lopolith, reducing to 0.005 m/d in the surrounding host granite and increasing to as much as 3 m/d in the geological structures.

The groundwater divide is considered to exist just to the west of the site close to the E4 highway, with steep hydraulic gradients through the low K host granite west towards Lake Vättern and east towards Lake Gyllinge. The more elevated K in the Norra Kärr lopolith will likely mean that groundwater gradients are slightly shallower. Given the fault in the SE of the site may extend in the direction of Lake Gyllinge it is possible that this fault acts as a conduit for groundwater discharging near the lake. It has been pointed out by WAI that the role of this fault may be reversed during the operational phase of the mine when the lake could act as constant pressure source, supplying water to the pit. However, this will remain conjectural until this structure has been properly evaluated.

WAI developed a numerical groundwater model for the PFS Groundwater Vistas v4.0 (MODFLOW) using the geological data and hydraulic parameters obtained from the site investigations conducted by LEM and by Golder Associates. The model was subdivided into 5 hydrogeological domains, namely the orebody, the host granite, the 'southeast fault', and an inferred north-south trending fault. Each domain was assigned a horizontal K ranging between 0.2 and 3 m/d, with the lowest conductivity value given to the granite and the highest to the fault in the SE corner of the pit. To simulate inflow to the open pit for each year of the 20-year life of mine, WAI produced 8 steady state models. These models predicted that inflow would steadily increase during the period of mining from 3.8 L/s (Year 3) to peak at 50.2 L/s in the last year of operation (Year 20).

23.5.4 Minewater Management Design Concepts

The following assumptions apply in respect of managing water at the Norra Kärr mine site:

- The pit dewatering schedule used for the present design concept assumes the mine will operate for 26 years, rather than the 20 years originally anticipated for the PFS.
- The present pit water management design uses the schedule of pit inflows produced by the WAI groundwater model but extends the period of peak flow (50.2 L/s) by an additional 6 years to 26 years. These flow rates and the pit depth determine the pipe size, pump power and capacity and pond dimensions. Note the comments in Section 23.5.6 on project risks, which point out concerns regarding the accuracy of the pit inflow rates produced by the WAI model.
- Most of the rock mass to be mined is strong enough not to require active pit dewatering using peripheral wells and gravity drains. However, the poor ground associated with the fault in the SE corner of the pit and, possibly some parts of the west wall where fenitised granite will be exposed may require some gravity drains to lower in-situ pore pressures. Otherwise, pit water will be managed using sumps and pumps located in the base of the pit and potentially at intermediate stations on upper benches, once the pit deepens.
- Surface run-off towards the pit and waste facilities from the surrounding catchment will be diverted using berms and ditches positioned around the periphery of these structures.
- The impact of pit dewatering on the surrounding aquifers will be monitored using a combination of standpipe wells and VWPs installed both inside and beyond the periphery of the pit.

With the exception of Year 1, Stage 1 of pit development, when there will be one duty pump and one sump, it is estimated there will be a requirement for two duty pumps and sumps for the entire 26 year period of operation, one for the southern pit and one for the northern one. Each sump shall also have a stand-by pump to cover periods of increased flow, usually associated with the spring thaw (freshet) at the end of winter. Each sump and pump shall have a rising main to lift the water out of the pit to a settling pond located just beyond the periphery of the excavation. Based on the initial assessment the majority of water supply will come from site collected water comprising of rainfall and pit dewatering. Additional makeup water will be required during operations from lake Våttern. Discharge of water from the site is unlikely but in the event of excess rain there maybe need to discharge rainwater. This will be non-contact water that can will be discharged to the catchment to the east. Any discharge water would pass through sediment lagoons and bio-filtration reedbed to ensure acceptable water quality. For the majority of the operational period no discharge is currently predicted although a detailed water balance will be required in the next phase of work to confirm this.

The pit dewatering design assumes that skid mounted centrifugal pumps will be deployed next to the sumps, capable of lifting water against a head of between 20m and 160m, depending on the stage of the project.

The rising main will comprise PN16 HDPE pipe with an internal diameter of 150mm to limit excessive friction head. The pipe length is estimated to be of the order of 800m to allow for the transmission of water from the pit to settling ponds and then on to the local water course.

The effect of dewatering on pore pressures in the rock mass around the pit and the environmental impact it has on more distant aquifer water levels will be measured using a combination of grouted vibrating wire piezometers (VWP) and standpipe wells arranged around the site. For budgeting purposes, it is assumed there will be a requirement for up to 3 VWPs drilled to the full depth of the final pit and a further 8 standpipe wells at various locations across the wider project site.

As with the PFS, water supply cannot depend on pit inflow (although this should be used to keep reliance on external water supply to a minimum) but should come from a more reliable source, ifor the purpose of this PEA, Lake Vattern. The supply assumes a lake-side offtake comprising a duty and a stand-by pump, each capable of pumping up to 13 L/s (which is the on-site process make-up requirement for the wet magnetic separation) and against a pressure head of 160m, the latter to cover ground elevation increases and friction loss in the pipe. The pipeline to convey the water to the site should be buried to a nominal depth of 1-2m and should comprise approximately 1900m of PN16 HDPE pipe with an internal diameter of 150mm. The pumping rate assumes that water for magnetic separation is recirculated, so long as water quality is adequate for the process. The water does not get discharged to the environment, but will require monitoring and treatment / settlement, for recycle in the process. Other alternatives to the lake that will be assessed include groundwater boreholes and abstraction of makeup water from Lake Gyllinge to the east.

23.5.5 Project Costs

The costs associated with the construction and operation of the water management infrastructure over the course of the mine's life are itemised in the Section 20. In terms of the present study, water infrastructure costs pertain to pit dewatering, groundwater monitoring and water supply.

23.5.6 Project Risks and Opportunities

The main water management risks and opportunities are considered as follows:

- The WAI groundwater model was developed in steady state and therefore does not account for changes to the storativity of water in the rock mass around the pit over time. Steady state models do not factor-in storage depletion and therefore may over-state long term inflow to the pit. In reality, there are few mine dewatering schedules that show a steady rise of inflow during the entire period of operation, but rather there is a flattening-off of the rate with inflow becoming increasingly a function of aerial recharge. Exceptions to this rule occur when the mine works continuously along strike, opening-up new, shallow areas of rock mass and new stores of water e.g. stratiform deposits and long wall mining of flat coal seams. The risk of lower inflow rates means that there should not be a complete reliance on pit inflows for on-site processing needs (wet magnetic separation). The potential upside is that lower inflows will likely mean lower operational costs and fewer concerns regarding treatment and if required disposal to the environment.
- Lake Vättern is regarded as the most logical source of make-up water for the mine. However, there could be some significant obstacles to realising this objective, including off-take permits from the authorities and land access and planning permission for the development of the pipeline across private land and a public highway.

The PFS hydrogeological model considered there might be a hydraulic connection between the pit and Lake Gyllinge via the fault located in the SE corner of the pit. An elevated K and continuous hydraulic link between the lake and the pit could give rise to much higher flow rates in the pit and mean that greater care is required to lower pore pressures in the structurally weaker portions of the eastern pit wall. Gyllinge could supply water to the mine but it is a small lake that is used by the local community for recreational purposes. Its catchment is small and would be particularly vulnerable during extended dry / drought periods when recharge would be expected to be negligible and very likely exceeded by evaporation losses. Gyllinge was not considered in earlier studies, SRK has chosen to stick with Vattern as a source for the same reasons at present, however further evaluation is required. Whilst Vattern is a Natura 2000 protected lake, it is vastly bigger.

24 INTERPRETATION AND CONCLUSIONS

24.1 Mineral Resources

A block model was generated and grades of each separate REE elements, zirconium, niobium, uranium and thorium were estimated into lithological domains separately. Variography was completed on the domained drillhole data and ordinary Kriging interpolation methodology was chosen for the grade estimate. The grades were validated using check estimates along with visual and statistical methods. No mining has been completed with which to verify the model through reconciliation. Density values were assigned to the model as an average per lithological unit in order to report tonnages.

Table 24-1: Norra Kärr Mineral Resource Statement (SRK, 18 August 2021)*

Mineral Resource Classification	Tonnes (Mt)	TREO (%)	ZrO_2 (%)	Nb_2O_5 (%)	Nepheline Syenite (%)
Inferred	110	0.5	1.7	0.05	65

**Notes:*

1. *Effective date 18 August 2021.*
1. *Qualified Person Mr Martin Pittuck MSc C.Eng*
2. *Mineral Resources are not Mineral Reserves until they have Indicated, or Measured confidence and they have modifying factors applied and they have demonstrated economic viability based on a Feasibility Study or Prefeasibility Study.*
3. *The Mineral Resources reported have been constrained using an open pit shell assuming the deposit will be mined using open pit bulk mining methods and above a cut-off grade of USD150/t., including a 30% premium on projected commodity prices and unconstrained by commodity production rates and the 260m highway buffer zone.*
2. *The Mineral Resources reported represent estimated contained metal in the ground and has not been adjusted for metallurgical recovery.*
3. *Total Rare Earth Oxides (TREO) includes: La_2O_3, Ce_2O_3, Pr_2O_3, Nd_2O_3, Sm_2O_3, Eu_2O_3, Gd_2O_3, Tb_2O_3, Dy_2O_3, Ho_2O_3, Er_2O_3, Tm_2O_3, Yb_2O_3, Lu_2O_3, Y_2O_3. Speciation of TREO is given in Table 13-11*
4. *Heavy Rare Earth Oxides (HREO) include: Eu_2O_3, Gd_2O_3, Tb_2O_3, Dy_2O_3, Ho_2O_3, Er_2O_3, Tm_2O_3, Yb_2O_3, Lu_2O_3, Y_2O_3.*
5. *HREO is 52% of TREO*

Table 24-2: Norra Kärr Rare Earth Element Distribution

Light REO proportion of Total REO (%)					Heavy REO proportion of Total REO (%)									
La_2O_3	Ce_2O_3	Pr_2O_3	Nd_2O_3	Sm_2O_3	Eu_2O_3	Gd_2O_3	Tb_2O_3	Dy_2O_3	Ho_2O_3	$Er2O_3$	Tm_2O_3	Yb_2O_3	Lu_2O_3	Y_2O_3
0.100	0.210	0.030	0.110	0.030	0.004	0.030	0.007	0.050	0.010	0.034	0.005	0.033	0.005	0.340
0.48					0.52									

24.2 Metallurgical Testwork and Process Recovery

Process evaluation has focused on the following approaches:

- Magnetic separation in two stages to separate eudialyte from the bulk of the mineralized rock. The bulk of this residue is nepheline syenite which will be sold as an industrial mineral. Second stage magnetic separation will separate fine eudialyte from aegirine that will be a waste on site but to be investigated for potential commercial use cases.
- Two stage sulfuric acid leaching to separate Zr, Nb and TREO from eudialyte.
- Separation and recovery of Zr, Nb and TREO into saleable products
- Future work will look at the feasibility of utilizing hydrochloric acid in place of sulfuric in a closed circuit with nanofiltration recovery of acid and also potential to recover Hf.

