Nordic Mining

Engebø Rutile and Garnet Project

Prefeasibility Study

2017-10-30	0	Approved for Use	P de Garis	D Nel	C Blake	I S Fossum
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1. Executive Summary

1.1 Introduction

This Technical Report has been prepared by Hatch Africa (Proprietary) Limited (Hatch) on behalf of Nordic Mining ASA. Hatch was commissioned to prepare a Technical Report compliant with the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves" (JORC Code 2012 Edition) for a Prefeasibility Study (PFS) of the Engebø Rutile and Garnet Project (the Project) located near Førde in Norway. The project involves the establishment of a mining and processing operation at the Engebø deposit. The mineral rights to the Engebø deposit are held by Nordic Mining's wholly owned subsidiary Nordic Rutile AS. Nordic Mining is a public company listed on Norway's Oslo Stock Exchange Axess list (OAX: NOM).

Two minerals, rutile (TiO_2) and garnet, will be produced from Engebø, which is a hard rock deposit with high grades of both rutile and garnet. Rutile is a titanium feedstock, primarily used in the production of titanium pigment, titanium metal and welding rods. The Engebø garnet, which is almandine, is used commercially in the abrasives and waterjet cutting industry.

The deposit is situated in a sparsely populated part of western Norway next to an existing deep-water ice-free port. The port is situated in a fjord adjacent to the North Sea, providing environmentally-friendly shipping to Europe and North America. The coastal climate with mild winters and summers will enable mining and processing operations to continue uninterrupted throughout the year.

The regulatory setting for the Project is driven by two key legislative requirements, namely the discharge permit and the zoning plan (planning permit). Both permits have been granted by Norwegian authorities, without further possibility for appeal.

1.2 Key Project Outcomes

1.2.1 Key Figures: Production and Financials

The foundation for the Project is the geological properties of the Engebø deposit including the Mineral Resource Statement, the mine plan and process flowsheet with estimates for capital expenditures and operating costs, and the Ore Reserve Statement. All these fundamentals are further described in this Executive Summary and in separate sections of the Technical Report.

The business case developed for the Project is based on two product revenue streams from a 1.5 Mtpa mining and processing operation, with open pit mining starting in 2021 and continuing for 16 years. Development of the underground mine will start in year 13 to enable underground production to take over from the open pit. The Life of Mine runs until 2049. The mining plan and mineral processing for the Project is based on ferro-eclogite only. Significant additional mineral resources (Inferred Resources) to extend the Life of Mine may be qualified through drilling. The mine plan has been developed in line with the guidelines of the JORC Code. Table 1-1 shows the key production and financial figures for the Project.

Table 1-1: Summary of Key Production and Financial Figures

Project Financials	Unit	Value
Pre-tax NPV @ 8.0%	US$ M	332
Pre-tax IRR	%	23.8
Payback Period	years	4.1
Net Project Pre-tax Cashflow (Undiscounted)	US$ M	1,613
Post-tax NPV @ 6.8%	US$ M	305
Post-tax IRR	%	20.8
Pre-tax Opportunity NPV @ 8.0%	US$ M	465
Production Capacity		
Initial Production Capacity ROM	Mtpa	1.5
Capital Expenditure		
Initial Capital Expenditure for Open Pit and Processing Plant	US$ M	207.2
Deferred Capital Expenditure for Underground Mine (Year 13)	US$ M	16.9
Operating Cost		
Total Operating Cost	US$/ROM tonne	16.28
Total Operating Cost	US$/Sales tonne	86.92
Mining and Processing		
Open Pit Life	years	16
Total Open Pit Ferro-eclogite Ore Production	Mt	22.6
Underground Life	years	13
Total Underground Ferro-eclogite Ore Production	Mt	19.4
Total Project Mine Life	years	29
Total Project Ferro-eclogite Ore Production (LOM)	Mt	41.9
Ferro Ore Grade – Rutile *	%	3.46
Rutile Recovery*	%	58.5
Ferro Ore Yield – Garnet *	%	17.47

Rutile and Garnet Sales (Average for First 5 Years of Operation)		
Rutile Sales	Tonnes per annum	30,525
Rutile Sales Revenue	US$ M per annum	32.7
Garnet Sales	Tonnes per annum	176,000
Garnet Sales Revenue	US$ M per annum	44.0
Garnet Basket Price	US$ per tonne	250
Rutile Price	US$ per tonne	1,070

** Diluted by waste rock: 4% for open pit and 6% for underground*

The key financials illustrate a profitable, robust and flexible business case with an Internal Rate of Return (IRR) of 23.8% and a Net Present Value (NPV_8) of US$ 332 M. The Project indicates a payback period of less than five years.

The initial capital expenditure for the Project is estimated at US$ 207 M. This includes the open pit mining operation and the processing plant facilities. A deferred capital expenditure of US$ 17 M related to the underground mining operation will accrue in year 13, and will, therefore, likely be financed from the operating cashflow.

The undiscounted net cashflow from the Project over the 29 year Life of Mine is around US$ 1.6 billion, indicating a sizeable business operation.

1.2.2 Key Project Characteristics

- Low Cost Mining Operation

The Engebø deposit is a large, outcropping high-grade resource which is open at depth and to the east and west. This allows for easy transition from open pit mining to effective underground bulk mining. The ridge profile of the deposit enables the use of a glory hole for open pit mining. Consequently, ore transportation costs are limited. The open pit stripping ratio is 1.34, with limited overburden. The geotechnical setting favours low operating costs due to low support requirements which allows for progressive mining both in the open pit and the underground operation. Hydrogeological conditions for the open pit and underground mine offer low risk of water inflows and low hydraulic conductivity through competent ore and country rocks.

- High-quality Products

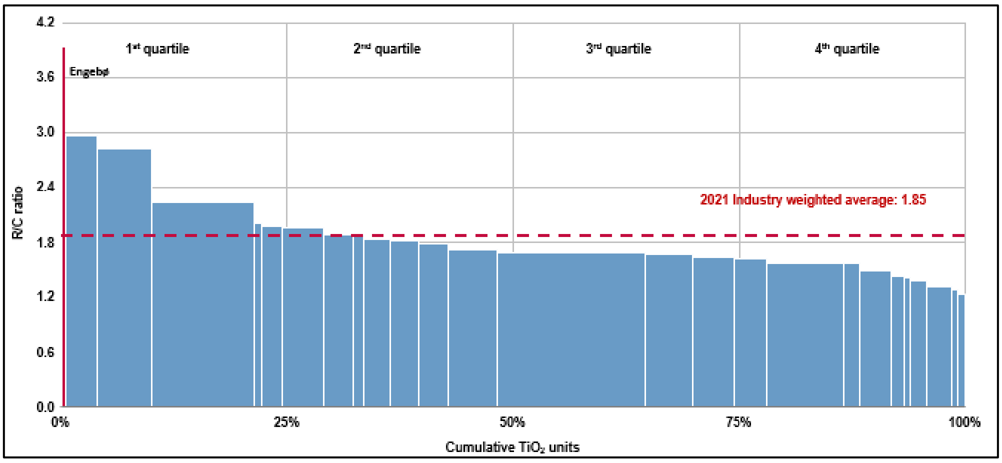
A comprehensive programme of comminution and process testwork has shown that standard technologies can be used to achieve high recoveries of rutile and garnet. The PFS programme has been successful in making pigment grade rutile at market specifications. Commercial products of both coarse and fine garnet have been made, meeting market specifications of 30/60, 80 and 100 mesh garnet products. There is potential for further optimisation of recoveries of rutile and coarse garnet.

- Favourable Infrastructure

The Project is surrounded by local infrastructure with reliable power and process water sources available within a few kilometres of the site. Natural gas for product drying is available from local suppliers. The existing deep-water port caters for simple and environmentally-friendly logistics during the construction and operational phases.

- High Revenue to Cost Ratio

The Project benefits from having high value products and relatively low operating costs. The dual mineral business case with shared production costs offers a revenue to cost ratio for rutile of approximately 3.9 over the first 10 years, ranking the Project in the first quartile amongst global titanium feedstock producers. This is illustrated in the figure below with estimations for 2021.



Source: TZMI ©

Figure 1-1: 2021 Revenue to Cash Cost Curve

Note: The Project revenue has been estimated using TZMI's long-term inducement price for rutile (US$ 1,070/t FOB) while the garnet price is assumed at US$ 250/t FOB. Operating cost estimates were provided by Nordic Mining.

1.2.3 Future Upside Opportunities and Flexibility

There are upside potentials in the business case related to, inter alia:

- Expanding the Run of Mine throughput to enable increased product sales
- Extending the Life of Mine through increased ore reserves from Inferred Resources
- Selling surplus garnet produced in the first years of production.

The above initiatives may offer a 40% improvement in the NPV_8 to US$ 465 M.

In addition to the ferro-eclogite which is the basis for the Project, the Engebø deposit also contains lower grade trans-eclogite. Possible utilisation of the trans-eclogite may offer flexibility and potential upside to the Life of Mine which could be investigated at a later stage.

1.3 Contributors

The PFS has been prepared through collaboration between a number of recognised consulting firms. The key project contributors are summarised in Table 1-2 below.

Table 1-2: Key Project Contributors

Company/Person	Primary Source of Services
Hatch Johannesburg, South Africa	• Main Technical Consultant • Mineral Processing and Comminution • Mining • Project Infrastructure and General Infrastructure • Engineering Design • Human Resources • Capital and Operational Cost Estimate • Financial Analysis • PFS coordination, report write-up and quality assurance
Adam Wheeler Independent Mining Consultant, Cornwall, United Kingdom	• Competent Person for Resource and Reserve Estimations in accordance with the guidelines of the JORC Code • Mineral Resource Statement • Mining • Ore Reserve Statement
Wardell Armstrong International Truro, United Kingdom	• Mining Geotechnical (Open Pit Mine Design)
SINTEF Trondheim, Norway	• Mining Geotechnical (Underground Mine Design) • Hydrogeology
IHC Robbins Brisbane, Australia	• Metallurgical testwork programmes and results • Flowsheet development and advisory
Mintek Johannesburg, South Africa	• Comminution testwork and results
COWI Fredrikstad, Norway	• Tailings Disposal
TAK Industrial Mineral Consultancy Gerrards Cross, United Kingdom	• Garnet market information
TZMI Perth, Australia	• Rutile market information

1.4 Property Description, Location and Access

The Engebø deposit is located between the towns of Førde and Florø in south-western Norway, with direct access to the North Sea. Engebø is on the northern side of the Førde Fjord in the Naustdal municipality, in the Sogn og Fjordane county.

The Fv 611 county road runs along the south side of the deposit before entering a 630 m long tunnel which runs through the deposit. The location of the Engebø deposit in south-western Norway is shown in Figure 1-2 and Figure 1-3 below; an aerial view of the site is shown in Figure 1-4 below.

Figure 1-2: Location of the Engebø Deposit

Figure 1-3: Location of the Engebø Deposit in the Sunnfjord Region

Figure 1-4: Aerial View of Engebø looking West towards the North Sea

Engebø is the local name of a small hill which varies in elevation from sea level to 335 masl. On the south side, the hill dips steeply into the Førde fjord, with more gentle slopes on the northern, eastern and western sides.

The climate at Engebø is characterised by long, warm days in summer and colder, darker and shorter days in winter. Snow is common in winter but proximity to the sea and relatively low altitude result in no permanent snow accumulation which allows for all-year operation. Annual rainfall is about 2,000 mm, through all four seasons. The fjord is permanently ice-free.

The town of Førde, with about 10,000 inhabitants, is located about 30 km east of Engebø. Førde has a regional airport nearby. The town of Florø with a population of 9,000 people lies 30 km west of Engebø, and has a regional airport. Between Engebø and Førde lies the municipality centre of Naustdal. The Naustdal municipality has about 2,500 inhabitants.

On the eastern part of the Engebø hill is a closed quarry and the harbour facilities constructed for rock shipment still exist. The harbour facility is designed for vessels with a capacity of up to 80 kt.

Power and water are easily accessible with a 22 kV power line close by and a reliable water supply within 9 km.

1.5 Deposit Mineralisation and Geology

The Engebø deposit is one of the world's highest-grade rutile deposits and is unique due to its substantial content of garnet. With negligible contents of radioactive elements and heavy metals, the deposit is a clean source of high-grade and high-quality titanium and garnet minerals. Unlike most rutile deposits, the Engebø rutile is contained in a hard-rock ore, a massive body of eclogite.

The deposit forms a 2.5 km long east-west trending lens that runs parallel with the Førde Fjord and the ridge, Engebøfjellet (Norwegian for Engebø Mountain). The deposit dips steeply towards the north with a dip of 60° to 85° degrees. Structural studies reveal many episodes of complex major folding and development of foliation. There is considerable exposure of eclogite on surface, although the overburden increases to the east and the country rocks frequently fold into the eclogite on its margins.

Geological investigations have determined that the eclogite can be subdivided into three different types, based on appearance and titanium content:

- Ferro-eclogite; dark and massive appearance, generally >3% TiO_2
- Transitional-eclogite; intermediate dark, generally 2 to 3% TiO_2
- Leuco-eclogite; light coloured and foliated, generally <2% TiO_2.

The contacts between the eclogite types are gradational, moving from ferro- to transitional- and leuco-eclogite. Figure 1-5 below shows the relationship between the different eclogite types.

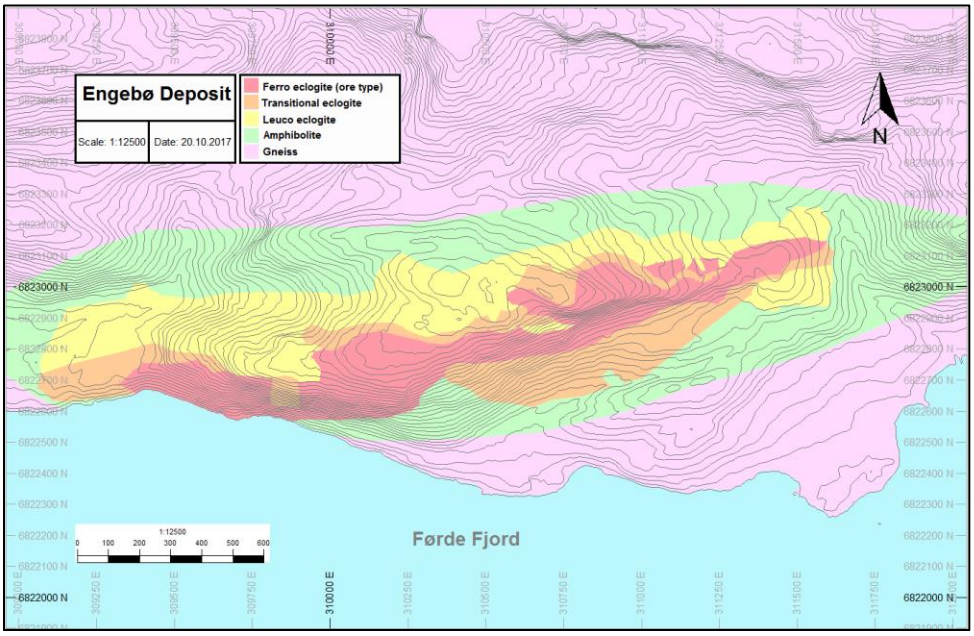


Figure 1-5: Geology of the Engebø Deposit

The main titanium bearing mineral is rutile. Only 5% of the titanium is found as ilmenite, and the presence of titanite/sphene is negligible. The rutile is practically free of uranium, thorium and other radioactive elements (less than 1 ppm). The mineral assemblage gives the rock a characteristic green and red colour. In general, the eclogite contains 40% to 50% almandine type garnet. The garnet content decreases gradually with the TiO_2 grade. Other major minerals present in the ore are pyroxene and amphibole.

1.6 Land and Mineral Tenure

Nordic Mining has access to the mining and processing plant areas both by land and sea as a result of option agreements with landowners, as well as an approved zoning plan for the area and a development agreement with Naustdal municipality.

Nordic Mining holds nine valid extraction permits covering the entire planned mining area, which gives the right to extract and utilise deposits of minerals within certain limits as described in section 32 of the Norwegian Minerals Act. The permits are valid until 12 November 2027. The Directorate of Mining may extend the duration period of the permits for up to ten years at a time. Extensions are normally granted if the deposit is deemed to be a reasonable reserve.

The land required for the processing plant is owned by three private landowners. Nordic Mining has entered into private option agreements with two of the landowners whereby the Company has an unconditional right to acquire the relevant area and exclusive right for mining operations. Renegotiations with the landowners are ongoing. By law, Nordic Mining has the right to acquire areas for operation through compulsory acquisition.

1.7 Mineral Processing

Comprehensive testwork programmes involving both comminution (crushing and grinding) and processing have been carried out, mainly in South Africa and Australia. The testwork has been successful in making a pigment grade rutile product with significantly increased recovery compared to historic testwork results. Coarse and fine garnet products that meet the market specifications of 30/60, 80 and 100 mesh products have also been achieved. Table 1-3 below shows the key results from the testwork.

Table 1-3: Summary of the Key Processing Testwork Results

Key Processing Results	Weight %
Rutile Product Grade (as TiO_2)	94.9
TiO_2 Recovery to Rutile Product	60.2
Coarse Garnet Product (>212 µm) Grade	95.4
Fine Garnet Product (<212 µm) Grade	95.5
Overall Garnet Yield	18.3

1.7.1 Comminution

Comprehensive comminution testwork has been undertaken in South Africa to develop a flowsheet capable of producing a suitable feed for garnet and rutile processing, while minimising overgrinding.

A three-stage comminution circuit was developed to produce a 550 µm to 212 µm feed for coarse garnet recovery, and a 212 µm to 45 µm fine feed for rutile and fine garnet recovery.

A series of different crushing and grinding technologies were evaluated based on:

- Mineral liberation (based on QEMSCAN analysis)
- Fines (<45 µm) generation

- Theoretical product recoveries
- Capital and operational cost
- Net Present Value
- Technology maturity and viability.

A circuit using an impact crusher in the tertiary crushing stage outperformed other methods due to better liberation and less fines production and was consequently selected for the project.

A simplified flowsheet for the selected comminution route is shown in Figure 1-6 below.

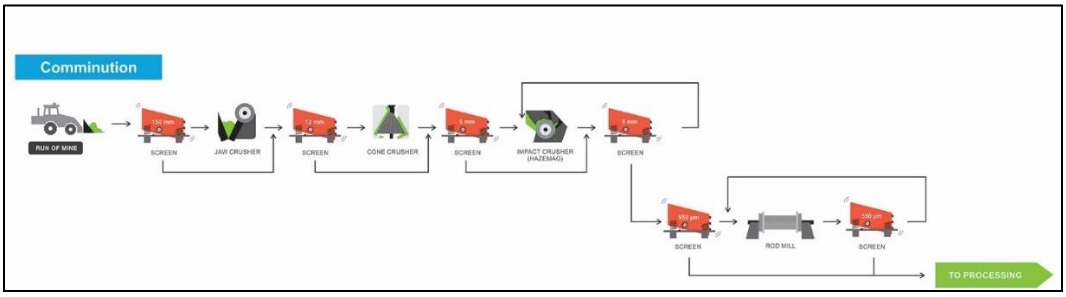


Figure 1-6: Simplified Schematic of the Selected Comminution Circuit

As illustrated, a primary jaw crusher is followed by a secondary cone crusher. The tertiary crushing stage is carried out with impact crushers operating in a closed circuit. Finally, a rod mill is employed to produce a <550 µm feed for the processing plant. The rod mill was selected as opposed to a ball mill due to the superior performance in terms of minimising fines and enhancing liberation of garnet and rutile.

1.7.2 Processing

IHC Robbins has been the main laboratory facility responsible for the process testwork. Comprehensive testwork programmes have been conducted with industrial scale equipment and run in 0.5-t to 4.0-t feed batches for testing of different process configurations. A viable process flowsheet has been developed using conventional and cost-effective process equipment.

Figure 1-7 below illustrates the developed flowsheet for garnet and rutile production.

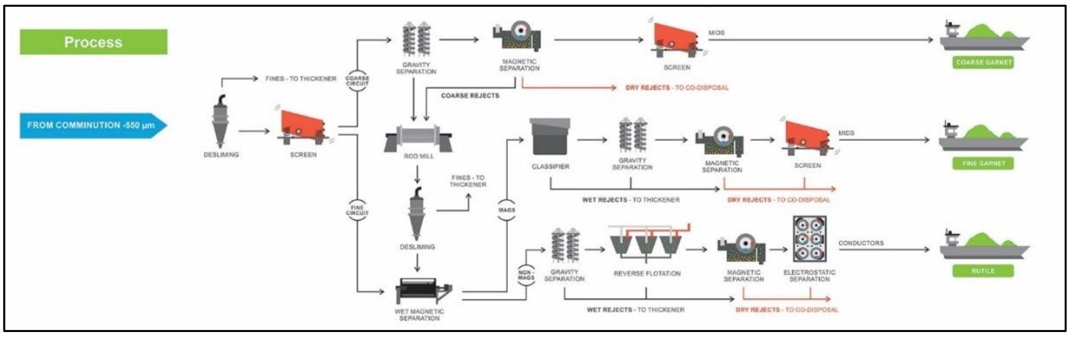


Figure 1-7: Simplified Schematic of the Selected Process Flowsheet

The developed flowsheet has been successful in making both marketable garnet and rutile products as well as maximising product recoveries.

The processing circuit, following comminution, consists of the following stages:

- De-sliming and primary screening to reject fines and to split the de-slimed feed into two size fractions for processing
- The >212 µm fraction reports to the coarse garnet process consisting of a wet gravity circuit followed by a dry magnetic circuit to recover a high-grade, coarse garnet product
- The rejects from the coarse garnet circuit are milled and combined with the <212 µm feed from the primary screening
- The <212 µm material containing the fine garnet and rutile reports to a wet magnetic separation circuit to split the feed into magnetic and non-magnetic concentrates containing the fine garnet and rutile respectively
- The magnetic concentrate is further processed through a wet gravity circuit followed by a dry magnetic circuit to produce a high-grade, fine garnet product
- The non-magnetic concentrate reports to a wet gravity circuit followed by a dry magnetic and electrostatic separation circuit, and finally to a flotation circuit to produce a high-quality rutile product.

The flowsheet employs tried-and-tested physical separation equipment extensively used in the mineral sands industry.

1.8 Mining

1.8.1 Geotechnical and Hydrogeology

Wardell Armstrong International has performed the geotechnical assessment and pit slope design for the Engebø open pit. A programme of diamond drilling and logging of oriented core, as well as geotechnical field mapping of the road tunnel exposure through the deposit, has been conducted. Recommended inter-ramp angles used for pit optimisation are between 54°and 67°, with one sector at 47°. Since the open pit is at high relief there is little risk of water inflows. In addition, hydraulic conductivity was found to be low with little to no ingress of water into excavations.

SINTEF Building and Infrastructure has undertaken a geotechnical analysis of potential underground mining methods suitable for Engebø. The rock mass quality was found to be high and all proposed underground mining methods will be stable. It is recommended to undertake in-situ stress assessments in the next phase of the underground study.

1.8.2 Mining

Norway is renowned for its hard rock mining, both open pit and underground. From a mining perspective, the deposit supports a low-cost production operation for the following reasons:

- The high relief location of the open pit enables the use of a glory hole for open pit mining. The ore pass (glory hole) will be raise-bored into the pit shell, with a crusher and silo system developed below the pit. As consequence, ore transport costs are minimal
- An attractive stripping ratio with almost no overburden. The waste is determined by the cut-off grade with the potential to stockpile low grade ore close to the pit for processing later
- A large resource which is open at depth and to the east and west. This allows for an easy transition to effective underground bulk mining
- The geotechnical setting does not require intensive support and further favours low operating costs in both open pit and underground mining.

The mining studies undertaken in this phase set out to determine the following:

- Plant capacity
- Mining cut-off grade
- Mining volumes and the balance between open pit and underground
- Mine design and scheduling for both open pit and underground.

The optimised business case results in an open pit mine at 1.5 Mtpa for 16 years followed by underground operations for a further 13 years. Figure 1-8 below shows the open pit and underground mine design used to derive the mine plan. Only the high-grade ore (ferro-eclogite) will be processed, with significant upside to the Life of Mine if transitional-eclogite is processed at a later stage.

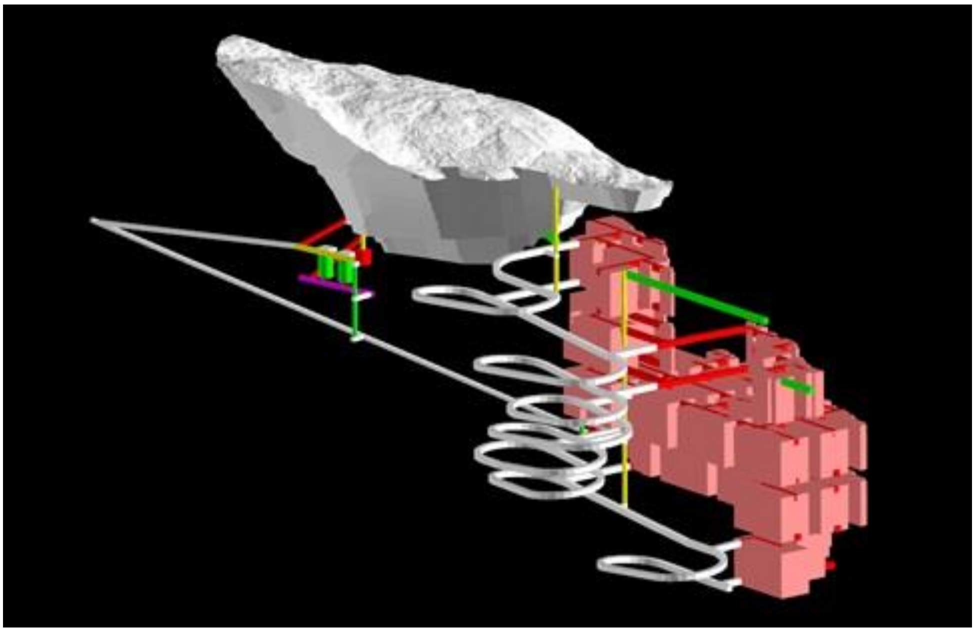


Figure 1-8: Open Pit and Underground Mine Design (looking South-East)

The key metrics for the mining operation are summarised in Table 1-4 below.

Table 1-4: Key Schedule Production Statistics

Activity	Units	Life of Mine
Open Pit Mining		
Production start		2021
Capacity	Mtpa	1.5
Life	years	16
Waste Mined	kt	30,446
Ferro Ore Mined	kt	22,616
Strip Ratio	tonne waste: tonne ore	1.34
Underground Mining		
Production start		2034
Capacity	Mtpa	1.5
Life	years	13
Waste Mined	kt	10,020
Ferro Ore Mined	kt	19,432
Feed to Plant *		
Ferro Ore Feed to Plant	kt	41,896
Ferro Ore Grade – Rutile	%	3.46
Ferro Ore Grade – Garnet	%	40.45

** Diluted by waste rock: 4% for open pit and 6% for underground*

1.8.3 Mining Trade-offs Performed

Mining optimisation started with the aim of determining the plant capacity to select, and thereafter, decide the best mine design and schedule. The mining optimisation process was iterative, with information updated as the process testwork and design dictated which recoveries and flowsheets were possible. A full description of the process is given in Section 12.3.

The following conclusions were reached as the study progressed through the iterative process:

1. Pit optimisation resulted in a final pit with very little difference between the different options. The size of the pit is constrained to the south-east and to the west by the owner's boundary (permitting boundary) and all pit shells extend to these boundaries
2. There are minor differences to the north for the different options. Since the north has mainly low grade leuco-eclogite, the higher cut-offs would exclude some of this ore
3. The final pit depth is determined by the slope angles and the boundaries in the southerly, easterly and westerly directions. There is no significant difference in pit depth for the different options

4. The results of the metallurgical testwork on recovery of coarse garnet will strongly impact the business case
5. The impact of higher feed grades and recoveries outweighs the effect of a higher stripping ratio. The business case is improved for higher grade and lower volume scenarios.

Based on the above, the final pit was selected for design and further scheduling optimisation. The pit design was performed with input parameters as per Table 1-5 below.

Table 1-5: Key Pit Design Parameters

Parameters	Unit	Value
Bench height	m	15
Berm width	m	5
Bench face angle	o	Varies per pit sector
Ramp grade	%	10
Ramp width	m	23
Turning clearance diameter	m	29
Top bench elevation	m	330
Bottom bench elevation	m	90

The scheduled was improved by using a two-stage pushback strategy as shown in Figure 1-9 below to defer waste stripping. The waste includes all Inferred material, as well as all leuco- and transitional-eclogite. This lower grade ore may be stockpiled for potential future use.

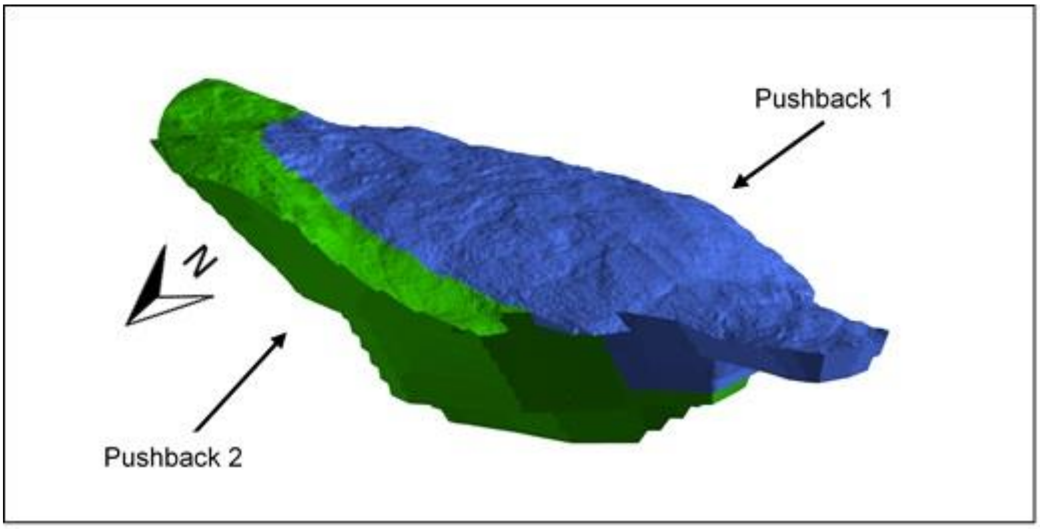


Figure 1-9: Pit Design Pushbacks

A key feature of the design is the use of a glory hole inside the pit. The effect of this is to reduce the ore haulage requirements since most benches are close enough to dump directly into the raise-bored hole. Ore is then crushed and transferred to underground silos before being conveyed to the plant. The underground infrastructure for both the pit and underground mining can be seen in Figure 1-10 below.

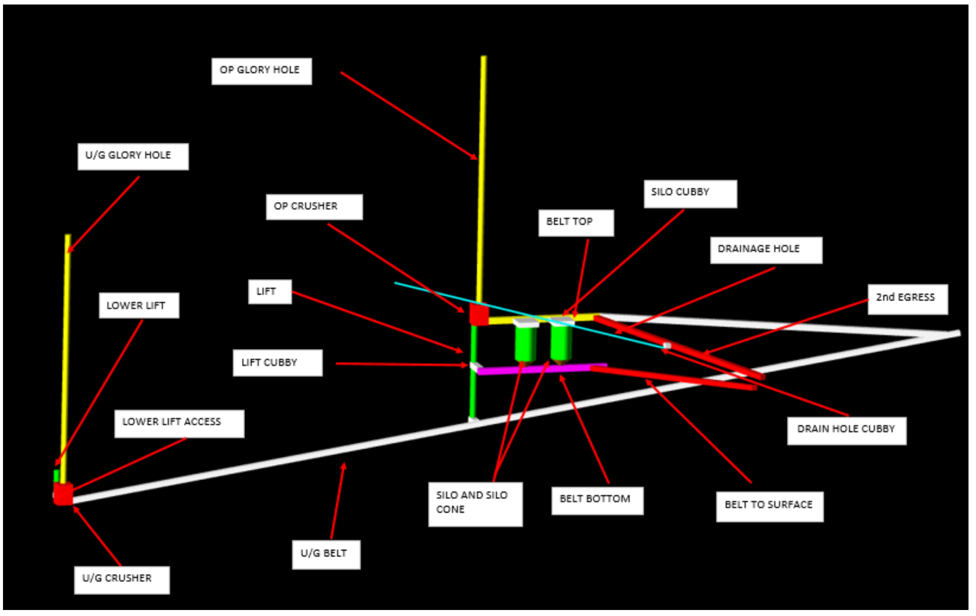


Figure 1-10: Underground Infrastructure Layout

1.8.4 Underground Mining

Underground mining will be performed in continuation of the open pit operation. A trade-off study of alternative underground mining methods has been performed and the shortlist as shown in Table 1-6 below compiled.

Table 1-6: Mining Methods Shortlist for Engebø

Mining Method	Applicable to Engebø	Rationale for Applicability
Sub-level Open Stoping	Yes	Low cost mining and highly productive and flexible
Sub-level Long Hole Open Stoping	Yes	Low cost mining, highly productive and flexible; modern drilling technology will most likely make it cheaper than sub-level open stoping
Vertical Crater Retreat	No	Highly constrained by sequence, but can be used to minimise development in waste
Cut and Fill	No	High cost of mining with backfill
Sub-level Caving	No	No surface subsidence permitted, high upfront development capital
Block Caving	No	No surface subsidence permitted, high upfront capital and long development time
Room and Pillar	Yes	Flexible mining, highly mechanised, medium to high productivity
Shrinkage Stoping	No	Low productivity and unsafe

Long hole open stoping was selected as the recommended mining method, primarily because it utilises the favourable geotechnical conditions and latest technology and, therefore, will have the lowest operating cost of the three potential methods identified.

The underground mine has been designed around the use of long hole open stoping whilst taking into consideration:

- The provincial road tunnel running through the deposit and the geotechnical constraints associated with it
- The geotechnical considerations, as specified by the geotechnical consultants, including boundary pillars around the pit and the fjord, and the recommended stope, pillar and sill dimensions
- The different ore types and grades available.

Underground access is from the pit in time to develop the decline, ramp and first level of stopes as shown in Figure 1-11 below. Ore handling is carried out in a similar manner to the open-pit with an ore pass down to a belt level which feeds the crushers.

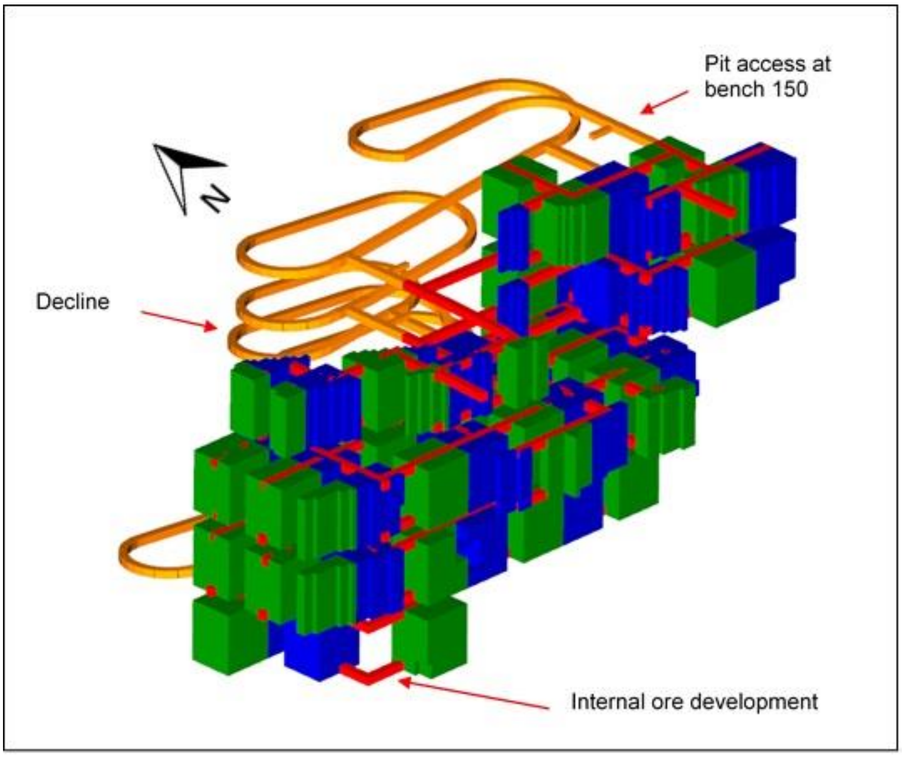


Figure 1-11: Decline Development and Ore Development

1.8.5 Plant Capacity Analysis

The objective of the capacity analysis study was to determine the project capacity range that delivered the most robust business case. Various aspects that influence the outcome of the business case were simulated as the PFS matured. The capacity analysis studies have a high degree of knowledge about variations in ore types and support a business case where the initial capacity of 1.5 Mtpa was chosen for the mining of ferro-eclogite only. The business case secures a suitable start-up and provides several possibilities for expansions in capacity and ore types as the project evolves.

A full description of the various processes through to the chosen capacity is given in Section 12.8.

1.9 Mineral Resource Estimate

In 2008, Nordic Mining assigned the independent Qualified Person, Adam Wheeler to make an updated resource estimation for the Engebø deposit in accordance with the guidelines of the JORC Code. The resource estimate was published in a scoping study and is summarised in Table 1-7 below.

Table 1-7: 2008 Resource Estimate (@ 3% TiO_2 Cut-off)

2008 Estimate	Tonnes (Mt)	TiO_2 Grade (%)
Indicated	31.7	3.77
Inferred	122.6	3.75

The mineral resource estimate for Engebø was updated in 2016 by Mr. Wheeler following completion of a diamond drilling and surface sampling campaign. The estimate substantially improved and increased the 2008 classification and enabled a qualified quantification of the garnet. The 2016 resource estimate, as shown in Table 1-8 and Table 1-9 below, is the basis for the PFS, as well as the Mineral Resource and Ore Reserve Statements made in this report.

A third party independent review of the mineral resource estimate has been carried out by SRK Consulting (UK) Limited (SRK) in December 2016. SRK concluded that the mineral resource estimate did not contain any fatal flaws and that the geological model produced for use in this PFS was fit for purpose.

Table 1-8: 2016 Mineral Resource Estimate (3% TiO_2 Cut-off)

TiO_2 Cut-off	Classification	Tonnes (Mt)	Total TiO_2 (%)	Garnet (%)
3%	Measured	15.0	3.97	44.6
	Indicated	77.5	3.87	43.6
	Total – Measured and Indicated	**92.5**	**3.89**	**43.7**
	Inferred	138.4	3.86	43.5

Table 1-9: 2016 Mineral Resource Statement (2% TiO_2 Cut-off)

TiO_2 Cut-off	Classification	Tonnes (Mt)	Total TiO_2 (%)	Garnet (%)
2%	Measured	19.0	3.68	43.9
	Indicated	105.7	3.51	43.0
	Total – Measured and Indicated	**124.7**	**3.53**	**43.2**
	Inferred	254.5	3.22	42.5

Notes:

- Grades presented above are total TiO_2
- Resource below sea level has been restricted by a boundary no closer than 50 m to the edge of the fjord
- Above Mineral Resources are inclusive of Ore Reserves.

At present, the Ore Reserve is defined with a 3% cut-off. The 2% cut-off includes ferro- and transitional-eclogite and has been included to facilitate the investigation of mining and processing options. The transitional-eclogite represents a potential future ore reserve. All leuco-eclogite (<2% TiO_2) has been excluded from the resource.

The resource remains open to the east, west and at depth, with potential to convert Inferred resources to Measured and Indicated Resources with additional drilling.

The updated resource delineates the garnet at an average grade of 43.7%.

The 2016 drilling and surface sampling campaign focused on the open pit area as indicated by the red drillholes in Figure 1-12 below, with most of the increases in the Resource Estimate occurring in the open pit.

The garnet content is 40% to 50% in the ferro-eclogite and transitional-eclogite, and 30% to 40% in the leuco-eclogite.

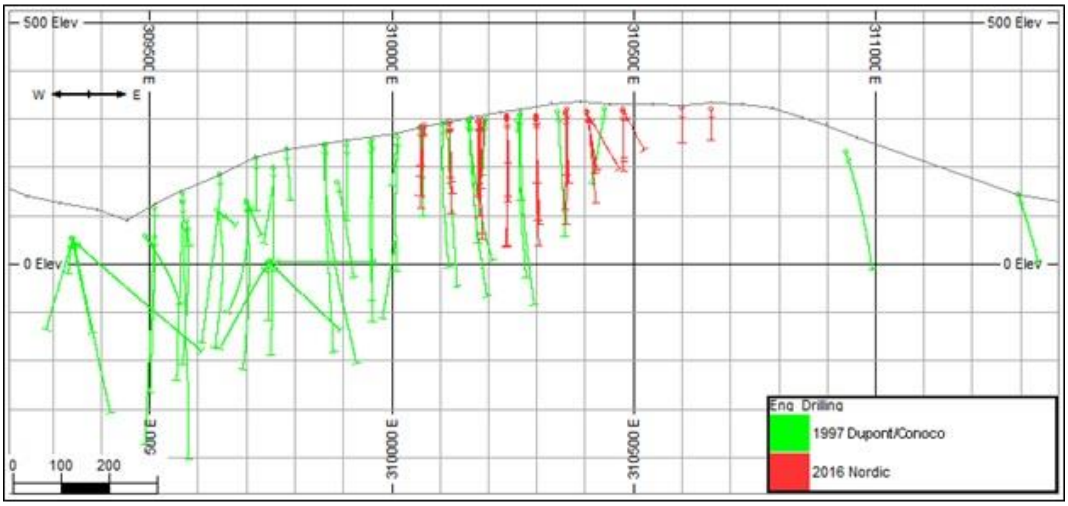


Figure 1-12: West-East Long Section of Drillhole Data

1.10 Ore Reserve Estimate

The ore reserve for the mine plan was estimated and qualified by Mr. Wheeler. The Ore Reserve Statement is presented in Table 1-10 below. The reserve estimation was carried out using Datamine and DESWIK software.

Table 1-10: Ore Reserve Statement

Ore Type	Proven Reserves			Probable Reserves		
	M Tonnes	TiO_2 %	Garnet %	M Tonnes	TiO_2 %	Garnet %
Ferro Ore - Open Pit	8.519	3.87	43.8	13.826	3.54	41.8
Ferro Ore - Underground	1.675	3.49	37.8	17.876	3.21	37.8
Ferro Ore - Total	10.194	3.81	43.4	31.702	3.35	39.5

The basis of conversion of Mineral Resources to Ore Reserves is as follows:

- Ore Reserve estimate is as of 30 September 2017
- Only Measured and Indicated Resources have been used to determine reserves; all Inferred Resources within the mineable envelope have been classified as waste
- Open pit mining is carried out for the first 16 years; thereafter the mining method is bulk underground mining (long hole open stoping)
- The open pit mine design is based on the recommendations of the geotechnical consultants for all pit design parameters
- The underground mine design is based on recommendations of the geotechnical consultants, assuming 100 m-long stopes, 45 m wide and 60 m high, with continuous pillars 20 m wide between stopes and sills 15 m thick above and below the stopes
- The garnet grades as reported above were not used to determine the final product volumes for garnet. Instead, a yield approach was used, which was considered to be more applicable for determining recoveries of a bulk mineral such as garnet. The yield approach assumed a yield of 17.5% garnet for ferro ore
- A rutile recovery of 58.5% was assumed
- A cut-off of 3% on TiO_2 has been applied to ferro ore
- Ore losses of 5% have been assumed throughout the mine plan
- Dilution of 4% for open pit and 6% for underground has been applied with a dilution grade of 0% for rutile and garnet.

1.11 Infrastructure

The Project's infrastructure includes open pit mine facilities, a haul road, underground mine facilities, process plant facilities, tailings disposal facilities and infrastructure (power, water, gas, access roads and communications).

The open pit mine facilities include a security fence, offices, preparation of the platform for a heavy vehicle service area and an LDV workshop, a haul road to the waste rock storage facility, a waste rock facility, agri-soil and subsoil stockpiles, a settling dam and a domestic and industrial waste site.

A new haul road will be constructed from the Fv 611 county road up to the open pit. The haul road will be a new construction, following the route of the existing gravel access road to the top of Engebø for part of its route.

Underground mine facilities will be built in two phases to support open pit mining from the start of production and underground mining once the open pit has been mined out. To support open pit mining, open pit dewatering facilities will be installed and underground excavations will be built, which include a glory hole plus grizzly arrangement in the pit, a primary crusher and crusher chamber, a silo and ore reclaim system, top and bottom access to the silo system, an ore conveyor belt from the silo reclaim system to the plant site, and a second egress from the top of the silo system to the plant site. For underground mining, a new ore pass (underground glory hole) and primary crusher chamber and crusher will be constructed to the east of the main underground mining areas. The crusher chamber will be connected to the existing silos and reclaim system by means of an underground conveyor belt system; electrical installations to support underground mining include switchgear, substations, cabling and transformers.

The process plant site facilities include water storage and process water reticulation facilities, natural gas storage facilities, compressed air systems, dust and off-gas handling systems, potable water and sewage treatment plants, fire protection and HVAC (Heating, Ventilation and Air Conditioning) systems, security fencing around the site, and an upgrade/rejuvenation of the existing quay. Administration and support buildings to be built at the process plant site include an administration office building, a process plant control room building, a change house, ablution and canteen building, a first aid building, and process plant laboratory and stores buildings.

A sea disposal system for tailings will be installed consisting of a mixing tank, where seawater is added to tailings, and an outfall line which transports the seawater/tailings mix via a gravity driven system to the seafloor. Equipment for continuous monitoring of the discharge system will be a part of the tailings disposal system.

Bulk power for the Project will be supplied by SFE, the regional power supply company. By upgrading the existing 22 kV grid, a new 22 kV grid line across the fjord and additional grid reinforcements, SFE will have sufficient grid capacity and reliability for the Project. The power intake transformers at Engebø will be supplied by SFE.

Bulk water supply for the Project will be purchased from SFE and will be sourced in a dedicated pipeline from the Skorven power plant, situated at the southern side of the fjord.

Natural gas for the Project will be supplied by a local gas supplier. An area has been located on the overall site layout for gas storage tanks and the supplier will transport gas to site by road regularly.

The main county road (Fv 611) providing access to the Engebø site will be diverted around the process plant facility.

For communications to site, it is envisaged that at the time of project implementation a 5G communication link will be available at the Engebø site. Facilities will be installed on site to ensure effective communications from the incoming 5G connection.

1.12 Environment and Permitting

Nordic Mining's overarching principle when operating the Engebø deposit is to adopt a good citizen approach and demonstrate that it can plan, build and operate Engebø in a manner that:

- Demonstrates environmental responsibility and adheres to the environmental terms of the permits and approvals requirements
- Continuously improves the environmental track record
- Commits to a sustainable long-term mining operation that will benefit the community.

1.12.1 Zoning Plan and Discharge Permit

The regulatory setting for the Project is driven by two key legislative requirements for Nordic Mining to establish a mining and processing operation at Engebø, namely the discharge permit and the zoning plan (planning permit). Both requirements have been fully met and the initiative is compliant with Norwegian environmental legislation.

The zoning plan was adopted by the local municipalities in 2011 and finally approved by the Ministry of Local Government and Modernisation on 17 April 2015. The zoning plan allows for, and provides guidelines on, the operation of the following activities:

- The processing site at Engebø
- The extraction of rock mass in open pit production and underground mining
- The service area at Engebø
- The waste rock disposal site in Engjabødalen
- Subsea area, tailings deposition in the Førde Fjord
- The works road running between the deposit and the processing plant
- The rerouting of county road Fv 611
- The rerouting of a 22 KV power line and the stringing of a new cable between the plant area and the deposit.

A final discharge permit for Engebø was issued on 29 September 2016. The discharge permit clearly states how the mining operation and the tailings deposition for the Project should be run and regulates areas such as:

- Dust and sound emission
- Mining, processing and blasting activity
- Tailings deposition
- Requirements for monitoring environmental footprints
- Requirements for environmental baseline studies.

The regulatory framework ensures that measures are put in place to minimise the environmental impact.

1.12.2 Environmental Impact Assessment

To obtain the discharge permit and zoning plan, a comprehensive Environmental Impact Assessment (EIA) programme with numerous environmental studies has been carried out between 2008 and 2015. Forty-four environmental and social responsibility studies/reports have been developed to date over the life of the Project.

A major topic for the EIA was the deposition of tailings. The permitted solution is a sea disposal system for transporting tailings through a pipeline down to the seafloor of the Førde Fjord, at a depth of 320 m. The fjord basin is a sedimentation environment confined by thresholds to the inner part of the fjord and by a glaciation sill to open sea. A 4.4 km^2 area of the fjord seafloor has been regulated for tailings deposition.

Detailed baseline studies were carried out to map the biodiversity in the fjord; this included test fishing, grab sampling and remotely operated vehicle (ROV) investigations. Currents, salinity, turbidity and temperature were measured in the fjord throughout a 12- month period to document the fjord environment.