Additional work on flotation has also been undertaken and whilst beneficial it would require use of flotation chemicals on site and magnetic separation does not.

The process will be undertaken at two sites;

- At Norra Kärr, primary and secondary crushing followed by two stage magnetic separation. No reagents will be used in this process. A concentrate will then be shipped for further processing off site
- For the purpose of this PEA, at Luleå in a brownfields industrial area the leaching separation in a hydrometallurgical facility will be undertaken along with metal separation and recovery.

Zr will be produced as chemical grade zirconium oxide, Nb as niobium oxide and the Rare Earth Elements as a Rare Earth Element mixed oxide concentrates for further separation. Nepheline syenite will be produced at the mine site as an industrial mineral. In future work the potential to generate Hafnium as a by-product should be evaluated once a resource is estimated. Also, the Luleå processing facility will produce approximately 2 million tons of gypsum over the estimated life of project. This may also be a potential by-product.

24.3 Mining Methods

The revenue factor 56% pit shell was selected as the basis for pit design resulting in 29.3 Mt of plant feed and 9.4 Mt of waste for a strip ratio of 0.32. The mine schedule produces 1.15 Mtpa of plant feed for some 26 years.

SRK has identified the following mining opportunities to the project:

- Sweden has a relatively low power cost which would prove beneficial to the project. Any opportunity to identify electric equipment would likely result in a reduced mining cost.
- In addition, electricity production in Sweden comes mainly from hydropower and nuclear, with an increasing build-out of wind power, all low carbon footprint alternatives. An all-electric mining fleet would hence allow the production from Norra Kärr to carry a minimal carbon footprint.

24.4 Environmental and Social Liabilities & Risks

- SRK is aware there is a vocal opposition locally and nationally regarding environmental concerns attributed to the Project and significant effort will be required to ensure all potential negative impacts are assessed, avoided, minimised and/or mitigated. SRK notes that the decision by the SAC to revoke the Mining Lease was the first based on the grounds of the EIS/MKB not considering ancillary areas required for mining; precedent before this case was that the MKB for Mining Lease applications consider only the area under consideration for the Mining Lease itself.
- SRK expects that the timescales for determination of all authorisations for this project will be extended to the limit, if not beyond normal regulatory timescales.
- The Project area is ecologically sensitive, with locally, nationally and internationally important sites nearby. Adequate mitigation measures and habitat restoration will be required during and post operations.
- Geochemistry tests will be required to complete initial characterisation of the waste material and to inform the future waste management plans for the Norra Kärr and Luleå sites. These tests may have long lead times and should be initiated as soon as practicable. This will also help inform the requirements for any water treatment facilities.
- The hiatus in the project development and associated community engagement may result in increased speculation and the spread of misinformation about the project. It may also result in increased local opposition affecting the Project's acceptance and a lengthy legal process to acquire the required land.
- SRK considers the closure cost estimate calculated in 2014 as low. Although a reduced project footprint may help mitigate closure costs, this will need to be confirmed as part of the next phase of the project. These include post closure water quality requirements, visual impact considerations and pit backfilling potential. Social; transitioning should also be considered as part of the overall closure cost.

24.5 Infrastructure & Logistics

Within the study infrastructure requirements and associated costs have been defined to an appropriate level of detail. Possible locations for infrastructure have been previewed. The project location is well situated with nearby national road, rail, and power infrastructure. A number of critical assumptions have been made in order to form a basis for the study and costs such as the location of the road to rail loading sidings (for Eudialyte transport), the location of the off-site processing facility, the assumptions around the off-site processing facility in respect of the provision of services and transport connections, and the requirements for bulk power supply, which will all be revisited in the next stage of study.

24.6 Economic Analysis

The economics of the Project are robust using the parameters detailed herein. The inclusion of co-product revenue from Zr, Nb and nepheline syenite products enhance the Project from both an economic and resource utilization perspective. Rare earth pricing is more complex compared to other commodities and is opaquer and the Project is most sensitive to commodity price changes but can absorb a 30% reduction in prices and still remain economically viable. The results of this PEA indicate that the Project is worthy of progressing to the next stage of study and assessment.

25 RECOMMENDATIONS

25.1 Resource Evaluation

Duplicate analysis and QA-QC work is required for Zr, Nb and Hf. Additional analysis to determine appropriate parameters to define in the resource Nepheline is also required and this needs modelling to optimise the estimate.

25.2 Mining Methods

- SRK has identified the following recommendations for the next phase of the project:
- Construction of a bund wall using mine waste would not have a significant impact of the mine operating costs, as haulage to this point would likely be equal to or shorter the distance to the external waste dump.
- Review loss and dilution quantities for larger block sizes.
- Engage with equipment manufacturers to investigate the potential availability in the near future for electric equipment at the small size required at Norra Kärr.
- Review the potential for a second mine operating shift to identify any potential cost savings.

25.3 Processing

The next phase of work needs to re-confirm estimates used on all stages in the proposed process and commodity recoveries. This will require on the collected bulk sample, comminution, magnetic separation (two stage approach), leaching testwork, solvent extraction and precipitation testwork to produce the final products for assay.

In addition, variability testing of the different ores is required to determine if any detrimental characteristics, elements or minerals exist in the different ore types that might influence process design.

For separation of Zr, Nb and Hf further work is required on solvent extraction and optimum conditions for uptake and stripping.

For the TREO work is required to determine the merit of establishing further separation and product of “final” product rare earth elements versus the current proposed mixed TREO phases.

In the case of Nepheline Syenite work is required to confirm quality of the project in order to determine valuation.

IN the case of the waste streams potential exists for the aegirine waste to be utilized in high temperature cements and the gypsum in plaster board. Both require demonstration studies to determine quality of the products in order to provide valuation.

25.4 Infrastructure & Logistics

In respect of infrastructure and logistics, and in addition to the typical infrastructure design scopes of work, key recommendations for studies to inform a PFS study are as follows:

- Undertake PFS level geotechnical investigations to inform design and costings at all sites;

- Determine the location for the off-site processing facility and the rail load-out facility (for Eudialyte);
- Discuss the rail load-out facility options with existing rail logistics providers, who may be able to provide a third-party supplier option;
- Within the discussion around the rail loading sidings, the Project team must engage with the Swedish rail authorities around timing and connection to the network for any new siding construction and the governance process around this, which can be time consuming;
- Invite discussions with logistics providers for consolidating road and rail transportation;
- Investigate alternative logistics solutions like road and sea freight;
- Investigation of up to three access road alignments to a PFS level of detail in order to facilitate environmental, social and governance surveys and inputs.
- Consider an off-site location for nepheline syenite storage and customer collection. This would assist in controlling road transport and other traffic impacts between the site to the main highway. Some preliminary (low level) traffic census studies on key roads would be useful;
- Power supply connecting points (i.e. existing substations) need to be confirmed as transmission line options. For this, the relevant utility companies and Svenska kraftnat need to be consulted. The estimated peak average demands and consumption will be needed.
- The local utility needs to be consulted to inform the electricity tariffs (all locations), which needs to incorporate any transmission charges and grid charges, taxes, and exemptions.
- Revisit and update the logistics options and costs once the locations for the off-site processing facility and road to rail loading sidings are determined.

25.5 Environment Social Governance

Leading Edge Materials should ensure that ESG factors are considered in the assessment and selection of project design alternatives, particularly the siting of infrastructure, water supply and mining and processing trade-off studies. Early ESG input can maximise opportunities for stakeholder engagement and avoiding key impacts and risks on the surrounding environment. This will require two-way communication between the project engineers and environmental and social specialists. Key recommendations include:

- Assess all opportunities for climate change considerations to be embedded in the project design. Design alternatives and option selection should take into consideration energy efficiency, energy supply, water use and project footprint to demonstrate the lowest practical carbon intensity for the overall project design. The project should look to commit to a 'net zero' carbon footprint.
- Other factors likely to be important for the future Environmental and Social Impact Assessment ("ESIA") will be landscape and visual assessments of the various project components. The risks and opportunities need to be considered in light of increased focus on key receptors and viewed from the perspective of environmental and human rights.
- Detailed studies of wastes to be generated and stmineralized rock d both at the Norra Kärr location and at the Luleå facility will be required. For waste material that meets the criteria of extractive wastes, a Waste Management Plan will be required, as will permitting of an Extractive Waste Facility.
- Detailed biodiversity mitigation and management measures are recommended to demonstrate a net positive impact from the project in the long term. As a minimum the project will have to demonstrate that there will be no significant environmental effects on the EU protected habitats in the vicinity of the project. A detailed Biodiversity Action Plan is a likely requirement as part of the final suite of management plans arising from the ESIA commitments.
- Local and national level stakeholders should be identified and mapped, appropriate engagement methods identified, and a stakeholder engagement strategy developed. Measures should be employed to improve local community's understanding and awareness of the project (including the positive and negative impacts of the project) through regular interactions and various methods of communication including newsletters and local media.
- Stakeholder engagement and meetings should be recorded and documented. Issues and concerns raised need to be formally documented, progress tracked, and a commitment made to feedback to the communities on these issues. This process can help improve the understanding of the positive and negative impacts on the social environment.
- A formal grievance process should also be developed and implemented in line with the UN Guiding Principles of Business and Human Rights. A formal grievance register should be kept with clear documentation on the grievance made, the steps taken to resolve the grievance and an option for third party resolution for any unresolved disputes.
- The anti-mining sentiment indicates a need for specific consideration on human rights, multi-stakeholder engagement platforms with open and transparent communication and dialogue, combined with increased capacity to mitigate any ongoing community opposition.

25.6 Economic Analysis

For the next phase of the Project SRK recommends the following is undertaken:

- Further design and cost estimation work to improve the accuracy of the capital and operating costs; and
- An updated market study is generated for each of the products assumed to be produced as well as letters of intent for offtake agreements where possible.

For and on behalf of SRK Consulting (UK) Limited

Rob Bowell,
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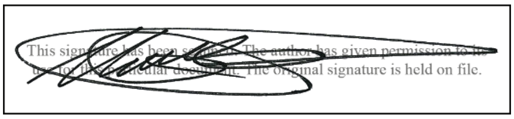

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26 REFERENCES

Ashley, P.M. (2009). Petrographic report of five rock samples from Norra Kärr, Jonkoping region, Sweden.

Adamas Intelligence, 2020. Rare Earth Magnet Market Outlook to 2030. Published in 2020 and updated in 2021.

Adamson, O.J. (1942) Eckermannite, a new alkali amphibole. Geologiska Föreningens i Stockholm Förhandlingar, 64, 329-334.

Adamson, O.J. (1944). The Petrology of the Norra Kärr District, An Occurrence of Alkaline Rocks in Southern Sweden.

ANSTO 2013a Norra Karr Project-Scoping Leaching testwork program. C1333. Report to Tasman Metals Ltd. June 2013. 142p.

ANSTO 2013b Norra Kärr Project – Single Stage Leach-Partial Neutralisation Note 1. Report to Tasman Metals Ltd. September 2013. 29p.

ANSTO 2013c Norra Kärr Project – Stage 2: Literature Review – Zirconium Solvent Extraction from Sulfate Solutions Note 2. Report to Tasman Metals Ltd. September 2013. 14p

ANSTO 2013d Norra Kärr Project – Norra Kärr Project Update - Zirconium Solvent Extraction with Alamine 336 Note 3. Report to Tasman Metals Ltd. September 2013. 15p

ANSTO 2014 Rare Earth Recovery flowsheet definition – Solvent Extraction and concentrate scoping test work program. C1383. Report to Tasman Metals Ltd. July 2014. 115p.

Arzamastsev, A., Yakovenchuk, V. & Ivanyuk, G.L. (2008). The Khibini and Lovozero alkaline Massifs: Geology and unique mineralization: 33 ICG excursion No. 47, July 22--August 2, 2008.

Asian Metal Ltd. 2021. Niobium Industry Annual Report 2020 and historical price series.

Atanasova, P., Marks, M.A.W., Heining, T., Krause, J., Gutzmer, J., and Markl, G. (2017) Distinguishing magmatic and metamorphic processes in peralkaline rocks of the Norra Kärr complex (Southern Sweden) using textural and compositional variations of clinopyroxene and eudialyte-group minerals. Journal of Petrology, 58, 361-384

Balomenos, E., Davris, P. Deady, E., Yang, J., Panias, D., Friedrich, B., Binnemans, K., Seisenbaeva, G., Dittrich, C. & Kalvig, P. (2017). The EURARE project: development of a sustainable exploitation scheme for Europe’s rare earth mineralized rock deposits. Johnson Matthey Technology Review 61:142–53.

Bansal, N. P. 1990. Influence of several metal ions on the gelation activation energy of silicon tetraethoxide. Journal of the American Ceramic Society 73 (9):2647–52.

Balinski, A., P. Atanasova, O. Wiche, N. Kelly, M. Andreas Reyter, and C. Scharf. 2019. Recovery of REEs, Zr(Hf), Mn and Nb by H2SO4 leaching of eudialyte concentrate. Hydrometallurgy 186:176–86.

Binnemans K and P. T. Jones, J. Sustain. Metall., 2015, 1, (1), 29-49, 186

Blaxland, A.B. (1977). Agpaitic magmatism at Norra Kärr Rb-Sr isotopic evidence: Lithos, volume 10, issue 1, pp. 1-8.

British Geological Survey (“BGS”) (2011). Rare Earth Elements. Commodity Profile generated by the BGS. Online only: http://www.bgs.ac.uk/mineralsUK/statistics/mineralProfiles.html.

Castor, S.B. & Hedrick, J.B. (2006). Rare Earth Elements: Industrial Minerals & Rocks, Commodities, Markets and Uses, 7th Edition, Society for Mining, Metallurgy and Exploration, Inc. (SME).

Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") (2010) Definition Standards for Mineral Resources and Mineral Reserves. [Online] 27 November 2010.

Canada's Mineral Production, Preliminary estimates 2019, 2019.

CRC (2016-2017) Abundance of elements in the Earth's crust and in the Sea, CRC Handbook of Chemistry and Physics, 97th edition (2016–2017), p. 14-17

Chanturia, V. A., E. L. Chanturia, V. G. Minenko, and A. L. Samusev. 2019b. Acid leaching process intensification for eudialyte concentrate based on energy effects. Obogashchenie Rud 3:29–36.

Chanturia, V. A., I. Z. Bunin, M. V. Ryazantseva, E. L. Chanturia, A. L. Samusev, E. V. Koporulina, and N. E. Anashkina. 2018a. Intensification of eudialyte concentrate leaching by nanosecond high-voltage pulses. Journal of Mining Science 54 (4):646–55.

Chanturia, V. A., I. Z. Bunin, M. V. Ryazantseva, E. L. Chanturia, A. L. Samusev, E. V. Koporulina, and N. E. Anashkina. 2019a. Intensification of eudialyte concentrate leaching by nanosecond high-voltage pulses. Journal of Mining Science 54:646–55.

Chanturia, V. A., V. G. Minenko, A. L. Samusev, E. L. Chanturia, and E. V. Koporulina. 2017. The mechanism of influence exerted by integrated energy impacts on intensified leaching of zirconium and rare earth elements from eudialyte concentrate. Journal of Mining Science 53 (5):890–96.

Chanturia, V. A., V. G. Minenko, A. L. Samusev, M. V. Ryazantseva, and E. V. Koporulina. 2018c. Influence exerted by ultrasound processing on efficiency of leaching, structural, chemical, and morphological properties of mineral components in eudialyte concentrate. Journal of Mining Science 54 (2):285–91.

Chizevskaya, et al.1994 Non – Traditional methods of treating high – Silicon containing are elements. Hydrometallurgy'94: papers presented at the international symposium 'Hydrometallurgy '94' organized by the Institution of Mining and Metallurgy and the Society of Chemical Industry, and held in Cambridge, England, from 11 to 15 July, 1994. Springer, Netherlands, Dodrecht, pp. 219–28.

City Population: https://www.citypopulation.de/en/sweden/admin/06__j%C3%B6nk%C3%B6ping/ (accessed Feb 2021)

Davidson, T., Short, M., Moseley, G., Mounde, M. & Digges La Touche, G. (2015a). Prefeasibility Study - NI 43-101 - Technical report for the Norra Kärr Rare Earth Element Deposit. NI 43-101 Technical Report prepared by GBM Minerals Engineering Consultants Ltd on behalf of LEM Metals Ltd.

Davidson, T., Short, M., Moseley, G., Mounde, M. & Digges La Touche, G. (2015b). Amended & Restated Prefeasibility Study - NI 43-101 - Technical report for the Norra Kärr Rare Earth Element Deposit. NI 43-101 Technical Report prepared by GBM Minerals Engineering Consultants Ltd on behalf of LEM Metals Ltd.

Davris, P., S. Stopic, E. Balomenos, D. Panias, I. Paspaliaris, and B. Friedrich. 2017. Leaching of rare earth elements from eudialyte concentrate by suppressing silica gel formation. Minerals Engineering 108:115–22

Davris P, E. Balomenos, D. Panias and I. Paspaliaris, Hydrometallurgy, 2016, 164, 125 20. P. "Report on Critical Raw Materials for the EU", Report of the Ad Hoc Working Group on Defining

Critical Raw Materials, May, 2014, Ares(2015)1819503, European Commission, Brussels, Belgium, 29th April, 2015 EURARE WP1-REE Market Analysis Report, 2016 (internal project report)

Eckermann, H. (1968). New contributions to the interpretation of the genesis of the Norra Kärr alkaline body in southern Sweden: Lithos 1, pp 76-88.

European Commission, Study on the EU's list of Critical Raw Materials – Final Report (2020) Joint Research Centre Directorate GROW.C | JRC.D, Brussels.

Fryer, B. & Edgar, A. (1977). Significance of Rare-earth distributions in coexisting minerals of peralkaline saturated rocks. Contrib. Mineral Petrol. 61, 35-48.

Forrester, K., M. Leijd, M. Oczlon, H. Holmström, and M. Saxon 2014, "Beneficiation of rare earth element enriched eudialyte from the Norra Kärr peralkaline intrusion with wet high intensity magnetic separation." In Conference of metallurgists, Canadian Institute of Mining, Metallurgy and Petroleum, Vancouver.

Gates, P.A., Horlacher, C.F. & Reed, G. (2012). Preliminary Economic Assessment NI 43-101 Technical Report for the Norra Kärr REE Zirconium Deposit, Gränna, Sweden. NI 43-101 Technical Report prepared by Pincock Allen and Holt on behalf of LEM Metals Ltd.Golder (2013). Norra Kärr Materialkaraktarisering av gråberg [Material Characterisation], Golder Associates 30th August 2013.

Golder Associates (2014) LEM PFS – Supplemental site visit, Environment and Social Scope

Högdahl, K., Andersson, U.B. & Eklund, O. (eds.), Andersson, U.B. (2004). The Transscandinavian Igneous Belt (TIB) in Sweden: a review of its character and evalution. Special Paper 37, s.l. :Geological Survey of Finland, 2004.

Holtstam, D. (1998) Jinshajiangite from the Norra Kärr alkaline intrusion, Jönköping, Sweden. Geologiska Föreningens i Stockholm Förhandlingar, 120, 373-374.

Hong NT, Lee M-S 2018 Review on the Separation of Niobium and Tantalum by Solvent Extraction, Mineral Processing and Extractive Metallurgy Review. V40, 1-13.

Imerys (2021) - Imerys, 2021. (**https://www.imerys.com/minerals/zirconia**) Retrieved 2021.

Industrial Mineral Management Consulting (IMMC), 2021. Norra Kärr Nepheline Syenite – General Market Summary report.

InfoMine Inc. (2020). Mine and Mill Equipment Cost Calculator. Retrieved January 23, 2018, from http://calc2020.costs.infomine.com/

33 IGC excursion No 51, August 2 – 5, 2008. The Sveconorwegian orogen of southern Scandinavia: setting, petrology and geochronology.

of polymetamorphic high-grade terranes

Jha M. K., A. Kumari, R. Panda, J. R. Kumar, K. Yoo, and J. Y. Lee. 2016. Review on hydrometallurgical recovery of rare earth metals. Hydrometallurgy 165:2–26.

Johnsen, O., G. Ferraris, R. Gault, J. Grice, A. Kampf, and I. Pekov. 2003. The nomenclature of eudialyte-group minerals. In *The Canadian Mineralogist* 41:785–94.

Jackson, W.D. & Christiansen, G. (1993). International Strategic Minerals Inventory Summary Report-Rare Earth Oxides: U.S. Geological Survey circular 930-N.

Johnsen , O., Feraris , G., Gault , R. A., Grice , J. D., Kampf , A. R., and Pekov , I. V. (2003). The nomenclature of eudialyte-group minerals. Canadian Mineralogist, 41, 785–794.

Jones, Adrian P., Francis Wall and C. Terry Williams, eds. (1996) Rare Earth Minerals: Chemistry, Origin and Mineralized rock Deposits, The Mineralogy Society Series #7, 372 pp.