The main conclusions regarding the fjord disposal solution from the EIA studies were:

- The tailings will sediment within the area regulated for disposal, which comprises 5% of the total fjord area
- The currents in the tailings area are moderate and there is limited risk of erosion currents that could potentially transport tailings outside the regulated area
- Limited effects are expected outside the regulated area and in the water column above the tailings outlet
- The tailings are benign, meaning they consist of non-harmful naturally occurring minerals with negligible contents of heavy metals
- The chemical additives that follow the tailings from flotation and thickening are biodegradable and in non-harmful concentrations
- The tailings consist of mainly sand and silt fractions and a little from the clay fraction; they are somewhat coarser than the sediments constituting the fjord bottom today
- The baseline studies showed that the fjord habitat has biodiversity that is typical for western Norwegian fjords
- There are no corals found in the tailings area
- The tailings deposits pose little threat to cod which have their breeding grounds in shallow fjord areas
- The tailings solution poses little threat to endangered fish that dwell in the fjord
- The tailings will affect bottom living organisms within the regulated area where the sedimentation rate is high. Mobile species such as fish will avoid areas with high turbidity

- The tailings will likely be recolonised within a few years after the tailings deposition ends. The biodiversity is expected to return to as good a state as it was before the depositing took place
- The fjord has no commercial fishing, but some recreational fishing. The tailings will not affect recreational fishing in the fjord
- The tailings will not affect fish farms that are operated in the fjord.

Norway has long-term experience with sea disposal of tailings. Currently there are five active tailings deposits in Norway and two (including Engebø) have recently been permitted. Experience with fjord deposition in Norway is, for the most part, positive. Advanced systems for continuous monitoring exist and there are established best practise guidelines for tailings deposition to limit the environmental footprint.

1.12.3 Socio-economic Studies

The social consequences of the planned mining activities have been assessed as part of the EIA. The EIA concludes that the Project will have positive effects on the local settlements as well as the local business community. The location of the Project, close to the cities of Førde and Florø, makes it attractive for the recruitment of local labour. Based on assumptions regarding local settlement for future employees, the study concludes that that the Project will have significant positive consequence for the economy of the Naustdal municipality.

It is expected that the regional contributions to the capital investments may represent up to 17%. It is estimated that during operation the local region will represent approximately 17% of regular supply and services, based on statistics from other mining industries in Norway. The direct and indirect employments from the Project are estimated to be approximately 110 direct employees with an indirect employment factor of 2.9. The Project is, therefore, likely to generate approximately 320 jobs in total, of which 60 to 90 will be indirect employees in the local region. Due to historic imbalances in the local labour market, relocation and commuting of the workforce has been a significant factor in the local region. The study indicates that the Project will have a positive impact on local settlement and the commuting trend.

1.12.4 Other Permits Required

Nordic Mining will need to apply to the Directorate of Mining for an operating license upon commencement of operations.

1.13 Capital Cost Estimate

The capital estimate to establish the open pit mining operation and the process plant is US$ 207 M, as summarised in Table 1-11 below.

Table 1-11: Capital Estimate

Capital Estimate	US$ M
Open Pit Mining	10.027
Comminution	16.802
Mineral Processing	61.573
Tailings Handling and Disposal	7.045
Product Storage and Loadout Facilities	13.108
Infrastructure	22.565
Indirects (excluding contingency)	41.862
Contingency	34.194
Total	**207.176**

The estimated cost of establishing the underground mine after 15 years of open pit operation is US$ 17 M (in current money terms), as summarised in Table 1-12 below.

Table 1-12: Deferred Capital Estimate for Underground Mining

Deferred Capital Estimate – Underground Mining	US$ M
Underground Mining	7.833
Comminution	2.970
Indirects (excluding contingency)	2.747
Contingency	3.381
Total	**16.931**

The contingency allowances in the above capital estimates has been calculated by means of a quantitative risk analysis (QRA) to determine the Project's capital risk profile. Contingency at an 80% confidence level has been allowed for, equating to 19.8% of the initial project capital cost.

The above estimates exclude taxes (general sales tax, fringe benefits tax, sales tax, and any government levies and taxes), working capital, sustaining capital and Stay in Business (SIB) capital.

1.14 Operating Cost Estimate

The operating cost estimate developed from the production schedule is summarised in Table 1-13 below. The total average operating cost over the Life of Mine is US$ 16.28/RoM t and US$ 86.92/product t (rutile and garnet combined, Free on Board).

Table 1-13: Operating Cost Summary

Item	Unit	Cost/t (US$)
Open Pit - Waste Mining	Waste tonne	1.89
Open Pit - Ore Mining	Ore tonne	1.82
Underground Decline Development – Waste Mining	Waste tonne	5.03
Underground Decline Development – Ore Mining	Ore tonne	3.28
Underground Horizontal Development – Waste Mining	Waste tonne	5.27
Underground Horizontal Development – Ore Mining	Ore tonne	4.59
Underground Stoping – Waste Mining	Waste tonne	3.67
Underground Stoping – Ore Mining	Ore tonne	2.93
Comminution	ROM tonne	3.93
Process	ROM tonne	5.39
Tailings Disposal	ROM tonne	0.17
Product Dispatch	ROM tonne	0.33
Overheads	ROM tonne	1.36
Total Cost	ROM tonne	16.28
Total Cost *	Sales tonne	86.92

** Cost per Sales tonne reflects cost for all sales tonnes (garnet and rutile combined)*

1.15 Project Economics

1.15.1 Key Project Financials

The key financials for the Project's business case are summarised in Table 1-14 below. The NPV of US$ 332 M is a real pre-tax value discounted by 8%, which is the assumed Weighted Average Cost of Capital (WACC). The IRR of 23.8% is real with no escalations applied. The payback period is the number of periods once operations start that generate positive cashflow equal to the capital invested. The Life of Mine is the number of operating years for the reserve derived in line with the guidelines of the JORC Code. The profitability index is a ratio of the NPV divided by the capital discounted to a present value using a WACC value of 8%.

Table 1-14: Key Project Financials

Metric	Unit	Value
NPV @ 8%	US$ M	332
IRR	%	23.8%
Payback Period	years	4.1
Life of Mine	years	29
Profitability Index	ratio	3.1

The post-tax key financials have been calculated using a post-tax WACC of 6.8%. A post-tax NPV of US$ 305 M and an IRR of 20.8% are estimated assuming a 60% debt financing of the Project and general implementation of accounting standards for depreciation and tax calculation.

There are upside potentials in the business case related to, inter alia:

- Expanding the Run of Mine throughput to enable increased product sales
- Extending the Life of Mine through increased ore reserves from Inferred Resources
- Selling surplus garnet produced in the first years of production.

The above initiatives may offer a 40% improvement in the NPV_8 up to US$ 465 M.

In addition to the ferro-eclogite which is basis for the Project, the Engebø deposit also contains lower grade eclogite. Possible utilisation of the trans-eclogite may offer flexibility and potential upside to the Life of Mine which possibly could be investigated at a later stage.

1.15.2 *Sensitivity*

Figure 1-13 below illustrates that the NPV is positively correlated to rutile revenue and garnet revenue, and negatively to CAPEX and OPEX. Garnet revenue has a larger influence than rutile. OPEX has a slightly larger influence on NPV than CAPEX.

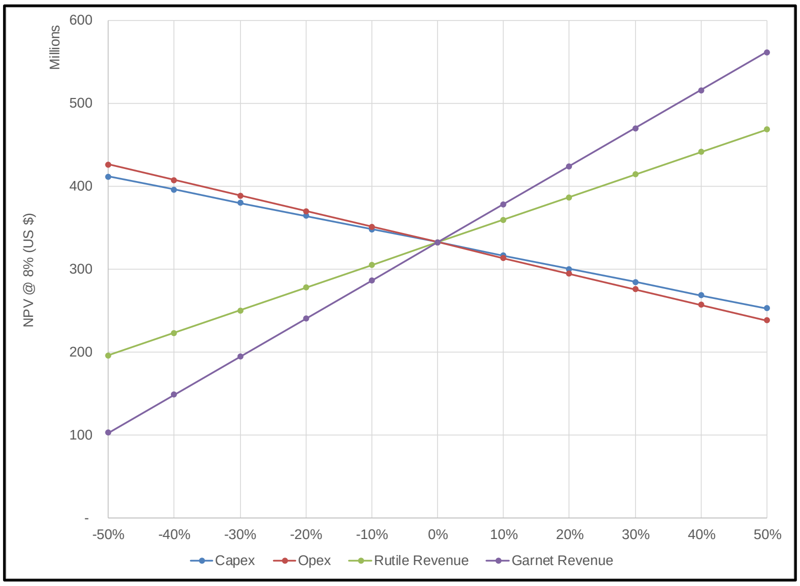


Figure 1-13: NPV Sensitivity

1.16 Markets

1.16.1 *Rutile*

The key minerals mined to supply titanium raw materials are ilmenite and rutile; Engebø plans to produce rutile. The global TiO_2 pigment market accounts for approximately 90% of all titanium feedstock demand, and is, therefore, the dominant driver of offtake.

TZMI, a leading global technical and marketing consultant, considers that the longer-term outlook for rutile production indicates that a significant supply deficit will develop if no new projects are commissioned. The global supply of rutile is set to decline considerably during the period 2017 to 2025, with output in 2025 expected to be 50% lower than 2016 levels.

Prices of rutile shipments into western markets are generally in the range US$ 750/t to US$ 850/t FOB at the current time (second half of 2017). In the longer term, TZMI expects prices to trend towards the inducement price level (US$ 1,070/t real 2016 dollars) by 2020/2021, along with other high-grade chloride feedstocks, to ensure there is sufficient new supply being induced to meet demand growth.

TZMI has used the preliminary product specifications generated during the PFS testwork campaign as a basis for assessing the Engebø rutile product quality. The preliminary quality of the rutile compares favourably to most other competing products. Based on product quality and particle size distribution, the Engebø rutile is considered being a suitable feedstock for chloride pigment and titanium metal applications.

Global demand for rutile for pigment and titanium metal end-use is estimated to reach 540 k TiO_2 units by 2020 and 710 k TiO_2 units by 2025. TZMI's current forecast indicates that supply deficit of the global rutile market could reach more than 250 k TiO_2 units by 2020 and 600 k TiO_2 units by 2025. As such, the planned output of approximately 30 ktpa from Engebø should easily be absorbed by the market by the time the Project comes on stream.

From a pricing perspective, TZMI estimates that the planned rutile product should be able to achieve the long-term price of a standard rutile (US$ 1,070/t FOB real 2016 dollars) if targeted at chloride pigment or as a feed for titanium sponge manufacture. US$ 1,070/t FOB has been used as the basis for financial evaluation in this study.

From a rutile revenue-to-cash-cost perspective, a primary measure of competitiveness used by TZMI for individual operations in the industry, the Project is positioned in the first quartile, which indicates a robust cost position.

1.16.2 Garnet

The primary markets for garnet are in abrasive blasting and waterjet cutting, although for some coarse grades there is also a market in water filtration. There is also a market in abrasion resistant materials such as in flooring, but this market is primarily restricted to China at the current time.

Whilst titanium feedstock production including rutile has developed in line with global economic growth ore many decades, garnet production has primarily developed over the last 20 to 25 years. The current world production of garnet is estimated at 1.4 Mtpa; India is the largest producer (estimated production is 450 ktpa to 500 ktpa); with Australia being the next largest producer at an estimated 280 ktpa production level. China is the third significant producer at an estimated 200 ktpa to 300 ktpa output. In line with the country production statistics, India and Australia are the primary exporters to world markets at estimated levels of 478 ktpa and 293 ktpa respectively.

Logistics is an important element of garnet marketing, with deliveries often expected within a few days of order. In this regard Engebø is very well placed with its direct access to the North Sea and, thereafter, major European waterways, resulting in lower transport costs and reduced time to market relative to the key global producers in India, Australia and China.

The three garnet products which Engebø will target are 80 mesh waterjet, 100 mesh waterjet and 30/60 mesh blast market. Whilst there are no terminal markets for garnet and no reliable published prices for products, average price trends show a clear uptrend from 2008 onwards. Shortages of supply from India at the current time continue to support higher prices, and TAK expects garnet prices in Europe going forward to average US$ 275/t to US$ 300/t on a CIF (Cost, Insurance and Freight basis) for an 80 or a 100 Mesh material range and US$ 300/t to US$ 320/t for 30/60 mesh grades. For the purposes of financial evaluation in this study, an FOB garnet basket price of US$ 250/t has been assumed. This number is in line with expected export prices as shown above, but assumes that some recovery in the growth rates of global economies will occur. The US$ 250/t price is based on an average price for the three products which Engebø is expected to produce in approximately equal volumes.

Nordic Mining has signed a Memorandum of Understanding with a leading, international producer of industrial minerals. The parties intend to establish long-term cooperation within development, production, sales, marketing and distribution of garnet products from Engebø. This may include an off-take agreement, joint marketing, and sales and distribution arrangements for garnet products to the international markets.

1.17 Project Schedule

A high-level Definitive Feasibility Study (DFS) schedule has been developed based on start in November 2017 with a duration of approximately 12 months. Thereafter, it is likely that the Project will proceed directly into a FEED (Front End Engineering Design) phase, where critical path engineering and procurement work will be continued to expedite the start of construction.

Once the DFS has been completed in Q4 2018, the following preliminary and indicative milestones apply:

- Start of FEED – Q4 2018
- Completion of FEED – Q2 2019
- Start of construction – Q2 2019
- End of construction – Q2 2021
- Start of commissioning and production ramp-up – Q2 2021
- End of commissioning – Q3 2021
- End of production ramp-up – Q4 2021.

2. Introduction

2.1 Terms of Reference and Purpose of the Report

Nordic Mining commissioned Hatch Africa (Proprietary) Limited (Hatch) to prepare a Technical Report which corresponds to the guidelines of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' (The JORC Code 2012 Edition) for a Prefeasibility Study (PFS) of the Engebø Rutile and Garnet Project (the Project) located near Førde, Norway.

This report provides Mineral Resource and Ore Reserve estimates using a classification of resources and reserves in accordance with the JORC Code 2012 Edition.

2.2 Qualifications of Consultants and Project Team

The information in this report that relates to Mineral Resources and Ore Reserves is based on information compiled by Mr. Adam Wheeler, who is an independent mining consultant. Mr. Wheeler is a Fellow of the Institute of Material, Minerals and Mining and has adequate experience, which is relevant to the style of mineralisation and type of deposit under consideration, and to the activity he is undertaking, to qualify as a Competent Person in terms of the JORC Code 2012 Edition. Mr. Wheeler consents to the inclusion of such information in this report in the form and context in which it appears.

2.3 Sources of Information

The persons and companies that have delivered information and results for the PFS are set out in Table 2-1 below.

Table 2-1: Sources of Information

Competent Person	Independent of Owner	Company	Primary Area of Information	Relevant Sections
	No	Nordic Mining in Oslo, Norway	• Executive Summary • Property Description • History • Land/Mineral Tenure and Licences • Geology • Drilling, Sampling and Ore Characterisation • Mining Geotechnical • Hydrogeology, Hydrology and Geochemistry • Market Information • Environment and Social Responsibility	Section 1 Section 3 Section 4 Section 5 Section 6 Section 7 Section 9 Section 10 Section 17 Section 19
	Yes	Hatch in Johannesburg, South Africa	• Introduction • Mineral Processing • Mining • Project Infrastructure	Section 2 Section 11 Section 12 Section 14

Competent Person	Independent of Owner	Company	Primary Area of Information	Relevant Sections
			• General Infrastructure • Engineering Design • Health and Safety • Human Resources • Capital Cost Estimate • Operating Cost Estimate • Financial Analysis • Feasibility Study Planning • Execution Planning • Risks and Opportunities • Value Improving Practices	Section 15 Section 16 Section 18 Section 20 Section 21 Section 22 Section 23 Section 24 Section 25 Section 26 Section 20
Mr. A Wheeler	Yes	Independent Consultant in Redruth, Cornwall, UK	• Mineral Resource Estimate • Mining • Ore Reserve Estimate	Section 8 Section 12 Section 13
	Yes	Kvale Advokatfirma in Oslo, Norway	• Land/Mineral Tenure and Licences	Section 5
	Yes	Wardell Armstrong International (WAI) in Truro, Cornwall, UK	• Mining Geotechnical (Open Pit Mine Design)	Section 9
	Yes	SINTEF in Trondheim, Norway	• Mining Geotechnical (Underground Mine Design) • Hydrogeology	Section 9 Section 10
	Yes	IHC Robbins in Brisbane, Australia	• Metallurgical testwork programmes and results	Section 11
	Yes	Mintek in Johannesburg, South Africa	• Comminution testwork and results	Section 11
	Yes	JKTech in Brisbane, Australia	• Metallurgical testwork programmes and results	Section 11
	Yes	Core in Brisbane, Australia	• Metallurgical testwork programmes and results	Section 11
	Yes	Mineral Technologies, Carrara, Australia	• Metallurgical testwork programmes and results	Section 11
	Yes	SGS in Vancouver, Canada	• QEMSCAN analytical results	Section 11

Competent Person	Independent of Owner	Company	Primary Area of Information	Relevant Sections
	Yes	SGS in Johannesburg, South Africa	• Comminution testwork and results	Section 11
	Yes	IMS in Johannesburg, South Africa	• Metallurgical testwork programmes and results	Section 11
	Yes	COWI in Fredrikstad, Norway	• Tailings disposal	Section 14
	Yes	DNV GL in Oslo, Norway	• Monitoring of subsea tailings disposal	Section 22
	Yes	TAK Industrial Mineral Consultancy in Gerrards Cross, United Kingdom	• Garnet market information	Section 17
	Yes	TZMI in Perth, Australia	• Rutile market information	Section 17
	Yes	Asplan Viak in Sandvika, Norway	• General permitting • County road re-alignment • Process water supply	Section 5 Section 14 Section 15

2.4 Reliance on Other Experts

This report is intended to be read as a whole, and sections should not be read or relied upon out of context, and any person using or relying upon this report agrees to be specifically bound by the terms of this Disclaimer and Limitations of Use. This report contains the expression of the professional opinions of Hatch, based upon information available at the time of preparation.

The report must be read in light of:

- The limited readership and purposes for which it was intended
- Its reliance upon information provided to Hatch by the Client and others which has not been verified by Hatch and over which it has no control
- The limitations and assumptions referred to throughout the report
- The cost and other constraints imposed on the report
- Other relevant issues which are not within the scope of the report.

2.5 Effective Date

The effective date of this report is 30 October 2017.

2.6 Units of Measure

The metric system has been used throughout this report. Tonnes are metric of 1,000 kilograms (kg), or 2,204.6 lb. All currency is in U.S. dollars (US$) unless otherwise stated.

2.7 Glossary

2.7.1 *Rutile (TiO_2)*

Rutile is composed essentially of crystalline titanium dioxide and, in its pure state, would contain close to 100% TiO_2. Naturally occurring rutile exhibits minor impurities and commercial concentrates of the mineral typically contain 94% to 96% TiO_2. Throughout this report the term rutile is used interchangeably with TiO_2.

2.7.2 *Mineral Resources and Ore Reserves*

As noted, the Mineral Resources and Ore Reserves have been classified according to the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves" (JORC Code 2012 Edition). Mineral Resources have been classified as Measured, Indicated or Inferred in line with the guidelines of the JORC Code; Ore Reserves have been classified as Proven and Probable in line with the guidelines of the JORC Code.

2.7.3 *Permits and Legislation*

Throughout this report:

- Extraction permits means the right to extract and utilise deposits of minerals within certain limits as described in section 32 of the Norwegian Minerals Act
- Norwegian Minerals Act means Act of 19 June 2009 No. 101 relating to the acquisition and extraction of minerals resources.

2.8 Abbreviations

The following abbreviations have been used in this report:

Table 2-2: Abbreviations

Abbreviation	Unit or Term
3D	three-dimensional
AACEI	Association for the Advancement of Cost Engineering - International
AI	Abrasion Index
amsl	above mean sea level
BBWI	Bond Ball Work Index
BRWI	Bond Rod Work Index
°C	degrees Centigrade
Ca	calcium
CAGR	Compound Annual Growth rate
CAPEX	Capital Expenditure
CIF	Cost, Insurance and Freight
cm	centimetre
cm^3	cubic centimetre
CP	Competent Person

Abbreviation	Unit or Term
CPI	Consumer Price Inflation
CWI	Crushability Work Index
DFS	Definitive Feasibility Study
DTM	digital terrain model
°	degree (degrees)
dia.	diameter
EBITDA	Earnings Before Interest, Tax, Depreciation and Amortisation
EIA	Environmental Impact Assessment
FOB	Free on Board
FS	Feasibility Study
g	gram
GDP	Gross Domestic Product
GNT	garnet
g/cm^3	grams per cubic centimetre
GSI	Geological Strength Index
g/t	gram per tonne
ha	hectare
HCl	hydrochloric acid
hp	horsepower
HPGR	High Pressure Grinding Roller
HTR	High Tension Roll
Ilm	Ilmenite
IRR	Internal Rate of Return
JORC	Australasian Joint Ore Reserves Committee
kA	kilo Amperes
kbar	Kilobar
kg	kilograms
kg/m^3	kilogram per cubic metre
km	kilometre
km^2	square kilometre
kr	Norwegian Krone or Swedish Krone
kt	thousand tonnes
kV	kiloVolt
kW	kiloWatt
kWh	kiloWatt-hour

Abbreviation	Unit or Term
kWh/t	kiloWatt-hour per metric tonne
l	litre
l/s	litre per second
lb	pound
LHD	Long-Haul Dump truck
LiDAR	Light detection and ranging
LoM	Life of Mine
m	metre
m/s	metre per second
m^2	square metre
m^3	cubic metre
m^3/s	cubic metre per second
masl	metre above sea level
Mg	magnesium
ml	millilitre
mm	millimetre
mm^2	square millimetre
mm^3	cubic millimetre
Mn	manganese
MoU	Memorandum of understanding
Mt	Million tonnes
Mtpa	Million tonnes per annum
MVAr	Mega Volt Amps reactive
MW	Million Watts
n/a	not applicable
n/d	not detected
NPV	Net Present Value
NTNU	Norwegian University of Science and Technology
OPEX	Operating Expenditure
%	percent
PEP	project execution plan
PFS	Prefeasibility Study
PPI	Producer Price Inflation
ppm	parts per million
PSD	particle size distribution

Abbreviation	Unit or Term
QA/QC	Quality Assurance/Quality Control
QEMSCAN	Quantitative Evaluation of Materials by Scanning
QXRD	Quantitative X-ray Diffraction
RED	Rare Earth Drum
RER	Rare Earth Roll
RoM	Run of Mine
rpm	revolutions per minute
RQD	Rock Quality Description
RMB	Renminbi (Chinese Yuan)
s	second
SG	Specific Gravity
SMC	SAG Mill Comminution
Sn	Tin
t	tonne (metric tonne) (2,204.6 pounds)
Th	thorium
Ti	titanium
TiO_2	titanium dioxide
tph	tonnes per hour
tpd	tonnes per day
t/m^3	tonnes per cubic metre
U	Uranium
UCC	Up-current Classifier
UCS	Uniaxial Compression Strength
UoM	Unit of Measure
µm	micron or microns
US$	U.S. Dollar
UTM	Universal Transverse Mercator
V	Volts
VDM	Value Distribution Model
VSI	Vertical Shaft Impactor
W	Watt
WAI	Wardell Armstrong International
WBS	Work Breakdown Structure
WHIMS	Wet High Intensity Magnetic Separator
wt	Weight

Abbreviation	Unit or Term
XRD	X-ray Diffraction
XRF	X-ray Fluorescence
y	year

3. Property Description

3.1 Property Location

The Engebø deposit is found within the Engebø hill, a ridge that runs parallel with the Førde Fjord. The site is located close to the town of Førde in western Norway, with navigable access to the North Sea. Engebø is on the northern side of the Førde Fjord in the Naustdal municipality, in the Sogn og Fjordane county. Its grid reference position is 310,200m E, 6,822,750m N, on the EU89-UTM zone 32 system. Its latitude is 61° 29' 35'' N with longitude 5° 25' 44'' E.

The Fv 611 county road, a single lane tarmac road, runs along the south side of the deposit (on the north side of the fjord) before entering a 630 m long tunnel which runs through the deposit.

The location of the Naustdal municipality near the south-western coast of Norway is shown in Figure 3-1 below.

Figure 3-1: Location of Naustdal Municipality (circled) in western Norway

The site location in Naustdal Municipality is shown in more detail in Figure 3-2 below.

Figure 3-2: Engebø Site (circled) in Naustdal Municipality

An aerial view of Engebø is shown in Figure 3-3 below.

Figure 3-3: Aerial View of Engebø Ridge looking West

3.2 Topography and Elevation

Engebø is the local name of a small hill which varies in elevation from sea level to 335 masl. On the south side, the hill dips steeply into the Førde Fjord, with more gentle slopes on the northern, eastern and western sides.

3.3 Climate

The climate at Engebø is characterised by long, warm days in summer and colder, darker and shorter days in winter. Snow is common in winter but proximity to the sea and relatively low altitude result in no permanent snow accumulation and there is no time of the year when operations are not possible. Over 2,000 mm of rain falls each year, through all four seasons. The fjord is permanently ice-free.

3.4 Accessibility and Infrastructure

The town of Førde (with a population of about 10,000 people), in Førde municipality, is located about 30 km east of Engebø, at the inner most part of the Førde Fjord. Førde has two regional airports nearby. It can also be reached by bus from Oslo, Bergen and Trondheim. West of Førde is the municipality centre of Naustdal, in Naustdal municipality. Naustdal municipality has a population of about 2,500 people. On the eastern part of the Engebø deposit is a closed quarry and harbour facilities constructed for shiploading of armours stone from the quarry, which still exist. The harbour facility is designed for vessels with a capacity of up to 80 kt and provides direct access to the North Sea and European ports.

Currently there is a small access/haul road from the Fv 611 county road to the top of the Engebø ridge. It is planned to upgrade this road to provide access to the mining area.

Small rivers and streams drain down to the sea at both the east and west end of Engebø, but the most likely source of fresh water for the process plant is a dam next to a hydroelectric power plant on the south side of the Førde Fjord. Process water will be transported to site via a dedicated pipeline across the fjord.

Electricity for the operations will be sourced from a connection to either the 22 kV grid that passes across the site, or a 132 kV national grid power line approximately four km from the site.

4. History

4.1 Prior and Current Ownership

The Engebø deposit was first recognised as a rutile deposit in the 1970s, after development of a local road tunnel. A small-scale quarrying operation was started in 1998 to produce armour stone from the western part of the deposit, due to the high density of the rock. The company Fjord Blokk built a deep-water quay and ran the operation until 1999.

The deposit was not systematically explored before the 1990s when DuPont, a major global titanium pigment producer, made claims for exploration of rutile in the deposit. DuPont initiated several exploration and beneficiation programs related to the Engebø deposit. In 1998, DuPont placed its interests in Engebø in the subsidiary Conoco (later ConocoPhillips). As a fossil energy focused Company, Conoco did not invest in any further exploration of the Engebø deposit. In September 2006, Nordic Mining acquired the claims from ConocoPhillips. Since 2011 the claims have been held by Nordic Mining's wholly owned subsidiary Nordic Rutile AS. The Norwegian Directorate of Mining granted Nordic Mining an extension of the extraction permits which are now valid until 2027.

4.2 Historic Development of the Project

DuPont carried out comprehensive drilling and sampling programmes that lasted between 1995 and 1998. The work was done in close cooperation with the Norwegian Geological Survey (NGU). A resource estimate was made based on the exploration results. The estimate showed 382 Mt of ore at an average grade of 3.96% TiO_2 for a cut-off at 3%. A range of comminution (crushing and grinding) tests were carried out by DuPont. The test work came short of systematically determining a recovery for rutile. A recovery of 47% was indicated and pigment grade rutile concentrates were achieved. At the time, little emphasis was put on recovering garnet.

In 2008 Nordic Mining assigned an independent Qualified Person, Mr. Wheeler, to make an updated resource estimation for the Engebø deposit in line with the guidelines of the JORC Code. The estimation was a more conservative approach than the earlier estimate made by DuPont. The resource estimate was published in a scoping study made by Mr. Wheeler together with independent Mining Engineer Mr. Dowdell.

Nordic Mining's strategy for the first years was to secure permits for the mining operation and for safe disposal of tailings. Comprehensive environmental impact assessment studies were carried out between 2008 and 2015. In 2015 an industrial area plan (zoning plan) and a discharge permit for the Engebø Project were approved. The permits are final with no possibility for appeals.

In early 2016, Nordic Mining started a significant drilling campaign to improve the resource classification and to quantify the garnet content of the deposit. Based on the results, Mr. Wheeler made an updated resource estimate. The estimate substantially improved and increased the 2008 classification and enabled a qualified quantification of the garnet. The 2016 resource estimate is the basis for the Prefeasibility Study and the resource and reserves statements made in this report. The 2008 and 2016 resource estimates are shown in Table 4-1 below.

Table 4-1: 2008 and the 2016 Resource Estimates (@ 3% TiO_2 Cut-off)

2008 Estimate	Tons (Mt)	TiO_2 Grade (%)	Garnet Grade (%)
Indicated	31.7	3.77	-
Inferred	122.6	3.75	-
2016 Estimate	**Tons (Mt)**	**TiO_2 Grade (%)**	**Garnet Grade (%)**
Measured	15.0	3.97	44.6
Indicated	77.5	3.87	43.6
Inferred	138.4	3.86	43.5

Notes:

- Grades presented above are total TiO_2
- Resource below sea level has been restricted by a boundary no closer than 50 m to the edge of the fjord
- Above Mineral Resources are inclusive of Ore Reserves.

A brief outline of the history of the Engebø deposit is as follows:

- 1970s and mid-1980s - Engebø was recognised as a rutile deposit by Elkem. Elkem and the Geological Survey of Norway (NGU) collaborated on additional sampling on various rutile-bearing eclogites in the area
- 1989 - DuPont and NGU started an evaluation of Norwegian rutile projects, aimed at deposits suitable for DuPont's chlorination process pigment plants. Engebø was identified as the most favourable project
- 1995 to 1997 - DuPont and the local company Fjord Blokk undertook a joint sampling and mapping exercise, with additional core drilling and beneficiation testing. NGU was involved as an external consultant. More than 15,000 m of drill cores and 40,000 assays were produced. DuPont discontinued the project after 1997 due to a change in company strategy. The daughter company Conoco maintained the mineral rights
- 2005 to 2006 - a number of mining companies visited Engebø, partly organised by "Rutilnett", an informal working group organised through Naustdal municipality. As a result, in 2006 several parties indicated their interest in purchasing the deposit from ConocoPhillips. Nordic Mining was the most successful and initiated further development of the Engebø deposit
- 2008 – a Scoping Study for the Engebø Project was completed for Nordic Mining by Messrs. Wheeler and Dowdell, independent mining consultants. This included an updated resource estimation, and preliminary underground and open pit mine planning. This enabled the approximate extent of a potential open pit to be defined. Mr. Wheeler, a Competent Person (CP), classified the deposit according to international standards

- 2008 to 2015 – a comprehensive Environmental Impact Assessments (EIA) was carried out, culminating in the granting of a complete zoning plan and discharge permit for the Project. Except for some smaller programmes related to re-assaying of old drill cores and garnet characterisation, no larger exploration programmes were undertaken before Nordic Mining initiated a drilling programme in 2016
- 2016 – a comprehensive diamond drilling and surface sampling campaign was carried out. The programme aimed at updating the open pit ore to reserve status as the basis for the PFS. In addition, data from the DuPont drilling programme was evaluated by relogging and re-assaying to confirm the quality of the data as the basis for an updated resource model which resulted in the issuing of a revised Mineral Resource estimate. The revised estimate forms the basis of the PFS as summarised in this report.

5. Land/Mineral Tenure and Licences

Nordic Mining (through Nordic Rutile AS) has access to the mining and processing plant areas both by land and sea as a result of option agreements with landowners, as well as an approved zoning plan for the area and a development agreement with Naustdal municipality.

5.1 Permits and Licences

Nordic Mining holds nine valid extraction permits covering the entire planned mining area, as shown in Figure 5-1 below.

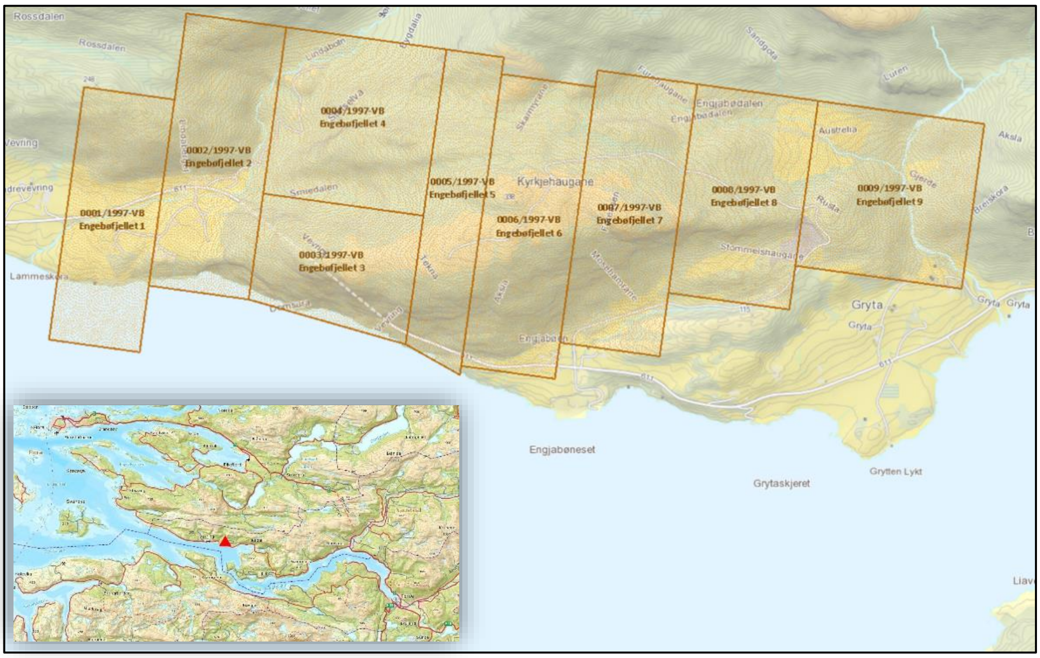


Figure 5-1: Extraction Permits held by Nordic Mining in the Vicinity of Engebø

The extraction permits give Nordic Mining the right to extract and utilise deposits of minerals within certain limits as described in section 32 of the Norwegian Minerals Act. The permits are valid until 12 November 2027. The Directorate of Mining may extend the duration period of the permits for up to ten years at a time. Extensions are normally granted if the deposit is deemed to be a reasonable reserve for the applicant’s operations.

The extraction permits entitle Nordic Mining to extract and utilise all deposits of minerals (rutile) owned by the Norwegian State in the extraction area. Minerals owned by a landowner (garnet, aggregates and possible other industrial minerals) may be extracted to the extent that this is necessary to extract deposits of minerals owned by the State.

The extraction permits are registered with the Land Register.

Nordic Mining will be required to apply for a regular operating license upon commencement of operations at Engebø, the threshold for application being the extraction of mineral deposits of more than 10,000 m^3 of mineral product. The operating license may be granted by the Directorate of Mining and is conditional on Nordic Mining being deemed to be qualified to extract the deposit. The license will be subject to standard conditions for mining.

5.2 Land Requirements and Associated Negotiations

The land required for the processing plant operations is owned by three private landowners. Nordic Mining has entered into option agreements with two of the landowners whereby Nordic Mining has an unconditional right to acquire the area for the planned processing plant and exclusive right to use the area for mining operations. The option agreements entitle Nordic Mining to acquire/take possession of the areas by giving the landowners 60 days' notice. These option agreements expire on 31 December 2017. Nordic Mining is currently discussing an extension of the option agreements with the landowners.

By law, Nordic Mining has the right to acquire areas for mining through compulsory acquisition of areas in accordance with the zoning plan. This process will not hinder the planned milestones for development of the Project.

5.3 Land Access

The Engebø deposit and the planned mining and processing plant areas are located adjacent to the Fv 611 provincial road and a deep water harbour facility. As noted, Nordic Mining has access to the planned mining and processing plant areas by means of an approved zoning plan. The zoning plan (planning permit), which was adopted by the Municipal Council for Naustdal Municipality in business item no. 022/11 on 11 May 2011 and the Municipal Council for Askvoll Municipality in business item no. 018/11 on 12 May 2011, and finally approved by the Ministry of Local Government and Modernisation on 17 April 2015, allows for and provides guidelines on the operation of the following activities:

- The extraction of rock mass in open pit production and underground mining
- The processing site at Engebø
- The service area at Engebø
- The waste rock deposition site in Engjabødalen
- Subsea area for tailings disposal on the sea floor of the Førde Fjord
- The works road running between the Fv 611 county road and the Engebø ridge
- The rerouting of county road Fv 611
- The rerouting of a 22 kV power line and the stringing of a new local cable at Engebø.

The zoning plan also ensures that measures are put in place to reduce the environmental consequences of the above activities for the local society and with respect to the landscape.

Nordic Mining has entered into a development agreement with Naustdal municipality regulating, inter alia, the improvement of the road infrastructure to facilitate the needs of the planned mining operations.

The harbour facility is covered by the option agreements entered into and under discussion with landowners; this will enable Nordic Mining to ship products directly to customers from the local deep-sea quay.

5.4 Relocation of People

There will be no need for relocation of people and moveable assets other than as prescribed by the option agreements with landowners as described in Section 5.2 above.

5.5 Compensation

The Project will affect three households, of which two are currently party to option agreements. There will, therefore, be no need for re-establishment of communities and their main livelihoods.

6. Geology

The Engebø deposit is one of the world's highest-grade rutile deposits and is unique due to its substantial content of garnet. Being almost free of radioactive elements and heavy metals, the deposit is a clean source of high-grade and high-quality titanium and garnet minerals.

6.1 Regional Geology

The rocks found in the Engebø area belong to the Western Gneiss Region, which is dominated by Proterozoic ortho-gneisses. These rocks have been subjected to varying degrees of pressure and temperature as revealed by the stages of metamorphism exhibited. There are several eclogite bodies in the Western Gneiss Region, among them the massive Engebø eclogite.

The Førde Fjord is located within the Western Gneiss Region, structurally situated in the footwall beneath rocks of Devonian age. The area has been subject to faulting and folding, resulting in regional east-west trending folds. These folds are the result of north-south compressional forces associated with the Caledonian orogenic event peaking some 400 million years ago. The rocks show different and complex deformation styles. The Western Gneiss Region is in general preserved in amphibolite facies with some isolated lenses preserved in eclogite facies.

There are two, mainly intrusive, units seen in the Førde Fjord area, the Hegreneset complex and the surrounding Helle complex. A geological map of the Førde Fjord area is shown in Figure 6-1 below. The Hegreneset complex consists of a variety of potassium-poor rocks, while Helle has more potassium-rich rocks. Hegreneset consists of basic to ultramafic, mainly eclogitic rocks with cross-cutting dioritic and granodioritic intrusives. The eclogites are best preserved in the central part of the dome structure which the complex exhibits. This is represented by the Engebø deposit. The Helle complex is comprised of mainly granitic to granodioritic gneisses, often migmatitic or banded and red to grey in colour. The rocks have been subject to strong deformation resulting in varied structures and textures.

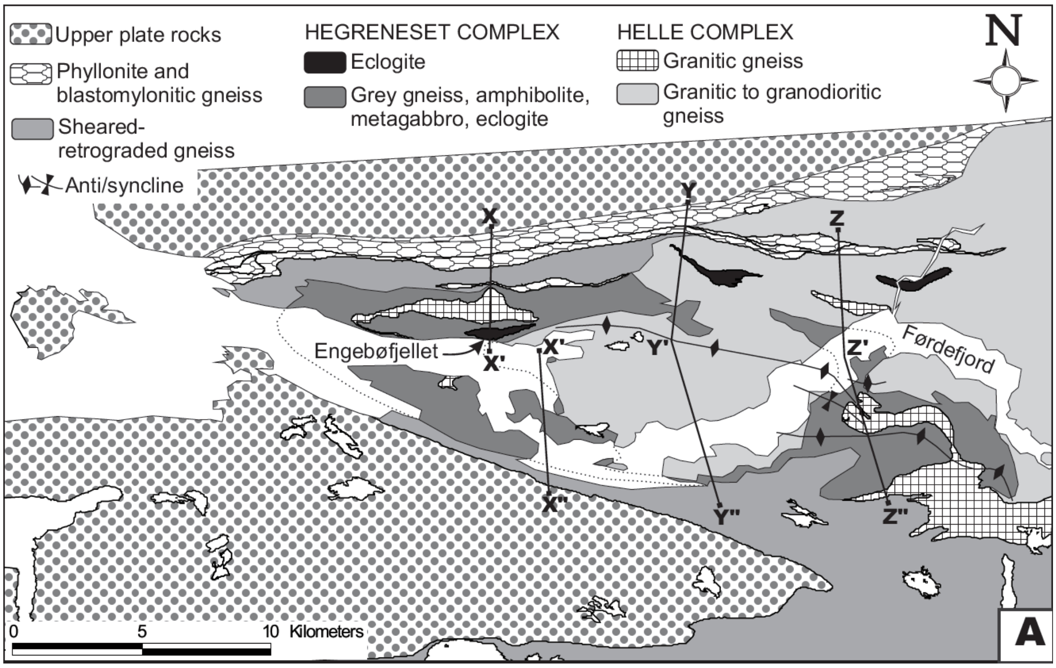


Figure 6-1: Geological Map of the Førde Fjord Area (NGU)

The Caledonian orogenic event is responsible for the eclogite facies metamorphism which has developed in the area. The early structures caused high-ductile deformation-zones to develop. These zones contain high amplitude, isoclinal and modified folds. The rapid exhumation of the rocks by structural means appears to have assisted in the preservation of the rutile grains. The late-Caledonian simple shear and asymmetric folding probably occurred after the eclogite was formed. The regional and local structures and general rock composition are readily evidenced on Landsat images.

DuPont's experience with Norwegian eclogites led it to conclude that those located within the Western Gneiss Region would have attractive rutile contents had they not been greatly affected by shearing. Exploration and drilling campaigns on other Norwegian eclogites within the Western Gneiss Region were unable to indicate potentially mineable material in sufficient quantities to justify development. Therefore, Engebø became the focus of the DuPont exploration effort as it has both the tonnage and grade to justify development.

6.2 Local Geology

The Engebø eclogite and the surrounding undifferentiated mafic and felsic rocks belong to the Hegreneset complex. The eclogite forms a 2.5 km long east-west trending lens with a distinctly massive character compared to the surrounding amphibolite facies rocks. The lens runs parallel with the Førde Fjord and the Engebø ridge. The eclogite is believed to represent a Proterozoic gabbroic intrusion that was enriched in iron and titanium (ilmenite) due to fractionated crystallisation. Due to the high pressure and temperature during the peak Caledonian metamorphism approximately 400 million years ago, the gabbro transformed to eclogite. During this process, ilmenite broke down to form rutile and excess iron that went in to the iron rich mineral garnet. The strike of the eclogite is generally east-west with a dip of 85° north. However, the dip varies from a steep angle northwards through vertical to southwards but for the most part is 60 to 85° to the north. Detailed structural studies reveal many episodes of complex major folding and development of foliation. In general, the eclogite may be considered an anticlinorium with a major fold axis trending about east-west. The limbs of the major fold are also highly contorted.

There is considerable exposure of eclogite on surface although the overburden increases to the east and the country rocks frequently fold into the eclogite on its extreme margins.

6.3 Deposit Type

Unlike most rutile deposits, the Engebø rutile is contained in a hard-rock ore, a massive body of eclogite. Rutile and garnet was formed deep into the earth's crust at high pressure and temperature. Rare circumstances led to the preservation of these minerals during rapid uplifting of the rock to its current position. The high quality of the minerals is owed to the unaltered state of the host rock.

Geological investigations of the Engebø deposit have determined that the eclogite can be subdivided into three different eclogite types, based on titanium content and appearance

- Ferro-eclogite; dark and massive appearance, >3% TiO_2
- Transitional-eclogite; intermediate dark, 3-2% TiO_2
- Leuco-eclogite; light coloured and foliated, <2% TiO_2.

The contact between the eclogite types are gradational, moving from ferro- to transitional- and leuco-eclogite. Figure 6-2 shows the relationship between the different eclogite types.

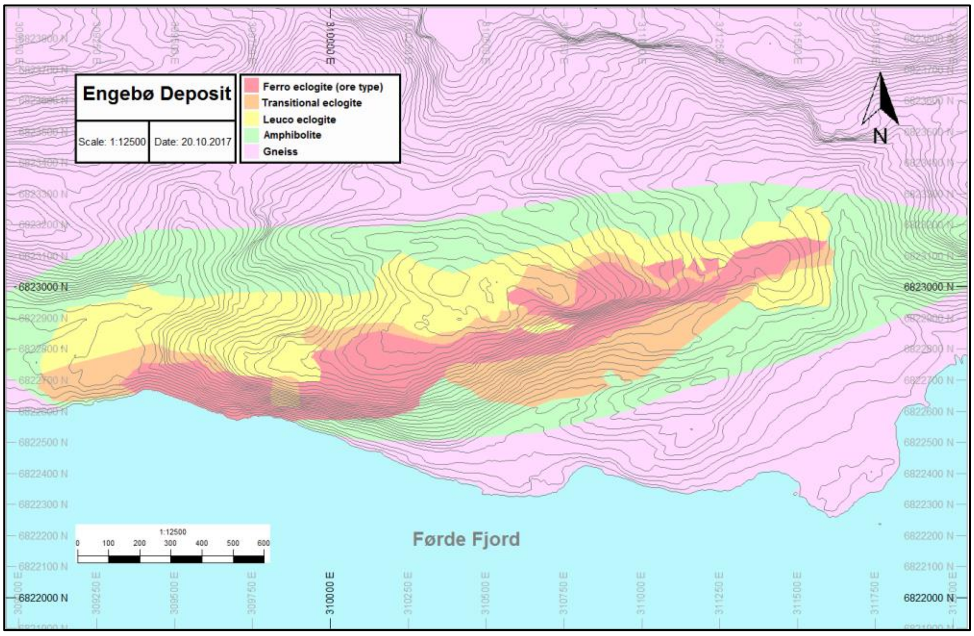


Figure 6-2: Geology of the Engebø Deposit

The economically significant ferro-eclogite is generally found in the southern portions of the deposit.

6.4 Mineralogy

Engebø originates from a gabbroic intrusion that was metamorphosed under eclogite-facies conditions of approximately 15 to17 kbar pressure and approximately 600°C temperature during the peak of the Caledonian metamorphism. Eclogitisation is defined as the mineralogical alteration where the plagioclase in the lower facies breaks down to form the sodium containing pyroxene omphacite. No relict magmatic minerals have been found.

The main titanium bearing mineral is rutile. Only 5% of the titanium is found as ilmenite and the presence of titanite/sphene is negligible. The rutile is practically free of uranium, thorium and other radioactive elements (less than 1 ppm). The mineral assemblage gives the rock a characteristic green and red colour. In general, the eclogite contains 40% to 50% almandine type garnet. The garnet content becomes gradually less for the low-grade eclogite types. Other major minerals are pyroxene (omphacite) and amphibole which gives the rock its green colour. Phengite and paragonite (white micas) are characteristic of leuco-and trans-eclogites, but minor amounts are also found in mafic ferro-eclogite. Other accessory minerals include epidote, carbonates (dolomite/ankerite), quartz, pyrite and apatite. Zircon occurs as trace mineral and typically as tiny inclusions in and close to rutile and garnet.

The texture is generally equi-granular but garnets are commonly coarser than other minerals. Garnet grain size is typically between 0.1 mm and 0.5 mm in diameter, but larger grains up of to 1 mm are not uncommon. Larger garnet grains (up to 1 cm) typically contain inclusions of other minerals.

6.5 Waste Rock Types

Except for low-grade eclogite rocks, the main waste rock types are:

- Amphibolite: this is the main side rock and encloses the eclogite lens. It is generally homogenous with no banding and with a sugarish texture. It is darker green than the eclogite, often with visible plagioclase feldspar. No garnets are visible
- Garnet Amphibolite: this is an amphibole type that is often found as internal zones in the eclogite. it is generally homogenous with no banding and with a sugarish texture. It is darker green than the eclogite, often with visible plagioclase feldspar and a distinct garnet content
- Gneiss: this generally occurs outside of the main eclogite body and the amphibole unit, and is dominated by felsic minerals like quartz, mica and feldspars. The gneiss is strongly foliated and sheared often with porphyroblasts of reddish feldspars.
- Alternating Mafic and Felsic rocks: the alternating rocks occur mostly in the contact between the eclogite and the side rock and are more or less altered sequences of mixed eclogite and gneiss
- Quartz veins: there are some occasional massive quartz veins, with thicknesses up to 1 m.

7. Drilling, Sampling and Ore Characterisation

A summary of the drilling and sampling campaigns, both historic and resent, for the Engebø deposit is given in this section. The summary includes a description of how drilling, sampling and assaying was carried out, and the Quality Assurance/Quality Control (QA/QC) system put in place. Different methods used to characterise the ore are described.

A summary of all diamond drilling carried out at Engebø both by DuPont/Conoco and Nordic Mining is shown in Table 7-1: below. All the DuPont/Conoco drilling produced BQ (36.5 mm) core. All the 2016 Nordic Mining drilling produced NQ2 core (50.7 mm).

Table 7-1: Summary of Drilling Campaigns

Drilling Campaign	Drillholes	Length (m)	Average Length/hole (m)
1997 DuPont/Conoco	49	15,198	310
2016 Nordic Mining	38	6,348	167

Figure 7-1 and Figure 7-2 below illustrate the historic drilling by DuPont (shown in green) and the 2016 drilling by Nordic Mining (shown in red). The historic drilling was concentrated in the western part of the deposit while the new drilling was centred in the planned open pit area located in the central part of the deposit.

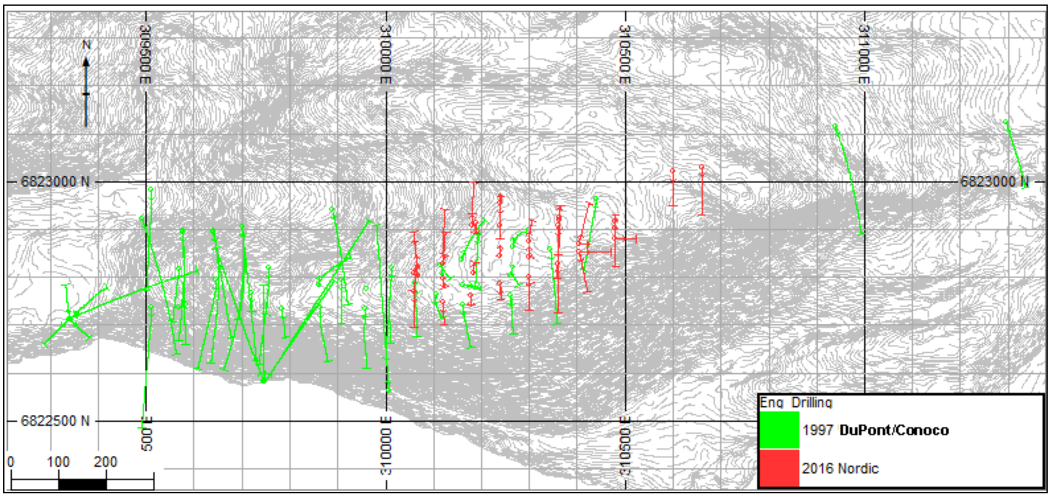


Figure 7-1: Plan of Drillholes

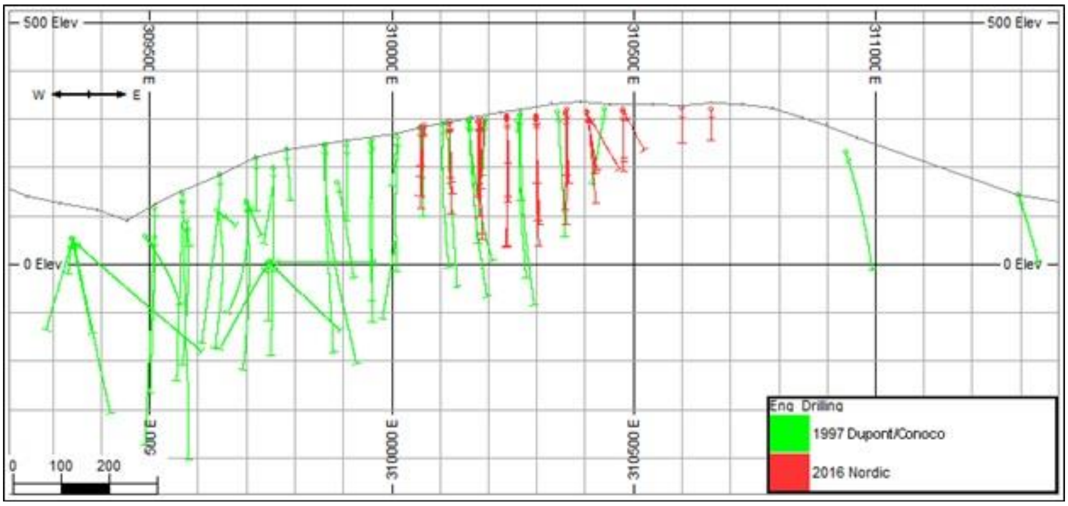


Figure 7-2: West-East Long Section of Drillhole Data

DuPont/Conoco carried out an extensive drilling campaign between 1995 and 1997. The drilling focused on the western part of the deposit which was regarded as the higher-grade part of the deposit. The road tunnel that runs through the deposit was also sampled through the full length (630 m). No geotechnical drilling was done at the time, nor was an attempt made to quantify the garnet content. Drill hole recoveries were excellent with negligible losses. NGU participated in the drilling campaign.

7.1 Sample Preparation and Analyses

The DuPont/Conoco core material was assayed by a portable, handheld, XRF (X-ray Fluorescence) instrument (X-Met). Analysis for TiO_2 and Fe_2O_3 was made directly on the core and recorded every 25 cm along the core. About 116 core laboratory composites (at 10 m intervals) were prepared and measured by XRF. These data were used to test and correct the X-Met analysis.

Additional measurements of total TiO_2 and Fe_2O_3 were obtained from samples taken from the sidewalls of the road tunnel that runs approximately through the middle of the deposit. These samples were taken by chip sampling or by obtaining the drill cuttings from small holes drilled into the walls. Drill cuttings were milled and the X-Met instrument was used for measurement.

Surface samples for measurement of total TiO_2 and Fe_2O_3 were taken by either chip sampling, drill dust sampling or direct X-Met measurement on the bedrock. X-Met measurements were made directly on milled chip samples or drill dust samples.

A summary of the X-Met and XRF sample assays from the DuPont/NGU campaign is shown in Table 7-2 below.

Table 7-2: DuPont/Conoco Sample Summary – for total TiO_2 and Fe_2O_3 Measurements

TYPE	DESCRIPTION		HOLES	LENGTH	NUMBER SAMPLES
Drillholes	Total Drilled		49	15,198	
	X-Met Measurements	Lokken TIO2	29	6,033	24,133
		Lokken FE2O3	29	6,045	24,180
		Engebo TIO2	30	4,306	17,225
		Engebo FE2O3	27	3,714	14,855
		Either TIO2 measurement	49	9,431	37,726
		Either FE2O3 measurement	48	9,070	36,279
	Lab Composite XRF		34	952	116
Tunnel				660	34
Surface Samples	Chip samples	Chip97-NGU			229
		chip96-NGU			44
	Drilldust samples	dd95-NGU			108
		dd96-DuPont			118
		dd96-NGU			76
	Direct X-Met	xmet96-NGU			680
		xmet97-DP			104

Additional procedures and measurements applied were:

- Extensive core logging of all drill-holes
- Photo-documentation of each drill-hole
- Bulk density measurements
- Magnetic susceptibility measurements, using a portable instrument
- Rutile/ilmenite content was determined for each laboratory composite, by additional measurement of acid-soluble TiO_2 by ICP-AES. Wt% Rutile = bulk wt% TiO_2 – acid soluble wt% TiO_2.

Although there was no specific QA/QC programme in place, the different X-Met methods were quality checked and evaluated against each other and XRF analysis.

The average X-met assay over an interval related to a laboratory composite sample was found to correlate well with the XRF measurements. However, the variation between the 25 cm X-met analysis within a composite was found to be large.

In the 2016 Nordic Mining drilling campaign, 709 m of the old core were re-sampled and re-assayed by ALS Minerals in Sweden to assist with verification of these data.

Although some computer modelling work was done previously by DuPont, the modelling was done completely anew as part of the Scoping Study initiated by Nordic Mining, starting from master database files (in Access) that were provided by DuPont/Conoco. The resource model obtained by use of the DuPont data was merged with the model created from the 2016 drilling. The data generation from the 2016 drilling was done to ensure that the data were comparable and as mergeable with the DuPont/Conoco data as possible. The main differences between the DuPont/Conoco and the 2016 data sets are:

- The 2016 drilling programme was done in a more structured manner based on a 60 m by 40 m grid spacing of the drill-holes rather than a more sporadic drilling pattern. The aim of the 2016 drilling was to classify existing resources from Inferred to Indicated/Measured categories
- Higher quality chemical assaying was employed in 2016 using 5 m laboratory composites for XRF measurements for all ore zones in all drill-holes
- A proper QA/QC system was set in place in the 2016 campaign
- Garnet was quantified in the 2016 campaign using QEMSCAN (scanning electron microscope analysis) and calculated from chemistry
- Measurement of acid-soluble TiO_2 by ICP-AES was carried out in the 2016 campaign for all laboratory composites to determine the ilmenite content and the ilmenite/rutile ratio.

7.2 Nordic Mining 2016 Drilling Campaign

The drilling campaign included recovery of 6,348 m of drill cores, collection of 77 surface samples and outcrop mapping. The cores were logged and sampled at Nordic Mining's core storage facility in Naustdal. 1,517 whole rock chemical analyses (XRF) and 336 rutile specific analyses (ME-ICP41) were carried out by ALS Minerals in Sweden. QEMSCAN was carried out by SGS Canada on 68 samples to investigate mineralogical, textural and petro-graphical variations within the deposit. Garnet was successfully quantified by correlating QEMSCAN data with iron content from chemical assays.

As part of the drilling programme, historical datasets have been re-assessed and old drill cores were re-logged and re-analysed. The results show a good correlation between new and historical data and thereby fully validate the historical datasets.

7.2.1 Nordic Mining Core Drilling

The 2016 drilling was carried out by Finnish contractors Kati Oy using Sandvik DE130 and DE140 drilling rigs. Both rigs used wireline drilling. Downhole survey measurements were taken every 5 m downhole. The majority of the holes were laid out on a regular 60 m by 40 m grid in the area demarcated as the potential open pit area in the 2008 scoping study. The holes were positioned using a total station. Seven holes were drilled using marking for orientation with the purpose of geotechnical logging.

The principal reasons for the 2016 drilling included:

- To provide a better coverage of sample data in the prospective open pit area, and thereby achieve at least an Indicated Resource category for the majority of the ore in this area
- To provide a bank of recent data which would help verify the 1997 drill hole data
- To provide samples for metallurgical testing in the potential open pit area
- To provide geotechnical samples and data to assist with selection of mine and slope design parameters

- To provide extensive additional data for assessment of garnet and different mineralisation qualities.

7.2.2 Nordic Mining Logging, Sampling and Assaying

All core material was brought to Nordic Mining's office in Naustdal where a logging and sampling facility was set up. The holes were carefully logged and separate logs were recorded, including lithology logs, geotechnical logs, retrogression zone logs, textural logs, pyrite and mica logs, and sample interval logs.

A handheld XRF (XMET) was used to aid the geologists in the logging and determination of different ore types. Samples were selected in accordance with major lithological breaks, and were restricted to a maximum length of 5 m. Core was sawn longitudinally in half, with one half being selected as a sample for chemical analysis and the other left in the core box for storage. All samples were continually labelled and trucked to ALS Minerals in Sweden for sample preparation and chemical analysis. Related to titanium, the principal assays determined were TiO_2 (total) by XRF, and TiO_2 (dissolvable in HCl) by ICP to determine the ilmenite content.

Other elements assayed were Fe_2O_3, MnO, Al_2O_3, BaO, CaO, Cr_2O_3, K_2O, MgO, Na_2O, P_2O_5, SO_3, SiO_2 and SrO.

Density measurements were taken at approximately 25 m intervals downhole. All holes were photographed.

7.2.3 Quality Control

A pre-planned QA/QC programme was used for all of the 2016 drilling campaign. The different types of quality control samples taken are depicted in Table 7-3 below. For field duplicates the core was additionally cut into four quarters in order be able to provide both additional samples and still have some core left in storage.

Coarse blanks were introduced by Nordic Mining into the sample batches, using standard blank material obtained from ALS Minerals. Fine blank material came from the same source after being ground by ALS for Nordic Mining to allocate into the sample stream.

A standard sample was purchased from the USGS, a Hawaiian basalt sample, code BHV0-2. This sample has certified grades of 2.73% TiO_2 and 12.3% Fe_2O_3.

Table 7-3: QA/QC Samples – Insertion Rates and Acceptance Criteria

Evaluation Parameter	Type of Sample	CODE	Frequency %	Process being evaluated	Acceptance Criteria
Precision	Field Duplicates	FD	2	Precision of taking samples	<=10% failed samples
	Coarse Duplicates	CD	2	Precision of sample preparation	<=10% failed samples
	Pulp Duplicates	PD	2	Precision of analysis	<=10% failed samples
Accuracy	Standard Samples	STD	6	Accuracy with respect to primary lab	Bias <=5%
	External Duplicates	ED	4	Accuracy with respect to secondary lab	Bias <=5%; adjusted R^2=1
Contamination	Coarse Blanks	CB	2	Contamination during sample preparation	Contamination <=2%
	Fine Blanks	FB	2	Contamination during analysis	Contamination <=2%
	Total		**20**		

7.2.4 Garnet Analysis

Associated with the 2016 drilling campaign, garnet was analysed by SGS Canada in two ways using QEMSCAN measurements:

a) **Thin Sections.** 10 core billets were selected over a range of different locations throughout the drilled areas. These were analysed by QEMSCAN using a textural analysis method.

b) **Coarse Pulp Rejects.** For the coarse rejects available from core sampling, 68 samples were selected over a range of grades and locations. The coarse reject material was ground carefully at the SGS laboratory in Canada with the aim of liberating grains and not over grinding. Slides were prepared for each of these samples by spreading a thin layer of pulp material onto each slide. The slides were then analysed by QEMSCAN. These results gave a percentage of garnet, which could be compared with the original assay data for the same sample.

Thin section analysis enabled:

- An assessment of the grain size distribution for each sample to be made. The QEMSCAN analysis indicates that the garnets have grain sizes typically between 0.1 to 0.5 mm (100-500 µm)
- The conclusion to be drawn that the textural feature that seems to mainly affect the distribution of garnet is a retrogression of the ore.

Observations from garnet analyses in the tests included:

- Typically, ferro- and trans -eclogite have between 40% and 50% garnet, leuco-eclogite has between 30% and 40% garnet

- The crystal shape of the garnet is euhedral to subhedral. In intensely foliated or sheared samples, the grains are somewhat more elongated and irregular
- The garnets typically have few inclusions but the larger garnets are more likely to have mineral inclusions
- In heavily altered zones, the garnet tends to break down and grain size is reduced. This constitutes a minor part of the deposit
- The principal garnet type is almandine.

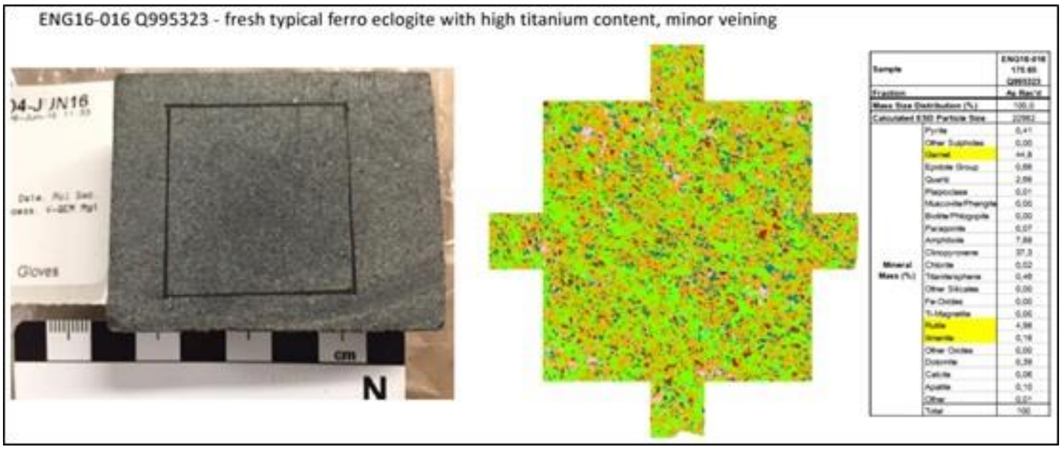


Figure 7-3: Example of QEMSCAN Results from Thin Section of Core Analysis

A way to estimate the grade of garnet was investigated by studying the relationship between garnet quantity from QEMSCAN data and the chemical analysis of the same sample. The best results were obtained by relating garnet to iron (Fe_2O_3) content. This was carried out using the following steps:

- The total iron content per sample is directly available from the assayed Fe_2O_3
- As well as garnet, it is known that other minerals containing iron will be ilmenite, pyrite (reflected by the SO_3 assay) and amphibole (reflected by the K_2O assay)
- Therefore, it can be reasoned that the amount of garnet will have some relationship as follows: Test (GNT) = Fe_2O_3 – (b x SO_3 + c x K_2O + d x ilmenite).

The test variable will be a number which can be correlated to the measured GNT values. After some analysis, it was also found that this it was best to split the analysis between different eclogite types.

For the ferro-eclogite sample and trans-eclogite sample sets, the best relationships were found to be:

Trans-eclogite: Test (GNT) = Fe_2O_3 – (4.1 x SO_3 + 3.0 x K_2O + 2.5 x Ilm)

Ferro-eclogite: Test (GNT) = Fe_2O_3 – (3.9 x SO_3 + 1.5 x K_2O + 2.5 x Ilm)

The %Ilmenite grade is derived from the assayed %TiO_2 in ilmenite value from the relationship below:

%Ilmenite = [TiO_2(total) x % TiO_2 (in ilmenite)] / 0.5265.

These test variables determined for the two sample sets plotted against the measured garnet grades gave the results as shown in Figure 7-4 below. The regression equations from these trendlines are:

Trans-eclogite: y = 1.988x + 17.167

Ferro-eclogite: y = 3.438x + 3.792

where: y = derived garnet wt%; x = test variable (as derived above).

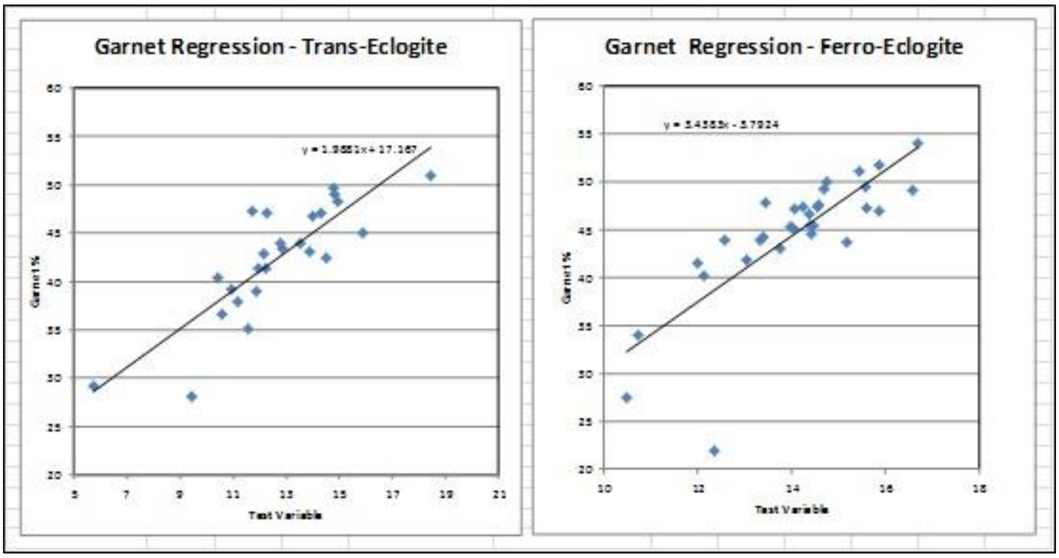


Figure 7-4: Graph of Test Variable vs. Garnet for Trans-Eclogite and Ferro-Eclogite

These regression coefficients were then used to derive a garnet assay for each sample. The pairs of the measured garnet and derived garnet values were then analysed using RMA (reduction to major axis) analysis. The analyses gave correlation values (R2) and low bias values of less than 5% when a very small number of outliers had been removed. This analysis, therefore, supported the use of these formulae in the derivation of garnet grades in the resource estimation.

7.2.5 Nordic Mining 2016 Surface Sampling

Additional surface samples were taken by Nordic Mining in 2016 using a handheld Makita drill. All the surface samples, each of which was approximately 100 g in mass, were sent to ALS Minerals for XRF analysis. The purpose of the surface sampling campaign was to provide additional surface grade information to assist with modelling, as well as to provide some verification of surface sampling made in previous campaigns.

The samples were located on 60 m section lines, with a spacing of 30 m to 60 m in the north-south direction, as shown in Figure 7-5 below. The figure shows both the Nordic samples and the historic samples taken by NGU. The Nordic samples were mostly located within the planned open pit area. The colour scheme in Figure 7-5 shows the different ore types of each sample.

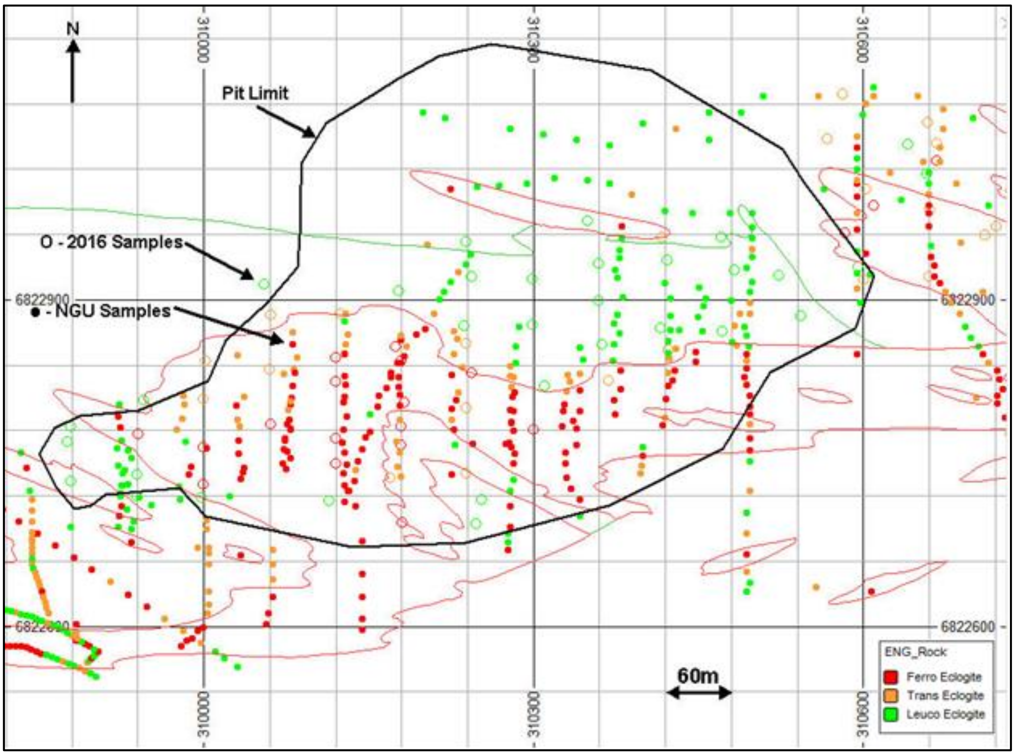


Figure 7-5: 2016 Surface Sample Locations

7.3 Data Verification

As noted, the Competent Person, Mr. Wheeler, has reviewed all the QA/QC results from Nordic Mining's 2016 drilling campaign. The results of this review are summarised below and are detailed in Mr. Wheeler's 2016 resource report.

7.3.1 Precision

Precision graphs for the results of field duplicates (FD), coarse duplicates (CD) and pulp duplicates (PD) were determined. No failures were encountered for any of the duplicate types, which is an extremely good result. Generally, up to 10% of failures would be considered a satisfactory result.

7.3.2 Accuracy

The standard assay results for 98 submitted samples were analysed. The results showed that there are only three outliers, and a very small bias value, well less than the 5% acceptable level. The accuracy level of the results is, therefore, very high.

7.3.3 Contamination

The TiO_2 level of the blank material was 0.11%, so this value became the lowest level of detection (LD). Of the 32 blank samples submitted by Nordic Mining, the highest coarse blank assay value was 0.13% TiO_2, well inside the 3x LD limit normally considered acceptable. The highest pulp blank assay from the 33 pulp blank samples submitted by Nordic Mining was 0.12% TiO_2, which is well inside the 3x LD limit normally considered acceptable. There is, therefore, no indication of any contamination during sample preparation or analysis.

7.3.4 Verification of Nordic Mining 2016 Surface Sampling

Seventy-nine primary surface samples were taken. Along with samples, nine field duplicates were taken (representing 12%). The field duplicates showed a very good correlation with the original samples, so it can be concluded that the surface samples are acceptable for use in resource estimation.

7.3.5 Verification of Historical Database

To assist with verification of the historical DuPont/Conoco diamond drilling data, samples were taken from 14 of the old holes. These were then prepared and re-assayed in the ALS laboratory in the same way as the samples from the 2016 drilling campaign. This re-assaying involved 709 m of core, representing approximately 6% of the eclogite core from the DuPont/Conoco drilling campaigns. Favourable results were obtained from the re-assay data. The results represent more than 25% of all original 49 NGU holes. It can be concluded, therefore, that the historical NGU data can be used for estimation in the updated resource estimation study for all resource category levels.

8. Mineral Resource Estimate

A summary of the 2016 mineral resource estimate for Engebø is given below. A third party independent review of the mineral resource estimate was carried out by SRK Consulting (UK) Limited (SRK) in December 2016. SRK concluded that the mineral resource estimate did not contain any fatal flaws and that the geological model produced for use in this PFS was fit for purpose.

8.1 Mineral Resource Model

The mineral resource estimation was completed using a 3D block modelling approach, with the application of Datamine software. The overall methodology used is depicted diagrammatically in the flowsheet in Figure 8-1 below. A combination of 79 surface samples and 87 drillholes, composing 21,546 m of drilling, was used to derive the 3D block model.

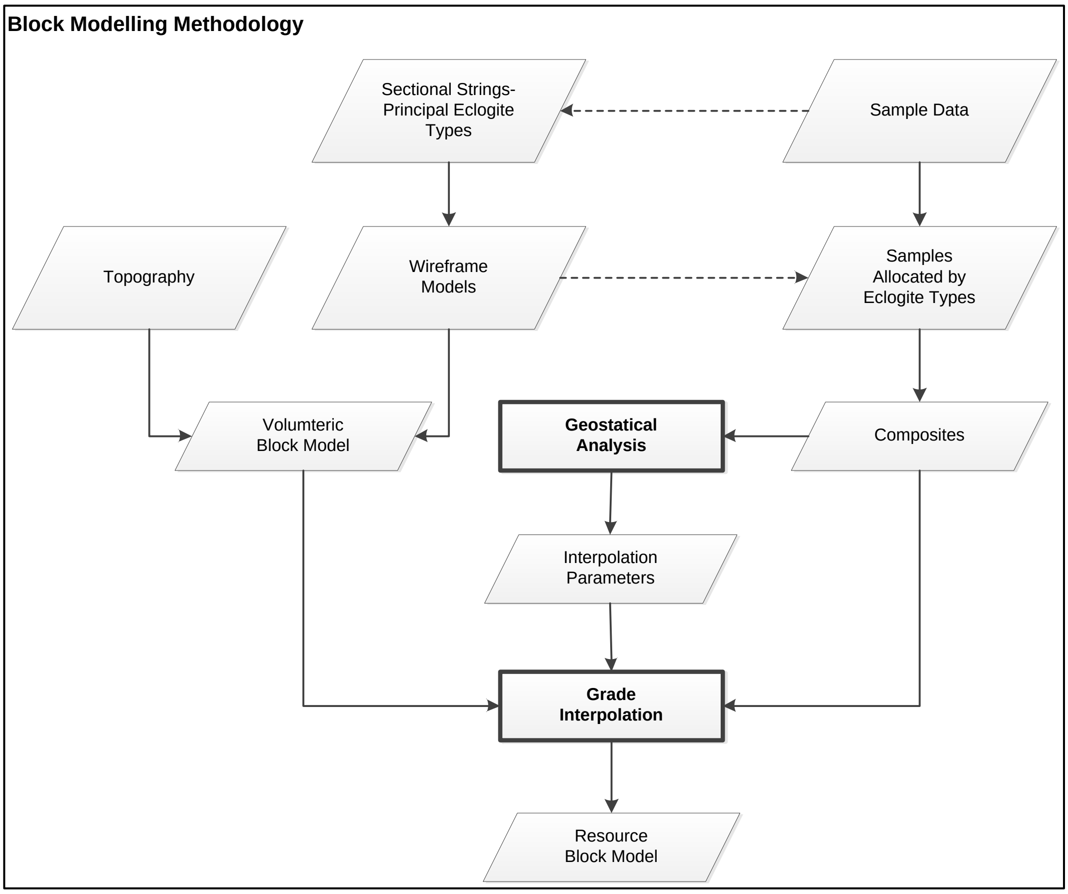


Figure 8-1: Block Modelling Methodology

Three different principal types of eclogite have been coded during the logging of the drillhole data. For each of these principal zones, sectional strings and perimeters were defined, based on all available lithological and sample data. Where possible, these perimeters were then converted into 3D wireframe envelopes. Along with topographical data, these wireframe data were used to create volumetric block models.

Samples associated with these overall interpreted zones were assigned logical codes, corresponding with the defined eclogite wireframe models. These sample data were then converted into approximately 5 m composites. The composite TiO_2 and other grade values were then used to interpolate grades into the block model according to the parent eclogite type to which they belonged. Geostatistical analysis was used to assist in the selection of interpolation parameters, as well as with subsequent resource classification.

Solid wireframe models were created for the three principal eclogite types, as well as some major zones of alternating mafic material, as shown in the 3D view in Figure 8-2.

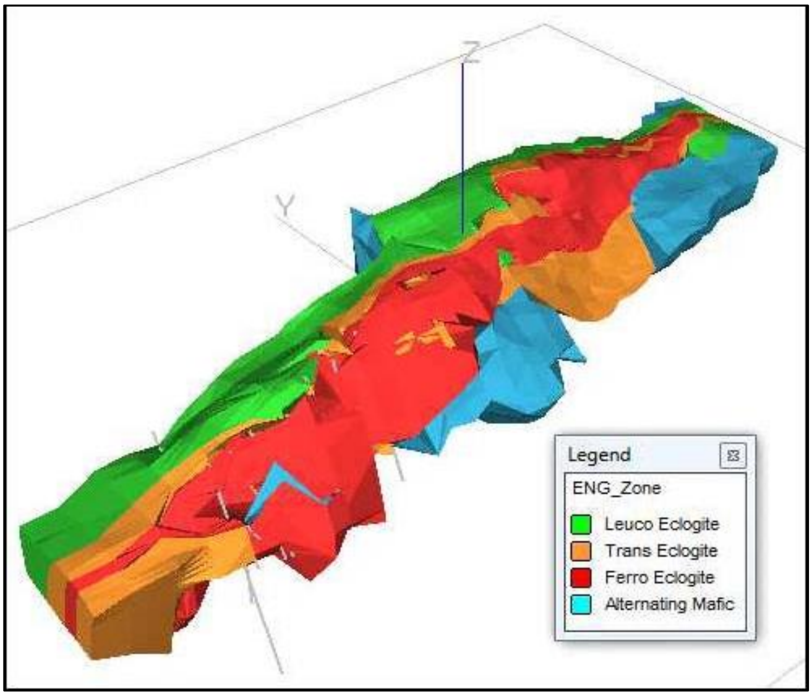


Figure 8-2: 3D View of Interpreted Wireframe Model showing main Eclogite Ore Types (looking North-East)

Sample data processing was then carried out using the eclogite wireframe models to select a sample set, after which top-cut levels were applied and 5 m downhole composites were created according to a zone classification. Thereafter geostatistical analysis of the downhole composites was carried out, followed by volumetric modelling and grade estimation.

The principal grade interpolation method used for TiO_2 was ordinary kriging, with nearest-neighbour and inverse-distance weighting methods being used for subsequent testing and validation.

Grade interpolation for garnet used a different method based on a formula which used Fe_2O3, K_2O, SO_3 and TiO_2 ilmenite values.

Density values were estimated into the block model using inverse-distance weighting.

8.2 Mineral Resource Statement

The August 2016 resource statement prepared in line with the JORC Code (2012 Edition) guidelines is shown in Table 8-1 and Table 8-2 below. Resource statements for two different cut-off grades are reported to provide flexibility for mine planning. Both resource statements given below are inclusive of Ore Reserves.

Table 8-1: Mineral Resource Statement (3% TiO_2 Cut-off)

TiO_2 Cut-off	Classification	Tonnes (Mt)	Total TiO_2 (%)	Garnet (%)
3%	Measured	15.0	3.97	44.6
	Indicated	77.5	3.87	43.6
	Total – Measured and Indicated	**92.5**	**3.89**	**43.7**
	Inferred	138.4	3.86	43.5

Table 8-2: Mineral Resource Statement (2% TiO_2 Cut-off)

TiO_2 Cut-off	Classification	Tonnes (Mt)	Total TiO_2 (%)	Garnet (%)
2%	Measured	19.0	3.68	43.9
	Indicated	105.7	3.51	43.0
	Total – Measured and Indicated	**124.7**	**3.53**	**43.2**
	Inferred	254.5	3.22	42.5

Notes:

- Grades presented above are total TiO_2
- Resource below sea level has been restricted by a boundary no closer than 50 m to the edge of the fjord. This is a vertical boundary that is defined 50 m horizontally from the fjord limit into the Engebø hill from sea level, and then vertically to the deepest part of the defined ore body. No ore within this 50 m zone is part of resource estimations
- Above Mineral Resources are inclusive of Ore Reserves as reported in Section 13 below.

8.3 Competent Person Sign-off

As noted, the geology and mineral resource study for Engebø was completed by an independent mining consultant, Mr. Adam Wheeler, who has been working on the Engebø Project since 2008. He has received full access to all available data and information connected with the deposit and project development, and has received unlimited assistance from all Nordic personnel connected with the Project. Mr. Wheeler has visited the site several times, including three times during 2016, in connection with the recent drilling campaign. Mr. Wheeler qualifies as a Competent Person in terms of the JORC Code 2012 Edition.

8.4 Future Work Programme

A drilling programme will be considered as part of the FS phase. The following objectives for drilling will be considered:

- To increase the amount of Measured ore in the ore resource
- To increase the amount of Indicated ore in the ore resource
- To aid the overall understanding of the structural geology of the orebody
- To enable an FS standard geotechnical model to be compiled, in part using additional geology drillholes.

9. Mining Geotechnical

Mining geotechnical studies for both the planned open pit and underground mining operations have been carried out to support the ore reserve estimate.

9.1 Open Pit Geotechnical Study

WAI was engaged to complete a PFS level geotechnical assessment and pit slope design for the Engebø open pit. A summary of the WAI studies is provided below.

Geotechnical site investigations to collect geotechnical information and laboratory samples were completed in May 2016. A programme of diamond drilling and logging of oriented core, as well as geotechnical field mapping of the road tunnel exposure through the deposit, was conducted. Hydrogeological information was interpreted as no detailed information was available at that time.

Based on a 2012 optimised pit shell provided to WAI, six geotechnical design sectors were identified for slope stability analysis. Kinematic analysis was performed for all design sectors using stereographic analysis to review failure modes that are kinematically possible at bench or inter-ramp scales. Inter-ramp scales wedge failures were identified in the North (1) Sector. In addition, bench scale wedge, planar and toppling features were identified in various geotechnical sectors. Bench face angles were selected to reduce the risk from structural controlled failures.

The overall slope rock mass stability was assessed using limit equilibrium models for inter-ramp and proposed North West, South, North East and South East walls (see Figure 9-1 below). The analysis indicates that the following wall designs, as summarised in Table 9-1, are achievable. The definitions of the terms used in Table 9-1 are provided in Figure 9-2 below.

Operational considerations to support the selected wall designs and to improve geotechnical stability recommended by WAI include controlled blasting practices, scale removal on bench faces, slope dewatering and slope monitoring.

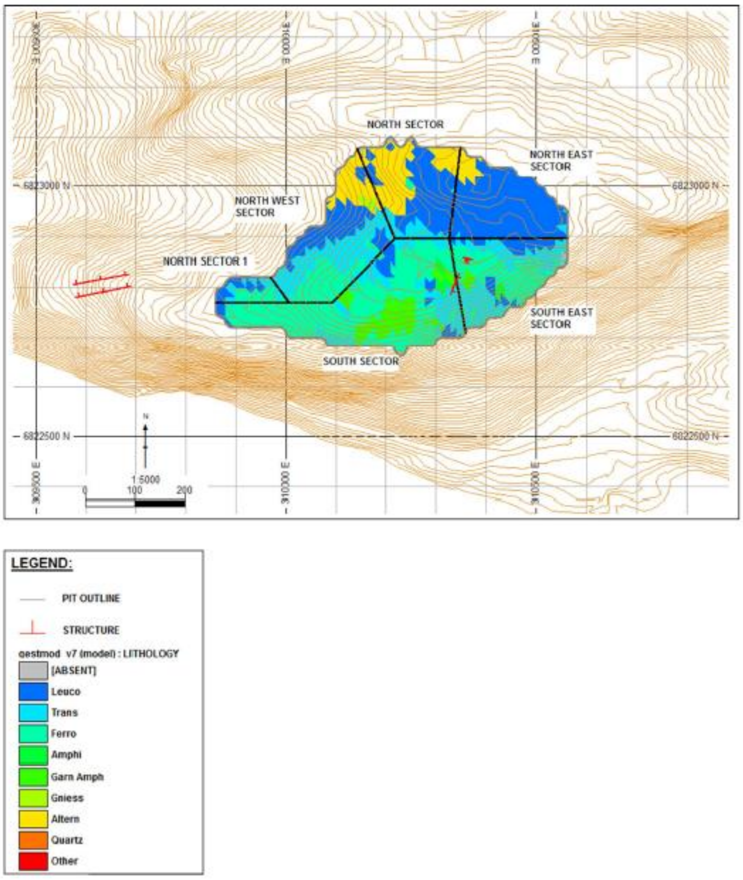


Figure 9-1: WAI Geotechnical Design Sectors

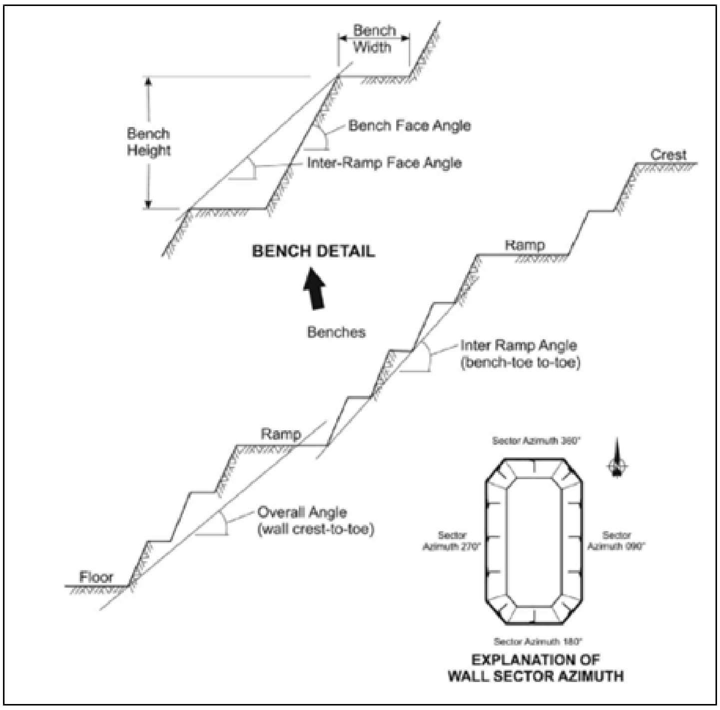


Figure 9-2: Pit Slope Configuration Illustration (Read and Stacey, 2009)

Table 9-1: Recommended Open Pit Slope Angles

Sector	Wall Orientation (°)	Max Slope Height (m)	Wall Lithology	Bench Height (m)	Bench Face Angle (°)	Berm Width (m)	Inter-Ramp Angle (°)	Max Inter-Ramp Slope Height (m)
North Sector	155-180	270	ALT, LEU	15	85°	5.0	67	200
North East Sector	180-270	260	LEU	15	68°	5.0	54	200
South East Sector	270-350	145	TRA, FER	15	68°	5.0	54	200
South Sector	350-030	215	FER, AMP	15	68°	5.0	54	200
North Sector 1	030-055	50	FER	15	85°	5.0	47*	200
	055-140	50	FER, LEU	15	74°	5.0	58	200
North West Sector	140-155	180	FER, LEU, TRA	15	74°	5.0	58	200

Notes:

1. MAX SLOPE HEIGHT IS BASED OPEN PIT OUTLINE PROVIDED BY A.WHEELER (MARCH 2012) OPTIMISED PIT SHELL
2. SLOPE ANGLES DETERMINED BY KINEMATIC AND ROCK MASS STABILITY ANALYSES
3. OVERBURDEN HAS BEEN CONSIDERED AS NEGLIGIBLE.
4. OVERALL SLOPE ANGLES WILL BE DEFINED BY STEP-OUTS AS REQUIRED FOR INTER-RAMP LIMITS AND HAUL RAMPS IN THE FINAL PIT WALL, ONCE DEIGN IS DEFINED.
5. NORTH SECTOR 1 (030-055) REQUIRES STEP-OUT.

The above recommended slope angles were used as the basis of the pit optimisation work and subsequent mine design carried out in this study.

9.2 Underground Geotechnical Study

SINTEF Building and Infrastructure, a Norwegian company affiliated to the Norwegian University of Science and Technology (NTNU) and the largest independent research organisation in Scandinavia, was commissioned to undertake a PFS standard geotechnical review of potential underground mining methods for Engebø. A summary of the SINTEF studies is given below.

Based on workshops and high-level evaluations, SINTEF were provided with three underground design concepts for geotechnical evaluation as follows; the design concepts were drawn up by the Project mining team:

- A long hole open stoping design, which tests the applicability of bulk mining methods at Engebø
- A room and pillar stoping design, which enables a more selective mining method to be tested
- A modified long hole open stoping design, which aimed to stretch the boundaries of the first long hole open stoping design provided to SINTEF.

The above mining layouts are summarised in below.

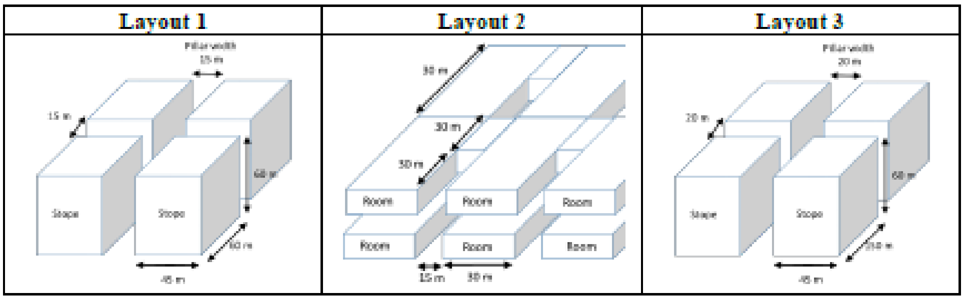


Figure 9-3: Mining Layouts Analysed by SINTEF for Geotechnical Stability

For more details of the process undertaken to derive the above underground design concepts, refer to Section 12 of this report.

For each of the above layouts numerical modelling was carried out in the FLAC3D (Itasca) finite element modelling software by creating simple 3D models to enable each layout to be evaluated from a stability/rock mechanics point of view. The numerical models were set up by analysing rock properties and rock mass, and by deriving key assumptions for *in-situ* stress measurements at Engebø based on regional data (in the absence of detailed *in-situ* stress measurements which will only be available for Engebø in the next phase of the Project). Key assumptions made for modelling include the following:

- *In-situ* stress conditions were extrapolated from previous measurements in the area (among others, one reading was taken 15 km south of the Engebø area across the fjord)
- The rock mass quality used in the modelling was considered to be 'Very Good" with a Geological Strength Index (GSI) of 80; this value is considered to be conservative
- No explicit discontinuities were included in the model
- Each mining level was assumed to be excavated at the same time and no consideration was taken of the possible influence of sequential excavations.

The results obtained from the numerical modelling can be summarised as follows:

- Layout 1 – long hole open stope design with 15 m wide pillars, 15 m wide sills between mining levels and stopes of 45 m width, 60 m height and 60 m length
  - Sigma 1, the main induced stress, reaches up to 20 MPa at its highest compared to a rock mass global strength of 80 MPa; this indicates that the selected vertical and sill pillar thicknesses are considered stable
  - Consideration should be given in the next round to reducing the pillar thickness to pillar height from the current ratio of 4 to 3; the easiest way to do this is to increase the pillar width to 20 m from the current 15 m
  - Consideration in future studies should be given to increasing the stope length from the current 60 m to 100 m; this will increase the extraction percentage

- Layout 2 – room and pillar stope design with 15 m wide by 15 m high by 30 m long pillars, 15 m wide sills between mining levels and rooms of 30 m width, 15 m height and 1,200 m length
  - The deepest vertical pillars are exposed to Sigma 1 stresses of 40 MPa compared to the global rock mass strength of 80 MPa; this indicates that instability might occur, which can be mitigated by increasing the pillar widths in the deeper parts of the mine to greater than 15 m
- Layout 3 – modified long hole open stope design with 20 m wide pillars, 15 m wide sills between mining levels and stopes of 45 m width, 60 m height and 150 m length
  - The deepest vertical pillars and sill pillars are exposed to Sigma 1 stresses of 20 MPa compared to the global rock mass strength of 80 MPa; this indicates that the layout is considered to be stable.

In conclusion, 3D numerical modelling indicates that from a stability/rock mechanics perspective all three layouts can be used for PFS design purposes, although on site *in-situ* stress measurements should be made in the next phase to support more detailed modelling. Based on this high-level SINTEF evaluation, both a bulk mining (long hole open stoping) and room and pillar layout are considered to be geotechnically stable alternatives for design of an underground mine for the PFS mine plan. More details on the selection process of the preferred option, long hole open stoping, are provided in Section 12.6 below.

9.3 Seismic Risk

With Norway being classified as low to intermediate on a global scale of seismicity, the seismic risk associated with the Project is considered to be low. Norway does, however, have the highest seismic risk in north-western Europe owing to its proximity to the continental shelf off its west coast.

9.4 Future Work Programme

To support the FS phase of the Project the following work programmes will be undertaken:

- Hydrogeological studies
- *In-situ* rock stress measurements
- Geotechnical laboratory testing programme
- Detailed geotechnical mapping
- Further geotechnical drilling
- Geotechnical modelling of the open pit and underground orebodies.

10. Hydrogeology, Hydrology and Geochemistry

During this phase of the Project, a geological drillhole dipping programme was undertaken by Nordic Mining to determine the water levels in boreholes drilled at Engebø. Data gathered during this programme, together with site observations and information provided in WAI's report were used by SINTEF to carry out a high level hydrological/ hydrogeological assessment of Engebø. A summary of the review is given below.

10.1 Hydrogeology

10.1.1 Open Pit

Since the planned open pit at Engebø is in an area with high relief and fresh outcrops with no regional fault zones/weakness zones it is not expected that water flow and water pressure will result in significant rock instability in the open pit. However, highly jointed zones may lead to infiltration of water and possible increased water pressure locally. If water pressure occurs in the pit slopes, a solution would be to drill drainage holes to release the water pressure and thereby reduce the risk of slope instability. Construction of ditches leading water outside of the open pit area is an additional mitigation measure to reduce water in-flow into the pit. Excessive water is expected to accumulate at the lowest part of the open pit and the design caters for removal of this water by means of a drainage hole through the underground workings and subsequent cleaning up prior to discharge to the environment.

10.1.2 Underground

Based on available information (which is limited at this stage) and observations in the nearby road tunnel, high water in-flows to the underground excavations are not expected. Water-bearing weakness zones may appear in parts of the underground mine but these are not expected to give severe stability challenges. The design caters for the removal of all water which enters the underground workings.

In summary, limited hydrology and hydrogeological data are available at present for the Engebø deposit. The likely impact of hydrology and hydrogeology on the final mine plans, however, is considered to be low for the following reasons:

- The Engebø deposit is situated on an isolated outcrop of the Engebø ridge above the Førde Fjord
- Limited surface water on the outcrop can be observed. Only minor water flows from horizontal fractures in the lower quarry exposures can be seen
- The only water flow which can be seen in the 0.63 km-long road tunnel, which runs through the western portion of the deposit, consists of minor drips near to each portal
- Hydraulic conductivity is anticipated to be low, pre-dominantly through fractures, and associated water levels are anticipated to be low owing to the location of the deposit in isolated high ground.

It is essential, however, that a more detailed hydrogeological study is undertaken in the FS phase to support the above conclusion that the impact of water on the mine plans (open pit slopes and dewatering) is considered to be limited. This comment is particularly important when deeper portions of the orebody are included in the mine plan where groundwater pressures are likely to be higher and where mining will take place below sea level. The volumes and qualities of water pumped out of the open pit and underground workings also needs to be considered. Hydrogeology considerations remain a risk to achievement of the overall mine plan although the risk is not considered to be high.

10.2 Hydrology

There is limited surface water at Engebø, with only one stream (leading to the Gryta river) running on the northern side of the proposed waste rock disposal area to the north-east of the pit, as shown in Figure 10-1 below.