Jönköping County https://www.jonkoping.se/byggabomiljo/planarbetestadsplaneringochutveckling/kommundelsutveckling/Grännakommundelsutveckling4.74fef9ab15548f0b800aa3.html (accessed Feb 2021)

Jönköping University (https://ju.se/en/about-us/jonkoping-university/facts-and-figures.html) accessed July 2021.

Jonsson, E. (2010) Drusiga sena natrolitassociationer från Norra Kärr. Berg & Mineral (Stockholm Amatørgeologiska Sallskap), 20(62), 12-13.

Karshigina, Z., Z. Abisheva, Y. Bochevskaya, A. Akcil, E. Sargelova, B. Sukurov, and I. Silachyov. 2018. Recovery of rare earth metals (REMs) from primary raw material: Sulphatization-leaching- precipitation-extraction. Mineral Processing and Extractive Metallurgy Review 39 (5):319–38.

Laznicka, P (2006). Giant Metallic Deposits; Future Sources of Industrial Metals. pp. 466,467.

Larsen, L. M., & Sørensen, H. (1987). The Ilímaussaq intrusion—progressive crystallization and formation of layering in an agpaitic magma. Geological Society, London, Special Publications, 30(1), 473-488.

Larsson, K. & Binnemans, K. (2015). Separation of rare earths by split-anion extraction. Hydrometallurgy, v.156, p.206-214.

Lerner BJ 1962 Process for recovery of Niobium from mineralized rock s in association with alkaline earth metals. US Patent 3025157. 5p.

Litvinova, T. E., and D. E. Chirkist, 2013. Physic-chemical foundations of the eudialyte processing. In: Proceedings of the 2nd Russian Conference with International Participation New Approaches to Chemical Engineering Minerals. The Use of Extraction and Sorption, pp. 84–87. St. Petersburg, Russia.

Ma, Y., S. Stopic, and B. Friedrich. 2019. Hydrometallurgical Treatment of an Eudialyte Concentrate for Preparation of Rare Earth Carbonate. In Johnson Matthey Technology Review 63:2–13.

Ma Y., S. Stopic, L. Gronen, and B. Friedrich. 2018. Recovery of Zr, Hf, Nb from eudialyte residue by sulfuric acid dry digestion and water leaching with H2O2 as a promoter. Hydrometallurgy 181:206–14.

Ma Y., S. Stopic, L. Gronen, M. Milivojevic, S. Obradovic, and B. Friedrich. 2018. Neural network modeling for the extraction of rare earth elements from eudialyte concentrate by dry digestion and leaching. Metals 8 (4):267.

Maksimainen T 2012 Metallurgical Tests on Norra Karr Ore. GTK C/MT/2012/1. GTK Eastern Finland Office, Outokumpu. 90p.

Maksimainen T 2015 Metallurgical Tests on Norra Karr Ore. GTK C/MT/2015. GTK Eastern Finland Office, Outokumpu. 110p. Marshall, W. L., and C.-T. A. Chen. 1982. Amorphous silica solubilities, VI. Postulated sulfatesilicic acid solution complex. Geochimica et cosmochimica acta 46 (3):367–70.

MinChem Ltd. 2021. Summary of the potential for a new source of Zr chemicals from Sweden.

Ministry of Environment and Energy (undated) https://www.government.se/495f60/contentassets/883ae8e123bc4e42aa8d59296ebe0478/the-swedish-climate-policy-framework.pdf

MSP-REFRAM, 2017. Towards a Strong and Sustainable European Refractory Metals Supply-chain – MSP-REFRAM Final Report, H2020 project.

Nayfonov T, Belborodov V, Zakharova I et al (1991) Flotation technology for beneficiation of eudialyte mineralized rock . In: XVII international mineral processing congress, Dresden, 23–28 Sept 1991, vol IV, pp. 131–138

Nockemann, B. Thijs, S. Pittois, J. Thoen, C. Glorieux, K. Van Hecke, L. Van Meervelt, B. Kirchner and K. Binnemans, J. Phys. Chem. B, 2006, 110, (42), 20978

Nebocat, J. (2009). Report on the Geology, Mineralization and Exploration Potential of the Norra Kärr Zirconium-REE Deposit, Gränna, Sweden. NI 43-101 Technical Report prepared by Pacific Geological Services on behalf of LEM Metals Ltd.

Oberti, R., Cámara, F., Hawthorne, F.C., and Ball, N.A. (2009) Fluoro-aluminoleakeite, NaNa2(Mg2Al2Li)Si8O22F2 a new mineral of the amphibole group from Norra Kärr, Sweden: description and crystal structure. Mineralogical Magazine, 73, 817-824.

Orica Limited. (2021, January 1). Australia National Price List. Retrieved from Orica Mining Services: http://www.oricaminingservices.com/au/en/section/products_and_services/price_list

Polido Legaria E S. Demirel Topel, V. G. Kessler and G. A. Seisenbaeva, Dalton Trans., 2015, 44, 31273

Price, WA 1997. Guidelines and Recommended Methods for the Prediction of Metal Leaching and Acid Rock Drainage at Minesites at British Colombia. Reclamation Section, Energy and Mineral Division, Smithers, British Colombia, 158 p

Rankin, L.R. (2011). Structural setting of the Nora Karr Intrusive Complex, Central Sweden. Geointerp. Confidential Report 2011/2014 for LEM Metals Ltd.

Rastsvetaeva, R. K. 2007. Structural Mineralogy of the Eudialyte Group: A Review. Crystallography Reports 52:47–64

Reed, G.C. (2011). NI 43-101 Technical Report, Norra Kärr REE - Zirconium Deposit Gränna, Sweden. NI 43-101 Technical Report prepared by Pincock Allen and Holt on behalf of LEM Metals Ltd.

Ribagnc P, Deblonde GJ-P, Blancher SB, Lengagne L, Donati L, Malimba C, Vourtaud B, Weigel V, Beltrami D 2017 Leaching of niobium-and REE bearing iron mineralized rock s: Significant reduction of H2SO4 consumption using SO2 and activated carbon. Separation and Purification Technology, 189, 1-10.

Sandström, F. (2002) Zinkblande från Norra Kärr. Litiofilen, 19(1), 15.

Saxon M, Leijd M, Berg J, Forrester K (2015). Geology, mineralogy, and metallurgical processing of the Norra Kärr heavy REE deposit, Sweden. November 2015. Conference: Symposium on Strategic and Critical Materials Proceedings, November 13-14, 2015, Victoria, British Columbia. British Columbia Geological Survey Paper 2015-3, 97-107.

SFS 2013:319 Regulation on the Extractive of Waste. Available at: https://www.ecolex.org/details/legislation/regulation-on-the-extraction-of-waste-sfs-2013319-lex-faoc126178/ (accessed 25 February 2021).

SFS 2018:396 The Radiation Protection Act (2018:396), issued by the Swedish Radiation Safety Authority. Available here: http://www.notisum.se/rnp/document/?id=20180396 (accessed 03 March 2021). SFS 2018:506 Radiation Protection Ordinance (2018:506). Available at: http://extwprlegs1.fao.org/docs/pdf/swe187077.pdf (accessed 04 March 2021).

SSMF 2018:4 The Swedish Radiation Safety Authority Guidance Collection, Guidance with Background and Motives to the Swedish Radiation Safety Authority Regulations (SSMFS 2018:4) on Naturally Occurring Radioactive Materials and Building Materials.

SGU Mapviewer Service: https://apps.sgu.se/kartvisare/kartvisare-mineralrattigheter.html

Sjöqvist, A. S. L (2020) Om Norra Kärr alkalina intrusions ursprung och utveckling. Norsk Mineralsymposium 2020: 36-43

Sjöqvist, A.S.L., Cornell, D.H., Andersen, T., Erambert, M., Ek, M., and Leijd, M. (2013) Three compositional varieties of rare-earth element mineralized rock : eudialyte-group minerals from the Norra Kärr Alkaline Complex, Southern Sweden. Minerals, 3, 94-120

Sjöqvist, A. S., Cornell, D. H., Andersen, T., Christensson, U. I., & Berg, J. T. (2017). Magmatic age of rare-earth element and zirconium mineralisation at the Norra Kärr alkaline complex, southern Sweden, determined by U–Pb and Lu–Hf isotope analyses of metasomatic zircon and eudialyte. Lithos, 294, 73-86. doi.org/10.1016/j.lithos.2017.09.023

Sorrensen, H. (1997). The agpaitic rocks; an overview: Mineralogical Magazine; August 1997; v. 61; no. 4; pp. 485-498

Shigematsu T, Tabushi M, Matsui M 1964 Separation of Zirconium-95 and Miobium-95 by solvent extraction using Acetylacetone Bull Chemical Research, 41, 1963-1966

Silin I (2015) Flotation of REE-bearing Zirconium silicates. Master Thesis. RWTH Aachen, Aachen

Stark T., Silin I. & Wotruba, H. (2017). Mineral Processing of Eudialyte Mineralized rock from Norra Kärr. J.Sustain. Metall. 3, 32-38.

Strand, U. (1999) Meddelande om nytt mineral för Sverige. Jinshajiangit från Norra Kärr, Gränna, samt något om fluorbritolit-(Ce). Litiofilen, 16(1), 22-23.

Tkaczyk AH, Bartl A, Amato A, Lapkovskis V, Petranikova M (2018). Sustainability evaluation of essential critical raw materials: cobalt, niobium, tungsten and rare earth elements. J. Physics Applied Physics, 51, 2-27.

Törnebohm, A. (1906) Katapleitsyenit, en nyupptakt varietet af nefelinsyenit i Sverige. Sveriges Geologiska Undersökning. Series C 1906, 199, 1-54.

Törnebohm, A. (1906) Katapleitsyenit, en nyupptakt varietet af nefelinsyenit i Sverige. Geologiska Föreningens i Stockholm Förhandlingar, 28, 415-417 [abstract of Törnebohm 1906].

Tranefors, H. and Olsson, K. (2017) Norra Kärr- ett återbesök och uppdatering från juni 2016. Litiofilen, 34, 7-10

USGS Mineral Yearbook - Feldspar and Nepheline Syenite Statistics, 2017.

Vaccarezza, V., and C. Anderson. 2018. Beneficiation and Leaching Study of Norra Kärr Eudialyte Mineral. In Rare Metal Technology 2018, ed. H. Kim, 39–51. TMS: Springer International Publishing.

Voßenkaul, D., A. Birich, N. Müller, N. Stoltz, and B. Friedrich. 2016. Hydrometallurgical processing of eudialyte bearing concentrates to recover rare earth elements via low-temperature dry digestion to prevent the silica gel formation. Journal of Sustainable Metallurgy 3:79–89.