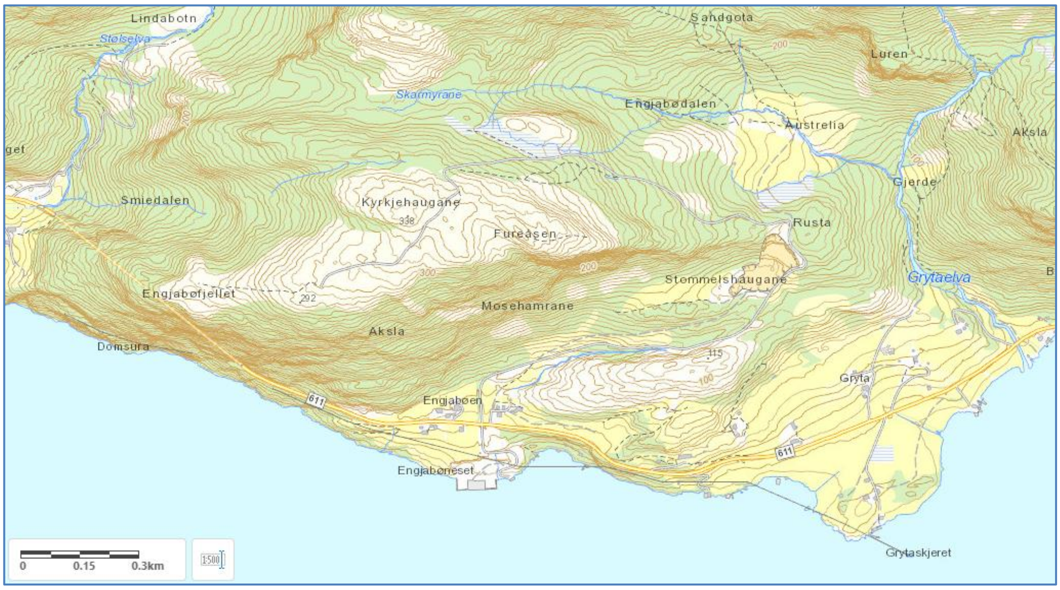


Figure 10-1: Map of Registered Rivers and Streams at Engebø

As a result of the lack of streams or rivers through the mining area and processing site, flood line determination does not apply; no river diversions will be required.

Conditions relating to hydrology which are specified in the zoning plan (planning permit) are as follows:

- Run-off from the open pit mining area must be secured by means of a sediment basin with the appropriate capacity
- In order to provide a safeguard against run-off from the waste rock disposal site into the Gryta river, a sedimentation pool with sufficient capacity must be established, as must a drainage ditch to carry water to and alongside the works road down to the processing site at the port.

For further details of the environmental requirements associated with the permitting process, refer to Section 19.

10.3 Geochemistry

The deposition of waste rock on land will not have a negative impact on the environment from a geochemistry perspective. This was determined through a testwork programme as summarised below.

To support the process of obtaining a discharge permit and zoning plan (planning permit), two of the key legislative requirements for Nordic Mining to construct and operate a mining and processing operation at Engebø, testwork was carried out on seven core samples from the drilling programme. ALS Global, an internationally accredited laboratory for analytical chemistry and testing services, carried out the testwork. The critical number for permitting purposes is the Neutralisation Potential Ratio (NPR). Norwegian standards follow the California and Nevada standards, which require an NPR of greater than three. The lowest number achieved for the Engebø samples was 101.60, more than 30 times higher than the minimum requirement. The Net Neutralisation Potential (NNP) is also an important number with typical requirement for readings to be higher than 20 kg/t $CaCO_3$. In the case of the Engebø samples, the readings are all above 500 kg/t, which exceeds the guideline by a factor of 25.

The testwork clearly demonstrates that there are no minerals present in the ore or waste rock that could cause acid drainage; as a result, Nordic Mining has permission to construct a standard waste rock site (a Category 3 site), with no extra requirement such as plastic lining to cater for acid mine drainage.

The following excerpt from the discharge permit summarises the findings of the Norwegian Ministry of Climate and the Environment, which led to the granting of the permit:

“In order to reach the eclogite ore the gangue material covering the orebody must be removed. Permission has been granted to deposit the gangue tailings in a separate fill in Engjabødalen, northeast of the open pit. In the course of the open pit phase there will be a production of 2-3 million tonnes of gangue tailings per year; in total, 35 million tonnes equalling 15 million m^3. Once there is a transition into underground mining, the need for removing additional gangue will be very limited. The disposal site at Engjabødalen will have a total surface area of 460 decares (0.46 km^2)”.

To mitigate any negative environmental impacts on the waste rock disposal site, the following conditions have been placed on the disposal process:

- Leachate from the disposal site shall be collected, which means that under normal operating conditions leachate and particulate material will not reach the Grytelva river and drinking water wells
- Ongoing monitoring must be undertaken
- An environmental risk assessment must be undertaken to determine pollution risks in the event of extreme weather and to establish design requirements for runoff water requirements in such situations
- The site must be closed in accordance with the plan submitted for closure and after-use.

10.4 Future Work Programme

To support the FS phase, the following work programmes will be undertaken:

- Structural geology (fault zones/weakness zones) study
- Joint sets, persistence, filling analysis
- Study of visible water in the Project area
- Rainfall regime and intensity determination
- Determination of the catchment area for the open pit
- Systematic study of the water level in selected drill-holes
- Observation and registration of the condition and current use of the water wells below Engebø
- Pumping tests of selected drill-holes
- Lugeon measurements (double packer tests to determine hydraulic conductivity in sections with joints)
- Flowmeter tests using an impeller to find the location of water-bearing joints)
- Hydrogeological analysis.

11. Mineral Processing

11.1 Introduction to Process Development and Results

11.1.1 Main Results

The main goal of the PFS process testwork was to develop an integrated flowsheet to produce both rutile and garnet products from the Engebø ore. The PFS programme was successful in making pigment grade rutile ~ 95% TiO_2 at market specifications. Recovery of TiO_2 was significantly increased from previous testwork to above 60%. Marketable products of both coarse and fine garnet were made from the Engebø ore that meet the current market specifications of 30/60, 80 and 100 mesh products.

The PFS testwork commenced in September 2016 and marked the start of a series of testwork programs to define suitable process technologies to recover rutile and garnet products that meet current market product specifications. Process test work was conducted with industrial scale equipment and tested in 0.5 t to 4.0 t feed batches for different process sequences. The sections that follow describe in detail the evolution of the testwork portfolio driven and shaped by testwork results that successfully developed a viable process flowsheet as shown in section 11.1.5 below using conventional and cost-effective process equipment. The salient results of the process development efforts are summarised in Table 11-1 below, and include the metrics used as inputs to the financial model. As shown in Table 11-1, two ore types were considered in the PFS, namely ferro ore and trans ore. Ferro ore is higher grade with a TiO_2 content greater than 3% while trans ore is lower grade with a TiO_2 content between 2% and 3% TiO_2. Although ferro ore is defined as the sole ore for the PFS, testwork was also conducted on trans ore to explore and develop mining and processing options that include a larger fraction of the deposit.

In addition to process testwork, comminution testwork was undertaken to develop a flowsheet capable of producing suitable feed to the process to maximise garnet and rutile liberation and ultimately lead to marketable products while minimising the generation of fines and overgrinding of rutile. A staged comminution circuit was selected which produces a feed at 212 µm to 550 µm for coarse garnet recovery and through regrinding, a <212 µm stream for rutile and fine garnet recovery.

Table 11-1: Summary of the Main Garnet and Rutile Process Results

Garnet			
Parameter	**Units**	**Ferro Ore > 3% TiO_2**	**Trans Ore 2% to 3% TiO_2**
Garnet grade of sample	Wt%	50.4	47.5
Garnet grade of coarse product	Wt%	95.4	94.1
Garnet grade of fine product	Wt%	95.5	93.3
Overall yield to coarse product	Wt%	11.4	6.9
Total garnet yield (62.5% coarse; 37.5% fine)	Wt%	18.3	11.0
Coarse product particle size distribution: D_{50}	µm	274	249
Rutile			
Parameter	**Units**	**Ferro Ore > 3% TiO_2**	**Trans Ore 2% to 3% TiO_2**
TiO_2 grade of sample	Wt%	4.7	2.7
TiO_2 grade of rutile product	Wt%	94.9	92.8
TiO_2 recovery to rutile product	Wt%	60.2	42.1
De-rating factor for scale-up*	%	97	-
De-rated Recovery	Wt%	58.4	-
Particle size distribution: D_{50}	µm	106	101
*A de-rating factor is applied to account for an expected decrease in recovery when operating a full-scale process, thereby providing a more conservative input to the financial model			

11.1.2 *Achieved Rutile Products*

The chemical composition of the rutile product from the ferro ore testwork programme is provided below in Table 11-2. The target rutile grade for a pigment-grade rutile product was met and elements are within the specifications.

Table 11-2: Composition of the Rutile Product from the Ferro Ore Testwork Programme Determined by XRF

Rutile Product Composition		
Compound	**Specification (Wt%)**	**Rutile Product (Wt%)**
TiO_2	>94.0	94.90
Fe_2O_3	<1.0	1.63
SiO_2	<2.5	1.53
Al_2O_3	<1.5	0.31
Cr_2O_3	-	0.01
MgO	<1.0	0.03
MnO	<1.0	0.02
ZrO_2	<1.0	0.06
P_2O_5*	<0.03	0.01
V_2O_5	<0.65	0.41
Nb_2O_5	<0.25-0.5	n/d
CaO**	≤0.8/0.15	0.35
K_2O	-	0.01
CeO_2	-	n/d
S*	<0.03	0.17
(FeS_2)	-	(0.30)
SnO_2***	<0.05	<0.02
U (ppm)	-	<10
Th (ppm)	-	<10

*Welding rod specification for P and S
**Non-sieve plate and sieve plate specification
***SnO_2 detection limit at 0.02%. SnO_2 limit applicable to the molten salt market.

The particle size distributions of the products from two ferro testwork programmes are provided below in Table 11-3 and Figure 11-1.

Table 11-3: Particle Size Distribution of the Rutile Product from Ferro Ore Testwork

Rutile Product Particle Size Distribution		
Aperture (µm)	Programme 1308* (Cum. Wt%)	Programme 1245** (Cum. Wt%)
250	100	100
212	99.9	100
180	93.1	75.2
150	81.7	52.1
125	65.0	31.0
106	50.0	18.3
90	29.7	6.6
75	15.1	2.3
63	6.9	n/a
45	0.7	0
0	0	0
D_{50} (µm)	**106**	**147**
D_{80} (µm)	**147**	**186**

*See section 11.4.3 for details of testwork programme 1308
**See section 11.4.2 for details of testwork programme 1245

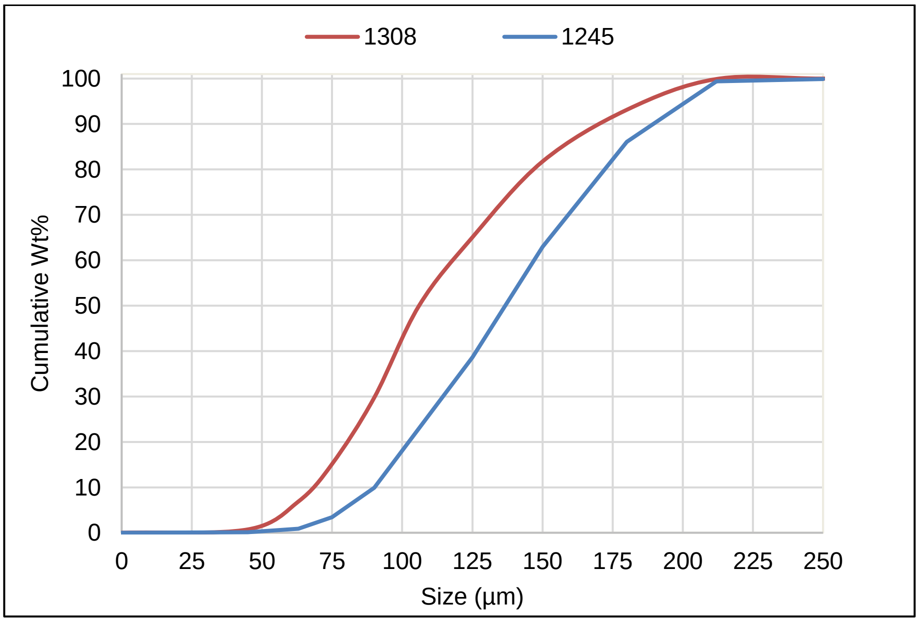


Figure 11-1: Particle Size Distributions of the Rutile Products from the Two Ferro Ore Bulk Testwork Programmes

Looking at the data in Table 11-3 and Figure 11-1, it can be seen that the PSD for the 1308 product is more fine-grained than the 1245 product. The product from programme 1308 contains approximately 12.8 percentage points more <75 µm material than the product from 1245. A pigment grade rutile product with minimal material below 75 µm is favoured and should in general be below 5%, although this varies between producers. Although the PSD of the 1308 product is finer than the 1245 product, the process development programme has demonstrated that a product conforming to the industry standard can be made. More work will be done as part of the Definitive Feasibility Study testwork programmes to optimise the comminution and screening of the rutile feed to produce a coarser product. The molten salt industry for titanium metal production could serve as a potential market for the finer rutile (45 µm to 75 µm) produced in the process. This should represent a smaller product stream that could easily be absorbed by this market. The tin (as SnO_2) level of the produced rutile is below 0.05% which is the threshold for the molten salt feed, as higher levels of tin make the metal brittle. Based on this information it is assumed that all produced rutile as presented above is a saleable product.

11.1.3 Achieved Garnet Products

The mineralogy of the coarse and fine garnet products achieved in the testwork programmes on ferro ore are provided below in Table 11-4. Both products are within the target of more than 92% garnet and have acceptable levels of quartz and other impurities.

Table 11-4: Mineralogy of the Coarse and Fine Garnet Products from the Ferro Ore Testwork Programme Determined by QEMSCAN Analysis (SGS Canada)

Garnet Product Mineralogy		
Mineral	**Coarse Garnet (Wt%)**	**Fine Garnet (Wt%)**
Garnet	95.4	95.5
Rutile	0.43	0.45
Pyrite	0.05	0.02
Quartz	0.47	0.28
Amphiboles	0.59	0.76
Clinopyroxenes	2.21	2.31
Other	0.85	0.68

The particle size distributions of the coarse and fine garnet products are shown below in Table 11-5.

Table 11-5: Particle Size Distributions of the Coarse and Fine Garnet Products from the Ferro Ore Testwork Programme

Garnet Product Particle Size Distributions		
Aperture (µm)	Coarse Garnet (Cum. Wt%)	Fine Garnet (Cum Wt%)
600	100	-
500	98.8	-
425	95.1	-
355	84.0	-
300	65.2	-
250	34.9	100
212	13.2	99.2
180	1.5	83.3
150	0.5	57.2
125	0.2	26.2
106	0	7.8
90	-	0.7
75	-	0
D_{50} (µm)	**274**	**144**
D_{80} (µm)	**342**	**176**

The particle size distributions of the two garnet products achieved in the PFS and the three target garnet products, namely 30/60, 80 and 100 mesh, are provided in Figure 11-2 and Figure 11-3 below (for a note on the meaning of mesh sizes, refer to Section 17.2.1). The target products will be derived from the coarse and fine garnet products by blending in the necessary ratios. The target is to achieve a blend such that 30/60 mesh, 80 mesh and 100 mesh are produced in the relative amounts of 35%, 32.5% and 32.5%, respectively. A surplus of fine garnet is produced; therefore, coarse garnet drives overall capacity and the expectation is that a coarse-to-fine ratio of 62.5:37.5 is sufficient to meet the target PSD of each product type.

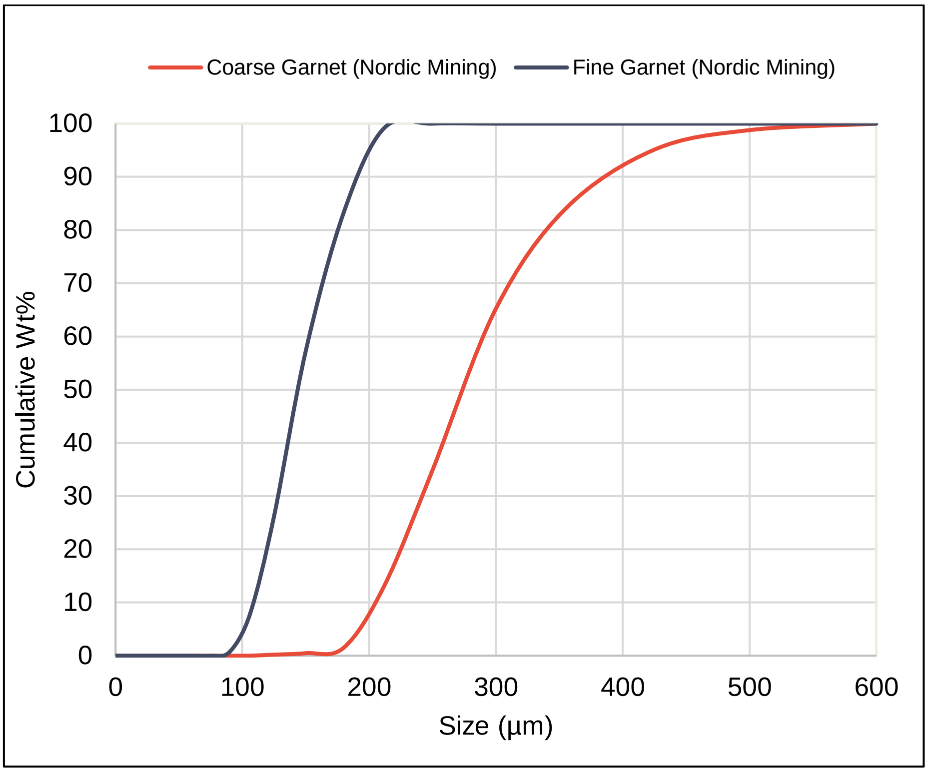


Figure 11-2: Particle Size Distributions of the Two Garnet Products achieved in the PFS

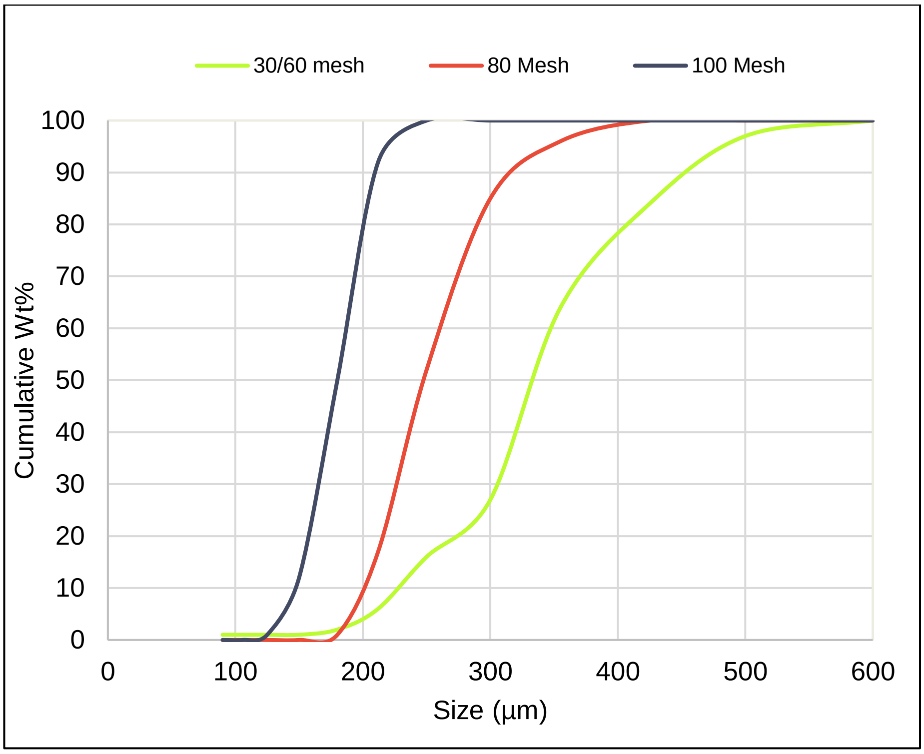


Figure 11-3: Particle Size Distributions of the Three Target Garnet Products

11.1.4 *Historic Test Work Programmes*

Previous testwork conducted by DuPont in the late nineties included work to understand and define comminution, wet magnetic separation, wet gravity separation, flotation, dry magnetic separation, electrostatic separation and agglomeration testwork. The information from historic testwork consisted of a hand drawn flowsheet, laboratory assay results, various memos from comminution tests and particle size distributions of various testwork fractions. Based on the test work, DuPont defined a flowsheet for rutile processing. A simple pilot setup revealed a rutile recovery of approximately 47%. DuPont stated that this was based on non-optimised process conditions and that the recovery most likely could be improved. No test work was done to recover garnet.

Nordic Mining acquired the asset in 2008 and undertook preliminary testwork programmes at several institutions to improve knowledge and understanding of the contained valuable minerals rutile and garnet. Several small-scale tests of different process stages were undertaken. Testwork included magnetic and electrostatic separation at Outotec, and several test programmes including flotation, at NTNU. The results did not show significant improvement of the processing performance and recoveries that were achieved by DuPont. A garnet concentration test by magnetic separation was done by a commercial garnet producer and was successful in producing a water-jet quality garnet product. All reports were made available for the PFS team.

11.1.5 *Introduction to the Developed Flowsheet*

The flowsheet developed in the PFS is shown below in Figure 11-4. Run of mine passes through a multi-stage comminution circuit that produces coarse (212 µm to 550 µm) and fine (<212 µm) products through jaw crushing, cone crushing, impact (Hazemag) crushing and rod milling. The material is de-slimed to reject material less than 45 µm. The coarse fraction (>212 µm) is processed through a gravity concentration circuit and the resultant concentrate is fed to a two-stage dry magnetic separation circuit. The final middlings from magnetic separation are screened at 212 µm and the oversize serves as the coarse garnet product. Meanwhile, the tails from the gravity circuit and a portion of the magnetic and non-magnetic rejects from the magnetic circuit are reground to <212 µm to liberate and recover additional rutile and garnet. This stream is combined with the fine stream from the comminution circuit and fed to a wet magnetic separation circuit that produces magnetic and non-magnetic streams. The non-magnetic stream serves as feed to the rutile circuit which comprises gravity concentration followed by flotation to remove pyrite and then dry magnetic and electrostatic separation, producing a final conductive, rutile product. The magnetic stream from wet magnetic separation is fed to a gravity concentration circuit where the resultant concentrate is fed to a dry magnetic separator. The middlings from dry magnetic separation are screened at 106 µm and the oversize serves as the fine garnet product. Wet rejects are fed to a dewatering circuit to recover process water and the underflow is combined with dry rejects and sent to the co-disposal system. The rejects are then diluted with sea water in the co-disposal system and discharged to the fjord bed.

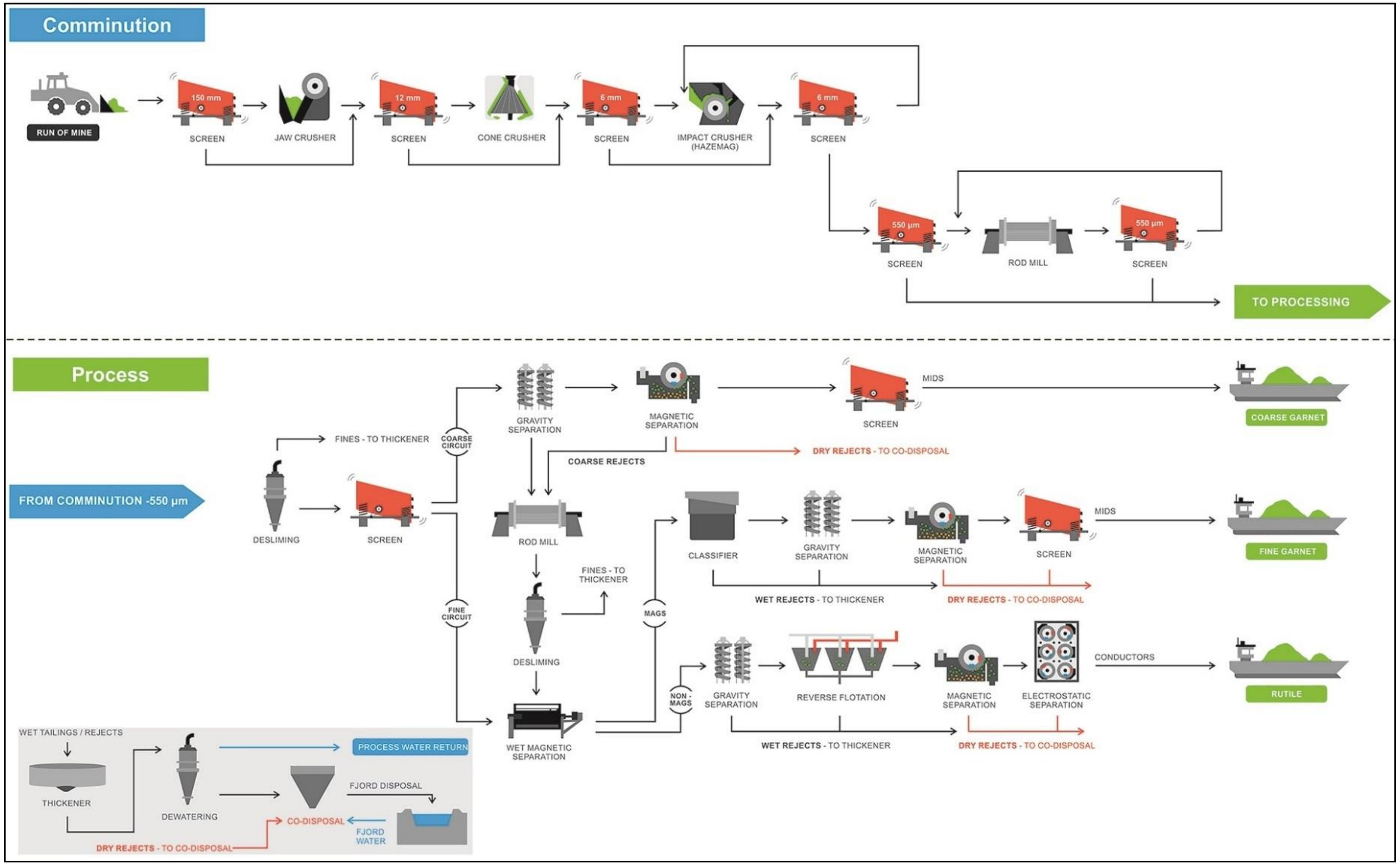


Figure 11-4: Block Flow Diagram of the Developed Flowsheet

11.1.6 *Selection of Yield as the Basis for Garnet Calculations*

The overall mass yield to the coarse garnet product was selected as the basis for reporting the garnet processing performance for the following reasons:

- The particle size distribution of the garnet products is an important factor considering the amount of garnet that can be retrieved from the process. Therefore, there is not a clear connection between garnet grade and recovery
- Using yield instead of recovery is common practice for a number of industrial minerals where quality of the product and not necessarily grade is important to predict the retrievable amount
- Garnet grades throughout the deposit are difficult to measure correctly because there is no chemical assay that can directly quantify the garnet content. For the deposit, mineralogical analysis of garnet was performed by QEMSCAN and a calculation based on chemistry was applied as explained in Section 7 of this report. For the quantification of garnet, a conservative approach was taken due to the limitation of the data. A yield approach for predicting the retrievable amount of garnet was determined to give a better result than recovery calculations based on head grade and a set recovery factor.

As part of the DFS work, a programme aiming to increase the knowledge of the grade and quality variations of garnet in the deposit, and to better understand how this affects the recovery and yield, will be investigated.

Mass yield (referred to simply as ‘yield’ in this report) is defined as the fraction of the feed mass reporting to a specified product or outflow stream. Therefore, the “overall yield” to a specified product is the fraction of RoM reporting to that product, which in the current context is the garnet product. This should not be confused with recovery, which refers to a particular species present in the ore. Recovery then, is the fraction of the total amount of the species (e.g. TiO_2) that reports to the specified product (e.g. rutile), and does not consider overall or total masses.

11.2 Sample Selection and Data Acquisition

In 2012 six (6) bulk samples were drilled and blasted at the Engebø mountain. The blasting was carried out by a local entrepreneur, Røyseth Maskin. The samples were selected based on being representative samples of the ore in the first years of open pit mining of the deposit. In total 110 t were obtained. Table 11-6 below lists the name, ore type, location, size and chemically assayed rutile grade and ilmenite content of each sample. Drill cuttings were obtained from each blast site and representative samples were obtained for chemical assays.

Table 11-6: Source Samples for Process Testwork

Sample	Lithology	TiO_2 Grade* (XRF)	% TiO_2 Ilmenite* (HCl)	X-Coordinate	Y-Coordinate	Sample Size (tonnes)
ENG1	Trans	2.60 %	0.40 %	310189	6822818	18.0
ENG2	Ferro	5.22 %	0.12 %	310170	6822852	21.0
ENG3	Ferro	3.81 %	0.12 %	310214	6822794	23.0
ENG7	Ferro	4.71 %	0.12 %	310062	6822788	14.0
ENG205	Trans	2.42 %	0.07 %	310285	6822833	16.0
ENG217	Ferro	5.20 %	0.13 %	310099	6822766	17.2

*All samples were assayed in 2016 except ENG01 which was assayed in 2013. ENG01 is not included in the PFS testwork.

The material was stored outdoors as blasted blocks at a local sand pit. The material does not seem to be affected by the short-term outdoor storage, and there is little evidence of weathering or oxidation. The iron oxidation surface can be found in freshly blasted samples and is believed to have been placed prior to storage.

Figure 11-5: ENG07 Sample in Storage

11.2.1 *Sample Preparation and Comminution*

In 2016 the blasted samples were crushed to < 200 mm and transferred to big bags and moved to Nordic Mining's office building in Naustdal.

Representative samples for further comminution and process tests were assembled by laying out material from each sample in an elongated heap, and then shovelling material by the length of the heap to create representative samples. The samples were continually weighed to achieve the right mass.

Table 11-7 below shows the blending ratios of the different source samples that were used to create blended samples for process and comminution testwork. Samples 1231, 1245 and 1308 are blends that represent ferro-eclogite, high-grade ore samples with a rutile grade above 3%; sample 1234 represents a transitional (trans) eclogite, a low-grade ore sample with a rutile content between 2% and 3%.

Table 11-7: Source Samples that Contributed to each of the Blended Samples for Process Testwork

Source Sample	Lithology	Processing Sample			
		1231	1245	1234	1308
ENG1	Ferro				
ENG2	Ferro	1	1		2.3
ENG3	Ferro	1	1		1
ENG7	Ferro	1	1		1
ENG205	Trans	1	1	1	1
ENG217	Ferro	2	2		0.7

11.2.2 Production of the 1231 Sample

The 1231 sample was made by crushing each source sample through a jaw crusher at NTNU to <8 mm and then mixing the samples as shown in Table 11-7. The combined <8 mm material was then crushed by a High Pressure Grinding Roller (HPGR) and screened on a Derrick screen at 212 µm. The oversize was further milled to <212 µm in a rod mill while continuously being mixed with the screen undersize. The 1231 is characterised as a finely comminuted ferro blend sample that was made to liberate rutile. The sample was shipped in plastic barrels to IHC Robbins for process testing to recover rutile and fine garnet.

11.2.3 Production of the 1245 Sample

The 1245 sample source material was jaw crushed to <30 mm at NTNU and blended according to Table 11-7 above. The combined <30 mm feed was crushed by a laboratory scale impact crusher and screened at 500 µm. The impact crusher at NTNU broke down so that only 70% of the material was crushed this way. The residual material was crushed using an HPGR. The oversize was milled to <500 µm in a rod mill while continuously being mixed with the screen undersize. The 1245 represents a coarsely comminuted ferro ore sample with the aim of recovering both coarse and fine garnet products as well as making a rutile product. The sample was sent to IHC Robbins in plastic barrels for process testing.

11.2.4 Production of the 1308 Sample

The 1308 sample is a carefully comminuted sample that was run through an optimised comminution route established through extensive testwork as described in Section 11.3.4. In brief, <200 mm material was stage crushed in a cone crusher at Mintek (South Africa) to <12 mm to make a suitable feed for the impact crusher. The <12 mm material was further crushed in a larger scale Hazemag Impact crusher at IMS (South Africa) to <550 µm. The crushing was monitored to ensure optimum crushing conditions. The

<550 µm material was collected as a crushed product and the >550 µm was further rod milled in closed circuit with a 550 µm screen to produce the <550 µm rod mill product. The rod milling was done at Mintek. The sample was then packed and sent to IHC Robbins for process testwork. The 1308 sample is an optimal comminuted sample with the aim of improving the recovery of coarse and fine garnet and rutile products by improved mineral liberation and minimised fines generation.

11.2.5 Production of the 1234 Sample

The 1234 sample represents a transitional (Trans) eclogite, low grade, ore sample for testing ore variations for the developed flowsheet. The 1234 sample was prepared in the same manner as the 1308 sample, but the coarse crushing to -12 mm was done by SGS Johannesburg, and the rod milling was carried out by JKTech in Brisbane, Australia. The impact crushing was done at IMS as for the 1308 sample.

Table 11-8: An Overview of the Different Bulk Samples

Feature	Processing Sample			
	1231	1245	1234	1308
Rock type (-eclogite)	Ferro	Ferro	Trans	Ferro
Type of sample	Blend	Blend	Single Source	Blend
Original processing sample size (kg)	4854	1255	879	608
Comminuted by	NTNU	NTNU	SGS/IMS/JKTech	IMS/Mintek
Processed by	IHC Robbins	IHC Robbins	IHC Robbins	IHC Robbins
Blast year	2012	2012	2012	2012
Sample year	2016	2016	2017	2017
Process year	2016/2017	2017	2017	2017

11.2.6 Sample Characterisation – Grain Size Distributions, Chemical Assays and QEMSCAN

The cumulative grain size distributions for the samples are presented in Table 11-9 and Figure 11-6 below.

Table 11-9: Prepared Bulk Sample Cumulative Particle Size Distributions

Size (µm)	Processing Sample			
	1231	1245	1234	1308
600	100	100	99.6	100
500	100	100	96.5	99.4
425	100	99.0	91.4	94.1
300	100	87.5	77.7	80.0
212	91.0	74.0	65.3	65.9
150	75.8	56.6	49.5	50.9
106	54.0	40.7	36.1	36.8
75	32.5	24.1	23.9	25.0
63	27.0	18.9	19.1	17.4
45	20.7	14.7	15.0	14.5

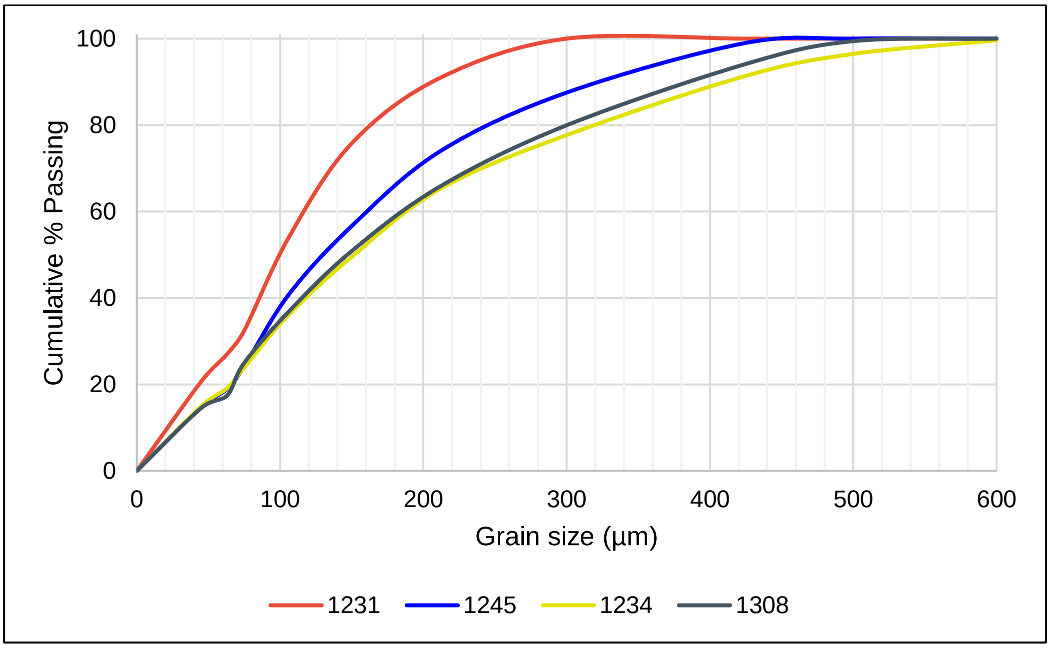


Figure 11-6: Prepared Bulk Sample Cumulative Particle Size Distributions

Table 11-10 below shows the most important chemical and mineralogical features of the different samples.

Table 11-10: Prepared Bulk Sample Properties

Feature	Processing Sample				
	UoM	1231	1245	1234	1308
Garnet grade in total sample*	wt%	43.2	47.3	47.5	50.4
Garnet distribution to +212 µm	wt%	n/a	26.8	34.5	35.1
Garnet distribution to +300 µm	wt%	n/a	10.5	20.0	20.7
Theoretical garnet recovery**	%	n/a	13.8	27.1	39.5
Theoretical garnet yield (at 92% product grade)	wt%	n/a	7.1	14.0	21.7
TiO_2 grade of total sample*	wt%	4.50	4.47	2.66	4.67
TiO_2 grade in -212/+45 µm	wt%	4.63	5.27	2.87	5.78
TiO_2 distribution to +212 µm	wt%	0	13.9	26.7	22.3
TiO_2 distribution to -212/+45 µm	wt%	79.3	72.4	56.1	65.8
TiO_2 distribution to -45 µm	wt%	20.7	13.7	17.2	11.9
Theoretical TiO_2 recovery	%	67.4	63.1	59.9	73.3
* Garnet mineralogy by QEMSCAN and TiO_2 chemistry by XRF; ** Based on free + liberated garnet					

All the comminuted samples were further sampled for chemical and mineralogical analysis by XRF and QEMSCAN. Where the comminution circuit consisted of both crushing and milling the two products were sampled individually before being combined into a total sample. In this way, important information regarding the performance of the two stages in the comminution circuit could be revealed. All the samples were assayed by both XRF (eight fractions) and QEMSCAN (six fractions). Analysing all the samples by the same setup every time gave good opportunities to compare the different comminution method and feeds for the processing testwork and made direct comparisons possible.

The samples were marked, and detailed information on all the samples were recorded in Nordic Mining's database together with tracking information from the couriers. The laboratories were informed about the estimated arrival time of the samples as well as information and procedures for treating the samples so that reliable data could be recorded rapidly with reduced risk of analytical or procedural errors.

Analyses of crush and rod mill products were copied into an MS Excel programme that combined the data correctly each time. For comparing and assessing different samples and comminution methods, pre-set parameters and criteria were investigated such as mineral and element distribution, liberation data and theoretical recoveries.

11.2.7 Theoretical Recovery Calculations and Theoretical Yield Figures

The theoretical garnet recovery calculations were based on the distribution of garnet to the coarse fraction (>212 µm) and the garnet liberation (>80% liberated) in this fraction. By multiplying the distribution of garnet with the liberation data, the potential garnet available for recovery is estimated, and dividing this by the head grade gives the theoretical recovery. The theoretical recovery of the coarse garnet is multiplied by 1.6 in order to include the specified amount of fine garnet (<212 µm) in the total garnet recovery that will constitute the three products: 30/60, 80 and 100 mesh. As there is a surplus of fine garnet, the coarse garnet recovery is the limiting factor in determining overall garnet recovery. The total garnet product should include 62.5% of the coarse garnet (>212 µm) and 37.5% of the fine garnet (<212 µm) products. The theoretical garnet yield is determined by multiplying the theoretical garnet recovery by the ratio of the garnet grade of the feed to the target garnet grade of the product.

The theoretical recovery of TiO_2 is calculated from the liberated (>80% liberation) TiO_2 available in the <212 µm product from the comminution circuit and the liberated TiO_2 from the regrind of the coarse garnet rejects (>212 µm). In both sources, rutile deportment to the <45 µm fraction is considered a loss of recovery.

The theoretical recoveries and yields were important factors in comparing different comminution methods when selecting the optimised comminution circuit as well as for predicting and comparing process testwork efficiencies.

11.3 Comminution Testwork

11.3.1 Introduction to Comminution Testwork

Comminution testwork was conducted in order:

- To characterise the ore
- To determine the optimal comminution route by:
  - maximising liberation of rutile and garnet
  - minimising production of fines
  - determining the comminution circuit with highest NPV.
- To determine key input figures for economic calculations.

Ore hardness characterisation testwork were conducted at Mintek, JKTech and Grinding Solutions. The following bench scale tests were conducted: Uniaxial Compression Strength testwork, Bond Crushability testwork, Bond Abrasion testwork, Bond Rod Work Index, Bond Ball Work Index, Grindmill testwork and SMC testwork.

Testwork at pilot scale using bulk samples was also conducted using different technologies in order to determine their performance and select the most cost-effective process route. The following pilot scale testwork were conducted: High Pressure Grinding Roll (HPGR), impact crusher using Hazemag technology, rod mill, cone crusher, Vertical Shaft Impactor (VSI).

The comminution circuit aimed to crush/mill the ore to <550 µm for coarser garnet beneficiation and re-grind the material to create a rutile and fine garnet feed at 212 µm. From the different circuits tested using different technologies, a three-stage crushing circuit comprising: (i) a jaw crusher as the primary crusher, (ii) a cone crusher as the secondary crusher, (iii) an impact crusher as the tertiary crusher and (iv) a rod mill to mill the ore to <550 µm, was found to be the most cost-effective process route. The success of this circuit is mainly due to: (i) a better liberation of coarser garnet produced by the impact crusher, (ii) a relatively significant proportion of liberated coarser garnet produced by the impact crusher, and (iii) the lower proportion of fines (<45 µm) generated.

11.3.2 Developed Comminution Circuit

The developed comminution circuit includes a jaw crusher as the primary crushing stage. The jaw crusher will operate in open circuit with pre-screening of the feed before the jaw crusher. The target size for the primary crushing stage is 80% passing 150 mm. The product of the primary crushing stage is fed to a secondary crushing stage which includes a bin and a double deck screen followed by a secondary cone crusher treating the oversize of the screen. The target size for the secondary crushing stage is 80% passing 50 mm. The product of the secondary crusher feeds the tertiary crushing stage. The tertiary crushing stage consists mainly of impact crushers operating in closed circuit with a 6 mm screen. The secondary crusher product is conveyed to a tertiary crusher bin. The tertiary crusher bin feeds screens ahead of the impact crushers. The screen oversize feeds the impact crushers operating in closed circuit with a 6 mm screen. The tertiary crushing stage product (<6 mm) is stored in a bin/silo before feeding the milling circuit. The crushing circuit product (<6 mm) is pre-screened to 550 µm. The oversize (>550 µm) feeds the primary rod mill operating in closed circuit with a 550 µm screen. The <550 µm generated from the pre-screening stage and rod-milling stage constitutes the feed to the coarse garnet beneficiation circuit.

11.3.3 Ore Hardness Characterisation Testwork

Comminution bench scale testwork was conducted on Nordic Mining's samples in order to characterise the ore and generate data for the design of the comminution circuit. The following testwork was conducted:

- The Uniaxial Compression Strength (UCS) test
- The Bond Crushability Work Index (CWI)
- The Bond Abrasion Index (AI)
- The Bond Rod Work Index (BRWI)
- The Bond Ball Work Index (BBWI)
- The SAG Mill Comminution (SMC) test
- The grind rod mill testwork.

Testwork was conducted at Geolabs Limited, Mintek and Grinding Solutions in the UK.

The Uniaxial Compression Strength (UCS) test was conducted at Geolabs Limited on different rock types and the results are given in Table 11-11 below. The average UCS value for the ferro and trans rock types which are mineralised is 141.2 MPa, while the 80th percentile is 172 MPa.

Table 11-11: UCS Results

No	Rock Types	UCS (MPa)		
		Minimum	Maximum	Average
1	Ferro	150	188	166
2	Trans	56.1	172	116.4
3	Leuco	72.2	133	97.3
4	Amphibolite	28.2	92.3	69.4
5	Alt	61.9	72.7	67.3
6	Gneiss			128

The Bond CWI test was conducted at Mintek and Grinding Solutions. The 80th percentile of the combined results obtained at both laboratories is 16.3 kWh/t. The summary results obtained at Mintek and Grinding Solutions are reported in Table 11-12.

Table 11-12: Bond Crushability Work Index Results Obtained at Mintek and Grinding Solutions

	Bond Crushability Work Index (kWh/t)			
Laboratory	Minimum	Maximum	Average	80th Percentile
Mintek	8.6	20.2	13.3	16.6
Grinding Solutions	4.8	18.6	10.9	14.3

The Bond AI test conducted at Grinding Solutions shows that the ore is abrasive with an index of 0.68. The AI results are summarised in Table 11-13 below.

Table 11-13: Abrasion Index Results

Bond Abrasion Index	0.68
Abrasiveness (French abrasiveness standard)	1,380

The BRWI test was conducted at Grinding Solutions at a limiting screen of 1,180 µm and at Mintek at a limiting screen of 550 µm. Results of 10.92 kWh/t and 10.90 kWh/t were achieved at closing screen of 1,180 µm and 550 µm respectively. The BRWI results are summarised in Table 11-14 below.

Table 11-14: Bond Rod Work Index Results

Sample ID	Limiting Screen (µm)	F80 (µm)	P80 (µm)	Net Production (g/rev)	Work Index (kWh/t)
Nordic Mining	550	10,851.20	323.02	6.63	10.90
Nordic Mining	1,180	10,751	778	11.87	10.92

The BBWI test was conducted at Mintek at a limiting screen of 212 µm. A BBWI value of 8.91 kWh/t classifying the ore as soft for ball milling was achieved as indicated in Table 11-15 below.

Table 11-15: Bond Ball Work Index Results

Sample ID	Limiting Screen (µm)	F80 (µm)	P80 (µm)	Net Production (g/rev)	Work Index (kWh/t)
Nordic Mining	212	2,522.53	182.42	3.77	8.91

An SMC test was conducted to generate the JK parameter A x b which is used for the modelling of the crushing circuit. The result of A x b of 35 was achieved classifying the ore as hard at coarser size. The SMC results are summarised in Table 11-16 below.

Table 11-16: SMC Results

Sample ID	DWI (kWh/ m³)	DWI (%)	MIi Parameters (kWh/t)			SG	A	b	A x b	ta
			Mia	Mih	Mic					
Nordic Mining	10.27	89	20.7	16.7	8.6	3.6	100	0.35	35	0.25

Laboratory batch rod mill testwork was conducted at different energies on a sample crushed to -13.2 mm. The objective of the testwork was to generate breakage parameters to be used for the design of the rod mill operating in closed circuit. The results are reported in Figure 11-7.

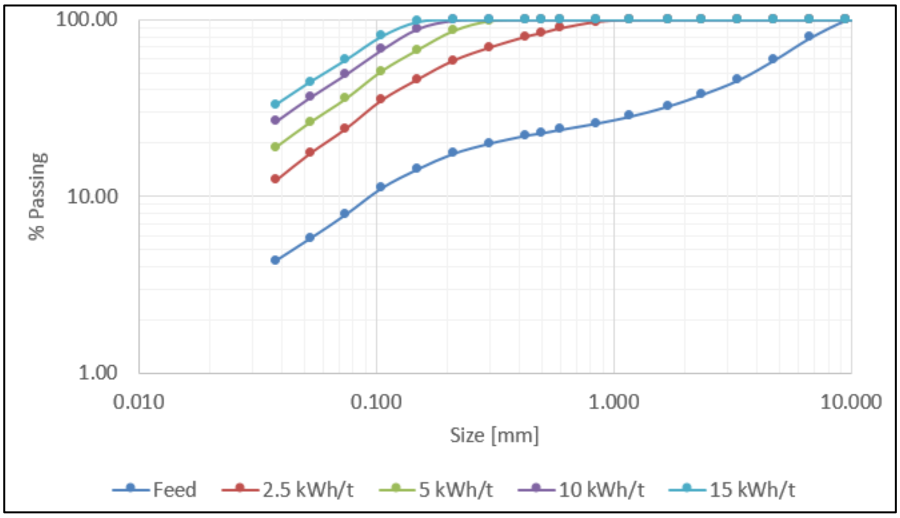


Figure 11-7: Laboratory Batch Rod Mill Testwork Results

The comminution bench scale results are summarised in Table 11-17 below.

Table 11-17: Summary of Bench Scale Comminution Testwork

	Testwork	Results
1	Uniaxial Compression Strength	172 MPa
2	Bond Crushability Work Index	16.3 kWh/t
3	Bond Abrasion Index	0.68
4	Bond Rod Work Index	10.90 kWh/t at 550 µm closing screen 10.92 kWh/t at 1,180 µm closing screen
5	Bond Ball Work Index	8.91 kWh/t at 212 µm closing screen
6	JK A x b parameter t_a	35 0.25

The results generated during this PFS were used to design the comminution circuit. Testwork recommendations for the FS were made in terms of ore hardness variability testwork and material flow properties testwork.

11.3.4 Comminution Process Route Determination

A comprehensive comminution testwork programme was conducted in order to determine the most cost-effective comminution process route to be used for the Project in order to extract the garnet and rutile.

The main comminution circuit involves crushing and milling the ore to <550 µm before the beneficiation process to extract garnet and rutile. The main characteristic of this Project is to maximise the production of garnet and rutile within specifications of grade and size distribution while minimising the fines (<45 µm) generation. This requirement makes the task of selecting the most cost-effective technology and circuit difficult.

The selection of a comminution circuit and technology is an economical choice influenced by factors such as: plant capacity, ore characteristics and product size. Secondary factors to the selection of a comminution circuit and technology such as the LoM, the geology/mining method, the process requirement and project specifics (client preference, commonality of equipment, lead time of major equipment, financial resources, risk profile of project, experience of work force, logistical equipment transport, perceived potential for expansion) also have a strong influence on the choice of a comminution circuit.

The following technologies and circuits were considered:

- Base case circuit: this is a three-stage crushing circuit using a jaw crusher – cone crusher – cone crusher followed by a rod mill in closed circuit with a 550 µm screen. The base case flowsheet is represented in Figure 11-8 below.

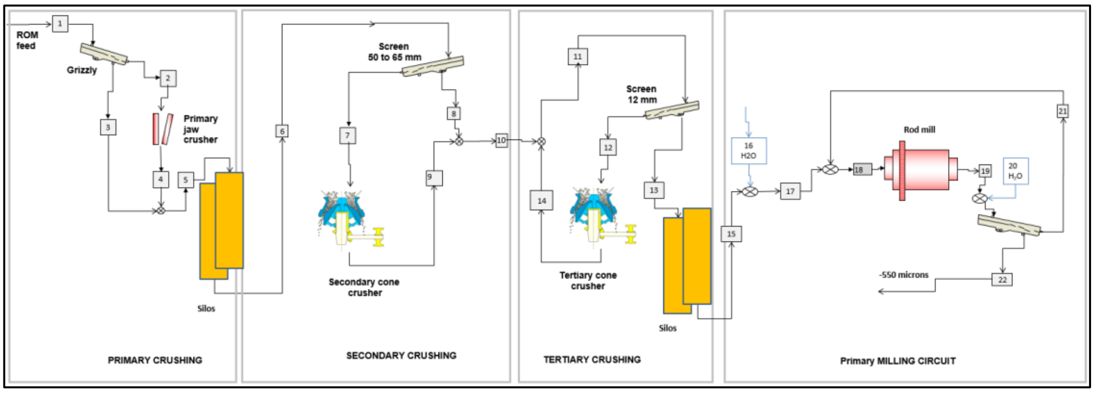


Figure 11-8: Base Case Flowsheet

- HPGR circuit: this circuit is similar to the base case but includes an HPGR in the tertiary crushing stage instead of a cone crusher as represented in Figure 11-9.

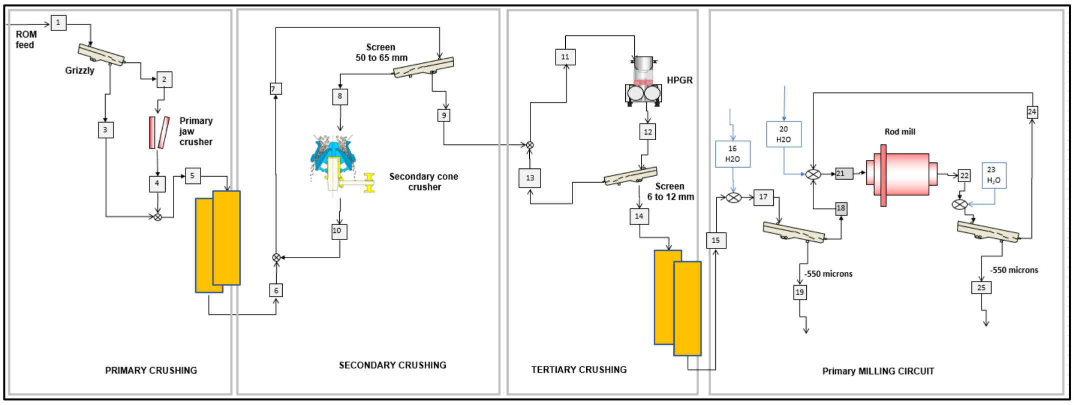


Figure 11-9: HPGR Flowsheet

- Hazemag impact crusher circuit: the Hazemag impact crusher can be used in secondary, tertiary and quaternary crushing stages. The results obtained in tertiary crushing application were satisfactory and the flowsheet is represented in Figure 11-12.

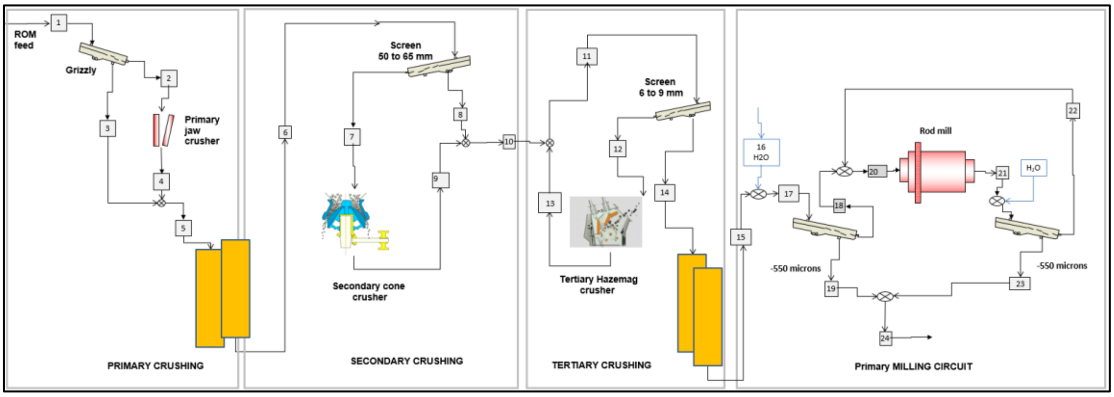


Figure 11-10: Hazemag Flowsheet

- Barmac/ VSI: the VSI is similar to the Hazemag in terms of mode of crushing. The flowsheet investigated is represented in Figure 11-11.

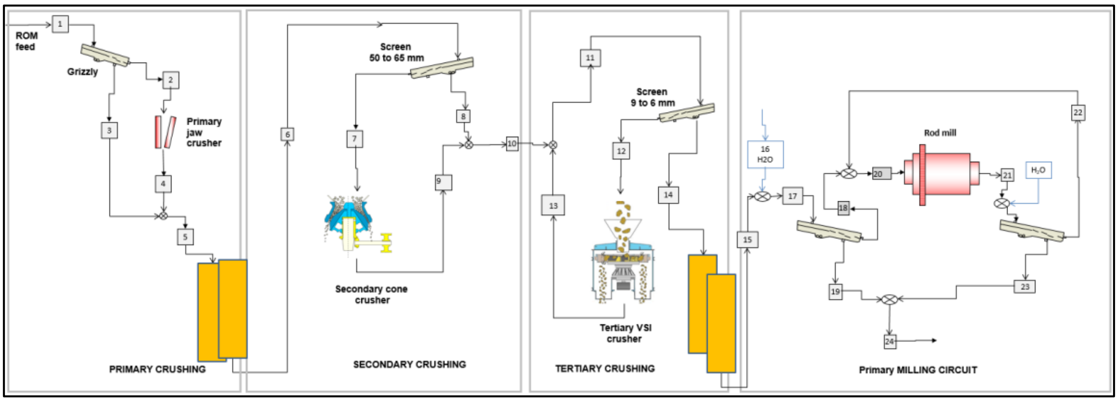


Figure 11-11: Vertical Shaft Impactor Flowsheet

Different technologies and circuit configurations were investigated during this PFS in order to determine the most cost-effective comminution process route. Since different technologies for the same target grind will achieve different liberation of garnet and rutile and hence different recoveries, Nordic Mining has developed and implemented a testwork programme with quantification of liberation using QEMSCAN results. Based on the liberation of garnet and rutile data provided, it was possible to determine a theoretical recovery of garnet and rutile that could be achieved. The recovery was used to determine the NPV for each case and the NPV was used as a criterion to select the most cost-effective comminution circuit.

The results obtained by using a plant capacity of 2.0 Mtpa, a 20-year LoM and an 8% discount rate indicate that the circuit using the impact crusher technology in the tertiary crushing application is the most cost-effective comminution process route for achieving a better NPV. Impact crushing gave a superior liberation of mineral particles and less fines production compared to other methods and therefore a better theoretical recovery for both rutile and garnet.

The most cost-effective comminution circuit is presented in Figure 11-12 below. The crushing circuit of the most cost-effective circuit includes a primary jaw crusher producing a product of <150 mm, a secondary cone crusher producing <50 mm and a tertiary impact crusher operating in closed circuit with a screen to produce <6 mm. In comparison to other technologies tested during this phase of work (cone crusher, HPGR, VSI), the Hazemag technology in tertiary crushing application has offered a better liberation of garnet and rutile, maximising its yield/recovery and grade while generating fewer fines. The wear rate of the impact crusher blow bars is higher requiring a rotation/replacement every three days. All technologies selected in the crushing circuit are mature technologies with many suppliers offering them across the world. The impact crusher has gained popularity in the last two decades with manufacturer such as Metso, Weir Minerals, Hazemag offering this technology. The primary milling circuit is conducted in a rod mill.

The rod mill was selected as opposed to the ball mill in order (i) to minimise fines (<45 µm) generation resulting in a loss of rutile and garnet, (ii) to improve the yield of coarser garnet. The rod mill is also a mature technology offered by many suppliers.

It is recommended that further testwork be conducted during the DFS phase to improve the accuracy of the study, improve the profitability of the Project and obtained process guarantee. The additional testwork includes more Hazemag wear testwork to reflect the circuit selected with pre-screening of fines to reduce the wear rate, variability testwork to understand better the deposit and design the circuit for the LoM and pilot testwork with a sample representing the first five years of operation.

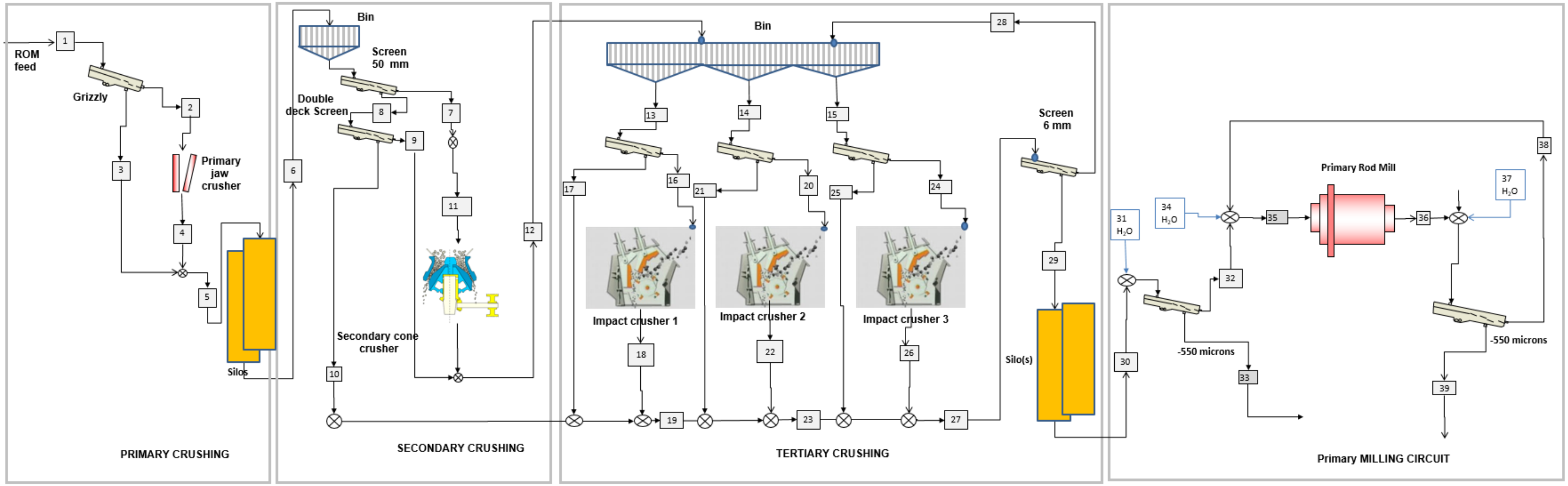


Figure 11-12: Selected Comminution Circuit

11.4 Bulk Process Testwork and Flowsheet Definition

Following a review of all the previous testwork undertaken by others, a scope of work was drafted to provide to suitable testwork laboratories in order to develop time and cost estimates to undertake the testwork. The scope of work provided insights to previous work undertaken including technology and technology sequence and endeavoured to relay the results to provide insight into successes and failures from previous testwork. It was acknowledged that process testwork was done previously and it was deemed important to draw from the results of this previous work to ensure that testwork undertaken as part of this PFS did not waste valuable resources exploring previous, similar testwork concepts.

Both Mineral Technologies and IHC Robbins were invited to participate in the testwork programmes. Since Mineral Technologies was unable to commit to the project time lines, IHC Robbins was selected as the primary laboratory to undertake the flowsheet development testwork. Both laboratories are considered competent and both are frequently involved in typical heavy mineral sands developments. In the Project team's opinion, both are equally competent to undertake this kind of testwork.

Table 11-18 below provides a summary of the bulk testwork programmes and the associated samples prepared and shipped to IHC Robbins in Australia for process definition testwork. The programmes in Table 11-18 below reflect the primary testwork programmes which served as the main focus of the process development. However, because suboptimal grades were obtained supplementary programmes were initiated to achieve grades and enhance recoveries and these are summarised in Table 11-19 below.

As shown in Table 11-18 below, four main samples were subject to process test work at IHC Robbins. The 1231 sample is a high-grade ore sample (ferro-eclogite) that was milled to a fine grain size (212 µm) with the intention to specifically liberate and recover rutile. The 1245 sample was the same high-grade material as 1231, but with a coarser grind to recover coarse garnet, and with regrinding to recover rutile. The 1308 and 1234 samples are both optimally comminuted samples using impact crushing and rod milling which proved to give the best liberation. The samples were processed to recover both coarse garnet and rutile through the enhanced process flowsheet developed through the 1231 and 1245 programmes. Figure 11-4 above shows the selected flowsheet as developed through extensive testing.

A summary of the performance of the bulk testwork programmes is included in Table 11-20 below, and Figure 11-13 and Figure 11-14 provide a graphical comparison of the different programmes. As can be seen in the table and the figures, the high-grade 1308 sample shows a significant increase in recovery and grade, and yield and grade for rutile and garnet products respectively. In addition, Figure 11-13 and Figure 11-14 show how each successive ferro ore programme improved upon the previous programme. Note that Figure 11-14 only includes the mass yield to the coarse garnet product and the overall mass yield to garnet (including fine garnet) is greater than that depicted in the figure. The process was successful in producing a pigment grade rutile product and garnet products meeting both coarse and fine market product requirements. The 1234 sample showed an inferior performance as can be expected when processing low grade ore. It should be noted that all the testwork reports from IHC Robbins contain QEMSCAN and QXRD results from Bureau Veritas Minerals (BVM) Australia, while final garnet products presented in this report are based on results from SGS Canada. A decision was made to use SGS Canada to align with the analytical procedures used by a prominent US garnet producer.

Table 11-18: Testwork Programmes for Process Definition and Metallurgical Performance

Bulk Testwork Programmes			
Programme/ Sample	**Sample Type**	**Grain Size**	**Objective**
1231	High grade. Fine grained ferro-eclogite sample	100% - 212 µm	To develop a flowsheet for production of rutile and garnet final products.
1245	High grade. Coarse grained ferro-eclogite sample	100% - 500 µm	To validate the flowsheets developed during programme 1231 and to define a flowsheet for production of a coarse garnet final product.
1308	High grade. Coarse grained ferro-eclogite sample, optimally comminuted	100% - 550 µm	To optimise metallurgical performance and validate the flowsheet developed in 1231, 1245 and the supplementary programmes using an optimally comminuted, high-grade sample.
1234	Low grade. Coarse grained trans-eclogite sample, optimally comminuted	100% - 550 µm	To determine the performance of the flowsheet developed in the previous bulk and supplementary testwork programmes, including comminution and beneficiation, on a typical lower grade Trans ore sample.

Table 11-19: Supplementary Testwork Programmes to Optimise Metallurgical Performance

Supplementary Testwork Programmes		
Programme	**Description**	**Objective**
1286-001	Coarse and fine garnet magnetic and size fractionation (T22 mag and T304 O/S)	To investigate magnetic and size fractionation of the fine and coarse garnet products in an effort to explore the upgrading potential.
1286-002	Rutile upgrade post pyrite flotation (using programme 1293 results)	To upgrade the post-pyrite flotation product and ensure that a rutile product of sufficient TiO_2 grade is achievable.
1286-003	High density attritioning of coarse garnet	To determine whether additional or improved garnet liberation could be obtained by high density attritioning.
1286-004	Optimally comminuted 1308 bulk sample scouting tests	To formulate a process flowsheet and conduct detailed scouting tests to ensure that a coarse garnet product that meets the grade specification can be generated.
1286-005	Optimally comminuted 1308 bulk sample scouting tests inclusive of a wet gravity circuit	To determine whether the inclusion of a wet gravity circuit upstream of the dry magnetic circuit enhanced garnet grades and yields
1293	Rutile upgrade optimisation (including IHC Robbins and Mineral Technologies programmes)	To validate independently the performance of the rutile upgrading circuit and to investigate the potential benefits of alternative operating conditions
1280	Rutile recovery from 1245 fines, 100% passing 45 µm	To determine whether there is potential to recover rutile from the <45 µm fraction, and produce a fine rutile product
JKTech (17059)	Reverse flotation to reject pyrite from rutile concentrate	To determine the potential of pyrite rejection and rutile upgrading through reverse flotation
Core Metallurgy (1090A)	Forward flotation and reverse flotation (including pyrite and silica flotation) of rutile concentrate	To determine the potential of pyrite rejection and rutile upgrading through forward and reverse flotation

Table 11-20: Summary of the Performance of Each Bulk Testwork Programme

Parameter		**Programme**			
	UoM	**1231***	**1245***	**1308**	**1234**
Garnet grade, coarse	%	-	86.0	95.4	94.1
Garnet grade, fine	%	93.1	92.6	95.5	93.3
TiO_2 grade in produced rutile	%	91.5	93.7	94.9	92.8
TiO_2 overall recovery	%	49.3	52.7	60.5	42.1
Garnet yield coarse product	%	-	6.6	11.4	6.0
Garnet yield fine product**	%	11.6	17.1	7.8	12.7

*The results from the supplementary programmes are included in determining the overall values
**Actual yield (i.e. not factored by the 62.5:37.5 coarse: fine garnet product specification)

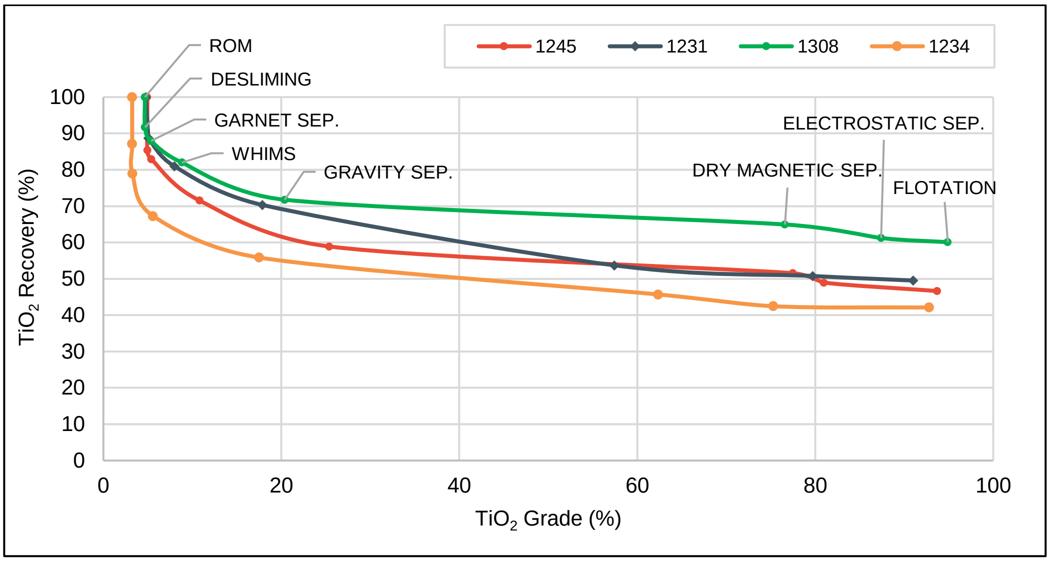


Figure 11-13: Grade-Recovery Curves for TiO_2 for the Different Testwork Programmes

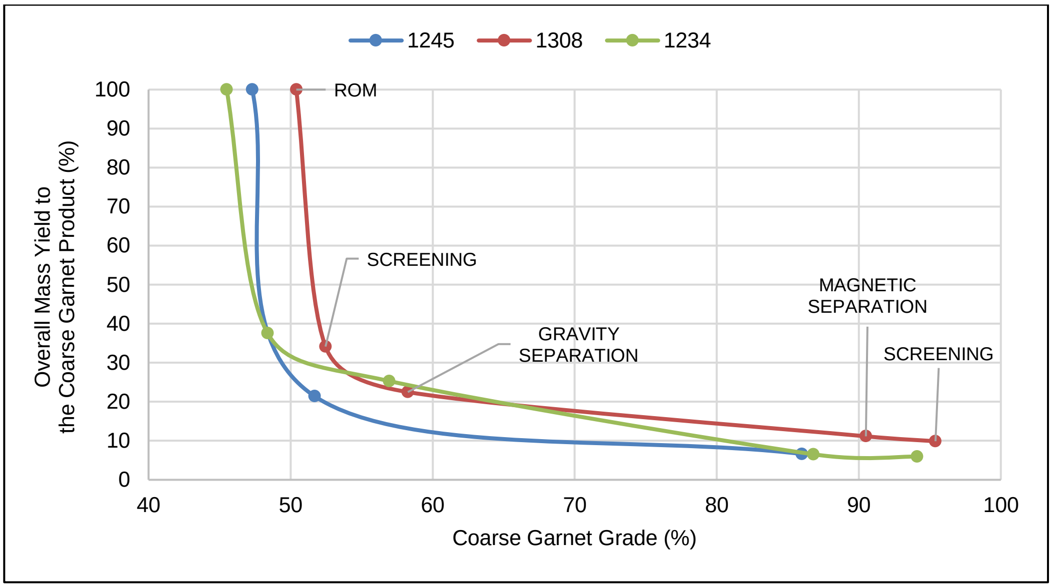


Figure 11-14: Grade-Yield Curves for Coarse Garnet for the Different Testwork Programmes

11.4.1 Programme 1231 – Fine Ferro Bulk Sample to Develop Process Flowsheets for Rutile and Garnet Final Products

Prior to programme 1231, all process knowledge held by the processing team was in the form of previous testwork results. Therefore, the 1231 programme was the first bulk sample programme undertaken and the objective of the programme was to define the process flow using standard heavy mineral sands process equipment and to produce rutile and garnet products that met market specifications.

A summary of the leading aspects of the testwork programme follows.

The Project scope of work called for a “clean sheet” approach to the testwork programme and the team reviewed most of the previous testwork conducted. Following the characterisation of a representative sample of the bulk sample, testwork got underway with a high-level comparison of gravity vs. magnetic separation as the first separation technology. Three representative sub-samples derived from the feed preparation process area sand fraction were used to compare the metallurgical performance of a spiral separator process, a three stage Wet High Intensity Magnetic Separator (WHIMS) process prior to gravimetric separation and a two stage WHIMS process. Metallurgical sighter test work confirmed that the three stage WHIMS process provided the most optimum metallurgical performance and consequently it was selected for the bulk processing of the sand fraction. This bulk processing delivered two products, a magnetic concentrate containing most of the contained fine garnet and a non-magnetic concentrate containing most of the contained rutile.

Sighter tests completed on the non-magnetic fraction comparing the performance of an Up-current Classifier (UCC)/spiral separator process to that of an all spiral separator process indicated no metallurgical advantage associated with the UCC/spiral separator process. Consequently, the non-magnetic concentrate upgrade process was developed using spiral separators. The primary process non-magnetic concentrate material was processed on a stage-by-stage basis through a five-stage spiral separator process.

Sighter tests conducted on the magnetic concentrate from the WHIMS circuit to develop the fine garnet circuit to produce a garnet product at >106 µm, evaluated a UCC and spiral separator circuit. Data confirmed that a UCC operating at a 60:40 (underflow: overflow) split followed by a spiral separator to remove residual pyroxene would produce a concentrate containing >80% garnet in the >106 µm size fraction. A bulk sub-sample derived from the primary process magnetic concentrate was processed on a stage by stage basis through a UCC and spiral separator operating at conditions as determined by the sighter tests, with resultant spiral separator concentrate upgraded into a garnet product by processing over two stages of rare earth drum magnetic separators and final screening at 106 µm. The mineralogy of the garnet product is provided in Table 11-21 and the garnet grade of 93.1% achieved exceeds the specification of 92%.

Finally, the concentrate from the gravity circuit that upgraded the non-magnetic product from the WHIMS circuit was used to develop a dry physical separation circuit to produce a final rutile product. This circuit would typically employ dry magnetic and electrostatic separation equipment to recover the rutile into a (non-magnetic and conductive) final product. Sighter tests were undertaken to establish the sequence of the magnetic and electrostatic separation technologies, results showed that superior upgrading of TiO_2 and TiO_2 distribution was achieved with a magnetic separation circuit ahead of the electrostatic separation equipment. However, the dry physical separation step did not achieve a high recovery of rutile and did not achieve product specifications as shown in Table 11-33 below. Flowsheet development testwork was undertaken on the spiral concentrate but this initial testwork programme failed to produce a final rutile product that met typical market chemical specifications. Effectively the “rutile” product still contained elevated levels of SiO_2 and mineralogical analysis confirmed the SiO_2 levels were contributed by the presence of amphiboles and pyroxenes. The concentrate also contained high levels of sulphur and the sulphur was confirmed to be contributed by the presence of pyrite (FeS_2). Recovery of TiO_2 to the concentrate of this circuit was suboptimal and as a result it was decided to undertake rutile flotation testwork as an alternative option to typical dry physical separation. The results of this work are summarised in section 11.4.5.6. However, results achieved were well below that achieved in the dry physical separation circuit and a decision was taken not to pursue this option further. It was further decided to provide a parallel sample to a competing laboratory (namely Mineral Technologies) to validate the metallurgical performance of the dry physical separation circuit. Results from Mineral Technologies confirmed that improved metallurgical performance was possible with improved focus on individual equipment testwork conditions (see section 11.4.5.5.1). These testwork conditions included feed rate, roll speeds, feed temperature, roll diameter (rare earth roll magnet) and voltage (electrostatic plate separator). IHC Robbins was also afforded the opportunity to repeat the dry circuit testwork now that detailed mineralogical analysis was available. The results of this work were undertaken as a separate proposal and are reported in section 11.4.5.5.2. Essentially, a significant improvement in the TiO_2 recovery of 16.3% was achieved through rigorous process equipment feed condition testing and will result in an increase of equipment quantities.

Table 11-21: Garnet Product Quality of Programme 1231 Determined by QEMSCAN (SGS Canada)

Mineral	Grade Obtained (%)
Garnet	93.1
Rutile	0.3
Quartz	0.4
Leucoxene	0.0
Ilmenite	0.0
Ti intergrowths	0.1
Pyrite	0.0
Pyroxene/Amphibole	4.6
Others	1.5

Table 11-22: Rutile Product Chemical Assay of Programme 1231 Determined by XRF

Major Oxide	Specification (%)	Adjusted Grades for FeS_2 (%)
TiO_2	>94.0	82.78
Fe_2O_3	<1.0	2.02
SiO_2	<2.5	3.56
Al_2O_3	<1.5	0.66
Cr_2O_3	-	0.01
MgO	<1.0	0.52
MnO	<1.0	0.02
ZrO_2	<1.0	0.03
P_2O_5*	<0.03	0.01
U (ppm)	-	<10
Th (ppm)	-	12
V_2O_5	<0.65	0.34
Nb_2O_5	<0.25-0.5	n/d
CaO**	≤0.8/0.15	0.87
K_2O	-	0.03
CeO_2	-	n/d
S*	<0.03	4.64
FeS_2	-	8.69
SnO_2***	<0.05	<0.02

*Welding rod specification for P and S
**Non-sieve plate and sieve plate specification
***SnO_2 detection limit at 0.02%. SnO_2 limit applicable to the molten salt market.

From the chemical analysis by XRF shown in Table 11-22, it can be seen that the rutile product still contained significant amounts of impurities mainly in the form of iron- and silicon oxides and pyrite (as per the adjusted data). The iron and silicon impurities can be attributed to the presence of amphibole and pyroxene minerals.

It is important that pyrite be removed from the product to further upgrade TiO_2 and to reduce the sulphur level to a value below the market specified maximum. To address this issue, reverse flotation was employed which was shown to remove pyrite effectively with trivial losses of TiO_2 to the flotation froth (Section 11.4.5.6). The optimisation work carried out in programme 1293 referred to above, focused on improving rutile recoveries and as such did not focus on achieving the required rutile grade. The resultant programme 1286-002 was initiated to upgrade the dry circuit rutile product where a TiO_2 grade of 94.9% was achieved, although at a recovery penalty. This demonstrated that a market-grade product was achievable. Details of programme 1286-002 are captured in section 11.4.5.2. It should be noted that the additional magnetic fractionation stage employed in

1286-002 will not form part of the final flowsheet and was merely used to determine and confirm final TiO_2 grade.

Table 11-23 summarises the overall mineral recoveries achieved in programme 1231:

Table 11-23: Overall TiO_2 and Fine Garnet Recovery for Programme 1231

Processing Area	TiO_2	Fine Garnet
	%	%
Feed Preparation Process (Desliming and screening)		
Screen, de-slimed sand	95.2	96.4
Primary concentration Process (WHIMS)		
Magnetic concentrate		65.6
Non-Magnetic concentrate	85.0	
Non-Magnetic/Rutile Concentrate Upgrade Process (gravity concentration)		
Concentrate	86.8	
Magnetic/Fine Garnet Concentrate Upgrade Process (gravity concentration and magnetic separation)		
Fine garnet product		38.5
Rutile Upgrade Process (magnetic and electrostatic separation)		
Rutile product	70.6	
Overall		
Rutile product	49.6	
Fine garnet product (+106 µm)		24.4

11.4.2 Programme 1245 – Coarse Ferro Bulk Sample to Validate Flowsheet for Rutile and Fine Garnet Final Products and Develop the Process Flow for Coarse Garnet Final Product

Based on the outcome from the first testwork programme (1231), IHC Robbins provided a detailed proposal to undertake testwork on a second bulk sample to validate the flowsheets developed during the 1231 testwork programme and to define a flowsheet for a coarse garnet final product. A significant amount was learned during the first testwork programme through both process testwork outcomes as well as detailed mineralogical analysis of intermediate and final products. These learnings were incorporated into the scope of this testwork programme. The 1245 sample was from the same source material as the 1231 sample, but with a coarser grind at <500 µm so as not to overgrind retrievable coarse garnet in the ore.

A summary of the leading aspects of the testwork programme follows.

The primary objective of this testwork programme was the development of a process flow suitable for production of a coarse garnet product. The 1245 sample was split in two parts, a +250 µm fraction for recovery of coarse garnet and a -250 µm fraction to recover rutile and fine garnet. After removal of garnet from the +250 µm material, the rejects were reground to -250 µm and added to the initial -250 µm material. Following sample characterisation, three representative samples were retrieved from the +250 µm bulk, de-slimed sample and used in scouting tests to explore different circuit configurations. The circuits evaluated to recover coarse garnet were:

- A three-stage WHIMS circuit followed by a wet gravity circuit and cleaning of the wet gravity circuit concentrate over a three-stage dry magnetic circuit as shown in Figure 11-15 below
- A three-stage dry magnetic circuit as shown in Figure 11-16 below
- A two-stage WHIMS circuit followed by a three-stage dry magnetic circuit as shown in Figure 11-17 below.

Metallurgical scouting test work confirmed the dry magnetic separation process using Rare Earth Drum (RED) magnetic separators and Rare Earth Roll (RER) magnetic separators as shown in Figure 11-16 to provide the most optimum metallurgical performance and consequently, was selected for bulk processing of the +250 µm fraction. This circuit was significantly less complicated than the two alternative circuits. Additionally, the more complex circuits had lower overall mass yields to the coarse garnet product, resulting in RoM tonnages that exceed Nordic Mining's mining license tonnage if aiming at meeting the target of producing 300 ktpa garnet.

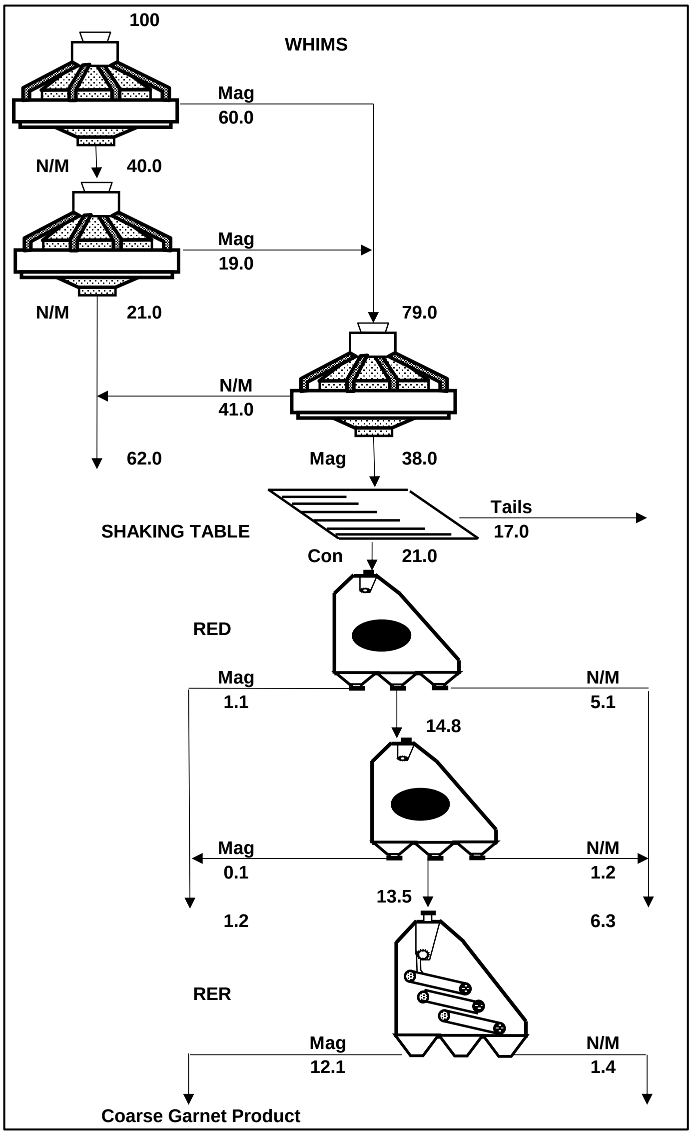


Figure 11-15: An Alternative Coarse Garnet Flowsheet Including Wet Magnetic and Gravity Separation. Mass Distribution Shown in Percentage

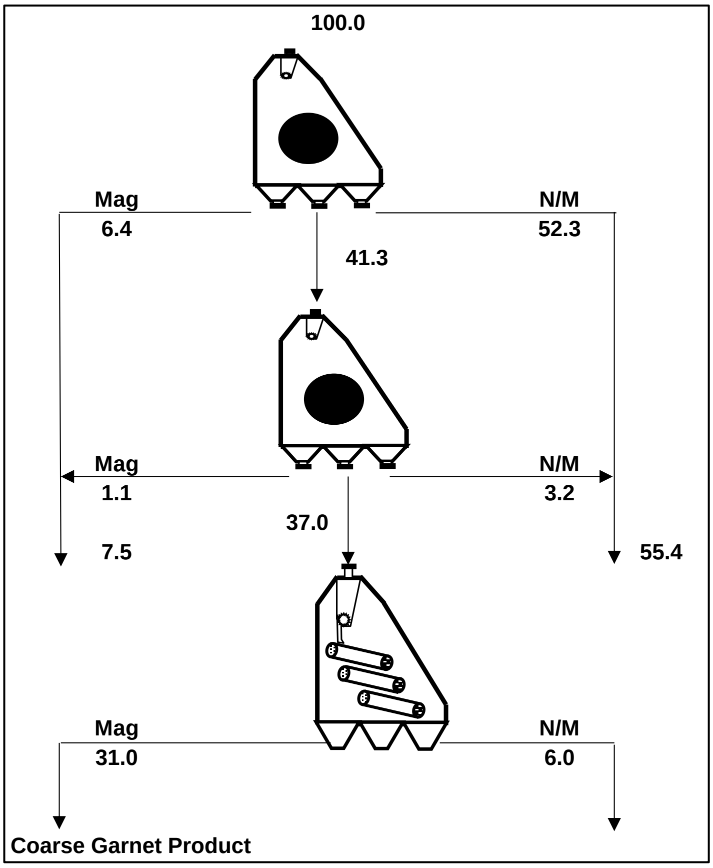


Figure 11-16: Coarse Garnet Flowsheet Selected as the Optimal for Further Test Work. Mass Distribution Shown in Percentage

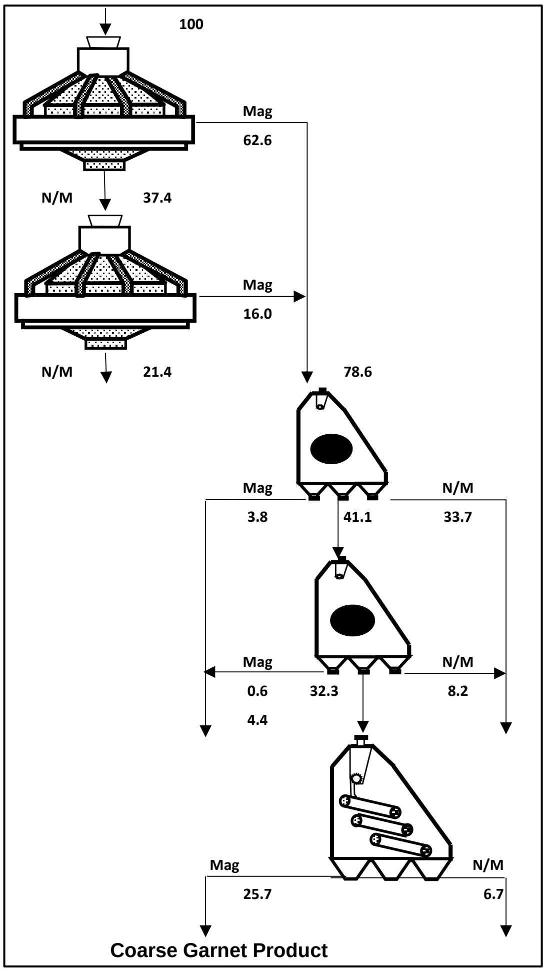


Figure 11-17: An Alternative Coarse Garnet Flowsheet Including Wet Magnetic Separation. Mass Distribution Shown in Percentage

The rejects from the coarse garnet circuit containing an appreciable amount of TiO_2 were milled to 100% passing 250 µm and mixed in the right proportion with the <250 µm of the feed sample. This sample was processed through the previously developed processes. Results obtained were similar to the results achieved during the 1231 testwork programme, but with improved performance of the dry magnetic circuit and a slightly poorer performance of the WHIMS stage.

Final garnet and rutile product qualities are provided in Table 11-24 and Table 11-25 respectively, and overall TiO_2 and garnet recoveries are provided below in Table 11-26 and Table 11-27, respectively.

Table 11-24: Garnet Product Quality Determined by QEMSCAN (SGS Canada) of Programme 1245

Mineral	Fine Garnet Grade Obtained (%)	Coarse Garnet Grade Obtained (%)
Garnet	92.6	86.0
Rutile	0.3	0.7
Leucoxene	0.0	0.0
Ilmenite	0.0	0.0
Ti intergrowths	0.1	0.2
Pyrite	0.1	0.1
Pyroxene/Amphibole	4.8	9.6
Quartz	0.5	1.0
Others	1.9	3.1

Table 11-25: Rutile Product Chemical Assay by XRF of Programme 1245

Major Oxide	Specification (%)	Adjusted Grade for FeS_2 (%)
TiO_2	>94.0	82.30
Fe_2O_3	<1.0	1.72
SiO_2	<2.5	1.68
Al_2O_3	<1.5	0.37
Cr_2O_3	-	n/d
MgO	<1.0	0.14
MnO	<1.0	0.01
ZrO_2	<1.0	0.04
P_2O_5*	<0.03	0.02
U (ppm)	-	<10
Th (ppm)	-	<10
V_2O_5	<0.65	0.34
Nb_2O_5	<0.25-0.5	n/d
CaO**	≤0.8/0.15	0.38
K_2O	-	0.01
CeO_2	-	n/d
S*	<0.03	7.13
FeS_2	-	13.34
SnO_2***	<0.05	<0.02

*Welding rod specification for P and S
**Non-sieve plate and sieve plate specification
***SnO_2 detection limit at 0.02%. SnO_2 limit applicable to the molten salt market.

Table 11-26: Overall TiO_2 Recovery of Programme 1245

Processing Area	TiO_2 %
Feed Preparation Process (desliming and screening)	
Screen, de-slimed sand (<250 µm)	73.3
Screened oversize (>250 µm)	12.1
Coarse Garnet Process (magnetic separation)	
Coarse garnet rejects	89.7
Primary Concentration Process (WHIMS)	
Non-magnetic concentrate	86.2
Non-Magnetic/Rutile Concentrate Upgrade Process (gravity concentration)	
Concentrate	82.3
Rutile Upgrade Process (magnetic and electrostatic separation)	
Rutile product	79.5
Overall	
Rutile product	47.5

Table 11-27: Overall Coarse and Fine Garnet Recovery of Programme 1245

Processing Area	Recovery %
Feed Preparation Process (desliming and screening)	
Screen, de-slimed sand (<250 µm)	59.1
Screened oversize (>250 µm)	17.8
Coarse Garnet Process (magnetic separation)	
Coarse garnet product	51.5
Coarse garnet rejects	48.5
Primary Concentration Process (WHIMS)	
Magnetic concentrate	80.0
Magnetic Concentrate Upgrade Process	
Fine garnet product	62.3
Overall	
Coarse garnet product	9.2
Fine garnet product	33.7

As in programme 1231, the rutile concentrate generated from the dry rutile separation circuit still contained significant levels of impurities especially pyrite. The pyrite grade in the dry rutile circuit concentrate was approximately 13.3% compared to 8.7% in the 1231 programme. Therefore, as in programme 1231, the rutile concentrate was subjected to a reverse flotation process to reject the pyrite. The flotation feed grade of TiO_2 was 82.3% and this increased to 93.7% after flotation with a sulphur concentration of 0.17%, equating to a pyrite concentration of 0.32%. However, the TiO_2 grade achieved was still less than the required grade of 95%.

Consequently, programme 1286-002 was initiated to upgrade the dry circuit rutile product where a TiO_2 grade of 95% was achieved, although at a recovery penalty. This confirmed that a market-grade product was achievable. Details of programme 1286-002 are captured in Section 11.4.5.2. It should be noted that the additional magnetic fractionation stage employed in 1286-002 will not form part of the final flowsheet and was merely used to determine and confirm final TiO_2 grade.

11.4.3 Programme 1308 – Processing of a Coarse Ferro Bulk Sample to Optimise Metallurgical Performance of the Selected Flowsheets for Rutile, Fine Garnet and Coarse Garnet

Based on the outcomes of programmes 1231, 1245 and the supplementary optimisation programmes, a further bulk testwork programme was undertaken to optimise metallurgical performance and validate the flowsheet on an optimally comminuted, high-grade sample. A significant amount was learned during the 1231, 1245 and optimisation test work programmes as well as detailed mineralogical analysis of intermediate and final products. These learnings were incorporated into the scope of this testwork programme. The 1308 sample consisted of a blend of source samples of which a sixth was transitional eclogite material but overall it was considered a high-grade ferro-eclogite sample. The sample was comminuted to 100% passing 550 µm according to the comminution flowsheet in Figure 11-12 above.

A summary of the salient points of programme 1308 is included below.

The first objective of the programme was to develop a robust flowsheet capable of producing a coarse garnet product at the required grade (>92%). To this end, the supplementary testwork programme 1286-004 was initiated. Details of the programme are provided in Section 11.4.5.4. In summary, coarse garnet grade was achieved in programme 1286-004 but it was apparent that the final separation stages (i.e. the rare earth rolls) offered limited upgrading potential which suggested that the process may lack robustness, especially given that a coarse garnet product of the required grade was not achieved in the previous bulk testwork programmes. During the course of programme 1286-004, observations from the mass and mineral balance (Section 11.6.4.1) indicated that a wet gravity circuit provided substantial upgrading potential of garnet through effective rejection of amphibole and pyroxene minerals and it was decided that a two-stage gravity circuit be tested up front of the dry magnetic circuit. This testwork was initiated as programme 1286-005 which ultimately formed the coarse garnet circuit of the bulk 1308 programme. The coarse garnet flowsheet of 1286-005 consisted of a two-stage spiral circuit with rougher and scavenger stages where the scavenger concentrate (i.e. cons) was returned to the rougher stage. The rougher cons were then fed to a dry magnetic circuit comprising two RED stages where the middlings (i.e. mids) from stage 1 were fed to stage 2, and the mids of stage 2 were screened with the oversize serving as the final coarse product. To remain consistent with 1286-004, the flowsheet of programme 1286-005 was tested using the following three samples to account for unknowns in the comminution results:

- Sample A – Hazemag crushed sample
- Sample B – Rod mill crushed sample and
- Sample C – Combined Hazemag and rod mill sample.

The QXRD results from programme 1286-005 are provided below in Table 11-28, Table 11-29 and Table 11-30 for samples A, B and C respectively. Note that QXRD was only available on the screened products of sample C, and therefore the mids of the second RED stage are reported as the product and used as a basis of comparison. As can be seen from the results, a product above 92% garnet was achieved in samples A and B, while screening was required in sample C before grade was achieved. However, QXRD readings for garnet are less accurate than QEMSCAN and tend to underestimate the garnet content. The final coarse garnet product for sample C, according to QEMSCAN, is 95.4% garnet as shown in Table 11-31 below, which is greater than the QXRD value of 92.5%. This product is considered the final coarse garnet product of programme 1308 and the decision was made to define the flowsheet for a combined Hazemag-rod mill feed to the coarse circuit. Images of the coarse garnet product are shown below in Figure 11-18 and Figure 11-19.

Comparing the overall garnet distribution of programme 1286-004 and 1286-005, it can be seen that including the wet gravity circuit enhances garnet recovery while still achieving grade. In addition, the mass yield to the garnet product in 1286-005 (33.3%) was greater than that in 1286-004 (25.5%), further reflecting the improvement achieved through the wet gravity circuit. Gangue removal upfront of the dry circuit has the added advantage of reducing the required capacity of the dryer. The overall mass yield to the coarse garnet product for programme 1308 was 11.4%, when corrected for recoverable garnet not included due to incorrect feed preparation as discussed below.

The flowsheet to produce a fine garnet product remained the same as that of programme 1245 and a fine garnet product of 95.5% was achieved as shown in Table 11-31 below. Images of the fine garnet product are shown below in Figure 11-20 and Figure 11-21.

Overall garnet yields and recoveries are included in Table 11-32 together with the yield input for the financial model. The garnet yield input to the financial model was calculated by applying an adjustment factor of 13.3% to the mass yield to the >212 µm fraction due to incorrect blending of the feed sample during testwork. Neglecting this adjustment negatively biases the amount of coarse material produced in the comminution circuit which underestimates the true amount of coarse garnet available for recovery. The adjustment increases the >212 µm yield to 38.6%. Furthermore, at the time of defining the financial model inputs, a mistaken mass yield of 67.2% was provided for the yield across the coarse garnet gravity circuit which was later corrected to 65.8%. Combining the adjustment factor and the original mass yield across the coarse garnet gravity circuit led to a coarse garnet yield of 11.4%. Applying the factor of 1.6 to account for the specified amount of fine garnet gives an overall garnet yield of 18.3% which was used as an input to the financial model.