Voßenkaul, D ; Voßenkaul, D ; Birich, A ; Birich, A ; Müller, N ; Müller, N ; Stoltz, N ; Stoltz, N ; Friedrich, B ; Friedrich, B 2018 Hydrometallurgical Processing of Eudialyte Bearing Concentrates to Recover Rare Earth Elements Via Low-Temperature Dry Digestion to Prevent the Silica Gel Formation. Journal of Sustainable Metallurgy, 2017-03, Vol.3 (1), p.79-89

Walter CS, Verner PH 1963 Sulfuric acid leaching of Niobium mineralized rock s. US Patent 3087809. 4p.

Wang LY, Lee HY, Lee MS 2013 Solvent Extraction of Zirconium and Hafnium from Hydrochloric Acid SolutionsUsing Acidic Organophosphorus Extractants and Their Mixtures with TOPO. Materials Transactions, 54, 1460-1466.

Waste Ordinance (2011:927) available here:

https://www.riksdagen.se/sv/dokument-lagar/dokument/svensk-forfattningssamling/avfallsforordning-2011927_sfs-2011-927 (accessed 25 February 2021).

Wilke, H.-J. (1976) Mineral-Fundstellen Band 4: Skandinavien. Christian Weise Verlag, München, 370 pp. (p. 159-160).

Zetterqvist, A. (1991) Zirkon från Norra Kärr. Stuffen Varpen, (Medlemstidning för Stockholms Amatørgeologiska Sallskap), 1(1), 22.

Zhang, J., and C. Edwards. 2013. Mineral decomposition and leaching processes for treating rare earth mineralized rock concentrates. Canadian Metallurgical Quarterly 52 (3):243–48.

ZIA (2019) - Zircon Industry Association, 2019. Technical Handbook on Zirconium and Zirconium Compounds, Fourth Edition, August 2019.

GLOSSARY, ABBREVIATIONS, UNITS

Glossary

Catapleiite	Zirconium-hosting mineral (formula: $Na_2ZrSi_3O_9{\cdot}2H_2O$)
Grennaite	Fine grained aegerine rich nepheline syenite (lithology)
Kaxtorpite	Local name for a microcline-pectolite-amphibole-aegerine-nepheline syenite (lithology)
Lakarpite	Local name for a coarse-grained variety of nepheline syenite comprising alkali feldspar, altered nepheline, arfvedsonite, aegirine and trace pectolite (lithology)

Abbreviations

CDN	Canadian dollars
HREO	Heavy rare earth oxides (Eu_2O_3, Gd_2O_3, Tb_2O_3, Dy_2O_3, Ho_2O_3, Er_2O_3, Tm_2O_3, Yb_2O_3, Lu_2O_3, Y_2O_3.)
LREO	Light rare earth oxides (La_2O_3, Ce_2O_3, Pr_2O_3, Nd_2O_3, Sm_2O_3).
masl	Metres above sea-level
MREO	Mixed Rare Earth Oxide phase containing both light (LREO) and heavy (HREO) oxides
REE	Rare earth elements
REO	Rare earth oxide
SGU	Swedish Geological Survey
SEK	Swedish Kroner
TREO	Total rare earth oxides
USD	Unite States Dollars
USGS	United States Geological Survey

Ca	Calcium
Na	Sodium
Zr	Zirconium
La	Lanthanum
Ce	Cerium
Pr	Praseodymium
Nd	Neodymium
Sm	Samarium
Eu	Europium
Gd	Gadolinium
Tb	Terbium
Dy	Dysprosium
Ho	Holmium
Er	Erbium
Tm	Thulium
Yb	Ytterbium
Lu	Lutetium
Y	Yttrium

Units

Kt	Thousand metric tonnes
Mt	Million metric tonnes
ppm	Parts per million
g/t	Grammes per tonne
%	Percent
Ga	Billion years
Ma	Million years