Table 11-28: 1286-005 Sample A – Hazemag Comminuted >212 µm Fraction (QXRD)

Test	Garnet Grade (%)	Overall Garnet Distribution (%)
Spiral Tails	27.5	19.0
RED 1 Mags	53.4	1.1
RED 1 Non-mags	15.4	5.2
RED 2 Mags	75.9	0.6
RED 2 Non-mags	46.7	0.7
RED 2 Mids	93.4	73.5

Table 11-29: 1286-005 Sample B – Rod Mill Comminuted >212 µm Fraction (QXRD)

Test	Garnet Grade (%)	Overall Garnet Distribution (%)
Spiral Tails	27.0	28.2
RED 1 Mags	56.2	2.0
RED 1 Non-mags	15.4	6.5
RED 2 Mags	73.4	1.2
RED 2 Non-mags	60.3	1.5
RED 2 Mids	93.4	60.7

Table 11-30: 1286-005 Sample C – Combined Comminuted >212 µm Fraction (QXRD)

Test	Garnet Grade (%)	Overall Garnet Distribution (%)
Spiral Tails	26.1	18.0
RED 1 Mags	54.7	2.4
RED 1 Non-mags	18.5	10.8
RED 2 Mags	76.0	2.0
RED 2 Non-mags	65.7	3.7
RED 2 Mids	90.5	63.2
Screen O/S	92.5	57.3

Table 11-31: Garnet Product Quality Determined by QEMSCAN (SGS Canada) of Programme 1308

Mineral	Fine Garnet Grade Obtained (%)	Coarse Garnet Grade Obtained (%)*
Garnet	95.5	95.4
Rutile	0.45	0.43
Mica	0.08	0.14
Ilmenite	0.07	0.05
Quartz	0.28	0.47
Pyrite	0.02	0.05
Pyroxene/Amphibole	3.07	2.8
Others	0.53	0.66
*Programme 1286-005 Sample C		

Table 11-32: Overall Coarse and Fine Garnet Recoveries and Mass Yields of Programme 1308

Processing Area	Garnet Recovery	Mass Yield
	%	%
Feed Preparation Process (desliming and screening)		
Screened oversize (+212 µm)	38.5	34.1
Screen, de-slimed sand (-212; +45 µm)	55.9	57.3
Fines (-45 µm)	5.6	8.6
Coarse Garnet Process (gravity and magnetic separation)		
Coarse garnet product	50.3	29.0
Coarse garnet rejects	10.5	7.5
Coarse garnet rejects to milling	36.6	57.1
Secondary de-sliming	2.6	6.4
Primary Concentration Process (WHIMS)		
Magnetic concentrate	75.4	38.8
Fine Garnet Process (gravity and magnetic separation)		
Fine garnet product	29.6	25.5
Overall		
Coarse garnet product	19.4	9.9
Fine garnet product	15.4	7.6
Financial model input		
Coarse garnet yield	n/a	11.4
Overall garnet yield	n/a	18.3

Figure 11-18: Coarse Garnet Product from Programme 1308

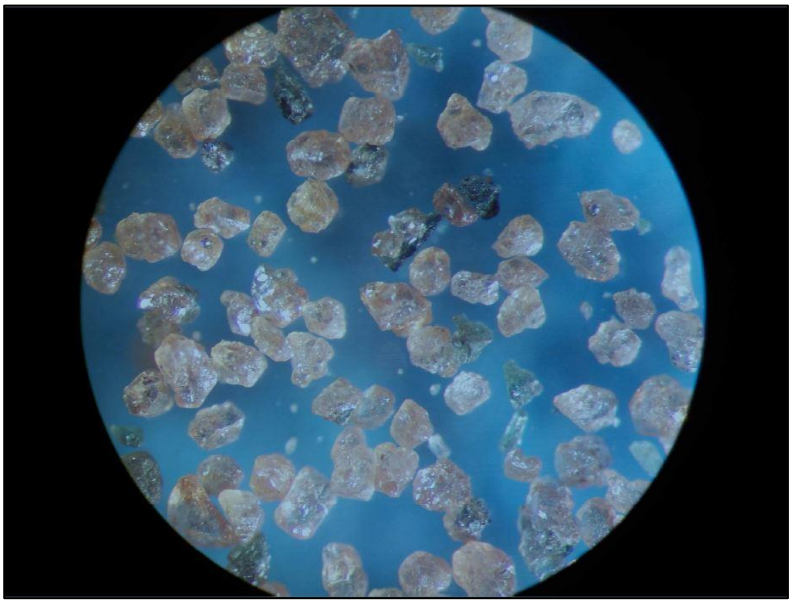


Figure 11-19: Microscope Image of the Coarse Garnet Product from Programme 1308

Figure 11-20: Fine Garnet Product from Programme 1308

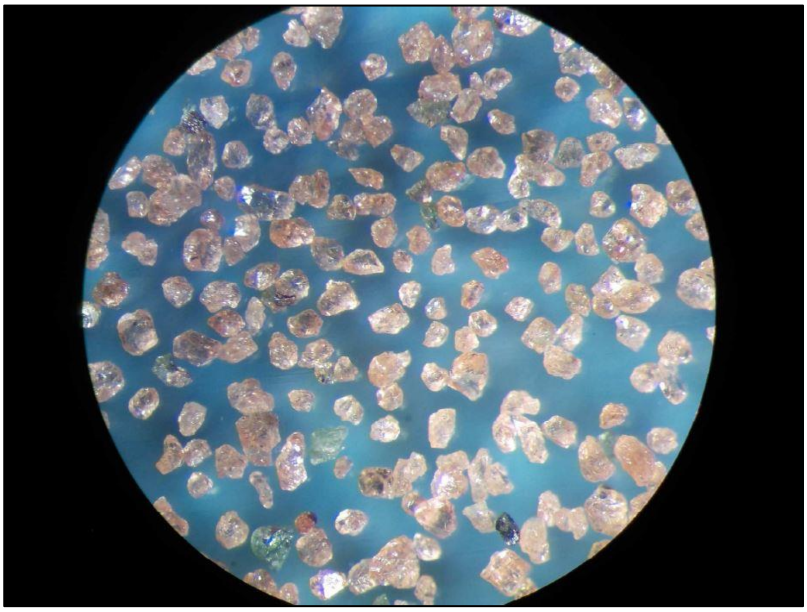


Figure 11-21: Microscope Image of the Fine Garnet Product from Programme 1308

The flowsheet to produce a rutile product combined all previous learnings and it was required that the position of flotation in the circuit be assessed. As such, two rutile concentrates were generated in programme 1308 to determine whether the location of the flotation circuit in the flowsheet has an impact on process performance. It was found that a rutile product of 94.9% TiO_2 was achieved when flotation is placed after the dry circuit and a product of 93.0% TiO_2 was achieved when flotation is placed after the wet spiral stage and before the dry circuit as shown in Table 11-33 below. In addition, the TiO_2 recovery from the spiral concentrate to the final rutile product is superior when flotation is placed after the dry circuit. The composition of the final rutile products for each case is provided in

Table 11-34 below and a breakdown of the TiO_2 recovery across the overall process is provided in Table 11-35 below. For the case where flotation is placed after the dry circuit, an overall TiO_2 recovery of 60.5% was achieved which is substantially greater than the previous bulk programmes. It should be noted that the overall TiO_2 recovery of 60.2% used as the basis for the financial model (and further de-rated to 58.4% for scale-up as shown above in Table 11-1), was calculated using a TiO_2 recovery of 98.1% (instead of 99.2%) for the post-dry circuit flotation stage as this was the only data available (from JKTech) at the time of performing the initial financial analysis. IHC Robbins had not yet issued their final results as contained in the report for programme 1308, where the overall TiO_2 distribution to the pyrite concentrate of 0.7% for the rutile upgrading process translates to a flotation recovery of 99.2% TiO_2. In addition to the different TiO_2 recovery for flotation, a different calculation approach was employed. IHC Robbins determined the overall recovery by multiplying the stage-wise recoveries together, whereas the 60.2% was calculated by using a continuous recovery through the process, and applying recovery values as calculated by feed and product grades, and stage mass splits. The use of a continuous recovery reduces rounding error which may contribute to the difference observed between the 60.2% and the 60.5%. However, the disparity of 0.3 percentage points is minor and the financial model is conservative as a consequence of the difference.

Despite the improved recoveries achieved in programme 1308, the particle size distribution of the final rutile product was finer than expected (for example compared to 1245 as shown in Figure 11-22 below) where ~15% of the material was below 75 µm (see Table 11-36 below). The PSD specification for pigment grade rutile should contain less than 5% <75 µm material. However, the fine fraction of the rutile product could potentially be absorbed by titanium metal producing customers located in Kazakhstan, Ukraine and Russia. According to market specialists, this market could consume between 5,000 tpa and 10,000 tpa, and from preliminary estimates, up to 3,200 tpa of <75 µm material could potentially be generated from this ore. Therefore, the amount of 3,200 tpa could easily be absorbed by this market. To ensure that a coarser product is generated, a detailed testwork programme is being developed for execution during the DFS. In short, the testwork comprises processing three samples generated from the same source rock to explore in detail the feed size distribution that leads to the optimal final rutile product particle size distribution at maximised grade and recovery. Images of the final rutile products are shown in Figure 11-23 and Figure 11-24.

Table 11-33: Comparison of the Location of Flotation in the Flowsheet

Position of Flotation	Final Product TiO_2 Grade (%)	TiO_2 Recovery Across the Rutile Upgrading Process* (%)
Before the dry circuit	93.0	77.1
After the dry circuit	94.9	83.9
*From spiral concentrate to final rutile product		

Table 11-34: Rutile Products Chemical Assay by XRF of Programme 1308

Main Compounds	Specification (%)	Final Rutile Product	
		Post-dry Circuit Flotation Route (%)	Pre-dry Circuit Flotation Route (%)
TiO_2	>94.0	94.90	93.02
Fe_2O_3	<1.0	1.63	1.62
SiO_2	<2.5	1.53	2.41
Al_2O_3	<1.5	0.31	0.56
Cr_2O_3	-	0.01	0.01
MgO	<1.0	0.03	0.27
MnO	<1.0	0.02	0.01
ZrO_2	<1.0	0.06	0.08
P_2O_5*	<0.03	0.01	0.01
U (ppm)	-	<10	<10
Th (ppm)	-	<10	<10
V_2O_5	<0.65	0.41	0.38
Nb_2O_5	<0.25-0.5	n/d	n/d
CaO**	≤0.8/0.15	0.35	0.52
K_2O	-	0.01	0.03
CeO_2	-	n/d	0.02
S*	<0.03	0.17	0.20
FeS_2	-	0.30	0.37
SnO_2***	<0.05	<0.02	<0.02

*Welding rod specification for P and S
**Non-sieve plate and sieve plate specification
***SnO_2 detection limit at 0.02%. SnO_2 limit applicable to the molten salt market.

Table 11-35: Overall TiO_2 Recovery of Programme 1308 (Post-dry Circuit Flotation)

Processing Area	TiO_2 Recovery	
	Cum. Wt%	Stage Wt%
Feed Preparation Process (desliming and screening)		
Deslimed sand	92.2	92.2
Coarse Garnet Process (gravity and magnetic concentration)		
Garnet Processing	89.9	83.5
Secondary de-sliming	88.5	91.8
Primary concentration Process (WHIMS)		
Non-magnetic concentrate	82.4	93.2
Non-Magnetic/Rutile Concentrate Upgrade Process (Gravity concentration)		
Concentrate	72.1	87.5
Rutile Upgrade Process (magnetic and electrostatic separation)		
Rutile product	61.0	84.6
Pyrite Flotation		
Flotation tails	60.5	99.2
Overall		
Rutile product	60.5	

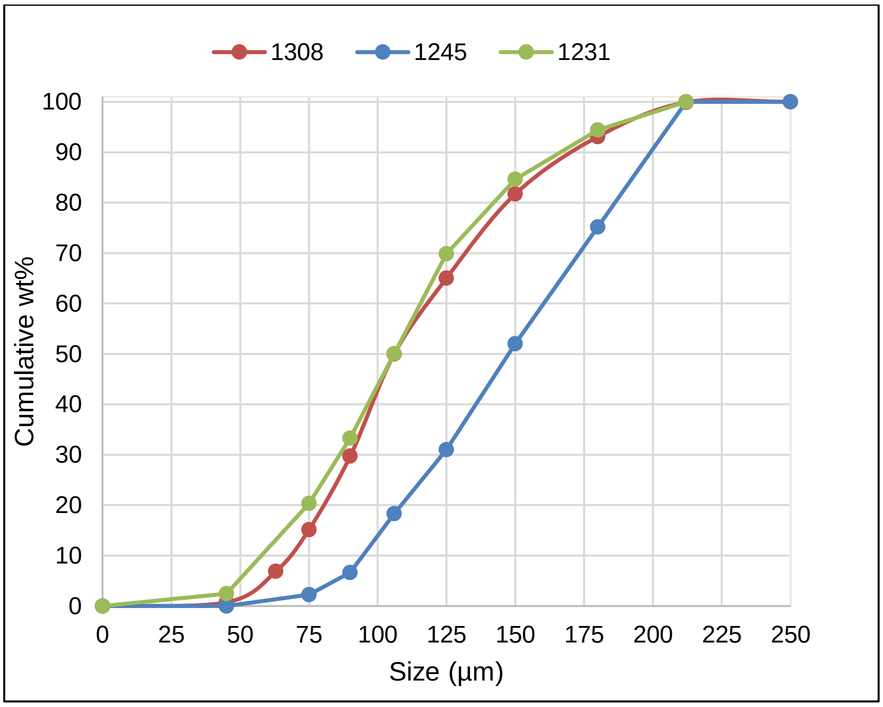


Figure 11-22: Comparison of the Particle Size Distributions of the Rutile Products from Different Testwork Programmes

Table 11-36: Particle Size Distribution of the Rutile Product from 1308

Aperture (µm)	Cum. Wt%
250	100
212	99.9
180	93.1
150	81.7
125	65.0
106	50.0
90	29.7
75	15.1
63	6.9
45	0.7
D_{50} (µm)	**106**
D_{80} (µm)	**147**

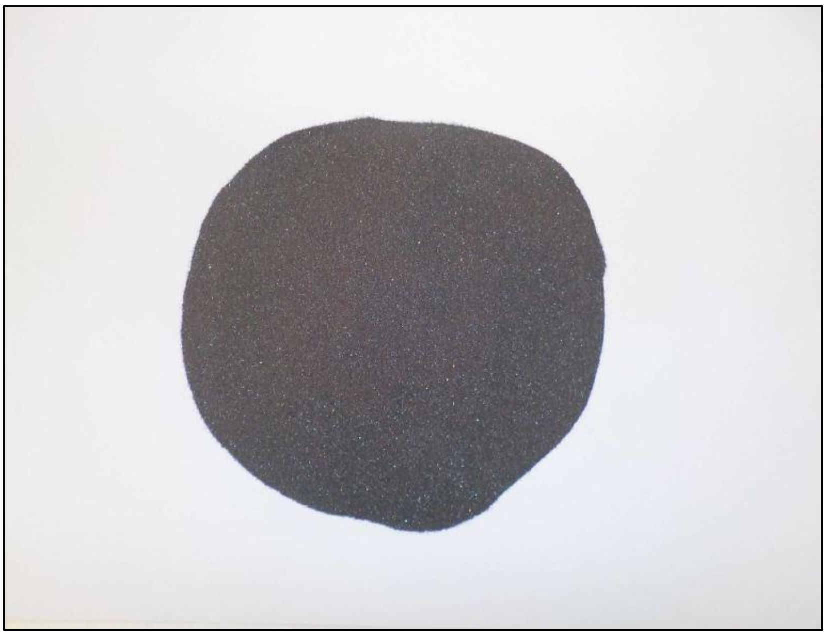


Figure 11-23: Image of the Rutile Product from Programme 1308

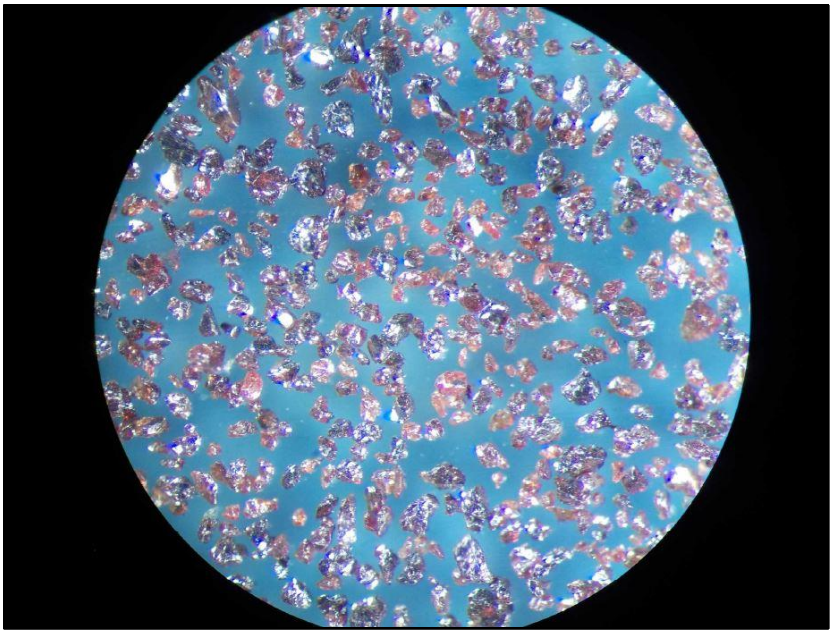


Figure 11-24: Microscope Image of the Rutile Product from Programme 1308

11.4.4 Programme 1234 – Processing of a Coarse Transitional Bulk Sample to Determine Metallurgical Performance of the Selected Flowsheets for Rutile, Fine Garnet and Coarse Garnet

In addition to ferro-eclogite, trans-eclogite is a major rock type in the Engebø deposit and can contribute substantially to garnet and rutile tonnages. However, transitional material has a lower garnet grade (around 47.5%) and a substantially lower TiO_2 grade (between 2% and 3%) compared to ferro-eclogite as shown in Table 11-10. Since trans ore is excluded from the PFS mine plan, understanding the performance of the developed flowsheet when processing trans ore, merely provides insight into process performance and does not contribute to the conclusions of the PFS, but may be instructive in developing options for future studies.

ICH Robbins provided a proposal to undertake detailed testwork on a bulk sample of Trans eclogite (1234). The objective of the programme was to determine the performance of the flowsheet developed in the previous bulk and supplementary testwork programmes, including comminution and beneficiation, on a typical lower grade Trans ore sample.

The results of programme 1234 still need to be finalised by IHC Robbins but the preliminary results are summarised below in Table 11-37, Table 11-38, Table 11-39 and Table 11-40.

So far, an overall mass yield to the coarse garnet product of 6% at a garnet grade of 94.1% has been attained and a rutile product of 92.8% TiO_2 has been achieved at a TiO_2 recovery of 42.1%. The grade is similar to that of 1308 and meets product requirements. However, the mass yield is substantially lower than that of 1308 and given that the theoretical garnet recovery for 1234 (27.1%) is lower than 1308 (39.5%), it suggests that the liberation performance for 1234 was poorer than that of 1308. This is due in part to a smaller grain size compared to the ferro-eclogite and possibly due to the bulk sample (~1 t) being below the optimal size (~3 t) for efficient Hazemag crushing. Note that after repeating testwork on the second RED stage of the coarse garnet circuit, the grade of 94.1% was achieved at the overall yield of 6.0%, however, the yield used in the financial model (6.9%) corresponds to the initial RED testwork which corresponds to a garnet product grade of 85.4% (QXRD).

Table 11-37: Garnet Product Quality Determined by QEMSCAN (SGS Canada) of Programme 1234

Mineral	Fine Garnet Grade Obtained (%)	Coarse Garnet Grade Obtained (%)
Garnet	93.3	94.1
Rutile	0.39	0.52
Mica	0.09	0.13
Ilmenite	0.05	0.10
Quartz	0.20	0.29
Pyrite	0.04	0.07
Pyroxene/Amphibole	5.1	4.07
Others	0.83	0.74

Table 11-38: Rutile Products Chemical Assay by XRF of Programme 1234

Main Compounds	Specification (%)	Final Rutile Product	
		Post-dry Circuit Flotation Route (%)	Pre-dry Circuit Flotation Route (%)
TiO_2	>94.0	92.79	89.90
Fe_2O_3	<1.0	1.89	2.27
SiO_2	<2.5	2.48	4.00
Al_2O_3	<1.5	0.57	0.83
Cr_2O_3	-	n/d	n/d
MgO	<1.0	0.33	0.62
MnO	<1.0	0.01	0.02
ZrO_2	<1.0	0.09	0.11
P_2O_5*	<0.03	n/d	n/d
U (ppm)	-	<10	<10
Th (ppm)	-	<10	<10
V_2O_5	<0.65	0.54	0.56
Nb_2O_5	<0.25-0.5	n/d	n/d
CaO**	≤0.8/0.15	0.60	1.03
K_2O	-	0.01	0.01
CeO_2	-	0.01	0.01
S*	<0.03	0.25	0.23
FeS_2	-	0.47	0.43
SnO_2***	<0.05	not available	not available

*Welding rod specification for P and S
**Non-sieve plate and sieve plate specification
***SnO_2 detection limit at 0.02%. SnO_2 limit applicable to the molten salt market.

Table 11-39: Overall TiO_2 Recovery of Programme 1234

Processing Area	TiO_2 Recovery	
	Cum. Wt%	Stage Wt%
Feed Preparation Process (desliming and screening)		
Deslimed sand	87.1	87.1
Coarse Garnet Process (gravity and magnetic concentration)		
Garnet Processing	81.5	82.2
Secondary de-sliming	79.0	90.5
Primary concentration Process (WHIMS)		
Non-magnetic concentrate	67.3	85.2
Non-Magnetic/Rutile Concentrate Upgrade Process (Gravity concentration)		
Concentrate	55.9	83.0
Rutile Upgrade Process (magnetic and electrostatic separation)		
Rutile product	42.4	76.0
Pyrite Flotation		
Flotation tails (and screening)	42.1	99.2
Overall		
Rutile product	42.1	

Table 11-40: Overall Coarse and Fine Garnet Recoveries and Yields of Programme 1234

Processing Area	Garnet Recovery	Mass Yield	
	%	%	
Feed Preparation Process (desliming and screening)			
Screened oversize (>212 µm fraction)	37.8	37.5	
Screen, de-slimed sand (<212 µm and 45 µm)	54.1	49.4	
Fines (<45 µm) (loss)	8.1	13.1	
Coarse Garnet Process (gravity and magnetic separation)			
Coarse garnet product	34.2	15.9*	18.4
Coarse garnet rejects	18.8	13.5*	10.9
Coarse garnet rejects to milling	47.0	70.7	
Secondary de-sliming**	93.4	90.5	
Primary Concentration Process (WHIMS)			
Magnetic concentrate	75.7	47.8	
Fine Garnet Process (gravity and magnetic separation)			
Fine garnet product	45.3	36.3	
Overall			
Coarse garnet product	12.9	6.0*	6.9
Fine garnet product**	24.2	12.7	

*IHC Robbins repeated the second RED stage in an attempt to increase the garnet grade. The higher mass yields were used as inputs to the financial model because the results of the repeat test were not available at the time. The final product grade corresponds to the lower yield values.
**Garnet recovery data for the secondary de-sliming stage was not available at the time of writing this report. Therefore, the same recovery as for the secondary de-sliming in 1308 is assumed for completeness. The overall garnet recovery to the fine garnet product is also dependent on this assumption.

11.4.5 Supplementary Testwork and Flowsheet Optimisation

The following supplementary testwork programmes were initiated during the course of the bulk sample programmes 1231 and 1245. As the title suggests, this testwork was supplementary in nature and was deemed necessary to achieve the objectives of the respective bulk sample programmes.

11.4.5.1 Programme 1286-001 - Coarse and Fine Garnet Magnetic and Size Fractionation (T22 mag and T304 O/S)

As part of the 1245 bulk sample programme, it was decided to investigate magnetic and size fractionation of the fine and coarse garnet products in an effort to explore the upgrading potential. Initial QXRD results suggested that both fine and coarse garnet products would be below quality specifications with regard to the garnet content. At that time, the QEMSCAN results had not been received and only the QXRD results were available. The garnet specification applicable to this project calls for final garnet products to contain greater than 92% garnet. This exercise therefore looked at splitting the garnet concentrates into several smaller fractions (magnetically and by size) in a bid to isolate impurities into discreet mass fractions. However, from the magnetic and size fractionation results it was clear that impurities were evenly distributed across all size and magnetic fractions rendering these mechanisms ineffective. Results are summarised in Table 11-41, Table 11-42 and Table 11-43 below.

Table 11-41: Magnetic Fractionation of the Coarse Garnet Product

		QXRD			
		Rutile	Amphiboles	Pyroxene	Garnet
	Wt %	%	%	%	%
Mag 1 (7,000 Gauss)	11.0	<1.0	10.0	9.0	75.0
Mag 2 (10,000 Gauss)	45.3	<1.0	7.0	6.0	85.0
Mag 3 (16,000 Gauss)	38.5	<1.0	6.0	6.0	84.0
N/M	5.2	<1.0	7.0	7.0	82.0
Coarse Garnet Product (T 22 Mag)	100.0	<1.0	6.9	6.4	83.4

Table 11-42: Magnetic Fractionation of the Fine Garnet Product

		QXRD			
		Rutile	Amphiboles	Pyroxene	Garnet
	Wt %	%	%	%	%
Mag 1 (7,000 Gauss)	15.4	<1.0	5.0	2.0	91.0
Mag 2 (10,000 Gauss)	63.6	<1.0	4.0	4.0	90.0
Mag 3 (16,000 Gauss)	20.1	<1.0	5.0	9.0	84.0
N/M	0.8	3.0	7.0	28.0	58.0
Fine Garnet Product (T 304 O/S)	100.0	<1.0	4.4	4.9	88.7

Table 11-43: Size Fractionation of the Coarse Garnet Product

		QXRD			
		Rutile	Amphiboles	Pyroxene	Garnet
Aperture (µm)	Wt %	%	%	%	%
425	1.1	<1.0	13.0	13.0	64.0
300	28.2	<1.0	7.0	7.0	82.0
250	32.1	<1.0	8.0	5.0	84.0
212	20.1	<1.0	8.0	6.0	84.0
180	11.6	<1.0	9.0	5.0	83.0
150	3.7	<1.0	7.0	4.0	86.0
125	1.9	<1.0	9.0	3.0	84.0
0	1.5	<1.0	12.0	7.0	77.0
Coarse Garnet Product (T 22 Mag)	100.0	<1.0	7.9	5.8	83.1

Table 11-44: Size Fractionation of the Fine Garnet Product

		QXRD			
		Rutile	Amphiboles	Pyroxene	Garnet
Aperture (µm)	Wt %	%	%	%	%
250	0.7	<1.0	5.0	5.0	87.0
212	11.9	<1.0	2.0	1.0	95.0
180	32.6	<1.0	3.0	1.0	94.0
150	25.5	<1.0	3.0	2.0	92.0
125	16.8	<1.0	2.0	7.0	88.0
106	9.0	<1.0	13.0	12.0	72.0
90	3.2	<1.0	7.0	21.0	69.0
0	0.2	<1.0	11.0	36.0	45.0
Fine Garnet Product (T 304 O/S)	100.0	<1.0	3.8	4.0	89.7

11.4.5.2 Programme 1286-002 - Rutile Upgrade Post Pyrite Flotation

During the processing of bulk samples 1231 and 1245, and treatment of the non-magnetic fraction for rutile flowsheet development, the emphasis was placed on mineral recovery (i.e. rutile as TiO_2) as it was known from previous testwork results that mineral recovery would be challenging. This resulted in the final rutile product grade not being achieved as shown above in Table 11-22 and Table 11-25 due to the presence of residual amphibole and pyroxene minerals. Rutile concentrate from the dry physical separation circuit was therefore subjected to a reverse flotation process for the removal of pyrite. Following pyrite rejection by flotation, the TiO_2 levels were still below (approximately 90%) the target grade of 95% TiO_2. Therefore, the underflow from this flotation process (rutile concentrate) was then subjected to additional electrostatic and magnetic separation processes to confirm that the final rutile product grade is achievable but with a corresponding recovery penalty. The results from the post-pyrite flotation physical separation testwork are shown below in Table 11-45, Table 11-46 and Table 11-47. Results showed that both magnetic and electrostatic separation were successful in achieving the final target TiO_2 grade for the rutile product. During weekly telephonic review sessions, it was decided to select the magnetic separation technology as the final upgrading technology. However, it is important to note that although this final upgrading step was necessary in these testwork rounds, it was acknowledged that in future programmes it would be endeavoured to achieve a grade high enough to avoid having this final magnetic separation stage.

In addition, it should be noted that during the 1231 and 1245 bulk sample programmes, it was neither known nor confirmed that flotation would be required to reject pyrite to reduce the sulphur content of the rutile product to market-acceptable levels. During the course of these testwork programmes, it became evident that flotation would be required but the actual position within the rutile processing train was not confirmed. Testwork was undertaken using both gravity circuit concentrate as well as rutile dry circuit product as feed to the flotation circuit. Following this testwork, a trade-off study (H352410-4000-210-030-0001) was undertaken to support the decision to place the flotation circuit ahead of the dry rutile separation circuit.

Table 11-45: Magnetic Fractionation of the 1231 Bulk Sample Rutile Concentrate

Magnetic Fractionation	Wt %	Assay									
		TiO_2	Fe_2O_3	SiO_2	MgO	MnO	P_2O_5	V_2O_5	Nb_2O_5	CaO	S
		%	%	%	%	%	%	%	%	%	%
T 1 Mag 1	7.8	54.9	6.77	23.1	3.3	0.04	0.00	0.3	0.01	5.8	0.30
T 1 Mag 2	10.6	76.1	4.96	10.8	1.5	0.03	0.01	0.3	0.01	2.7	0.50
T 1 Mag 3	8.2	89.7	3.18	3.4	0.4	0.02	0.01	0.4	0.01	0.8	0.46
T 1 N/M	73.4	95.5	1.56	0.72	0.1	0.00	0.01	0.4	0.01	0.2	0.49
Feed	100	89.8	2.5	3.7	0.5	0.01	0.01	0.4	0.01	0.9	0.48
T 1 N/M	73.4	95.5	1.56	0.72	0.05	0.00	0.01	0.4	0.01	0.2	0.49
T1 N/M + Mag 3	81.6	94.9	1.7	1.0	0.09	0.00	0.01	0.4	0.01	0.2	0.49
T 1 N/M + Mag 3 + Mag 2	92.2	92.8	2.1	2.1	0.25	0.01	0.01	0.4	0.01	0.5	0.49
Distribution											
T 1 Mag 1		4.8	21.4	47.9	52.7	39.2	6.0	5.9	3.4	49.8	5.0
T 1 Mag 2		9.0	21.4	30.6	32.9	40.1	12.3	9.1	10.2	31.0	11.1
T 1 Mag 3		8.2	10.6	7.4	6.8	20.7	11.1	7.9	9.3	7.0	8.0
T 1 N/M		78.1	46.6	14.1	7.6	0.0	70.7	77.1	77.1	12.1	76.0
Feed		100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
T 1 N/M		78.1	46.6	14.1	7.6	0.0	70.7	77.1	77.1	12.1	76.0
T1 N/M + Mag 3		86.3	57.2	21.5	14.4	20.7	81.7	85.0	86.4	19.2	84.0
T 1 N/M + Mag 3 + Mag 2		95.2	78.6	52.1	47.3	60.8	94.0	94.1	96.6	50.2	95.0

Table 11-46: Electrostatic Separation of the 1231 Bulk Sample Rutile Concentrate

Electrostatic Separation	Wt %	TiO_2	Fe2O3	SiO_2	MgO	MnO	P2O5	V2O5	Nb2O5	CaO	S
Assay											
		%	%	%	%	%	%	%	%	%	%
Cond	86.0	93.7	2.2	1.5	0.2	0.01	0.01	0.4	0.01	0.4	0.5
Mids	11.7	73.3	3.4	13.2	1.8	0.01	0.01	0.3	0.01	3.3	0.2
N/C	2.2	30.6	6.4	38.2	5.1	0.03	0.03	0.2	0.00	9.0	0.0
Feed	100.0	89.9	2.4	3.7	0.5	0.01	0.01	0.4	0.01	0.9	0.5
Distribution											
Cond	86.0	89.7	77.6	35.1	30.0	82.3	80.1	88.8	92.0	34.6	95.7
Mids	11.7	9.6	16.5	41.8	45.4	11.2	10.9	9.9	8.0	42.8	4.1
N/C	2.2	0.8	5.9	23.1	24.6	6.4	9.0	1.3	0.0	22.6	0.1
Feed	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Table 11-47: Magnetic Fractionation of the 1245 Bulk Sample Rutile Concentrate

Magnetic Fractionation	Wt %	TiO_2	Fe_2O_3	SiO_2	Al_2O_3	MgO	MnO	P_2O_5	V_2O_5	Nb_2O_5	CaO	S
Assay												
		%	%	%	%	%	%	%	%	%	%	%
T 3 Mag 1	3.4	73	7.0	10.4	2.5	1.3	0.1	0.03	0.3	0.01	2.05	0.9
T 3 Mag 2	11.1	86.8	3.9	4.8	1.1	0.7	0.0	0.01	0.4	0.01	1.06	0.4
T 3 Mag 3	9.0	91.4	2.9	2.9	0.7	0.4	0.0	0.01	0.4	0.01	0.63	0.3
T 3 N/M	76.5	96.6	1.3	1.0	0.2	0.1	0.0	0.01	0.4	0.02	0.19	0.1
Feed	100	94.2	1.9	1.9	0.4	0.2	0.0	0.01	0.4	0.01	0.39	0.2
T 3 N/M	76.5	96.6	1.3	1.0	0.2	0.1	0.0	0.01	0.4	0.02	0.19	0.1
T3 N/M + Mag 3	85.5	96.1	1.4	1.2	0.2	0.1	0.0	0.01	0.4	0.02	0.24	0.1
T 3 N/M + Mag 3 + Mag 2	96.6	95.0	1.7	1.6	0.3	0.2	0.0	0.01	0.4	0.01	0.33	0.2
Distribution												
T 3 Mag 1	3.4	2.7	12.7	18.6	21.0	20.2	12.3	6.9	2.8	2.6	18.1	16.0
T 3 Mag 2	11.1	10.2	22.7	27.9	30.5	31.5	26.3	9.6	10.2	6.0	30.1	21.8
T 3 Mag 3	9.0	8.7	13.5	13.7	14.9	14.9	16.0	6.5	8.5	8.0	14.5	13.7
T 3 N/M	76.5	78.4	51.2	39.8	33.6	33.4	45.4	77.0	78.5	83.4	37.2	48.5
Feed	100	100	100	100	100	100	100	100	100	100	100	100
T 3 N/M	76.5	78.4	51.2	39.8	33.6	33.4	45.4	77.0	78.5	83.4	37.2	48.5
T3 N/M + Mag 3	85.5	87.1	64.7	53.5	48.5	48.3	61.4	83.5	87.0	91.4	51.8	62.2
T 3 N/M + Mag 3 + Mag 2	96.6	97.3	87.3	81.4	79.0	79.8	87.7	93.1	97.2	97.4	81.9	84.0

11.4.5.3 Programme 1286-003 - High Density Attritioning of Coarse Garnet

Programme 1286-003 was initiated during the coarse garnet circuit flowsheet development of bulk sample 1245. Refer also to Section 11.4.2 that describes the review of the three coarse garnet circuits that were considered. At the time of flowsheet development, chemical analysis was used as a guide and compared to microprobe chemical analysis of the coarse garnet. The chemical analysis obtained compared well to the microprobe analysis provided by Nordic Mining but it was acknowledged that mineralogical analysis would be required to confirm the garnet grade. Once it was confirmed by QXRD results that the coarse garnet grade was below specification, programme 1286-003 was commissioned.

The scope of work called for three representative coarse garnet circuit feed samples to be retrieved from the bulk +250 µm sample. It is important to note that at the time the coarse garnet circuit consisted only of the dry magnetic circuit as shown in Figure 11-25 below. The primary objective of this work was to establish if additional or improved garnet liberation could be obtained by high density attritioning (effectively determining if composite particles could be liberated into discreet mineral particles). The three samples were labelled A to C where sample A would not be attritioned and processed through the coarse garnet circuit, sample B would be attritioned for five minutes at 80% solids and processed through the circuit and sample C would undergo the same process as B, but attritioned for ten minutes. It was also requested that the particle size distribution (PSD) of each sample be determined after attritioning to get an understanding of the degree of change that took place following the attritioning treatments.

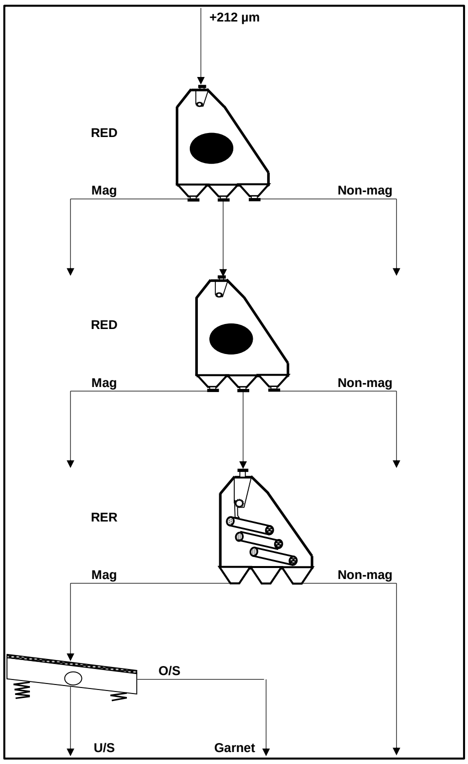


Figure 11-25: Coarse Garnet Flowsheet of Programme 1286-003

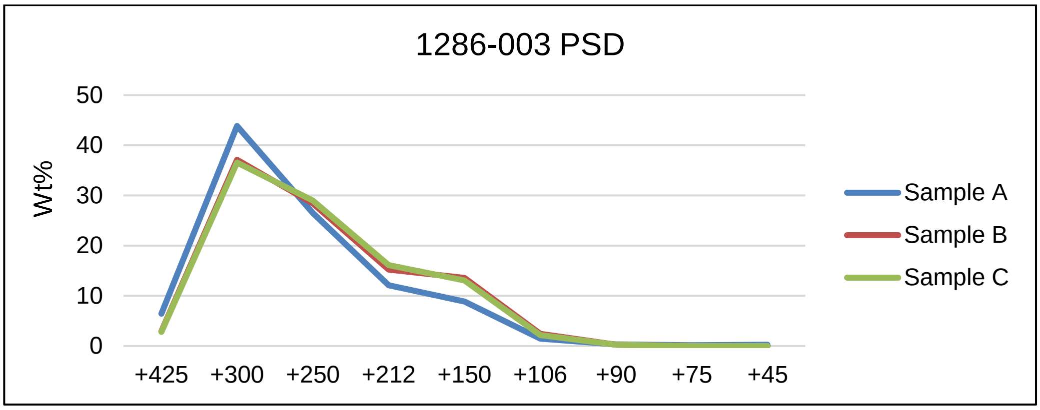


Figure 11-26: PSDs of Samples A, B and C of Programme 1286-003

The mass percentage per size class for samples A, B and C suggest that no significant difference in the PSD for 5- and 10-minute attritioning times as shown in Figure 11-26 above and only a minor reduction in the PSD is effected by the attritioning process.

From the results in Table 11-48 below, it can be seen that at best only a marginal improvement in the garnet grade was achieved by the five- and ten-minute attrition times. Most importantly, results showed that attritioning was not a successful mechanism for achieving coarse garnet grade.

Table 11-48: Summary of the Results for Samples A, B and C of Programme 1286-003

Assay						
	Wt %	Rutile	Garnet	Pyroxene	Amphiboles	Others
		%	%	%	%	%
Sample A	28.5	<1	82.2	7.2	5.5	5.0
Sample B	26.7	<1	87.0	3.9	6.0	3.1
Sample C	23.9	<1	84.1	6.4	5.0	4.6
Distribution						
Sample A	28.5	-	53.4	6.2	18.5	10.3
Sample B	26.7	-	49.6	3.4	18.3	6.4
Sample C	23.9	-	45.4	4.7	12.8	8.6

11.4.5.4 Programme 1286-004 – Optimally Comminuted 1308 Bulk Sample Scouting Tests

This testwork was commissioned specifically to formulate a process flow sheet and conduct detailed scouting tests to ensure that a coarse garnet product that meets the grade specification can be generated. Furthermore, it was required that the circuit be sufficiently robust to handle variable feed conditions and ensure that consistent metallurgical performance is achieved across a range of feed grades and liberation characteristics.

As seen from the results from programme 1286-003, at this point a coarse garnet product containing >92% garnet was not achieved and programme 1286-004 was specifically set up to achieve this goal. As such, a comprehensive set of scouting tests was undertaken to test systematically all known feed and equipment set-up conditions to determine the optimal testwork conditions to maximise both grade and recovery. At this time, the comminution testwork results were being processed and it was now known that the Hazemag crusher delivered the most optimally comminuted sample with the highest garnet and rutile liberation compared to the other crusher options. Outflows from the comminution testwork were kept separate as results showed that the Hazemag crusher delivered superior liberation of minerals compared to the rod mill. At this point it was not yet decided to process the output from the crushing and milling circuit as a single stream. As such, three samples were generated to be used in this testwork programme, they were:

- Sample A – Hazemag crushed sample
- Sample B – Rod milled sample and
- Sample C – Combined Hazemag and rod mill sample.

Each of the above samples were then subjected to a matrix of scouting tests consisting of variations of feed rates and roll speeds with variations in the mass distribution to the different products of the coarse garnet circuit consisting of a dry magnetic separation circuit only (two RED stages followed by a single RER stage, as shown in Figure 11-25 above). Scouting test performance curves were generated and reviewed once the QXRD results were available. IHC Robbins included the perfect separation line (line from the zero point to the average of the feed grade) as well as the no separation line (diagonal line connecting the zero and 100% points) in the graphical representation, where data points were scattered between these two lines. Results showed that following the initial RED stage, very little separation benefit existed as the next stage feed grade increased and the difference between actual separation and no separation was becoming increasingly small. Once the results from the scouting tests were received for sample A (Hazemag +212 µm), optimal condition selection could be made and the bulk sample was processed at the selected conditions. During the processing of samples B and C, learnings from sample A were used and it was decided to fast track samples B and C by using mass splits similar to those of sample A. At the time, due to the large number of QXRD samples generated in this process, BVM experienced a back log in the QXRD sample analysis. As a result, it was decided to prioritise the bulk sample analysis over the sighter tests for samples B and C. Bulk sample results are shown below in Table 11-49, Table 11-50 and Table 11-51. In each instance, Test- 101, 201 and 301 refer to the first stage RED, the second RED and the RER magnet performance respectively.

Table 11-49: Sample A – Hazemag Comminuted +212 µm Fraction

Test	Garnet Grade	Garnet Distribution	Overall Garnet Distribution
Test 101A	89	83	83
Test 201A	94	90	75
Test 301A	94	68	51

Table 11-50: 1286-004 Sample B – Rod Mill Comminuted +212 µm Fraction

Test	Garnet Grade	Garnet Distribution	Overall Garnet Distribution
Test 101B	82	64	64
Test 201B	83	91	58
Test 301B	93	71	41

Table 11-51: 1286-004 Sample C – Combined Hazemag and Rod Mill Comminuted +212µm Fraction

Test	Garnet Grade	Garnet Distribution	Overall Garnet Distribution
Test 101C	84	76	76
Test 201C	88	90	68
Test 301C	93	65	44

Overall circuit results for samples A, B and C are shown in Table 11-52 below.

Table 11-52: 1286-004 Overall Circuit Performance for Samples A, B and C

Test	Garnet Grade	Garnet Distribution
Test 301A	94	53
Test 301B	93	42
Test 301C	93	45

With limited benefit achieved from the subsequent separation stages, in particular the final RER stage, it became increasingly clear that perhaps an abbreviated wet gravity circuit may add value to the coarse garnet circuit and it was decided to investigate this in programme 1286-005 which was incorporated into programme 1308.

11.4.5.5 Programme 1293 - Rutile Upgrade Optimisation

11.4.5.5.1 Mineral Technologies

As discussed in Section 0, following below-target metallurgical performance results from the dry rutile processing circuit, it was decided to provide a parallel sample to a competing laboratory, namely Mineral Technologies (MT), to validate the performance and to investigate the potential benefits of alternative operating conditions. The scope of work was split into two parts. Part A consisted of sighter tests to confirm optimal operating conditions including high density attritioning of the feed material prior to physical separation; part B covered the processing of a bulk sample once optimal conditions were established in part A.

A short letter style report (A.12 Mineral Technologies 83207 Item A Report) covering part A of the scope reported results inferior to that obtained by IHC Robbins. Based on these results, it was decided not to go ahead with part B. Mineral Technologies reported that closely following similar mass splits to that developed by IHC Robbins was the root cause for the poor performance. As such Mineral Technologies took it upon themselves to repeat the magnetic separation testwork following a different operating philosophy and returned results significantly improved to their first attempt as well as a notable improvement compared to the first attempt by IHC Robbins. These results provided confidence that with improved operating conditions, improved metallurgical performance was possible.

Table 11-53 below shows that a significant improvement was realised by Mineral Technologies over the first attempt by IHC Robbins. However, it must be noted that these results exclude electrostatic separation as Mineral Technologies could not match the results achieved by IHC Robbins.

Table 11-53: Dry Magnetic Circuit Performance Comparison for Rutile Upgrading

Laboratory	% Wt	TiO_2	Fe_2O_3	SiO_2	Al_2O_3	MgO	ZrO_2	CaO	TiO_2 Dist
MT	22.0	69.2	7.7	12.7	2.0	1.6	0.08	3.1	86.7
IHCR	23.0	57.6	8.3	18.9	3.1	2.7	0.03	4.7	76.4

11.4.5.5.2 IHC Robbins

Following the decision to conduct parallel testwork for the rutile dry circuit at a competing laboratory, it was further decided to provide IHC Robbins the opportunity to repeat the testwork of this circuit now that valuable mineralogical analysis was available.

During the 1231 bulk sample programme, particularly during the dry rutile processing, it became increasingly clear that dry physical separation at standard operating conditions would result in substantial TiO_2 losses due to overlapping physical properties of the mineral assemblage. Therefore, it was decided to undertake extensive sighter testwork to establish the optimal operating conditions in a bid to improve the TiO_2 recovery across the dry rutile circuit. The operating variables investigated included:

- Feed rate
- Roll speed
- Feed temperature
- Roll diameter (rare earth roll)
- Voltage (electrostatic plate separator).

The above operating variables were used to define an overall matrix of tests to establish the optimal operating conditions for each separation technology. A bulk sample was processed through the developed flow sheet at the optimal conditions and a significant increase in the overall TiO_2 recovery of 16.3% was observed for the dry rutile circuit, as shown in Table 11-54 below. The final grade of the rutile concentrate did not change from the approximate mark of 80% TiO_2 as the focus of this effort was placed on TiO_2 recovery. High density attritioning of the rutile concentrate was included in the scope of work and the results indicated very limited improvement in the grade and recovery performance.

Table 11-54: Comparison of the Rutile Dry Circuit Before and after Supplementary Test Work

Processing Area	Programme 1231	Programme 1293
Rutile Upgrade Process		
Rutile product TiO_2 recovery (%)	70.6	86.9
Overall		
Rutile product	49.6	61.0

11.4.5.6 Flotation to Reject Pyrite and Upgrade Rutile

The investigation of the use of both forward and reverse flotation in the Nordic Mining flowsheet was conducted in two separate programmes:

- Core Metallurgy (Australia) investigated both forward and reverse flotation of the spiral concentrate stream, prior to the dry mill for programme 1231
- JKTech (Australia) investigated flotation of the dry mill rutile concentrate for programmes 1231 and 1245, focusing on reverse flotation of pyrite only.

This testwork evaluated the potential for removal of gangue minerals such as pyroxene and pyrite, from the valuable minerals, rutile and garnet. Two process routes were considered: forward flotation of rutile, or reverse flotation of firstly pyrite, then silica. It was found that garnet tends to follow the rutile in most flotation tests.

The reagents investigated are summarised in Table 11-55 below.

Table 11-55: Flotation Reagents Tested

Process	pH Range	Activators	Collectors
Rutile flotation	2-12	MgOH CaOH None	Flotinor SM15 (Phosphoric acid) Flotinor FS2 (Carboxylic acid) Flotinor FS3 (Carboxylic acid) Flotinor 3635 (Carboxylic acid ester) Hydroxamic acid
Pyrite flotation	Natural 6.0	$CuSO_4$ None	SIBX PAX
Silica flotation	3-12	MgOH CaOH None	Flotigam 2835-2 (alkyl ether diamine) Flotigam EDA 3 (alkyl ether amine)

The following was concluded from the Core Metallurgy testwork programme:

Rutile flotation:

- Flotigam FS2 was the best performing collector, which produced high rutile recoveries (75% to 89%) at concentrate rutile grades of 21% to 25%. Rutile grades were low due to the high proportion of garnet in the concentrate
- Unfortunately, repeat tests using FS2 demonstrated that the repeatability of this reagent was poor. The method of preparation of the FS2 reagent was also a factor in determining the results.
- Forward flotation could not match conventional physical separation in terms of upgrading of TiO_2 and TiO_2 recovery to the final concentrate.

Pyrite flotation:

- The pyrite tests performed well with ~97% total sulphur (ST) recovery at a concentrate ST grade of ~46%. This test was polished grind and had copper sulphate and SIBX addition
- A test was trialled without copper sulphate addition, which produced a slightly lower ST recovery (86%) and ST grade (~44%)
- Tests on dry mill product indicated that copper sulphate was not required to activate the pyrite. Recoveries of 97.2% were obtained at natural pH using 200 g/t of SIBX. It is believed that the sample of spiral concentrate may have been oxidised during the laboratory procedure, thus requiring copper sulphate to activate the surface.

Pyrite/silica flotation:

- The pyrite/silica tests failed to produce a high-grade rutile tails stream with high rutile recoveries. A test with Flotigam 2835 silica collector and CaOH addition had the highest rutile grade tails stream at ~23.5% but at an average rutile recovery of ~54%.

Overall, the pyrite flotation process was considered substantially more effective compared to the rutile and silica flotation processes and was incorporated into the flowsheet with the results summarised in Table 11-56 below. Pyrite recoveries were similar for both spiral concentrate and rutile concentrate. A high-level trade-off study (H352410-4000-210-030-0001) was conducted to determine the best position for the flotation process, looking at both the capital and operating costs. Locating the flotation process on the spiral concentrate, prior to the dry mill, was found to be significantly more cost effective due to the additional drying costs required if the flotation circuit is located after the dry mill.

Table 11-56: Pyrite Flotation Performance

Laboratory	Sample	Programme	Feed Grade		TiO_2 Recovery to Tails	Tails Grade	
			%TiO_2	%S	%	%TiO_2	%S
Core	Spiral Concentrate	1231	17.5	1.1	99.6	18.1	0.03
JKTech	Dry Mill Product	1231	82.5	4.7	99.4	91.5	0.14
	Dry Mill Product	1245	82.3	7.1	97.3	93.7	0.17

Tails grades from the spiral concentrate sample were substantially lower than from the dry circuit product, since the garnet tends to follow the rutile in the flotation process. However, this material is then directed to the dry magnetic separation process for further removal of garnet and pyroxene.

11.4.5.7 Rutile Recovery from 1245 Fines, 100% passing 45 µm

Although recovery of TiO_2 from the fines (particles smaller than -45 µm) did not specifically form part of the PFS testwork, the team identified early in the programme that this stream could contain a significant percentage (10 to 15%) of the total TiO_2, should the necessary caution not be exercised during the comminution process.

Introductory testwork was undertaken with limited success but it is the intention to explore this opportunity further during the next study phase to confirm firstly if a fine rutile product can be produced and once this is confirmed, to determine the recovery of such a product.

To understand the value of this opportunity, a financial analysis based on high level assumptions was made during the PFS to confirm if this opportunity warranted the expenditure of financial and human resources. Based on the parameters below it is evident that this opportunity should be explored during the next study phase and hence this work is included in the future work programme.

The following input parameters were used in the financial analysis:

- Capital investment of US$ 3 M
- Annual operating cost of US$ 300,000
- Rutile price based on 70% of the coarse rutile price (US$ 735)
- Annual fine rutile production of 2,205 t.

The following financial performance indicators were derived from the analysis:

- NPV of US$ 8 M over a 20 year period
- IRR of 24%
- NPV/capital ratio of 1.57.

11.5 Plant Operating Factor

An Operating Factor (OF) definition and basis were developed in order to estimate the achievable operating factor and plant utility for the three major product streams, being coarse garnet, rutile and fine garnet.

Ideally this would have entailed Reliability-Availability-Maintainability (RAM) modelling to ascertain predicted availabilities for the various unit operations, but a first approximation based on the number, and type of equipment involved, as well as assumed planned maintenance periods was believed to be sufficient at this stage due to limited final equipment details and vendor information with regards to maintenance requirements, frequencies and durations.

An Operating Factor for a facility is an indication of the percentage of time the production will be at instantaneous design rate and required quality. Operating factor is therefore dependant on the amount of time a facility is available for operation, the percentage of available time utilised, the achieved rate of operation/production, and the quality of the product. Mathematically the operating factor is expressed as:

$$Operating\ Factor = Availability\ x\ Utilisation\ x\ Rate\ Factor\ x\ Quality\ Factor$$

Where:

Availability = available time/calendar time

Utilisation = utilised time/available time

Rate Factor = average instantaneous rate/design rate

Quality Factor = percentage of first pass quality achieved

Plant utility is defined as the total utilised or operating time of the facility, and is calculated by the product of availability and utilisation.

Thus:

$$Plant\ Utility\ = Availability\ x\ Utilisation$$

Plant utility does not have a linear relationship with production, as rate and quality are not considered; it does, however, give an indication of expected operating and therefore production time (uptime) per annum. On the assumption of always running to design capacity and achieving 100% first pass quality, the operating factor will be equal to the plant utility.

An operating factor estimation will ideally involve availability calculation or modelling on a higher level of granularity, i.e. planned maintenance as per recommended maintenance intervals and tasks from supply or fabrication vendors on individual equipment items, synchronised to a certain starting point, and a best common interval selected to capture most preventative and scheduled maintenance activities. For the purposes of this study, four- and six-week intervals were selected for process areas, as these are medians in industry ranging from three- to twelve-week intervals for scheduled maintenance shutdowns.

On a similar level of detail, failure or breakdown events could be estimated on an individual equipment item level, supported by typical modes of failure, the (industry average) frequency of expected failure for the type of item, its application and environment of operation (MTBF), corrective action durations (MTTR) and spares requirements.

Vendor information and recommendations for the study were not available at the time of operating factor estimation, and industry best practice maintenance and failure events were therefore assumed on the basis of complexity, number of unique equipment items and type of equipment. These values were also compared to the performance of organisations in similar industries and locales, to validate the assumptions.

It is recommended that a more detailed assessment of expected availabilities and other secondary contributors to operating factor is done when vendor information is available during the next study phase. For now, the operating factors listed in Table 11-57 below have been determined for each process area based on the above approach. Using unique operating factors for each process area is reasonable due to the presence of buffers between each main process area, allowing increased availability.

Table 11-57: Operating Factors of Each Process Area

Process Area	Operating Factor (%)
Crushing	95.7
Primary Milling	94.3
Primary Desliming and Screening	94.3
Coarse Garnet Gravity Separation	97.0
Coarse Garnet Dry Mill	96.8
Secondary Milling and Desliming	96.8
Rutile Gravity Separation	96.8
Rutile Flotation	96.8
Rutile Dry Mill	96.5
Fine Garnet Magnetic Separation (WHIMS)	96.8
Fine Garnet Gravity Separation	96.8
Fine Garnet Dry Mill	96.8
Tails and Co-disposal	96.5

11.6 Process Design

11.6.1 Equipment Selection

The processing equipment selected was based on typical, industry-standard equipment that has an established presence in heavy mineral sands-type applications. The only major difference is the equipment required for the comminution circuit, as comminution is typically not required in mineral sands operations. Table 11-58 below summarises the equipment selected for the testwork programmes together with the separation feature of each.

Table 11-58: Equipment Types Selected for the Process

Equipment Type	Separation Feature	Duty
Screen	Particle size	Wet
Cyclone	Particle size/density	Wet
Wet High Intensity Magnetic Separator (WHIMS)	Magnetic susceptibility	Wet
Gravity Spiral	Particle density	Wet
Up-Current Classifier	Particle size & density	Wet
Rare Earth Drum (RED)	Magnetic susceptibility	Dry
Rare Earth Roll (RER)	Magnetic susceptibility	Dry
Electrostatic Separator	Conductivity	Dry
Electrostatic Plate Separator	Conductivity	Dry
Flotation	Surface properties	Wet

11.6.2 Process Design Basis / Criteria

11.6.2.1 Plant Capacity, Design Life and Production Profile

The processing plant is based on a RoM of 1.5 Mtpa dry solids. Total garnet production is expected to average 176 ktpa over the first five years and 224 ktpa over the first ten years.

11.6.2.2 Feed Grades

The feed grades of rutile and garnet for the two primary ore types, ferro ore and trans ore, are shown in Table 11-59 below.

Table 11-59: Average Grades of Rutile and Garnet for the Engebø Deposit

Ore Type	Rutile*	Garnet**
	wt%	wt%
Ferro eclogite	3.73	44.6
Trans eclogite	2.56	42.4
Leuco eclogite***	-	35.7
*Approximate grades from the mine plan for the open pit and underground schedules		
**Grades as stated in "*Technical Report – Resource Estimation for the Engebo Rutile/Garnet Deposit, Norway*" by A. Wheeler		
*** Not included in the current study but considered a potential future ore if economic considerations call for lower grade material		

11.6.2.3 Final Products Production Capacity

The plant production capacity for rutile, fine garnet, coarse garnet, total garnet and the potential product, pyrite, is shown in Table 11-60 for the 1.5- and 2.0 Mtpa cases. These values are also used in the financial model. A short description of their origin is provided below.

In Table 11-60 below, the total garnet yield is based on a blend of ferro- and trans ore in the ratio of 1.95 parts ferro to 1 part trans, as per the mine plan at the time of writing. Furthermore, a total garnet yield on ferro ore of 18.3% is used as determined from the coarse yield in programme 1308 of 11.4% and multiplied by 1.6 to account for the coarse/fine garnet ratio requirement of 62.5% to 37.5%. A de-rating factor of 97% was applied to the TiO_2 recovery of the ferro and trans programmes to account for a potential decrease in process performance as a result of scaling and to adopt a conservative position with respect to rutile recovery. Annual rutile production values are determined for ferro ore using a TiO_2 recovery and grade of 58.4% and 4.66%.

Table 11-60: Annual Product Capacity for 1.5 Mtpa Ferro Ore (excluding mining ore losses and dilution)

Product	Expected Product Yield	Annual Production: 1.5 Mtpa ROM
	Wt%	kt/annum
Total Garnet*	18.3	274.5
Coarse Garnet (212 to 550 µm)	11.4	171.0
Fine Garnet (106 to 212 µm)	6.9	103.5
Rutile** (45 to 212 µm)	2.32	34.4

*Based on a coarse: fine garnet ratio of 62.5:37.5
**Based on a de-rated (at 97%) recovery of 58.4%, ahead grade of 3.73% and a rutile product grade of 95% TiO_2

11.6.2.4 *Final Product Grades Achieved*

The rutile and garnet products achieved in the two representative bulk test work programmes for the ferro-eclogite (1308) and trans-eclogite (1234) ore types are included in the following tables below:

- Ferro-eclogite, rutile product: Table 11-61
- Ferro-eclogite, coarse garnet product: Table 11-62
- Ferro-eclogite, fine garnet product: Table 11-63

Table 11-61: Rutile Product Properties Achieved in Test Work Programme 1308

Rutile Product Composition (XRF)	
Compound	**Wt%**
TiO_2	94.90
Fe_2O_3	1.63
SiO_2	1.53
Al_2O_3	0.31
Cr_2O_3	0.01
MgO	0.03
MnO	0.02
ZrO_2	0.06
P_2O_5	0.01
V_2O_5	0.41
Nb_2O_5	n/d
CaO	0.35
K_2O	0.01
CeO_2	n/d
S	0.17
(FeS_2)	(0.30)
SnO_2	<0.02
U (ppm)	<10
Th (ppm)	<10

Rutile Product Particle Size Distribution	
Aperture (µm)	**Cum Wt%**
250	100
212	99.9
180	93.1
150	81.7
125	65.0
106	50.0
90	29.7
75	15.1
63	6.9
45	0.7
0	0
D_{50} (µm)	**106**
D_{80} (µm)	**147**

Table 11-62: Coarse Garnet Product Properties Achieved in Test Work Programme 1308 (i.e. 1286-005)

Coarse Garnet Product Composition (QEMSCAN)	
Mineral	**Wt%**
Garnet	95.4
Rutile	0.43
Pyrite	0.05
Quartz	0.47
Amphiboles	0.59
Clinopyroxenes	2.21
Other	0.85

Coarse Garnet Product Particle Size Distribution	
Aperture (µm)	**Wt%**
600	100
500	98.8
425	95.1
355	84.0
300	65.2
250	34.9
212	13.2
180	1.5
150	0.5
125	0.2
106	0
D_{50} (µm)	**274**
D_{80} (µm)	**342**

Table 11-63: Fine Garnet Product Specifications Achieved in Test Work Programme 1308

Fine Garnet Product Composition (QEMSCAN)	
Compound	**Wt%**
Garnet	95.5
Rutile	0.45
Pyrite	0.02
Quartz	0.28
Amphiboles	0.76
Clinopyroxenes	2.31
Other	0.68

Fine Garnet Product Particle Size Distribution	
Aperture (µm)	**Wt%**
250	100
212	99.2
180	83.3
150	57.2
125	26.2
106	7.8
90	0.7
75	0
D_{50} (µm)	**144**
D_{80} (µm)	**176**

11.6.2.5 *Process Plant Battery Limits*

The battery limits for the processing plant are as follows:

Incoming

- Secondary crushing feed conveyor discharge into secondary screen feed bin
- Receipt of raw / fresh water from the source
- Receipt of natural gas for the driers and re-heaters via trucks[1]
- Receipt of reagents via trucks into storage sheds, including flocculants and flotation reagents (collector and frother).

Outgoing

- Load out of primary rutile product from ship loader into ocean going vessel
- Load out of 30/60 mesh garnet product from ship loader into ocean going vessel
- Load out of 80 mesh garnet product from ship loader into ocean going vessel
- Load out of 100 mesh garnet product from ship loader into ocean going vessel
- Wet plant tailings and coarse plant rejects discharge into blending chamber via pumping system.

11.6.3 *Process Overview – Process Flow Diagrams*

The circuit design of the process plant comprises three stages: crushing, milling and beneficiation to extract rutile and garnet. Figure 11-27 below summarises the key plant areas.

[1] Delivery to be confirmed

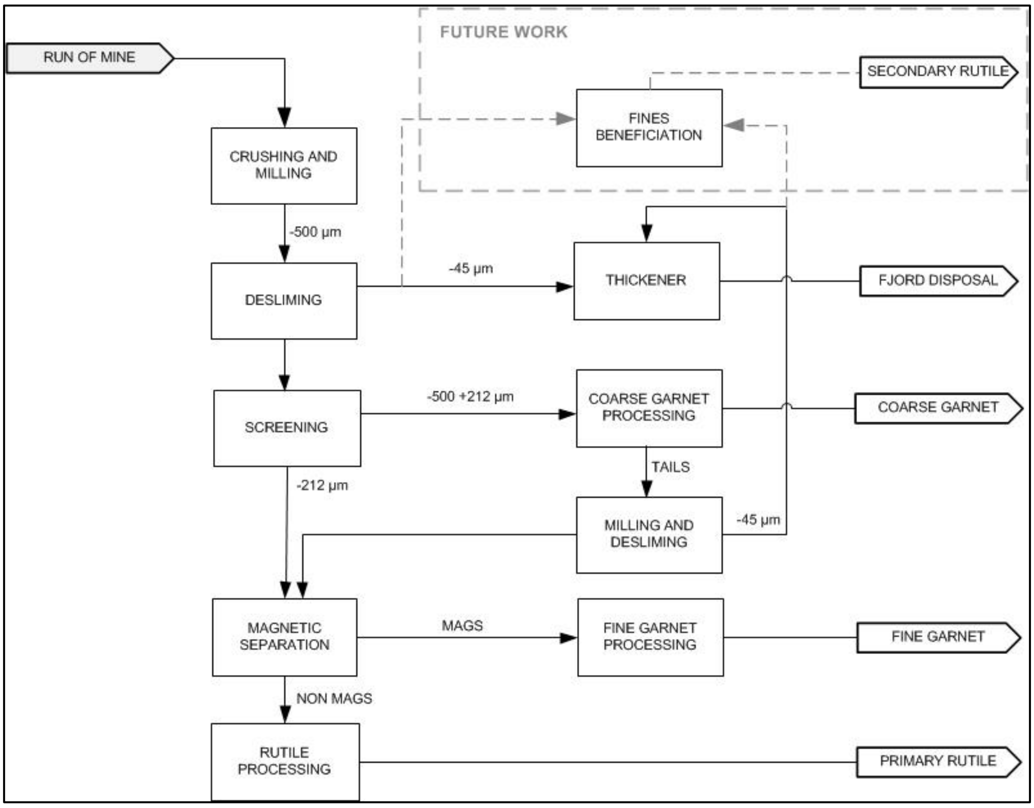


Figure 11-27: Simplified Block Diagram

11.6.3.1 Primary Crushing and Screening – Areas 3110 and 3120

The primary crusher will be supplied with ore from a glory hole in the pit on the Engebø mine. A static bar grizzly will be installed at the top of the glory hole. Oversize will be broken by a fixed rock breaker. Material will gravitate through the hole onto a pile and will be withdrawn at a controlled rate from the pile by an apron feeder and fed onto the 150 mm grizzly screen. A fixed rock breaker will also be available here to break any large oversize material.

The grizzly screen oversize will fall into the primary (jaw) crusher, where it will be reduced to -150 mm. The screen undersize will join the primary crushed ore and discharged onto a conveyor belt that will transfer the material to RoM silos with an estimated capacity of 40,000 t for both silos. The silos will be used both to store ore and to blend it if required prior to processing. A magnet will be installed over the conveyor for removal of any tramp steel that may damage the equipment downstream.

From the RoM holding/blending silos, material will be drawn onto a conveyor belt by feeders at the required rate to achieve the RoM blend required at the time. The conveyor belt will transfer the ore through a tunnel to the processing site. The primary crushing station will not operate between the hours of 22h00 and 06h00 daily including week-ends and public holidays.

A water spray dust suppression system will be installed at the crusher area. Spray nozzles will create a fine mist of spray water to lay dust.

11.6.3.2 Secondary and Tertiary Crushing - Areas 3130, 3140 and 3150

The purpose of the secondary and tertiary crushing circuit is to produce <6 mm material that will be fed to the milling circuit. A simplified schematic of the circuit is given in Figure 11-28 below.

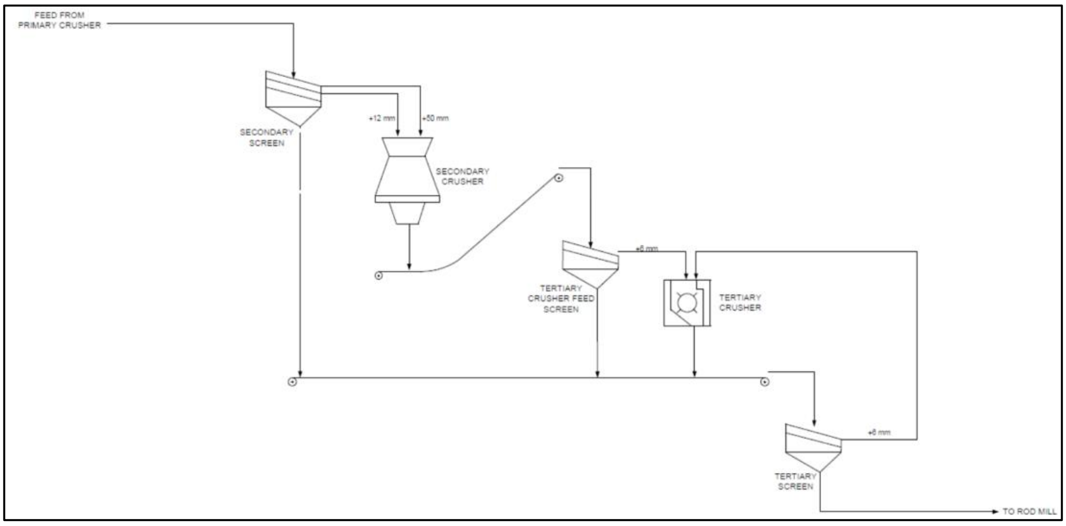


Figure 11-28: Schematic of the Secondary and Tertiary Crushing Circuit

The secondary crushing feed conveyor will deliver crushed ore (<150 mm) to the secondary screen buffer bin. Material will be withdrawn from the bin by a vibrating feeder and fed to a double deck secondary screen. The secondary screen will produce two oversize fractions (>50 mm and >12 mm) and an undersize fraction (<12 mm). The two oversize fractions will gravitate into the secondary (cone) crusher, while the undersize discharges onto the tertiary screen feed conveyor (not to be confused with the tertiary crusher feed conveyor). The secondary cone crusher discharges onto the tertiary crusher feed conveyor and reports to the tertiary crusher feed bin. The purpose of the feed bin is to ensure that the tertiary crushers are choke fed. The screens ahead of the tertiary crushers will remove the -6 mm undersize fractions to minimise the amount of fines reporting to the tertiary crushers. The screen undersize together with the tertiary crusher (Hazemag) product will join the secondary screen undersize and be conveyed to the tertiary screen.

The tertiary screen feed conveyor will deliver the material into a tertiary screen feed bin. The purpose of this bin is to facilitate even feeding of this relatively wide screen. This screen is a single deck fitted with a 6.0 mm wire mesh screen. The >6.0 mm fraction will be conveyed back to the tertiary crusher feed conveyor while the <6.0 mm fraction is conveyed to a buffer silo/s (primary rod mill feed silo) with a capacity of approximately 1,500 t. The tertiary crushers will thus operate in closed circuit with the tertiary screen.

A dedicated dust extraction system will be installed in the circuit to ensure the dust levels are kept low.

11.6.3.3 Primary Rod Milling - Area 3210

The mill feed conveyor will deliver crushed ore to the primary rod milling circuit. Material will discharge onto the rod mill feed screen distribution bin. The purpose of this bin is to distribute feed evenly to two sizing screens. The purpose of the feed screen is to reject the <550 µm fraction, this is required to minimise generation of fines (-45 µm solids). Oversize from this screen will report to the rod mill. Inlet dilution water will be added to the mill feed chute to control mill slurry density.

The mill is in closed circuit with another 550 µm screen to return only the >550 µm fraction while the <550 µm from the rod mill joins the <550 µm from the rod mill feed screen and is fed to the desliming circuit. Both these <550 µm streams report to the desliming cyclone feed sump.

A spillage sump pump will be provided in the mill bunded spillage area. Spillage will be pumped to the mill feed inlet.

11.6.3.4 Primary Feed Preparation - Area 4110

The <550 µm streams from the primary rod milling circuit report to the desliming cyclone feed sump where dilution water is added before the slurry is pumped by a variable speed pump to the desliming cyclone cluster for fines (<45 µm solids) removal, as shown in Figure 11-29 below. The cyclone overflow will gravitate to the slimes thickener to separate the fine solids from the process water, typically a conventional thickener underflow achieves a solids concentration ranging between 32% to 35% solids by mass. Clear water will overflow the thickener overflow rim and will be returned to the process water tank. Underflow from the de-sliming cyclone cluster will be pumped to the primary (Derrick) screen where dilution water will be added to achieve a slurry solids concentration of approximately 35% solids by mass. The slurry will report to a three-way distributor to split the feed into equal flows to feed the three Derrick screens. The cut size on the Derrick screens will be 212 µm.

Oversize (>212 µm) from primary sizing screen will gravitate to the coarse garnet rougher spiral feed sump while the undersize will be transferred to the primary WHIMS feed sump. During times when the wet gravity circuit is off line, the feed to the wet gravity circuit will be dewatered by belt filter and stored in a bin. Conversely, when the primary WHIMS circuit is off line, the feed to the WHIMS will be diverted to a dewatering belt filter and stored in a bin.

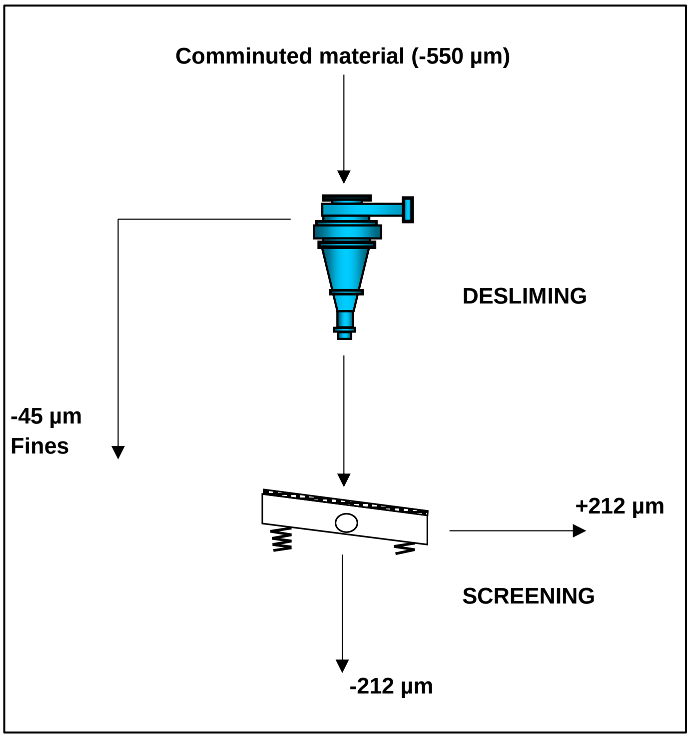


Figure 11-29: Primary Feed Preparation Area

11.6.3.5 Coarse Garnet Processing – Areas 4310 and 4320

The function of the coarse garnet circuit is to produce a high grade coarse garnet product by means of spiral gravity concentrators and dry RED magnetic separators. The coarse garnet in the ore is concentrated by using the difference in relative densities (specific gravity) of the valuable mineral and the gangue. A schematic of the circuit is given in Figure 11-30 below.

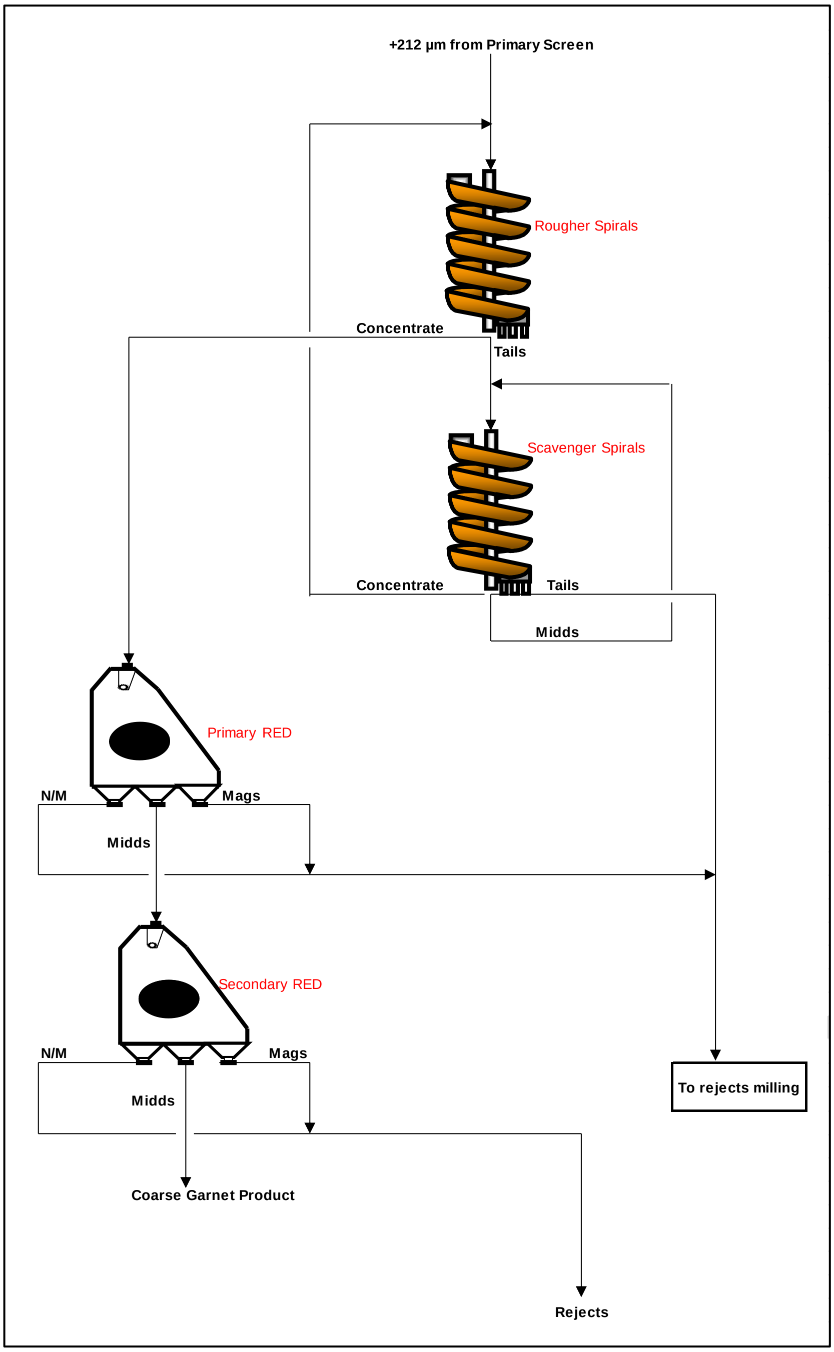


Figure 11-30: Schematic of Coarse Garnet Processing Circuit

The gravity circuit has two stages of HG10i spirals, namely rougher and scavenger. The rougher spirals produce two products, a concentrate that gravitates to the final concentrate sump and a tailing that gravitates to the scavenger spiral feed sump. The scavenger spirals produce three products, a concentrate that gravitates to the rougher spiral feed sump, a middling that gravitates back to the scavenger spiral feed sump and a tailing that gravitates to the final tails sump. The concentrate is laundered to a sump and pumped via a dewatering cyclone onto a vacuum belt filter. The vacuum belt filter reduces the moisture to below 5% (by mass) and the concentrate is dried through a fluid bed dryer. The tailings are pumped to the secondary milling circuit before feeding the WHIMS circuit.

The dried concentrate discharges from the dryer into a bucket elevator that feeds the mineral cooler. The hot mineral (~100 ºC) gravitates through multiple tube nests fed with cooled water and cooled mineral will discharge from the mineral cooler. Hot water will flow from the tube nests and be cooled in an adiabatic water cooler before re-entering the mineral cooler inlet port. Cooled mineral (~60 ºC) will report to the process consisting of two stages of RED magnetic separators. The primary RED magnets produce three products, a mag, a middling and a non mag. The mag and non mag contains recoverable amounts of TiO_2 and as such will also report to the secondary milling circuit as is the case with the coarse garnet wet gravity tails. The middlings from the primary RED report to the secondary RED magnets via a bucket elevator. This stage also generates three products where the mag and non mag report to tails and the middling is directed to final product. In the event that the secondary milling circuit is off line, the primary RED mag and non mag can be diverted to the final tails system as well.

11.6.3.6 *Secondary Feed Preparation - Area 4120*

The secondary feed preparation area is made up of secondary rod milling and WHIMS. The function of this area is to produce feed for the fine garnet and rutile processing circuits. A schematic of the process area is given in Figure 11-31 below.

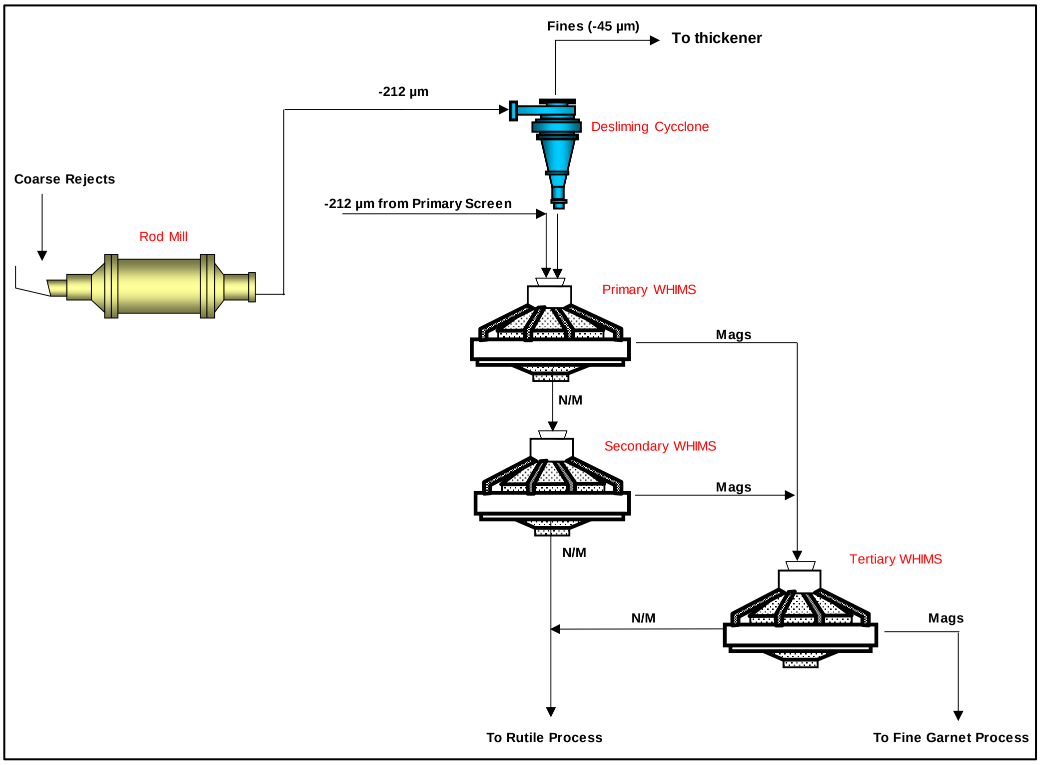


Figure 11-31: Schematic of Rod Milling and WHIMS Circuit

Feed to the secondary milling is received from two sources; namely the coarse garnet wet gravity circuit tails and the coarse garnet dry circuit primary RED rejects. These streams will report to the rod mill that is in closed circuit with a 212µm screen. The >212µm fraction is recirculated back to the rod mill while the <212µm from the rod mill reports to the secondary desliming cyclone cluster feed sump. Slurry is transferred to the desliming cluster where most of the <45µm solids will report to the cyclone overflow. The cyclone overflow will gravitate to the slimes thickener to separate the fine solids from the process water, while the underflow joins the <212µm from the Derrick screen undersize and fed to the WHIMS circuit feed sump. The primary WHIMS produces two products, a mag and a non mag. The mags gravitate to the tertiary WHIMS feed sump while the non mags gravitate to the secondary WHIMS feed sump. The secondary WHIMS feed reports to the dewatering cyclone before the underflow gravitates to a two-way gravity distributor. This WHIMS stage produces two products: a mag and a non-mag. The mags gravitate to the tertiary WHIMS feed sump while the non mags gravitate to the non mag concentrate feed sump. The tertiary WHIMS are fed via a two-way gravity distributor and produce two products: a mag and a non-mag. The mags gravitate to the mags dewatering feed sump and are pumped to the fine garnet processing circuit, while the non mags is combined with the secondary WHIMS non-mags and are pumped to the rutile gravity separation circuit.

11.6.3.7 Rutile Processing – Areas 4210, 4220 and 4230

The non mags from the WHIMS non mag sump are pumped to the rougher spirals. The rougher spirals produce three products: a concentrate that gravitates to the cleaner feed sump, a middling that gravitates to the mids scavenger spiral feed sump and a tailing that gravitates to the rougher scavenger spiral feed sump. The mids scavenger spirals are fed by the rougher spiral mids, the mids scavenger spiral mids and the rougher scavenger spiral concentrate and is split to four banks of spirals via a four-way distributor. The mids scavenger spirals produce three products: a concentrate that gravitates to the cleaner feed sump, a middling that gravitates to the mids scavenger spiral feed sump and a tailing that gravitates to the rougher scavenger spiral feed sump. Feed to the cleaner spirals consisting of the rougher spiral concentrate, mids scavenger spiral concentrate and cleaner spiral mids is pumped to a three-way pressure distributor feeding the cleaner spirals. The cleaner spirals produce three products: a concentrate that gravitates to the gravity circuit concentrate sump, a middling that gravitates to the cleaner spiral feed sump and a tailing that gravitates to the cleaner scavenger spiral feed sump. The rougher scavenger spirals produce three products, a concentrate that gravitates to the mids scavenger feed sump, a middling that gravitates to the rougher scavenger spiral feed sump and a tailing that gravitates to the tails sump.

Feed to the cleaner scavenger spirals consisting of the cleaner spiral tails is pumped to a two-way pressure distributor feeding the cleaner scavenger spirals. The cleaner scavenger spirals produce two products; a concentrate that gravitates to the gravity circuit concentrate sump and a tailing that gravitates to the rougher scavenger spiral feed sump. The rutile concentrate is pumped to the pyrite flotation bank while the tailings are pumped to the tails de-watering process.

The spiral concentrate is pumped from the rutile concentrate sump to the feed box of the flotation bank where it is combined with frother and collector. Air from the air compressor supplies the flotation cells. Spray water is added to the lip and launder of each cell for froth washing and mobility. Frother is delivered to the plant via tanker trucks and pumped to a frother storage tank. This frother is then pumped in batches to an agitated mixing tank where it is diluted with raw water from the raw water tank. This diluted mixture is pumped to the frother header tank which overflows back to the agitated mixing tank to maintain a constant level. A peristaltic pump draws frother from the frother header tank and pumps to the flotation bank.

Collector (xanthate) is delivered to the plant in bags via delivery trucks. A hoist is used to lift bags off the truck into the storage area and from the storage area to the mixing tank. A chute and bag breaker are installed above the mixing tank which breaks the bags and directs the dry xanthate pellets into the agitated tank. Raw water is added from the raw water tank and the solution is allowed to mix until the pellets are fully dissolved. Fumes are collected in a specialised vent to collect any carbon disulphide gas before being released. The xanthate mixture is pumped to an agitated holding tank. The collector is then pumped to the collector header tank which overflows back to the agitated holding tank. A peristaltic pump draws collector from the collector header tank and pumps to the flotation bank.

The reagents area is fitted with a spillage pump, eye wash stations and safety showers. Spillage is collected in drums for hazardous waste disposal.

The flotation froth phase (containing the pyrite) is directed to the flotation froth sump and pumped to the tails dewatering circuit. The flotation slurry product (containing rutile) is directed to the flotation product sump and pumped to the dewatering cyclone located above a vacuum belt filter. The vacuum belt filter reduces the moisture to below five percent (by mass) and the concentrate is dried through a fluid bed dryer. A schematic of the rutile wet circuit is given in Figure 11-32 below.

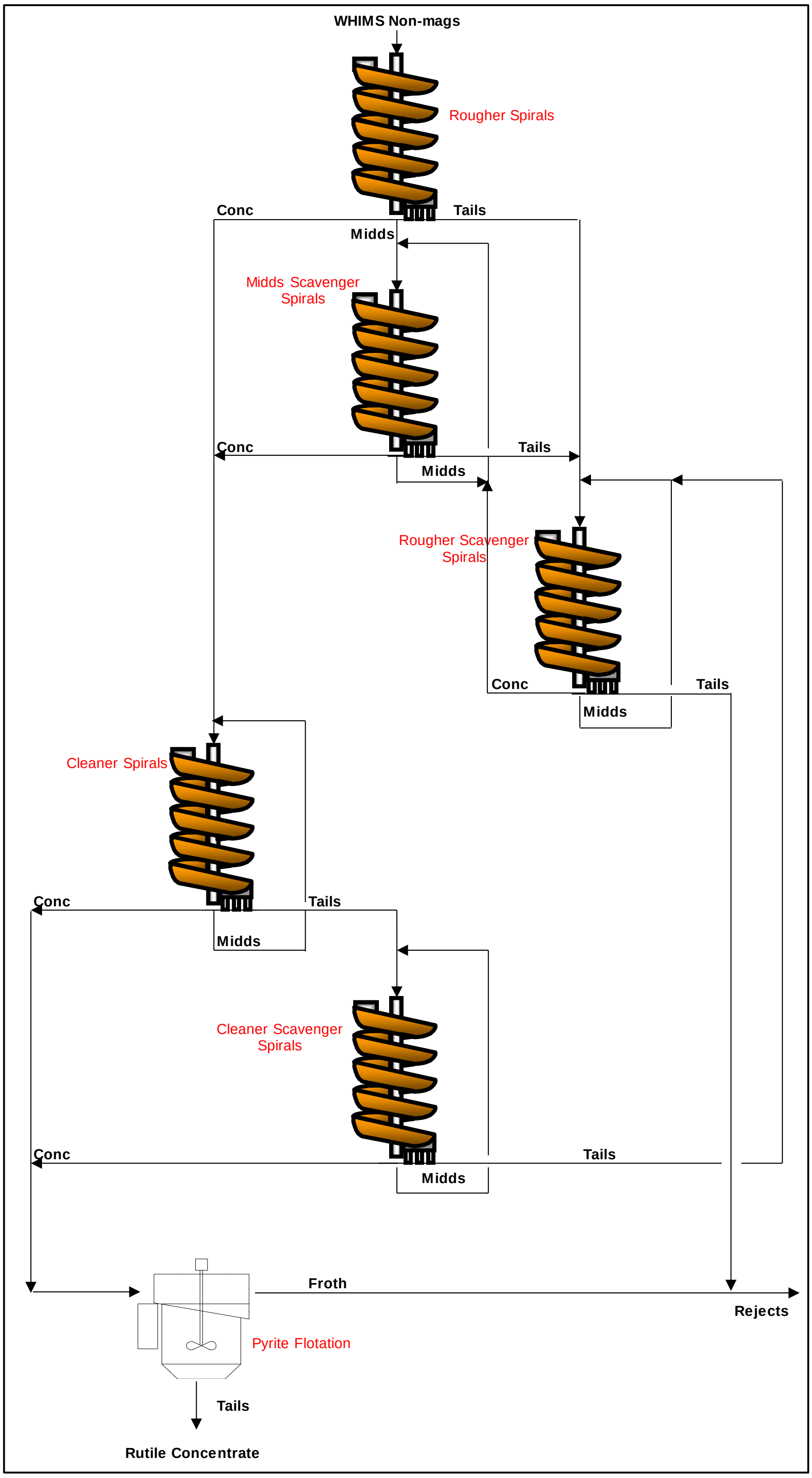


Figure 11-32: Schematic of Rutile Wet Circuit

The dried concentrate discharges from the dryer into a bucket elevator that feeds the mineral cooler. The hot mineral (~100 ºC) gravitates through multiple tube nests fed with cooled water and cooled mineral will discharge from the mineral cooler. Hot water will flow from the tube nests and be cooled in an adiabatic water cooler before re-entering the mineral cooler inlet port. Cooled mineral (~60 ºC) will report to a single stage of RED magnetic separators where magnetic and non-magnetic products are generated. The magnetic fraction reports to the primary RER stage to scavenge misplaced non-magnetic minerals from the magnetic feed. The RER generates two products; a magnetic fraction that reports to rejects and a non-magnetic fraction that joins the non mags from the primary RED magnet stage.

The combined non-magnetic fraction from the first two stages report to the secondary RER stage where two products are generated; a magnetic fraction that reports to the tertiary RER magnet and a non-magnetic fraction that reports to the re-heater ahead of the electrostatic separation circuit. The tertiary RER generates two products: a magnetic fraction that reports to rejects and a non-magnetic fraction that joins the non mags from the secondary RER to the re-heater. Secondary and tertiary RER non-magnetic fractions along with the electrostatic plate separator conductors report to a fluid bed re-heater. The re-heater is required to elevate the mineral temperature to approximately 80 to 100ºC for optimal electrostatic separation efficiency. Hot mineral will be transferred to the primary High Tension Roll (HTR) machines where three products will be generated; a conductor reporting to the cleaner HTR machines, a middling reporting to the scavenger HTR machines and a non-conductor reporting to the scavenger cleaner HTR machines.

The scavenger HTR machines generate two products; a conductor reporting to the cleaner HTR machines and a non-conductor reporting to the scavenger cleaner HTR machines. The scavenger cleaner HTR machines generate two products; a conductor reporting to the scavenger re-cleaner HTR machines and a non-conductor reporting to the electrostatic plate separator re-heater. The scavenger re-cleaner HTR machines generate two products: a conductor reporting to the cleaner HTR machines and a non-conductor reporting to the electrostatic plate separator re-heater. The cleaner HTR machines generate two products: a conductor reporting to the rutile product bin and a non-conductor reporting to the electrostatic plate separator re-heater. Electrostatic plate separators generate two products: a conductor reporting to the primary HTR re-heater and a non-conductor reporting to the rejects conveyor. A process equipment baghouse is included in the design to capture dust generated in the process. Fine solids captured in the bag house will be directed to rejects while the clean air is released back into the atmosphere. A schematic of the rutile dry circuit is given in Figure 11-33 below.

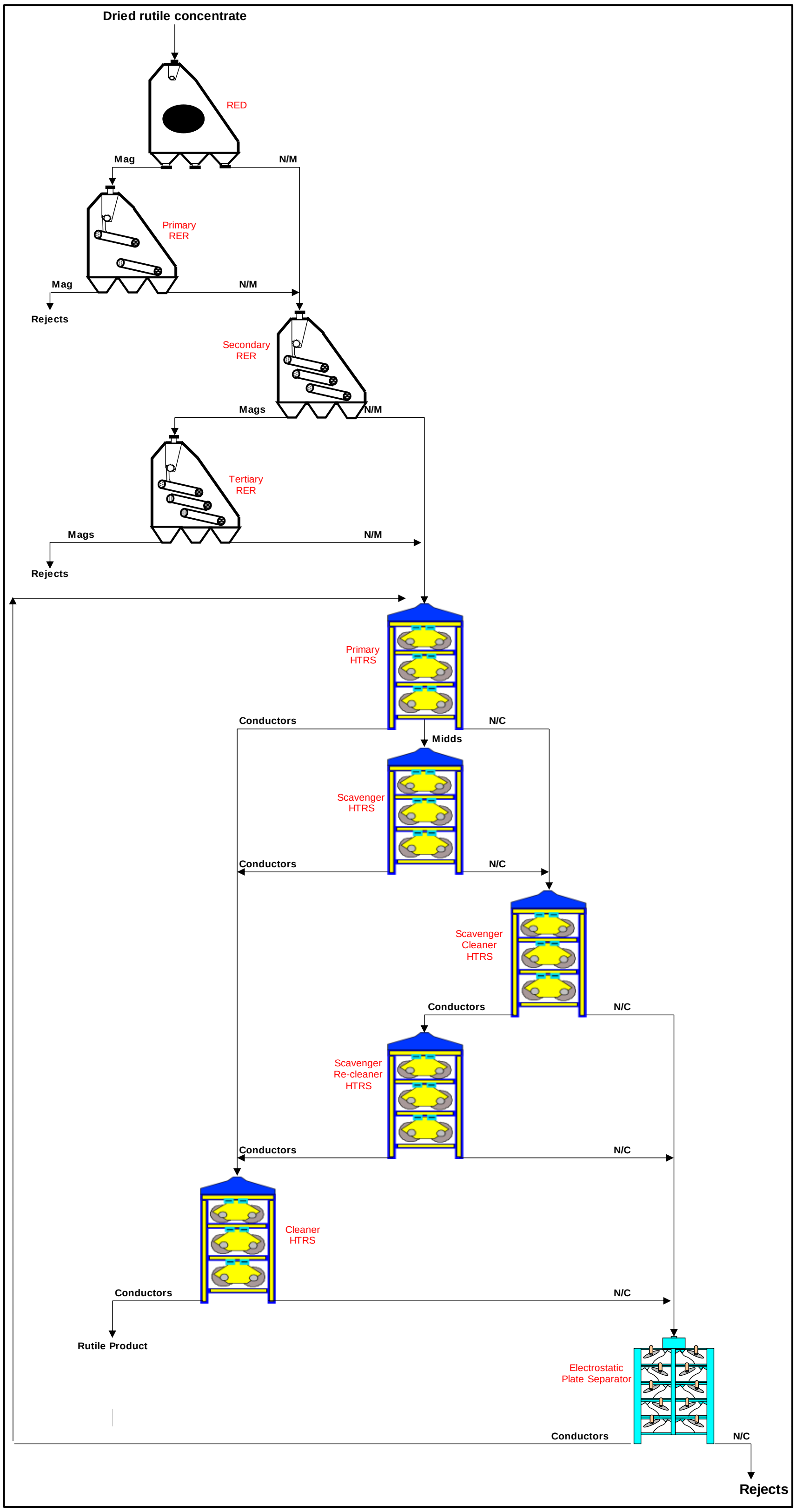


Figure 11-33: Rutile Dry Processing Circuit

11.6.3.8 Fine Garnet Processing – Areas 4330 and 4340

Magnetic product from the WHIMS circuit is pumped to a de-watering cyclone located above the UCC. The cyclone underflow reports to the UCC while the overflow reports to the process water distribution box and all excess water gravitates to the thickener feed well. The UCC separates on size and density and is operated to prepare feed for spiral separation. The teeter water establishes an upward current and a solids bed form in the unit that acts as a dense media separation zone. Large heavy particles overcome the upward current as well as the “dense bed” while lighter (and smaller dense) particles cannot penetrate the “dense bed” and is swept away by the up current. The UCC overflow reports to the final tails while the underflow is fed to the spirals. Two banks of spirals make up the gravity section of the fine garnet circuit. The spiral concentrate gravitates into the concentrate transfer sump which delivers the slurry to a dewatering cyclone located above the vacuum belt filter. The vacuum belt filter reduces the moisture to below five per cent and the concentrate is dried through a fluid bed dryer. The tails from the spirals joins the overflow from the UCC and pumped to the tails dewatering area. A schematic of the gravity circuit is given in below.