APPENDIX

A LIFE OF MINE CASHFLOW SUMMARY

Description	Units	LoM Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
Production																	
Mining																	
Ore	(t)	**29,315,383**	-	-	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000
Waste	(t)	**9,371,933**	-	-	416,854	370,320	476,973	475,512	502,028	500,000	474,735	388,710	509,205	307,529	173,814	90,989	74,193
TMM	(t)	**38,687,316**	-	-	1,566,854	1,520,320	1,626,972	1,625,512	1,652,028	1,650,000	1,624,735	1,538,710	1,659,205	1,457,528	1,323,814	1,240,988	1,224,192
Processing																	
Magnetic Separation																	
Ore Feed	(t)	**29,315,383**	-	-	862,500	1,092,500	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000
Magnetic Concentrate	(t)	**2,667,700**	-	-	78,488	99,418	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650
Aegirine Waste	(t)	**7,592,684**	-	-	223,388	282,958	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850
Non-Magnetic [Nepheline Syenite Product 1]	(t)	**7,143,791**	-	-	210,180	266,229	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241
Non-Magnetic [Nepheline Syenite Product 2]	(t)	**10,480,983**	-	-	308,365	390,596	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154
Non-Magnetic [Nepheline Syenite Product 3]	(t)	**1,430,225**	-	-	42,079	53,300	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106
Leaching																	
Mag Con Feed	(t)	**2,667,700**	-	-	78,488	99,418	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650
REO Product	(t)	**138,862**	-	-	3,852	5,215	5,519	5,101	5,102	5,479	5,486	5,660	5,318	5,098	5,394	5,251	5,378
Zr Oxide Product	(t)	**265,210**	-	-	6,378	9,314	10,924	11,166	10,892	11,207	10,085	10,851	10,172	9,950	10,181	10,486	11,013
Nb Oxide Product	(t)	**13,662**	-	-	409	549	565	513	502	530	508	537	503	482	506	503	514
Neutralised Waste	(t)	**2,249,965**	-	-	67,848	84,340	87,642	87,870	88,154	87,434	88,571	87,602	88,657	89,119	88,568	88,410	87,745
Gypsum Waste	(t)	**2,389,814**	-	-	70,312	89,062	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749
Waste Streams																	
Magnetic Separation Aegirine Waste [Aegirine]	(t)	**7,592,684**	-	-	223,388	282,958	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850
Neutralised Waste	(t)	**2,249,965**	-	-	67,848	84,340	87,642	87,870	88,154	87,434	88,571	87,602	88,657	89,119	88,568	88,410	87,745
Gypsum Waste	(t)	**2,389,814**	-	-	70,312	89,062	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749
Total	(t)	**12,232,464**	**-**	**-**	**361,547**	**456,359**	**479,241**	**479,469**	**479,753**	**479,033**	**480,170**	**479,201**	**480,256**	**480,718**	**480,167**	**480,009**	**479,344**
Products																	
REO Product																	
Ce_2O_3	(t)	**28,477**	-	-	797	1,030	1,067	972	992	1,077	1,182	1,162	1,161	1,116	1,196	1,126	1,119
Dy_2O_3	(t)	**6,437**	-	-	178	248	267	250	247	262	244	259	236	227	235	238	248
Er_2O_3	(t)	**4,449**	-	-	120	169	184	173	170	180	167	178	162	158	161	164	171
Eu_2O_3	(t)	**514**	-	-	14	20	21	19	19	21	20	21	19	18	19	19	20
Gd_2O_3	(t)	**4,632**	-	-	128	178	190	176	174	187	181	190	173	164	173	169	177
Ho_2O_3	(t)	**1,433**	-	-	39	55	60	56	56	58	54	57	53	51	53	54	56
La_2O_3	(t)	**12,668**	-	-	356	456	466	419	440	468	529	508	519	504	536	509	503
Lu_2O_3	(t)	**575**	-	-	16	22	24	23	22	23	21	23	21	21	21	22	22
Nd_2O_3	(t)	**15,037**	-	-	414	554	581	533	543	591	614	623	600	573	614	581	591
Pr_2O_3	(t)	**3,730**	-	-	103	136	142	130	133	144	154	154	152	146	156	148	149
Sm_2O_3	(t)	**4,157**	-	-	118	159	168	156	154	167	163	171	158	150	161	155	161
Tb_2O_3	(t)	**943**	-	-	26	36	39	37	36	38	36	38	35	33	35	35	36
Tm_2O_3	(t)	**682**	-	-	18	26	28	27	26	28	26	27	25	24	25	25	26
Y_2O_3	(t)	**50,930**	-	-	1,411	1,968	2,108	1,965	1,928	2,067	1,938	2,082	1,849	1,761	1,854	1,849	1,938
Yb_2O_3	(t)	**4,197**	-	-	114	159	174	166	162	169	156	166	154	150	153	157	162
Total REO	(t)	**138,862**	**-**	**-**	**3,852**	**5,215**	**5,519**	**5,101**	**5,102**	**5,479**	**5,486**	**5,660**	**5,318**	**5,098**	**5,394**	**5,251**	**5,378**
Zr Oxide Product																	
ZrO_2	(t)	**265,210**	-	-	6,378	9,314	10,924	11,166	10,892	11,207	10,085	10,851	10,172	9,950	10,181	10,486	11,013
Total	(t)	**265,210**	**-**	**-**	**6,378**	**9,314**	**10,924**	**11,166**	**10,892**	**11,207**	**10,085**	**10,851**	**10,172**	**9,950**	**10,181**	**10,486**	**11,013**
Nb Oxide Product																	
Nb_2O_5	(t)	**13,662**	-	-	409	549	565	513	502	530	508	537	503	482	506	503	514
Total	(t)	**13,662**	**-**	**-**	**409**	**549**	**565**	**513**	**502**	**530**	**508**	**537**	**503**	**482**	**506**	**503**	**514**
Nepheline Syenite																	
Product 1	(t)	**7,143,791**	-	-	210,180	266,229	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241
Product 2	(t)	**10,480,983**	-	-	308,365	390,596	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154
Product 3	(t)	**1,430,225**	-	-	42,079	53,300	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106
Total	(t)	**19,054,999**	**-**	**-**	**560,625**	**710,125**	**747,500**	**747,500**	**747,500**	**747,500**	**747,500**	**747,500**	**747,500**	**747,500**	**747,500**	**747,500**	**747,500**
Gross Revenue																	
Summary																	
REO	(USDk)	**7,366,963**	-	-	179,870	256,525	284,193	272,165	276,739	299,306	289,198	302,105	280,832	269,398	282,719	279,674	288,597
Zr Oxide	(USDk)	**1,060,842**	-	-	25,513	37,255	43,697	44,666	43,567	44,828	40,341	43,403	40,689	39,801	40,725	41,943	44,052
Nb Oxide	(USDk)	**478,172**	-	-	14,331	19,228	19,765	17,938	17,560	18,538	17,780	18,799	17,603	16,887	17,725	17,619	18,000
Nepheline Syenite	(USDk)	**1,055,616**	-	-	31,058	39,340	41,410	41,410	41,410	41,410	41,410	41,410	41,410	41,410	41,410	41,410	41,410
Total Gross Revenue	**(USDk)**	**9,961,593**	**-**	**-**	**250,771**	**352,348**	**389,066**	**376,180**	**379,276**	**404,083**	**388,730**	**405,718**	**380,534**	**367,496**	**382,580**	**380,647**	**392,058**
Operating Costs																	
Mining	(USDk)	**164,960**	-	-	6,310	6,267	6,476	6,491	6,629	6,715	6,683	6,599	6,883	6,451	6,152	5,925	5,941
Processing	(USDk)	**1,501,216**	-	-	46,757	56,141	58,941	58,817	58,658	59,052	58,418	58,955	58,374	58,120	58,421	58,512	58,880
G&A	(USDk)	**146,577**	-	-	4,313	5,463	5,750	5,750	5,750	5,750	5,750	5,750	5,750	5,750	5,750	5,750	5,750
Transport	(USDk)	**144,544**	-	-	4,173	5,363	5,700	5,689	5,675	5,710	5,655	5,702	5,651	5,629	5,656	5,663	5,695
Sales (toll separation)	(USDk)	**2,638,378**	-	-	73,184	99,077	104,859	96,921	96,940	104,100	104,234	107,543	101,047	96,863	102,487	99,765	102,181
Royalty	(USDk)	**21,898**	-	-	721	829	864	813	832	909	842	909	816	794	851	835	873
Total	**(USDk)**	**4,617,572**	**-**	**-**	**135,457**	**173,140**	**182,590**	**174,481**	**174,485**	**182,235**	**181,582**	**185,459**	**178,522**	**173,607**	**179,316**	**176,451**	**179,321**
Unit Costs																	
Per tonne ore mined (total)	(USD/t)	**157.51**	**-**	**-**	**117.79**	**150.56**	**158.77**	**151.72**	**151.73**	**158.47**	**157.90**	**161.27**	**155.24**	**150.96**	**155.93**	**153.44**	**155.93**
Per tonne ore mined (ex toll separation)	(USD/t)	**67.51**	**-**	**-**	**54.15**	**64.40**	**67.59**	**67.44**	**67.43**	**67.94**	**67.26**	**67.75**	**67.37**	**66.73**	**66.81**	**66.68**	**67.08**
Minng - per tonne ore mined	(USD/t)	**5.63**	**-**	**-**	**5.49**	**5.45**	**5.63**	**5.64**	**5.76**	**5.84**	**5.81**	**5.74**	**5.99**	**5.61**	**5.35**	**5.15**	**5.17**
Processing - per tonne ore processed	(USD/t)	**51.21**	**-**	**-**	**54.21**	**51.39**	**51.25**	**51.15**	**51.01**	**51.35**	**50.80**	**51.27**	**50.76**	**50.54**	**50.80**	**50.88**	**51.20**
G&A - per tonne ore processed	(USD/t)	**5.00**	**-**	**-**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**
Transport - per tonne ore processed	(USD/t)	**4.93**	**-**	**-**	**4.84**	**4.91**	**4.96**	**4.95**	**4.94**	**4.97**	**4.92**	**4.96**	**4.91**	**4.89**	**4.92**	**4.92**	**4.95**
Per kg REO (no by-product credit)	(USD/kg)	**33.25**	**-**	**-**	**35.17**	**33.20**	**33.08**	**34.20**	**34.20**	**33.26**	**33.10**	**32.77**	**33.57**	**34.05**	**33.24**	**33.60**	**33.34**
Per kg REO (ex toll separation, no by-product credit)	(USD/kg)	**14.25**	**-**	**-**	**16.17**	**14.20**	**14.08**	**15.20**	**15.20**	**14.26**	**14.10**	**13.77**	**14.57**	**15.05**	**14.24**	**14.60**	**14.34**
By product credit per kg MREO	(USD/kg)	**18.68**	**-**	**-**	**18.41**	**18.38**	**19.00**	**20.39**	**20.10**	**19.12**	**18.14**	**18.31**	**18.75**	**19.24**	**18.51**	**19.23**	**19.24**
Per kg REO (ex toll separation, by-product credit)	(USD/kg)	**- 4.43**	**-**	**-**	**- 2.24**	**- 4.17**	**- 4.92**	**- 5.19**	**- 4.90**	**- 4.86**	**- 4.04**	**- 4.54**	**- 4.18**	**- 4.19**	**- 4.27**	**- 4.63**	**- 4.89**
Per kg REO (incl. toll separation, by-product credit)	(USD/kg)	**14.57**	**-**	**-**	**16.76**	**14.83**	**14.08**	**13.81**	**14.10**	**14.14**	**14.96**	**14.46**	**14.82**	**14.81**	**14.73**	**14.37**	**14.11**
Capital Costs																	
Project Development	(USDk)	**487,160**	243,580	243,580	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital (Mining Specific)	(USDk)	**9,894**	-	-	-	-	-	-	1,107	-	-	-	-	-	-	214	-
Sustaining Capital (General)	(USDk)	**74,300**	-	-	-	-	3,377	3,377	3,377	3,377	3,377	3,377	3,377	3,377	3,377	3,377	3,377
Closure	(USDk)	**35,000**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total	(USDk)	**606,354**	**243,580**	**243,580**	**-**	**-**	**3,377**	**3,377**	**4,484**	**3,377**	**3,377**	**3,377**	**3,377**	**3,377**	**3,377**	**3,592**	**3,377**
Cashflow																	
Gross Revenue	(USDk)	**9,961,593**	0	0	250,771	352,348	389,066	376,180	379,276	404,083	388,730	405,718	380,534	367,496	382,580	380,647	392,058
Operating Costs	(USDk)	**(4,617,572)**	0	0	(135,457)	(173,140)	(182,590)	(174,481)	(174,485)	(182,235)	(181,582)	(185,459)	(178,522)	(173,607)	(179,316)	(176,451)	(179,321)
Net Operating Cashflow (EBITDA)	(USDk)	**5,344,021**	**0**	**0**	**115,313**	**179,208**	**206,476**	**201,699**	**204,791**	**221,848**	**207,148**	**220,259**	**202,012**	**193,890**	**203,264**	**204,196**	**212,738**
Capital Costs	(USDk)	**(606,354)**	(243,580)	(243,580)	0	0	(3,377)	(3,377)	(4,484)	(3,377)	(3,377)	(3,377)	(3,377)	(3,377)	(3,377)	(3,592)	(3,377)
Working Capital	(USDk)	**0**	0	0	(15,330)	(8,187)	(3,440)	727	(453)	(2,167)	1,818	(1,570)	2,228	1,042	(1,203)	(19)	(1,149)
Net Project Cashflow (Pre-Tax)	(USDk)	**4,737,667**	**(243,580)**	**(243,580)**	**99,983**	**171,021**	**199,659**	**199,049**	**199,854**	**216,304**	**205,588**	**215,312**	**200,863**	**191,554**	**198,684**	**200,586**	**208,211**
Cumulative Cashflow (Pre-Tax)	(USDk)		**(243,580)**	**(487,160)**	**(387,177)**	**(216,156)**	**(16,497)**	**182,553**	**382,407**	**598,711**	**804,299**	**1,019,611**	**1,220,473**	**1,412,028**	**1,610,712**	**1,811,297**	**2,019,508**
Corporation Tax	(USDk)	**(985,074)**	0	0	(11,210)	(31,899)	(37,447)	(36,393)	(36,938)	(40,382)	(37,284)	(39,915)	(36,087)	(34,344)	(36,206)	(36,324)	(38,083)
Net Project Cashflow (Post-Tax)	(USDk)	**3,752,593**	**(243,580)**	**(243,580)**	**88,773**	**139,122**	**162,213**	**162,656**	**162,916**	**175,922**	**168,304**	**175,396**	**164,776**	**157,210**	**162,478**	**164,262**	**170,128**
Cumulative Cashflow (Post-Tax)	(USDk)		**(243,580)**	**(487,160)**	**(398,387)**	**(259,265)**	**(97,053)**	**65,603**	**228,520**	**404,442**	**572,746**	**748,143**	**912,918**	**1,070,129**	**1,232,607**	**1,396,869**	**1,566,997**

Description	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052	2053
Production																
Mining																
Ore	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	565,387	-	-	-
Waste	11,136	0	594,739	669,924	674,727	725,273	718,255	464,316	309,870	226,839	136,646	61,212	18,134	-	-	-
TMM	1,161,136	1,150,000	1,744,739	1,819,923	1,824,727	1,875,272	1,868,254	1,614,316	1,459,870	1,376,839	1,286,645	1,211,212	583,520	-	-	-
Processing																
Magnetic Separation																
Ore Feed	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	1,150,000	910,383	-	-	-
Magnetic Concentrate	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	82,845	-	-	-
Aegirine Waste	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	235,789	-	-	-
Non-Magnetic [Nepheline Syenite Product 1]	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	221,849	-	-	-
Non-Magnetic [Nepheline Syenite Product 2]	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	325,485	-	-	-
Non-Magnetic [Nepheline Syenite Product 3]	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	44,415	-	-	-
Leaching																
Mag Con Feed	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	104,650	82,845	-	-	-
REO Product	5,528	5,547	5,477	5,530	5,715	5,358	4,649	4,766	5,190	5,661	5,873	6,385	5,329	-	-	-
Zr Oxide Product	11,040	11,533	11,102	10,744	10,594	11,070	9,120	8,962	9,223	9,841	10,082	10,682	8,598	-	-	-
Nb Oxide Product	526	547	534	524	545	526	447	466	499	587	604	667	565	-	-	-
Neutralised Waste	87,555	87,023	87,537	87,852	87,796	87,695	90,433	90,456	89,738	88,561	88,090	86,916	68,352	-	-	-
Gypsum Waste	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	74,215	-	-	-
Waste Streams																
Magnetic Separation Aegirine Waste [Aegirine]	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	297,850	235,789	-	-	-
Neutralised Waste	87,555	87,023	87,537	87,852	87,796	87,695	90,433	90,456	89,738	88,561	88,090	86,916	68,352	-	-	-
Gypsum Waste	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	93,749	74,215	-	-	-
Total	**479,154**	**478,621**	**479,136**	**479,451**	**479,395**	**479,294**	**482,032**	**482,055**	**481,337**	**480,159**	**479,689**	**478,515**	**378,357**	**-**	**-**	**-**
Products																
REO Product																
Ce_2O_3	1,145	1,076	1,084	1,133	1,177	1,068	985	1,029	1,130	1,164	1,185	1,261	1,042	-	-	-
Dy_2O_3	256	268	261	257	263	257	212	214	228	258	272	300	252	-	-	-
Er_2O_3	176	187	182	179	181	179	150	149	157	178	188	208	177	-	-	-
Eu_2O_3	21	21	21	20	21	20	16	17	18	21	22	24	20	-	-	-
Gd_2O_3	184	188	186	185	193	184	152	152	166	187	197	217	181	-	-	-
Ho_2O_3	57	60	58	57	58	57	48	48	50	57	60	66	56	-	-	-
La_2O_3	511	477	479	499	509	461	451	472	516	528	528	560	467	-	-	-
Lu_2O_3	23	24	23	23	23	23	20	20	21	23	24	26	22	-	-	-
Nd_2O_3	606	586	585	593	623	562	495	519	574	613	629	675	563	-	-	-
Pr_2O_3	151	143	143	146	152	137	125	132	145	152	155	165	137	-	-	-
Sm_2O_3	165	166	165	165	173	160	135	139	153	169	176	191	160	-	-	-
Tb_2O_3	38	39	38	37	38	37	31	31	34	38	40	44	37	-	-	-
Tm_2O_3	27	29	28	27	27	28	23	23	24	27	29	32	27	-	-	-
Y_2O_3	2,002	2,107	2,054	2,041	2,105	2,014	1,662	1,678	1,823	2,079	2,193	2,426	2,030	-	-	-
Yb_2O_3	167	176	172	168	170	171	145	143	150	168	175	191	160	-	-	-
Total REO	**5,528**	**5,547**	**5,477**	**5,530**	**5,715**	**5,358**	**4,649**	**4,766**	**5,190**	**5,661**	**5,873**	**6,385**	**5,329**	**-**	**-**	**-**
Zr Oxide Product																
ZrO_2	11,040	11,533	11,102	10,744	10,594	11,070	9,120	8,962	9,223	9,841	10,082	10,682	8,598	-	-	-
Total	**11,040**	**11,533**	**11,102**	**10,744**	**10,594**	**11,070**	**9,120**	**8,962**	**9,223**	**9,841**	**10,082**	**10,682**	**8,598**	**-**	**-**	**-**
Nb Oxide Product																
Nb_2O_5	526	547	534	524	545	526	447	466	499	587	604	667	565	-	-	-
Total	**526**	**547**	**534**	**524**	**545**	**526**	**447**	**466**	**499**	**587**	**604**	**667**	**565**	**-**	**-**	**-**
Nepheline Syenite																
Product 1	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	280,241	221,849	-	-	-
Product 2	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	411,154	325,485	-	-	-
Product 3	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	56,106	44,415	-	-	-
Total	**747,500**	**747,500**	**747,500**	**747,500**	**747,500**	**747,500**	**747,500**	**747,500**	**747,500**	**747,500**	**747,500**	**747,500**	**591,749**	**-**	**-**	**-**
Gross Revenue																
Summary																
REO	297,448	303,639	297,410	295,962	304,890	291,276	246,326	250,728	270,898	300,744	314,234	343,858	288,231	-	-	-
Zr Oxide	44,160	46,133	44,407	42,974	42,374	44,282	36,482	35,848	36,890	39,363	40,330	42,726	34,392	-	-	-
Nb Oxide	18,425	19,162	18,704	18,352	19,083	18,423	15,642	16,307	17,472	20,555	21,156	23,349	19,770	-	-	-
Nepheline Syenite	41,410	41,410	41,410	41,410	41,410	41,410	41,410	41,410	41,410	41,410	41,410	41,410	32,782	-	-	-
Total Gross Revenue	**401,443**	**410,344**	**401,931**	**398,699**	**407,757**	**395,391**	**339,859**	**344,293**	**366,671**	**402,072**	**417,130**	**451,343**	**375,175**	**-**	**-**	**-**
Operating Costs																
Mining	5,781	5,796	6,887	7,163	7,241	7,460	7,039	6,591	6,404	6,330	6,054	5,821	2,872	-	-	-
Processing	58,983	59,281	58,996	58,818	58,847	58,910	57,397	57,384	57,776	58,431	58,689	59,336	50,323	-	-	-
G&A	5,750	5,750	5,750	5,750	5,750	5,750	5,750	5,750	5,750	5,750	5,750	5,750	4,552	-	-	-
Transport	5,704	5,730	5,705	5,690	5,693	5,697	5,566	5,565	5,599	5,656	5,679	5,735	4,562	-	-	-
Sales (toll separation)	105,035	105,389	104,066	105,069	108,584	101,800	88,337	90,549	98,614	107,565	111,584	121,323	101,260	-	-	-
Royalty	894	916	882	877	907	864	694	748	816	911	933	1,030	537	-	-	-
Total	**182,146**	**182,862**	**182,286**	**183,366**	**187,022**	**180,482**	**164,783**	**166,587**	**174,959**	**184,643**	**188,689**	**198,996**	**164,106**	**-**	**-**	**-**
Unit Costs																
Per tonne ore mined (total)	**158.39**	**159.01**	**158.51**	**159.45**	**162.63**	**156.94**	**143.29**	**144.86**	**152.14**	**160.56**	**164.08**	**173.04**	**290.25**	**-**	**-**	**-**
Per tonne ore mined (ex toll separation)	**67.05**	**67.37**	**68.02**	**68.08**	**68.21**	**68.42**	**66.47**	**66.12**	**66.39**	**67.02**	**67.05**	**67.54**	**111.16**	**-**	**-**	**-**
Minng - per tonne ore mined	**5.03**	**5.04**	**5.99**	**6.23**	**6.30**	**6.49**	**6.12**	**5.73**	**5.57**	**5.50**	**5.26**	**5.06**	**5.08**	**-**	**-**	**-**
Processing - per tonne ore processed	**51.29**	**51.55**	**51.30**	**51.15**	**51.17**	**51.23**	**49.91**	**49.90**	**50.24**	**50.81**	**51.03**	**51.60**	**55.28**	**-**	**-**	**-**
G&A - per tonne ore processed	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**5.00**	**-**	**-**	**-**
Transport - per tonne ore processed	**4.96**	**4.98**	**4.96**	**4.95**	**4.95**	**4.95**	**4.84**	**4.84**	**4.87**	**4.92**	**4.94**	**4.99**	**5.01**	**-**	**-**	**-**
Per kg REO (no by-product credit)	**32.95**	**32.97**	**33.28**	**33.16**	**32.73**	**33.69**	**35.44**	**34.96**	**33.71**	**32.61**	**32.13**	**31.16**	**30.79**	**-**	**-**	**-**
Per kg REO (ex toll separation, no by-product credit)	**13.95**	**13.97**	**14.28**	**14.16**	**13.73**	**14.69**	**16.44**	**15.96**	**14.71**	**13.61**	**13.13**	**12.16**	**11.79**	**-**	**-**	**-**
By product credit per kg MREO	**18.81**	**19.24**	**19.08**	**18.58**	**18.00**	**19.43**	**20.12**	**19.63**	**18.45**	**17.90**	**17.52**	**16.83**	**16.31**	**-**	**-**	**-**
Per kg REO (ex toll separation, by-product credit)	**- 4.86**	**- 5.27**	**- 4.80**	**- 4.42**	**- 4.27**	**- 4.75**	**- 3.68**	**- 3.68**	**- 3.74**	**- 4.28**	**- 4.39**	**- 4.67**	**- 4.52**	**-**	**-**	**-**
Per kg REO (incl. toll separation, by-product credit)	**14.14**	**13.73**	**14.20**	**14.58**	**14.73**	**14.25**	**15.32**	**15.32**	**15.26**	**14.72**	**14.61**	**14.33**	**14.48**	**-**	**-**	**-**
Capital Costs																
Project Development	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital (Mining Specific)	-	-	2,657	84	-	-	-	-	3,320	1,287	1,225	-	-	-	-	-
Sustaining Capital (General)	3,377	3,377	3,377	3,377	3,377	3,377	3,377	3,377	3,377	3,377	3,377	-	-	-	-	-
Closure	-	-	-	-	-	-	-	-	-	-	5,833	5,833	5,833	5,833	5,833	5,833
Total	**3,377**	**3,377**	**6,035**	**3,461**	**3,377**	**3,377**	**3,377**	**3,377**	**6,697**	**4,664**	**10,435**	**5,833**	**5,833**	**5,833**	**5,833**	**5,833**
Cashflow																
Gross Revenue	401,443	410,344	401,931	398,699	407,757	395,391	339,859	344,293	366,671	402,072	417,130	451,343	375,175	0	0	0
Operating Costs	(182,146)	(182,862)	(182,286)	(183,366)	(187,022)	(180,482)	(164,783)	(166,587)	(174,959)	(184,643)	(188,689)	(198,996)	(164,106)	0	0	0
Net Operating Cashflow (EBITDA)	**219,296**	**227,482**	**219,645**	**215,333**	**220,735**	**214,909**	**175,077**	**177,706**	**191,712**	**217,429**	**228,441**	**252,347**	**211,069**	**0**	**0**	**0**
Capital Costs	(3,377)	(3,377)	(6,035)	(3,461)	(3,377)	(3,377)	(3,377)	(3,377)	(6,697)	(4,664)	(10,435)	(5,833)	(5,833)	(5,833)	(5,833)	(5,833)
Working Capital	(832)	(1,015)	1,049	445	(696)	772	5,103	(402)	(1,796)	(3,250)	(1,310)	(3,113)	5,376	27,371	0	0
Net Project Cashflow (Pre-Tax)	**215,087**	**223,090**	**214,659**	**212,316**	**216,661**	**212,304**	**176,802**	**173,927**	**183,219**	**209,515**	**216,696**	**243,401**	**210,612**	**21,538**	**(5,833)**	**(5,833)**
Cumulative Cashflow (Pre-Tax)	**2,234,595**	**2,457,685**	**2,672,344**	**2,884,660**	**3,101,321**	**3,313,625**	**3,490,427**	**3,664,354**	**3,847,573**	**4,057,087**	**4,273,784**	**4,517,184**	**4,727,796**	**4,749,334**	**4,743,500**	**4,737,667**
Corporation Tax	(39,434)	(41,143)	(39,474)	(38,584)	(39,697)	(41,006)	(35,309)	(35,851)	(38,672)	(43,943)	(46,066)	(50,940)	(42,441)	0	0	0
Net Project Cashflow (Post-Tax)	**175,653**	**181,946**	**175,185**	**173,732**	**176,964**	**171,298**	**141,493**	**138,076**	**144,547**	**165,571**	**170,630**	**192,460**	**168,170**	**21,538**	**(5,833)**	**(5,833)**
Cumulative Cashflow (Post-Tax)	**1,742,649**	**1,924,596**	**2,099,780**	**2,273,512**	**2,450,477**	**2,621,775**	**2,763,267**	**2,901,343**	**3,045,890**	**3,211,462**	**3,382,091**	**3,574,551**	**3,742,722**	**3,764,260**	**3,758,426**	**3,752,593**

APPENDIX

B QUALIFIED PERSON CERTIFICATES

SRK Consulting (UK) Limited
5th Floor Churchill House
17 Churchill Way
City and County of Cardiff
CF10 2HH, Wales
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CERTIFICATE OF AUTHOR

Robert John Bowell

Corporate Consultant (Geochemistry)SRK Consulting (UK) Ltd

Email: rbowell@srk.co.uk

I, Robert John Bowell, BSc PhD CChem CGeol EGeol, do hereby certify that:

1. I am Corporate Consultant (Geochemistry&Geometallurgy) of:

SRK Consulting (UK) Limited, Churchill House, Churchill Way, Cardiff CF10 2HH, UK

2. I graduated with an honours degree in Geology and Chemistry (class i) in 1987 from Manchester University and a PhD in Geochemistry from Southampton University in 1991.

3. I am a chartered chemist of the Royal Society of Chemistry, a Chartered Geologist of the Geological Society of London and a Registered European professional Geologist of the European Federation of Geologists in good standing in Europe in the areas of Chemistry and Geology

4. I have worked as a professional Geochemist, Geologist and Geometallurgist for a total of 34 years. My experience includes mineralogy, process chemistry, metallurgy, geometallurgy, geochemistry and engineering controls.

5. 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 fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.

6. I am a contributing author for the preparation of the technical report titled " PRELIMINARY ECONOMIC ASSESSMENT OF NORRA KÄRR RARE EARTH DEPOSIT AND POTENTIAL BY-PRODUCTS, SWEDEN”

" (the “Technical Report”), dated August 19th, prepared for Leading Edge Materials and I have overall responsibility for the report particularly 1-8, 12, 16, 20-26. I have visited the project site and core facilities June 28-30 2021.

7. I have had no prior involvement with the property that is the subject of the Technical Report.

8. I have had no prior involvement with the project or collaboration with the Client

9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

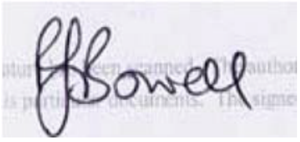
10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 19 day of August 2021.

Dated in Cardiff, United Kingdom, August 19th, 2021



Dr Robert John .Bowell

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CERTIFICATE OF QUALIFIED PERSON

I, Martin Frank Pittuck, MSc., C.Eng, MIMMM do hereby certify that:

a) I am Corporate Consultant (Mining Geology) of SRK Consulting (UK) Ltd with an office at 5th Floor, Churchill House, Churchill Way, Cardiff CF10 2HH.

b) This certificate applies to the technical report titled "PRELIMINARY ECONOMIC ASSESSMENT OF NORRA KÄRR RARE EARTH DEPOSIT AND POTENTIAL BY-PRODUCTS, SWEDEN" with the Effective Date of August 19th 2021

c) I am a graduate with a Master of Science in Mineral Resources gained from Cardiff College, University of Wales in 1996 and I have practised my profession continuously since that time. Since graduating I have worked as a consultant at SRK on a wide range of mineral projects, specializing in rare metals and igneous deposits. I have undertaken many geological investigations, resource estimations, mine evaluation technical studies and due diligence reports. I am a member of the Institution of Materials Mining and Metallurgy (Membership Number 49186) a Fellow of the Geological Society and I am a Chartered Engineer;

d) I have not visited the Norra Kärr site

e) I am co-author of this report and have responsibility for the Mineral Resource Estimation method and classification described in Executive Summary section 4 and Section 13 of the Technical Report.

f) I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

g) I have not had prior involvement with the property that is the subject of the Technical Report.

h) 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 fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101; the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.

i) As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 19th Day of August, 2021.

This signature has been scanned. The author has given permission to its use for this particular document. The original signature is held on file.

Martin Frank Pittuck, MSc., C.Eng, FGS, MIMMM
Corporate Consultant (Mining Geology)

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CERTIFICATE OF AUTHOR

Richard Martindale

Principal Consultant (Geotechnical/ Tailings Engineering)SRK Consulting (UK) Ltd
Email: rmartindale@srk.co.uk

I, Richard Martindale, BSc MSc CEng, do hereby certify that:

1. I am Principal Consultant (Geotechnical and Tailing Engineering) of:
SRK Consulting (UK) Limited, Churchill House, Churchill Way, Cardiff CF10 2HH, UK.
2. I graduated with an honours degree in Geology (class 2i) in 2001 from Durham University and an MSc in Mining Geology from Camborne School of Mines, Exeter University in 2002.
3. I am a Chartered Engineer of the Engineering Council, with registration made through the Institute of Mining Minerals and Materials (IMMM).
4. I have worked as a professional Geotechnical Engineer for a total of 18 years. My experience includes soil and rock mechanics, tailings engineering, slope design and risk assessment, and expert witness investigations.
5. 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.
6. I am a contributing author for the preparation of the technical report titled " PRELIMINARY ECONOMIC ASSESSMENT OF NORRA KÄRR RARE EARTH DEPOSIT AND POTENTIAL BY-PRODUCTS, SWEDEN"
" (the "Technical Report"), dated August 19th 2021, prepared for Leading Edge Materials and I have responsibility for Sections 23.1 and 23.2.
7. I have had no prior involvement with the property that is the subject of the Technical Report.
8. I have had no prior involvement with the project or collaboration with the Client.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated in Cardiff, United Kingdom, August 19th, 2021

This signature has been scanned. The author has given permission to its use for this particular document. The original signature is held on file.

SRK CONSULTING (UK) LIMITED Co. No: 01575403

Richard Martindale, CEng

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www.srk.com

CERTIFICATE OF QUALIFIED PERSON

To Accompany the report entitled: **PRELIMINARY ECONOMIC ASSESSMENT OF NORRA KÄRR RARE EARTH DEPOSIT AND POTENTIAL BY-PRODUCTS, SWEDEN**

I, Colleen MacDougall, PEng, do hereby certify that:

1) I am a Principal Consultant (Mining Engineering) with SRK Consulting (Canada) Inc. (SRK) with an office at Suite 1500, 155 University Avenue, Toronto, Ontario, Canada.
2) I am a graduate of McGill University in Montreal, Quebec, Canada with a BEng in Mining in 2006. I have practiced my profession continuously since 2006. I focus on open pit mining engineering projects worldwide. I have been directly involved in technical reviews, audits, and technical studies for precious metal, base metal, bulk commodities, and industrial mineral projects and operations.
3) I am a Professional Engineer registered with the Professional Engineers Ontario (PEO#100530936);
4) I have not personally visited the project site for this Technical Report, and this was primarily due to current travel restrictions related to the global COVID-19 pandemic;
5) 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 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
6) I, as a Qualified Person, am independent of Leading Edge Minerals as defined in Section 1.5 of National Instrument 43-101;
7) I am the co-author of the technical report titled, “PRELIMINARY ECONOMIC ASSESSMENT OF NORRA KÄRR RARE EARTH DEPOSIT AND POTENTIAL BY-PRODUCTS, SWEDEN (the “Technical Report”) dated 19th August relating to NORRA KÄRR Project (the “Project”) and responsible for Sections 16 and 21 and accept professional responsibility for those sections of this technical report;
8) I have had no prior involvement with the subject property;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10) That, as of the 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.
11) 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, for which the omission to disclose would make the Technical Report misleading.

Local Offices:
Saskatoon
Sudbury
Toronto
Vancouver
Yellowknife

Group Offices:
Africa
Asia
Australia
Europe
North America
South America

12) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them or Leading Edge Minerals for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public.

Toronto, Ontario
August 19, 2021

Colleen MacDougall, PEng, (PEO#100530936)
Principal Consultant (Mining Engineering)

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CERTIFICATE OF AUTHOR

Colin Michael Chapman

Principal Consultant (Mining Infrastructure & Logistics)

SRK Consulting (UK) Ltd

Email: cchapman@srk.co.uk

I, Colin M Chapman, a Chartered Professional Engineer, do hereby certify that:

1. I am responsible for the preparation of the technical report titled, "PRELIMINARY ECONOMIC ASSESSMENT OF NORRA KÄRR RARE EARTH DEPOSIT AND POTENTIAL BY-PRODUCTS, SWEDEN (the "Technical Report") dated 19th August relating to NORRA KÄRR Project (the "Project"). In particular, Sections 17 and 20.
2. I did not visit the Project site for this Technical Report and this was primarily due to current travel restrictions related to the global COVID-19 pandemic.
3. I am currently employed as a consulting engineer to the mining and mineral exploration industry, as a Principal Consultant (Mining Infrastructure & Logistics)with SRK Consulting (UK) Ltd, with an office address of 5th Floor Churchill House, 17 Churchill Way, Cardiff, CF10 2HH, UK.
4. I graduated with a Master's Degree in Applied Environmental Geology from Cardiff University, UK in 2007. I was directly involved in mining operations and exploration prior to my Masters Degree. Since my master's degree, I've practiced my profession for 14 years with my career focused on civil engineering and mining project related consulting services worldwide. I have been directly involved in technical review and audit, due diligence, and providing various technical services for many base metal, precious metal, bulk commodity and specialist and industrial mineral mines / projects;
5. I am a Professional Engineer registered with the IOM3, Institute of Mining & Metallurgy, UK (#460270).
6. I have been employed as a geochemist in the mining and mineral exploration business and in applied academia, for the past 29 years, since my graduation from university.
7. I have read the definition of "qualified person" set out in National Instrument 43-101 of the *Standards of Disclosure for Mineral Projects* ("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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. The Technical Report is based upon my personal review of the information provided by the Issuer. My relevant experience for the purpose of the Technical Report is:

102. Principal Consultant (Mining Infrastructure & Logistics), SRK Consulting from 2012 to date;

103. Principal Geotechncal Engineer, WSP Group Plc, 2007-2012;

- Amcorp Ireland Ltd (as subsidiary of Anglo American) and the Lisheen Mine, 2000-2006.
- Experience in the above positions, and in particular with SRK Consulting, working with and reviewing mining projects with similar or comparable mining infrastructure and product logistics systems requirements in concert with mining and processing engineers, financial modelling specialists, and environmental and social specialists.
- As a Consultant, I have been involved in numerous competent person's reports for Projects with similar mining infrastructure and product logistics systems including NI 43-101 technical reports, 2012 to date.

8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. 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, for which the omission to disclose would make the Technical Report misleading.
10. I am independent of Leading Edge Minerals applying the test in section 1.5 of NI 43-101.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them or Leading Edge Minerals for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public.

Dated in Cardiff, United Kingdom, August 19th, 2021

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SRK CONSULTING (UK) LIMITED Co. No: 01575403

("signed") ("sealed")

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