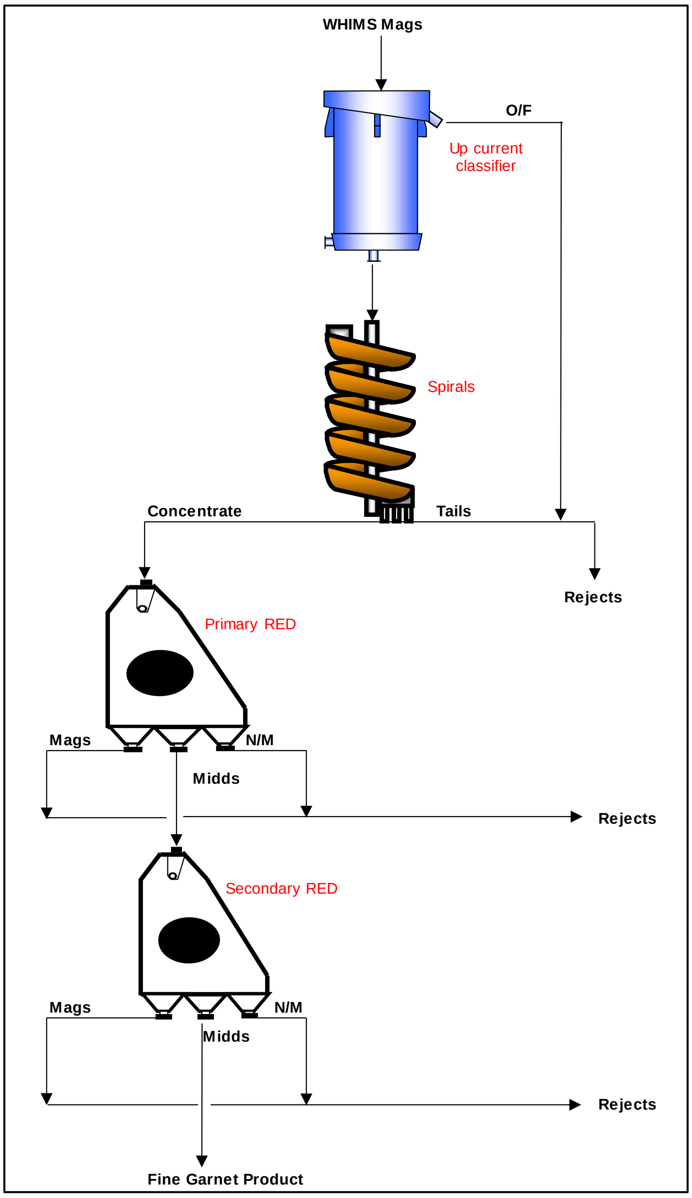


Figure 11-34: Schematic of Fine Garnet Processing Circuit

The dried concentrate discharges from the dryer into a bucket elevator that feeds the mineral cooler. The hot mineral (~100 ºC) gravitates through multiple tube nests fed with cooled water and cooled mineral will discharge from the mineral cooler. Hot water will flow from the tube nests and be cooled in an adiabatic water cooler before re-entering the mineral cooler inlet port. Cooled mineral (~60 ºC) will report to the process consisting of two stages of RED magnetic separators. The primary RED magnets produce three products: a mag, a middling and a non mag. The primary RED mag and non mag reports to the rejects while the middling from the primary RED reports to the secondary RED magnets via a bucket elevator. This stage also generates three products where the mag and non-mag report to tails and the middling is routed to final product.

11.6.3.9 Product Handling and Load-out Plant

Final products from the rutile, coarse garnet and fine garnet circuits are stored in large product silos. The size of the product silos will be better defined once shipping schedules are better defined. Prior to storage, fine and coarse garnet will pass over two sizing screens to split the two products into three final size fractions. The coarse garnet will report to a 355 µm screen (to be confirmed) where the oversize reports to the coarse garnet product bin and the undersize to the intermediate garnet bin. Similarly, the fine garnet will report to a 106 µm screen where the oversize reports to the fine garnet product bin and the undersize to the dry rejects conveyor.

11.6.3.10 Tails Handling

Tailings from production will be safely placed in a deep fjord deposit at a depth of 300 m. The tailings material consists of natural minerals and process reagents. It is a requirement to remove as much fresh water as possible, not only to limit the fresh water consumption but also to ensure limited low density water (fresh) enters the co-disposal system.

Wet plant tails consist of thickener underflow with a solids concentration of approximately 35% (m/m) as well as tails from the wet gravity separation plants. The tails from the wet gravity circuits will be dewatered by dewatering cyclones. The cyclone underflow will report to a dewatering linear screen while the overflow gravitates to a water collection sump. The screen underflow gravitates to a water collection sump which transfers the water to the cyclone overflow water sump. The dewatering screen discharges the moist tails onto a belt conveyor that will transfer the material to a mixing sump upstream of the co-disposal system. Discard from the dry processing areas will be combined onto a single belt conveyor system that will carry the rejects to the same mixing sump as above. The underflow from the slimes thickener will also be pumped to the mixing sump where all three rejects stream will be diluted with sea water then pumped to the feed chamber of the co-disposal system. Further dilution by sea water will take place in the co-disposal chamber after which the stream will be deposited in the deep see fjord disposal area.

11.6.3.11 Services and Utilities

11.6.3.11.1 Plant Water

Raw water from the source will be discharge into the fire water tank. Only the fire water pumps are connected to the fire water tank. The fire water tank overflows into the raw water tank and the raw water reservoir overflows into the process water tank. Three water pump systems draw water from the raw water reservoir, one delivering water to the gland seal water distribution, one delivering raw water to various process plant areas via a raw water ring main system and the other supplying the potable water treatment plant.

The process water tank receives process water from thickener overflow. Make up water to process water tank will be supplied by the raw water from the source. Five pump systems (quantity to be confirmed during next study phase) draw water from the process water tank; one supplying comminution and feed preparation, one supplying dust handling, one supplying the central processing header, one supplying the coarse garnet circuit and one supplying the tails dewatering process area. The fire water tank will supply only the fire water pumps consisting of an electrical pump system consisting of a supply pump and jockey pump as well as a diesel fire water pump in the event that the electrical supply is off.

11.6.3.11.2 Plant and Instrument Air

The processing site will be serviced by a single compressed air facility comprising two compressors, two air receivers and two air dryers. It is anticipated that a single train will be sufficient to provide compressed air to the plant and the other train will act as a stand-by unit. The compressors, air receivers and dryers will be piped to provide maximum flexibility in the event of one of the components in a system becoming unavailable. Piping of the systems will also allow connection of a mobile compressor to the compressed air system.

11.6.3.11.3 Natural Gas

Natural gas will be consumed at various process areas within the process plant. The fuel will be utilised as a source of energy in the fluid bed dryers and re-heaters.

11.6.3.11.4 MSP Dry Process Area Dust and Off-Gas Handling

The dry process areas will generate dust during the normal processing (and drying) and this dust and off gas will be captured in dedicated bag houses to ensure the working environments are safe for operators to work in. The design philosophy for dust capturing and handling is to keep bag houses treating exhaust off gases from liquid fuel burning dryers and re-heaters separate from those connected to process equipment and materials handling equipment in the process areas.

As a preliminary basis, each bag house will discharge dust via a rotary valve and/or possibly a screw feeder (depending on the number of discharge points) into the dust pump system. The dust pump system will consist of a typical froth pump system. Each bag house will, therefore, have a dedicated froth pump pumping to a central baghouse dust collection sump before the combined dust laden slurry is pumped to the slimes thickener. Process water will be supplied to each froth pump system as make-up including the central baghouse dust collection sump.

11.6.4 Mass and Mineral Balances

11.6.4.1 High-level Mass and Mineral Balance

A high-level mass and mineral balance was developed in MS Excel to guide process development by tracking the distribution of mineral species through the proposed flowsheet, and to determine indicative solids flow rates through each stage of the process. The balance was based on data from the IHC Robbins test work programmes including mass splits, XRF, QXRD and QEMSCAN results. The grade and recovery of TiO_2 (i.e. rutile) and garnet were of primary interest and allow for direct comparisons of the different testwork programmes, as shown in Figure 11-35 below.

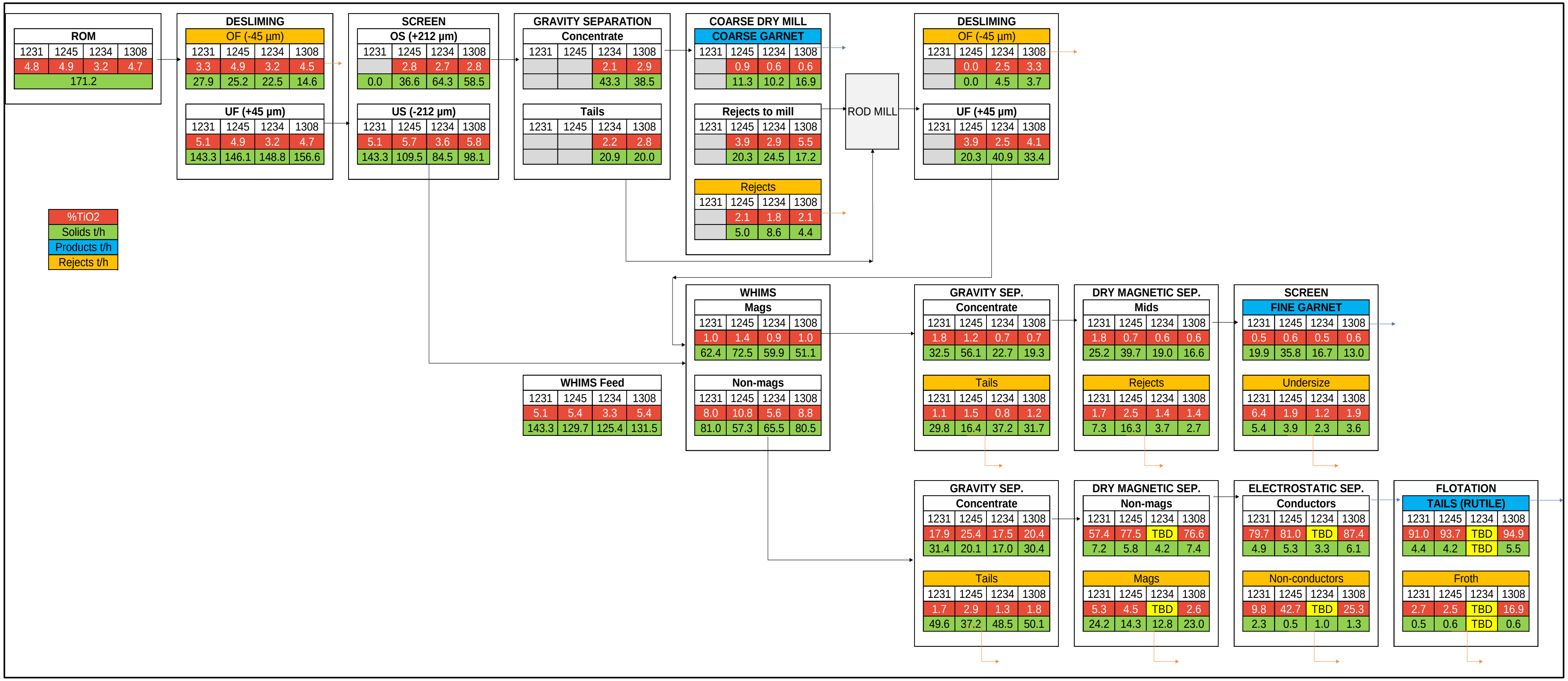


Figure 11-35: Summary Block Flow Diagram of the Process Including TiO_2 Grades

In addition to TiO_2 and garnet, gangue minerals were tracked (where available) in an attempt to understand how effectively each process step upgrades the target minerals and rejects unwanted minerals. Importantly, the mineral balance provided insight into the inclusion of a wet gravity stage in the coarse garnet circuit. In essence, the balance highlighted that spirals achieved the best upgrading ratio of garnet through effective rejection of amphiboles and pyroxenes. These insights guided development of the supplementary testwork programme 1286-005.

11.6.4.2 Detailed Mass Balance

The high-level balance described in Section 11.6.4.1 was used to guide process development. In addition to the high-level balance, a detailed mass balance, including solids and water, was developed in MS Excel. The balance provides sufficient detail for a PFS-level study and is used to generate stream tables which inform equipment sizing and mechanical design. The mass balance was linked directly to the Process Design Criteria (PDC) which include all relevant test work data.

The mass balance was based on an average hourly flow rate (using 8760 hours per annum) and a specified RoM of 1.5 Mtpa. Using an average flow rate allows the balance to close without disparity across process areas due to different operating factors. From the average hourly flow rate, nominal flow rates were calculated using unique operating factors for each process area. Each main process area has a unique operating factor due to the inclusion of buffers which segment the process. A second mass balance was developed for a RoM of 2.0 Mtpa to provide insight into process requirements for the future expansion case.

For the spiral circuits, modelling of the recycle/circulating streams led to different overall mass yields compared to test work data, as these streams were simulated by using shaking tables in the laboratory. To ensure that overall mass yields match those of the test work data, the overall mass yields of the spiral circuits were fixed while the mass yields across the spiral stages that determine recycle stream flow rates were allowed to vary. It is recommended that a dynamic model of the spirals that includes release curve data be employed in the next study phase to more accurately capture feed grade variation on spiral stage mass splits. However, the methodology employed here is deemed sufficient for PFS-level detail. The output of the mass balance is captured in the stream tables that are included with the PFDs.

11.6.5 Tailings Evaluation

The tailings were subject to extensive testing as part of the permitting of the tailings disposal. The tailings contain gangue minerals that are typically found in Norwegian bedrock. The minerals are evaluated as benign, meaning that the levels of sulphides, heavy metals and radioactive elements are low and will not pose a risk to the environment. The PFS testwork has identified three reagents that are recommended in the processing of the ore, namely: sodium isobutyl xanthate (SIBX), polypropylene glycol and Magnafloc 5250. These reagents will be present in low concentrations in the tailings. A summary of each reagent is included in the following sections with the annual utilisation rates of each provided below in Table 11-64.

Table 11-64: Summary of the Process Reagents

Reagent	Purpose	Annual Utilisation (t/a)
Sodium isobutyl xanthate	Flotation Collector	26.0
Polypropylene glycol	Flotation Frother	10.5
Magnafloc 5250	Thickener Flocculant	1.4

11.6.5.1 *Flotation Collector – Sodium Isobutyl Xanthate*

Collector is required in the flotation process to assist with the attachment of minerals to air bubbles. In the current process, SIBX will be used as the collector. SIBX is widely used in the mining industry. The expected dosage of SIBX to the flotation circuit is 100 g/t which leads to the concentrations shown in Table 11-65 below. As can be seen from the table, the discharge concentration of SIBX is substantially lower than the LC_{50} values identified in literature, before and after dilution in the fjord. Note that SIBX is not one of the permitted chemicals in Nordic Mining's discharge permits. Therefore, the substance must pass through the formal legislation procedure to be utilised in the process.

Table 11-65: Expected Concentration of SIBX at Different Locations in the Process

Location	SIBX Concentration	
	g/m^3	ppm
Thickener underflow	61.4	48.4
Co-disposal stream	2.13	2.0
In fjord 50 m around the discharge point - dilution factor of 30*	0.071	0.067
LC_{50} values identified in literature	-	56 to 100
*Recommended value from EIA study		

11.6.5.2 *Flotation Frother – Polypropylene Glycol*

Testwork conducted by Core Metallurgy identified DF400 as the optimal frother required for reverse flotation of pyrite. Additional testwork by JKTech suggests a dosage of 40 g/t to be sufficient. This dosage results in the concentrations in Table 11-66 below. DF400 is not one of the permitted chemicals in Nordic Mining's discharge permits. Therefore, the substance must go through the formal legislation procedure to be utilised in the process.

Table 11-66: Expected Concentration of DF400 at Different Locations in the Process

Location	DF400 Concentration	
	g/m^3	ppm
Thickener underflow	24.5	19.3
Co-disposal stream	0.85	0.80
In fjord 50 m around the discharge point - dilution factor of 30*	0.028	0.027
LC_{50} values identified in literature	1,000	-
*Recommended value from EIA study		

11.6.5.3 *Flocculant – Magnafloc 5250*

After preliminary testwork, BASF recommended the use of Magnafloc 5250 as the appropriate flocculant for the slimes thickener. Magnafloc 5250 is a high molecular weight anionic polyacrylamide flocculant supplied as a free-flowing granular powder. The optimal dose was found to be 8.6 g/t, which translates to the concentrations presented in Table 11-67 below.

Table 11-67: Expected Concentration of Magnafloc 5250 at Different Locations in the Process

Location	Magnafloc 5250 Concentration	
	g/m³	ppm
Thickener underflow	3.3	2.6
Co-disposal stream	0.11	0.11
In fjord 50 m around the discharge point - dilution factor of 30*	0.004	0.004
LC_{50} values identified in literature	100**	-
* Recommended value from EIA study **For fish and aquatic invertebrates (MSDS)		

Nordic Mining has received a permit for the use of 60 tpa of Magnafloc 155 which is also classified as a polyacrylamide. Current estimates place the annual usage of Magnafloc 5250 at 1.4 t which is well below the maximum value of the permit of 60 tpa. However, Magnafloc 5250 specifically is not part of the permitted chemicals and must go through the formal legislation procedure to be utilised in the process.

11.7 Process Functional Description

A functional description was developed for use in subsequent study phases, which describes the high-level control philosophy of the various process components of the Project.

The Functional Description (FD) is an intermediate artefact of the software engineering process. The purpose of the FD is to facilitate the design process by:

- Communicating requirements, summary criteria/constraints, context and knowledge to team members
- Acting as a catalyst to focus development systematically to encompass all aspects of requirements
- Visualising the proposed realisation by:
  - Enumerating and expanding requirements
  - Developing and capturing salient design principles
  - Outlining the proposed user/system interfaces

- Presenting displays/examples and operational scenarios/examples to facilitate validation of the proposed design with the team and functional validation with key stakeholders.

- Serving as a record of the engineering process, summarising salient requirements, design approach and operating scenarios for use in test scenarios and preparing formal system documentation.

The sections that make up the FD document includes a brief description of the process area followed by a high-level summary of the primary and secondary (if included) control requirement. This document will become the foundation of the Functional Specification which will develop the control philosophy in detail during the next study phase. Essentially each PFD is used to describe the driving control requirement/s necessary to ensure that optimal control of that section is achieved to ensure the design intent is achieved.

11.8 Basic Plant Layout and Plant Design

As illustrated in Figure 11-36 and Figure 11-37 below, the plant layout consists of three geographical levels. The overall plant is divided across the three levels as follows:

Level 1

- Primary crushed ore conveyor (from underground primary crusher)
- Secondary crushing and screening
- Tertiary crushing and screening
- Primary rod milling and screening
- Administration buildings
- Buffer storage and dewatering circuits
- Process water storage.

Level 2

- Primary feed preparation (de-sliming and screening at 212 µm)
- Tails thickening
- Wet garnet circuits (coarse and fine)
- Dry garnet circuits (coarse and fine)
- Secondary feed preparation (coarse rejects milling (i.e. secondary milling), secondary desliming, and WHIMS)
- Wet rutile circuit
- Dry rutile circuit
- Tails dewatering
- LPG storage.

Level 3

- Product storage (rutile, 100 mesh garnet, 80 mesh garnet and 30x60 mesh garnet)
- Product loadout for shipping.

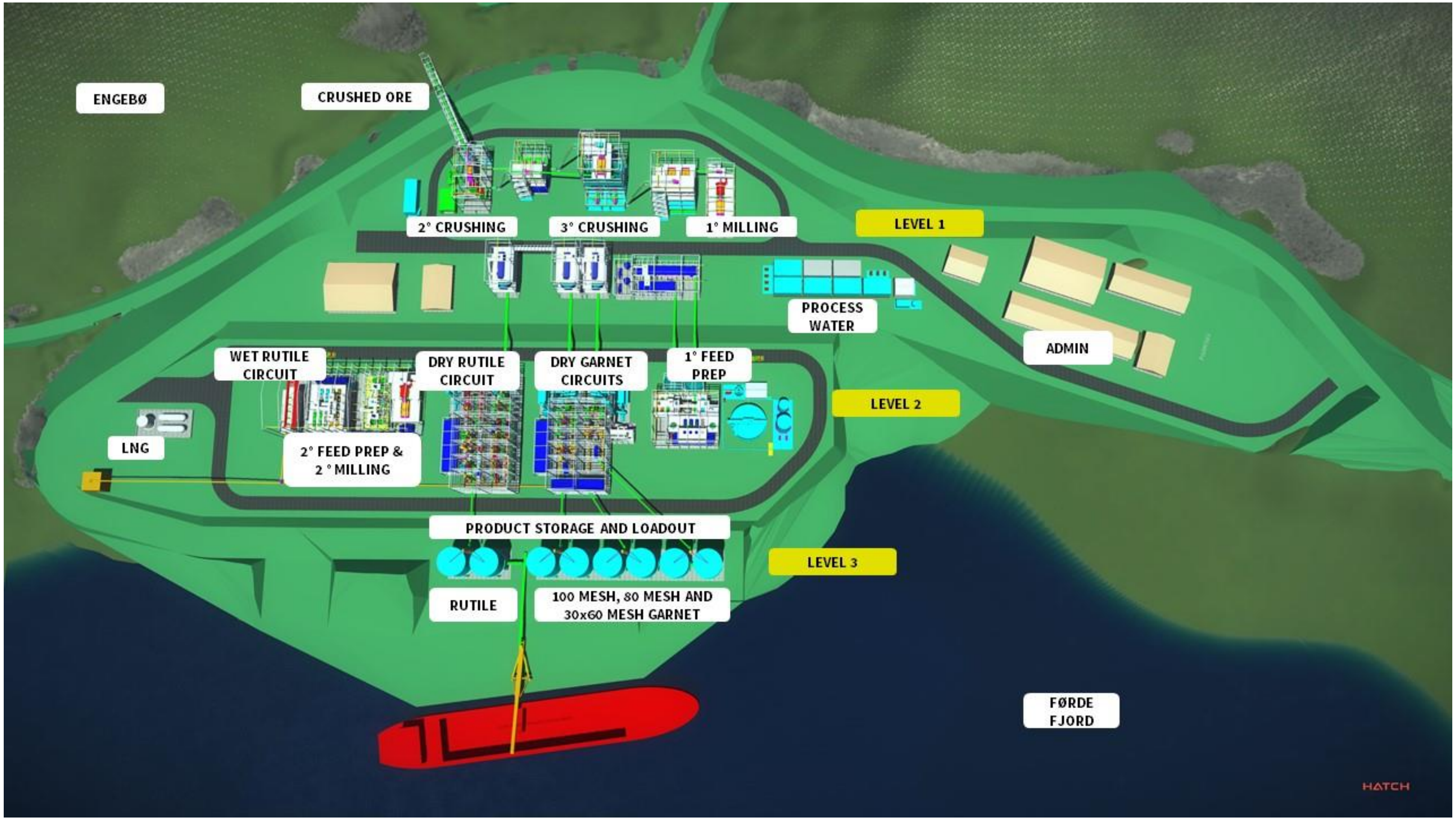


Figure 11-36: Plan View of the Proposed Plant Layout

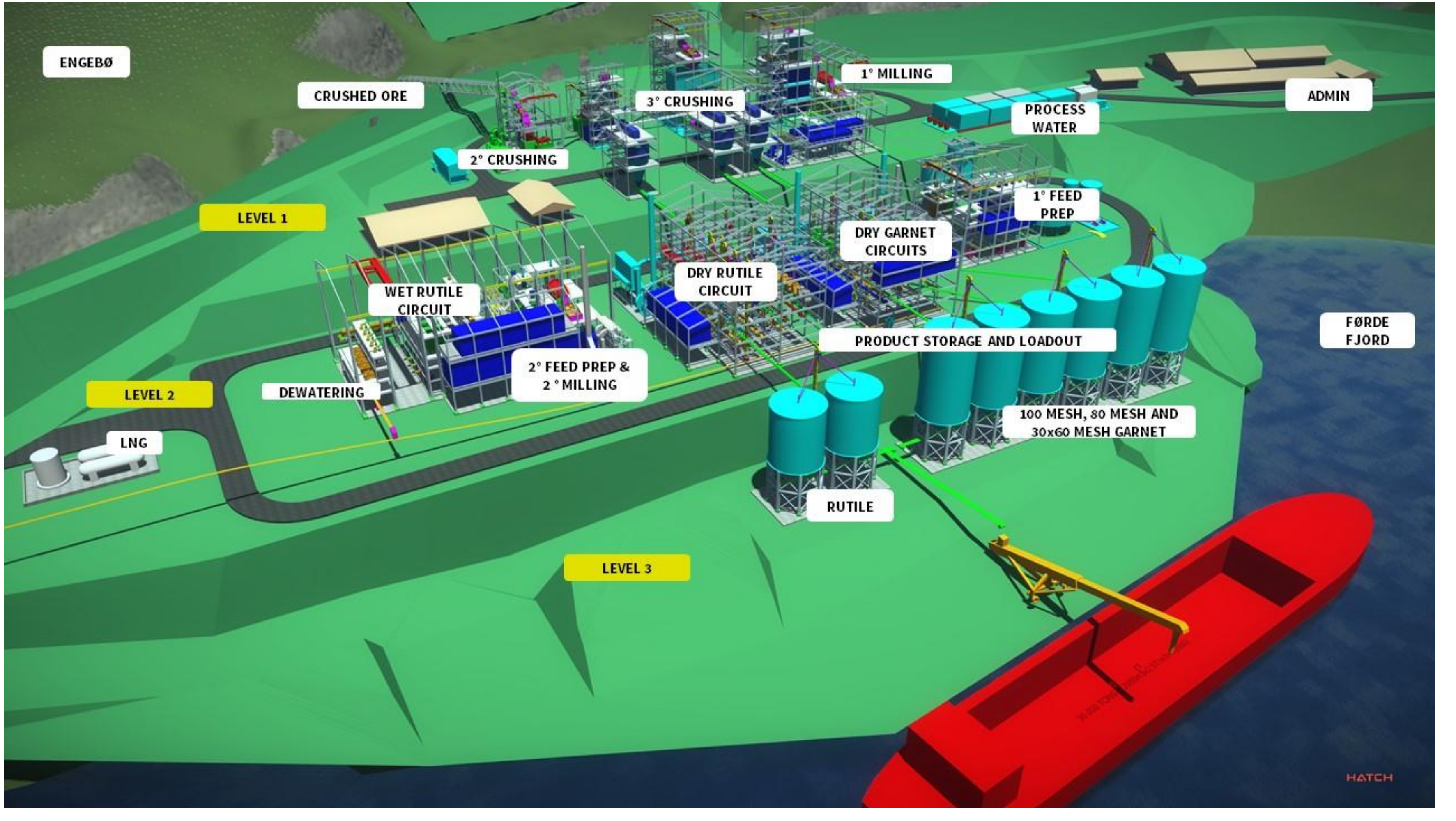


Figure 11-37: Overview of the Proposed Plant Layout

11.9 Future Work Programme

List all metallurgical testwork including any other testwork such as comminution work required to demonstrate ore variability.

Following completion of the extensive testwork programme undertaken as part of the PFS, the following testwork should be undertaken as part of the DFS for the reasons listed below:

- Risk Mitigation – Areas were identified during the PFS that will require additional testwork to deliver technical data to understand ore variability better
- Opportunity Definition – Opportunities were identified during the PFS and will require additional testwork to enable the Project Team to interpret and progress these opportunities into the design.

The following testwork for the comminution and the processing areas was identified:

11.9.1 Comminution Circuit

11.9.1.1 Introduction

Additional comminution testwork will be required in order to improve the understanding of the deposit and the design of the comminution circuit.

11.9.1.2 Material Flow Properties Testwork

Material flow properties testwork are required in order to design correctly chutes, bins, silos and conveyors.

The following testwork is typically required:

- Direct shear test
- Wall friction test
- Compressibility test
- Moisture content test
- Surface roughness test
- Particle size test
- Angle of repose test.

11.9.1.3 Ore Hardness Variability Testwork

Ore hardness variability testwork is required in order to understand the deposit better. Data generated together with the mining plan will be used to design the circuit and predict the performance as a function of the LoM.

The variability assessment will include a spatial variability assessment and a rock type variability assessment. The following ore hardness characterisation testwork will be conducted per sample:

- Uniaxial Compression Strength test
- Bond Crushability Work Index test

- Bond Abrasion Index test
- Bond Rod Work Index test
- Bond Ball Work Index test
- SMC test
- Laboratory batch rod mill testwork at different energy inputs on fresh feed and coarser garnet reject product.

11.9.1.4 Pilot Plant Testwork

The following pilot plant testwork is required:

- Hazemag pilot plant testwork
- Hazemag wear testwork: This testwork will be required to confirm the OPEX. Testwork will conducted to replicate the selected flowsheet. The effect of ore moisture and feed particle size distribution to the Hazemag will also be investigated
- Rod mill pilot testwork at different percentage circulating load
- Rod mill pilot testwork with recovery of coarser garnet on the screen oversize, as shown in Figure 11-38 below.

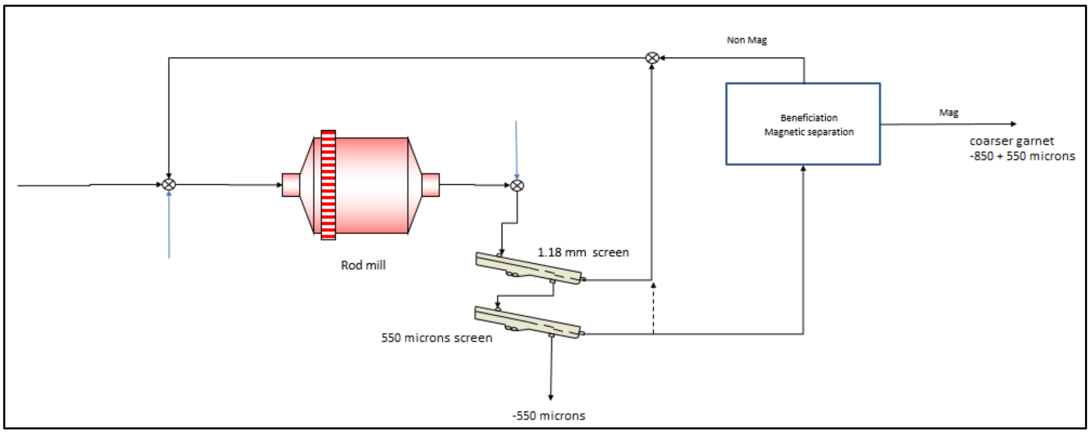


Figure 11-38: Rod Mill in Closed Circuit with a Screen with the Option of Producing a Coarser Garnet (-850/+550 µm) Fraction

11.9.1.5 New Technologies

All comminution technologies investigated during this phase of study cover all mode of crushing and milling that are used in different crushing and milling technologies. It is therefore not expected that a new technology will provide significant benefits in terms of liberation. However, from an operating cost point of view, it will be interesting to monitor the progress achieved by technologies such as the Vero liberator to assess if it can be used in the future in the circuit.

11.9.2 Processing Circuits

During the course of the PFS, four bulk testwork programmes were conducted to develop the process flow sheet and to gain a better understanding of how process parameters and ore characteristics influence material behaviour in the defined processing circuits. These testwork programmes are detailed in Table 11-18 above. During the execution of the PFS, the following testwork was identified to be completed as part of the DFS and is listed in no particular order:

- Bulk sample testwork programmes to confirm metallurgical performance that could be expected from ferro-eclogite ores
- Testwork to determine the optimal screen size to prepare feed for the rutile processing circuit. This testwork programme is specifically developed to deal with the unexpected fine rutile product achieved in the 1308 testwork programme
- Coarse garnet recovery from wet gravity coarse garnet tails and dry coarse garnet circuit tails, QEMSCAN will be required from these streams to reveal where the most liberated coarse garnet is as well as which stream requires milling to 100% passing ~400 µm before a slightly less coarse garnet can be recovered
- Testwork to be specified to determine if the Hazemag and Rod mill -550 µm fractions to be processed separately or combined. Comminution testwork has shown these streams to have different liberation characteristics and there may be benefit in treating these separately
- Reworking of the fine WHIMS to limit the mags tonnage to satisfy the fine garnet production. Screening may have limited success due to it being a difficult screening application and a small mass will be rejected to undersize. This work may lead to not requiring the fine garnet gravity circuit
- From the comminution variability testwork, glean data to define the variability in the coarse circuit PSD better. In particular, define the variability in the mass balance due to misplaced -212 µm material due to expected screening efficiencies and or variation in the proportion of near size material. This will directly impact the requirement for design factors to ensure equipment loadings remain in line with testwork
- In line with the point above, glean data from the comminution testwork results to define the variability of the crushed and milled ore as a function of the comminution process and as a result of ore hardness
- Develop laboratory procedures to mimic the currently developed process to determine key performance parameters from drill core samples. This work will be key to kick off the geo-metallurgical testwork programme
- Based on the above, develop a geo-metallurgical testwork programme to define the correlation between the geology and metallurgy of the processing circuits. This work programme will be vital to predict metallurgical performance based on the ore geology.

12. Mining

12.1 Introduction

This section summarises the mining work undertaken during the PFS. The key mining related work packages undertaken are as follows:

- Development of a mining block model to be used as the basis for the open pit and underground mine design
- Pit optimisation studies
- Open pit mine design
- Underground mining method selection
- Underground mine design (including underground infrastructure)
- Incorporation of updated from ongoing process testwork and other studies into mine optimisation
- Plant capacity determination
- Production schedule, including equipment selection, operating and capital cost estimates.

Subsequent to completion of most of the mining studies in this phase, financial modelling of a number of mining options using updated OPEX and CAPEX numbers indicated that the development of a high-grade option to mine and process ferro ore only at a capacity of 1.5 Mtpa provided a better IRR than options which mined both trans ore and ferro ore. One of the main factors driving the value of the ferro ore option was the change in understanding of the garnet contribution to the Project revenue; recent testwork results indicated that the expected garnet recovery for ferro ore would be significantly higher than for trans ore, which changed the Project economics. For these reasons, the final mine plan developed considered the mining and processing of ferro ore only, with trans ore being treated as waste. The bulk of the write-up in this section of the report reflects early work carried out before the Project decision was made to exclude trans ore.

12.2 Open Pit Mining Block Model Development

12.2.1 Definition of Mining Model Shape for the Open Pit

The first step in the development of the open pit block model was to regularise and reconcile the block model received to 5 m by 5 m by 15 m (x, y and z) blocks. This resizing of the blocks allowed for more efficient design in terms of both processing power and geometry.

Figure 12-1 below provides an isometric view of the block model as supplied of the Engebø deposit, coloured on TiO_2 grade.

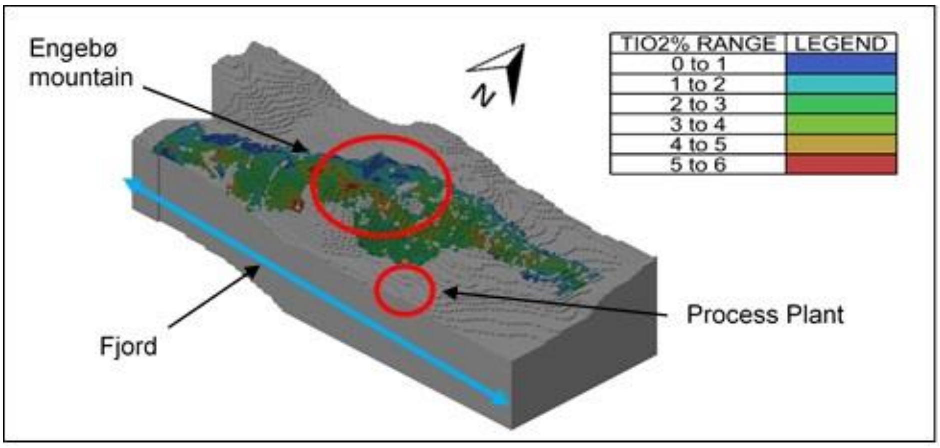


Figure 12-1: Initial Geological Block Model showingTiO$_2$ Grade

After regularisation the distribution of garnet and TiO$_2$ grades in the block model is shown in Figure 12-2 below.

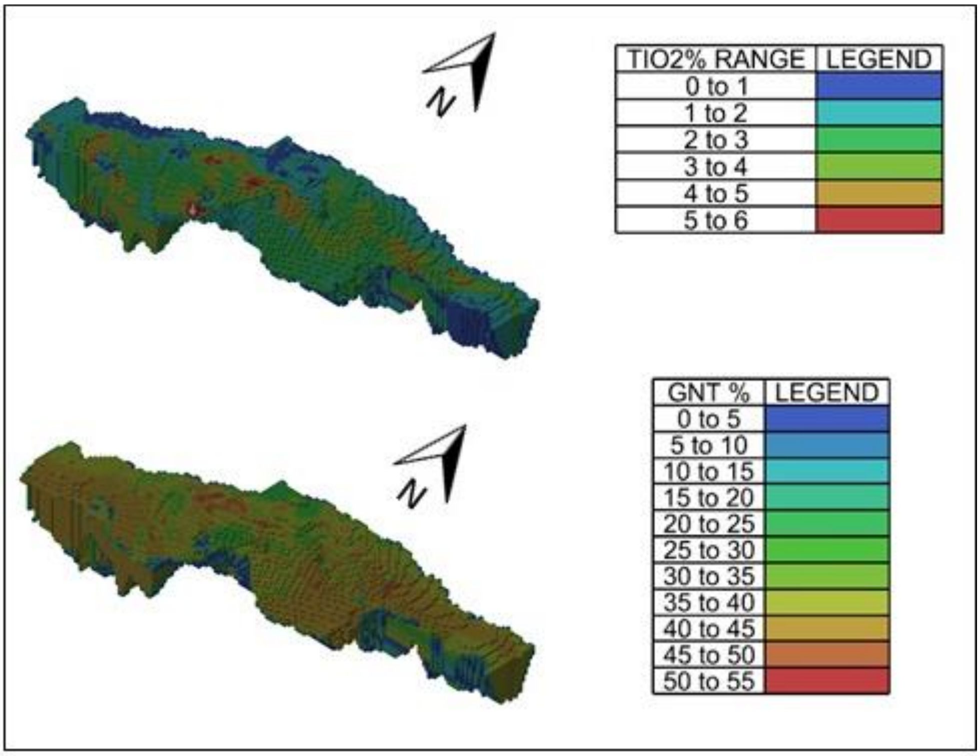


Figure 12-2: Block Model Grades

The next step in the design process was to filter the block model on material class to target the mining area to ensure that only mineral resources in line with the JORC guidelines were used as the basis of the mine design.

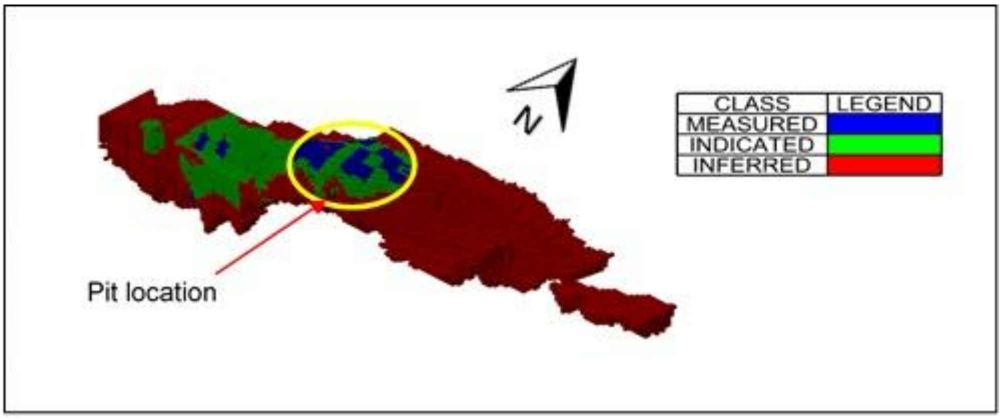


Figure 12-3: Orebody Block Model Filtered on JORC Material Class

The above model formed the basis for the pit and underground design used in the study.

12.3 Pit Optimisation Studies

12.3.1 Introduction

The following sections summarise the open pit optimisation analysis undertaken and the recommendations which resulted from the work in terms of a practical pit design to be used as the basis of the PFS mine plan.

12.3.2 Background

The Project is being developed in line with the guidelines of the JORC Code. However, opportunities have been investigated outside of the JORC standard to include Inferred Resources, and are reported separately. This is to support project decisions related to exploration, mine capacity, processing, infrastructure and the market. However, only the open pit optimisation studies undertaken to support a mine plan which corresponds to the guidelines of the JORC Code are discussed in this section.

It is foreseen that two products will be produced from the Engebø deposit, namely rutile concentrates and garnet. Mining will commence from an open pit which will provide access to an underground mine. Ore will be drilled, blasted, loaded and hauled to a glory hole (ore pass) in the pit, after which it will undergo primary crushing underground. Thereafter, the ore will be transported via a conveyor belt to the secondary and tertiary crushing facilities in the processing area next to the deposit. Finally, processing of the ore to extract rutile and garnet products will take place. Waste will be stripped and hauled to a waste disposal site to the north-east of the pit. Three ore types have been modelled, namely:

- Ferro-eclogite, which generally contains >16% Fe_2O_3 and >3% TiO_2
- Trans-eclogite, which generally contains 14% to 16% Fe_2O_3 and 2% to 3% TiO_2
- Leuco-eclogite, which generally contains <14% Fe_2O_3 and <2% TiO_2.

Open pit optimisation formed the initial step within the reserve estimation process. During the early stages of the Project, limited information and resolution was available, which impacted negatively on the pit optimisation process. The following limitations had a significant impact on the open pit optimisation:

- Permitting limitations: to limit the visual impact of the open pit, a permitting string as supplied by Nordic Mining defined the outermost physical limit of the pit. The final pit designs mined to the permit string in the eastern, western and southern directions; it is only in the northern direction that the pit does not extend to reach the permitting string
- Capacity decision: the capacity options investigated range from 1.0 Mtpa to 4.0 Mtpa RoM ore processed through the plant. For this range, the mining Operating Expenditure (OPEX) varies by 82% per RoM tonne from the low to high capacity (US$ 4.6/ROM t to US$ 2.6$/ROM t respectively)
- Market demand: the selling of two products with a limited understanding of garnet market volumes and prices. The Project's view on the garnet price dropped over the duration of this study phase (from US$ 300/t of product to an interim price of US$ 220/t, with a final price of US$ 250/t). The first optimisation runs considered 100 ktpa of garnet sales, after which 200 ktpa was considered before the most recent runs which used 300 ktpa garnet as maximum sales per year. The average over the LoM is 250 ktpa with a total of almost 8 Mt that can be placed in the market
- Metallurgical results: initially, the recovery for garnet was not well defined; in addition, the lower limit cut-off grade for TiO_2 was not well understood in terms of metallurgical recovery. The current view from a process perspective of the lowest grade ore which can be treated varies from 1% to 2% TiO_2; this implies that ore below a 2% TiO_2 grade will have a very unpredictable recovery. A lower cut-off grade limit for TiO_2 of 1% has been used in open pit optimisation. This has a significant impact on the waste: ore definition based on cut-off grade; there could be a significant number of blocks that are economical based on garnet content that could be classified as waste
- It became evident during the study that garnet, and in particular recovery of coarse garnet, plays a significant role in improving the business case.

To accommodate these limitations, a first pass iteration process was adopted consisting of reconciliation, design sensitivities, ultimate pit sensitivities and pushback pit optimisation. The iteration process incorporated:

- Owner's boundary limit
- Geotechnical design specifications
- Rock types
- Consideration of Measured and Indicated Resources, and exclusion of Inferred Resources
- Rutile cut-off grade
- Mining method
- Mining factors for dilution and recovery
- Mine design and pit access implications

- Mining equipment size and operability
- Mine infrastructure and waste rock disposal location
- Process and infrastructure cost.

12.3.3 Approach

The approach followed to define the ultimate pit, initial pushback and design criteria is laid out in Figure 12-4 below. The main aspects of the approach are described in more detail within this report. During the evaluation process, various ultimate pit shells were developed to evaluate the sensitivity of the mentioned limitations.

The first pass iteration compared this study's optimisation model with the pit optimisation reported as part of the Resource Estimate report conducted by Mr. Wheeler and reported within the Technical Report – Resource Estimation for the Engebø Deposit.

During the second pass, the aim was to obtain the ultimate pit shell considering additional detail. The base case input parameters were adjusted to incorporate:

- A ramp design: this was done by adjusting the overall slope angle to accommodate various haul truck sizes
- Update operational cost over the total value chain.

In addition to the above changes, a cut-off-grade analysis has been conducted on both JORC and opportunity business cases.

The final pass iteration focused on pushback analysis. The aim was to optimise the business case focusing on JORC requirements. The main aspect evaluated was the influence of resource classification on the business case, as well as a strategy to defer waste stripping.

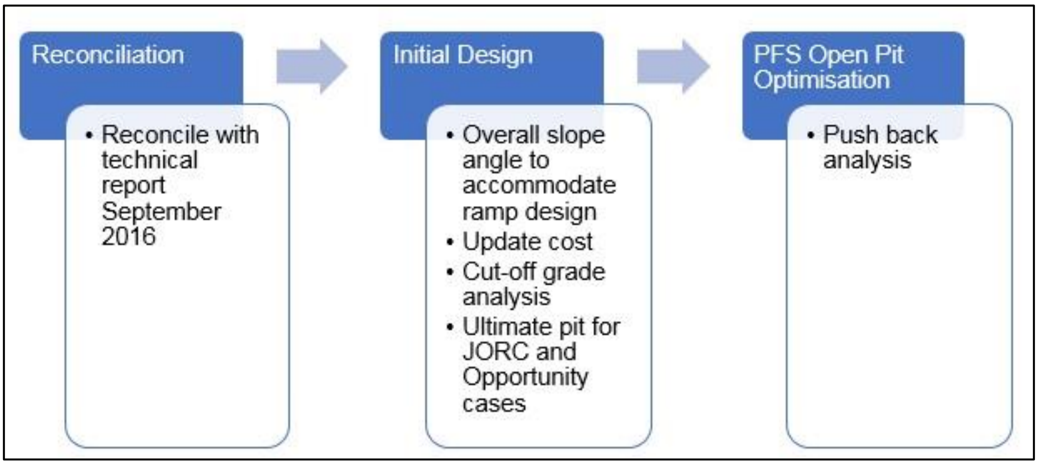


Figure 12-4: Open Pit Optimisation Iteration Steps

12.3.4 *Options Evaluated*

The following main pit optimisation cases were evaluated:

- Option 1: reconciliation between the optimisation model developed for this study and the model defined for the Resource Estimation report. Run 3 from the above report has been used for reconciliation purposes
- Option 2: incorporation of a ramp design within Option 1 by adjusting the overall slope angle to accommodate 90 t haul trucks (CAT 777 or similar), in line with the anticipated equipment which would be used on a deposit of the size and configuration of Engebø
- Option 3: an update of operational cost and recovery factors based on PFS information including mining, processing, product handling and overheads
- Option 4: a cut-off grade analysis. Cut-off grade analysis (decision for rock to be classified either as ore or waste) has been conducted based on TiO_2 grade values
- Options 4b: recovery sensitivity analysis. Garnet recovery has not been well defined. To investigate the effect of garnet recovery on the ultimate pit shell, recoveries per ore rock type have been evaluated
- Option 5: a final ultimate pit shell in line with the JORC Code guidelines. The aim was to include up to 10% non-Measured and Indicated Resources within the initial open pit design and schedule – this amount was considered at the time to be an acceptable upper limit to develop a mine plan which corresponded to the guidelines of the JORC code; the Project Team subsequently agreed to consider only Measured and Indicated Resources in the mine plan
- Option 6: consider ferro-eclogite as the primary ore type with high-grade transitional-eclogite and remove leuco-eclogite as ore. This option has been included to verify the impact on metallurgical results received towards the end of the PFS as well as to investigate options to improve the Project's Internal Rate of Return (IRR).

12.3.5 *Inputs*

The open pit optimisation was conducted using the 2016 regularised resource model supplied by the client. NPV Scheduler was used to perform the open pit optimisation. A base case pit design was created using DESWIK software, classifying ore based on Measured and Indicated resources for rock type 1, 2 and 3. Rock types 4 to 8 were classified as waste, together with all ore blocks below the specified cut-off grade. The classification of the above rock types in the resource model is summarised in Table 12-1 below.

Table 12-1: Rock Type Classifications in Resource Model

Rock Type	Category	Description
1	Leuco-eclogite	<14% Fe_2O_3 and <2% TiO_2; often light coloured, but can be dark green; often more coarser grained
2	Transitional-eclogite	14% to 16% Fe_2O_3 and 2% to 3% TiO_2; a mix between ferro and leuco, no clear boundary, a transitional change
3	Ferro-eclogite	>16% Fe_2O_3 and >3% TiO_2; often dark and fine grained, often has a homogenous appearance; abundant garnet and rutile
4	Amphibolite	Homogenous, no banding; moss green with no garnets
5	Garnet Amphibolite	Homogenous, no banding; darker green than eclogite
6	Gneiss (including felsic rocks)	Usually internal zones within main eclogite body; quartz vein like
7	Alternating mafic and felsic rocks	Usually country rock surrounding main eclogite body; mixing of mafic and felsic rocks
8	Quartz	Massive quartz vein of more than 1 m

The open pit optimisation runs aligned to the JORC constraints (i.e. Inferred Resources classified as waste) were limited within the perimeter for which the owner has permission to evaluate as a potential open pit.

The following table provides the pit optimisation parameters used during this study:

Table 12-2: Open Pit Optimisation Input Parameters

Parameter		Unit	Option 1	Option 2	Option 3	Options 4 and 5
Prices	TiO_2	US$/t product	1,000	1,000	1,050	1,050
	Garnet	US$/t product	300	300	220	220
Mining OPEX	Waste	US$/t rock	2.64	2.64	2.5	2.5
	Ore	US$/t ore	2.64	2.64	2.1	2.1
Processing OPEX	Direct	US$/t ore	9.14	9.14	10.00	10.00
	G&A	US$/t ore	0.85	0.85	In Processing OPEX	In Processing OPEX
	Shipping TiO_2	US$/t product			3.00	3.00
	Shipping Garnet	US$/t product			3.00	3.00
	Total	US$/t ore	10.00	10.00	10.30	10.30
Mining Factors	Dilution	%	5	5	5	5
	Recovery	%	95	95	95	95
Cut-Off Grades	TiO_2	%	2.0	2.0	2.0	0 1.5 2.0 2.5
Process Factors	TiO_2 Recovery	%	55	55	55	55
	Garnet Recovery	%	16	16	21.2 (Leuco & Trans), 33.5 (Ferro)	21.2 (Leuco & Trans), 33.5 (Ferro)
Discount Rate		%	10	10	8	8
Overall Slope Angles*	0	°	56	49.72	49.72	49.72
	90	°	55	45.20	45.20	45.20
	180	°	55	49.14	49.14	49.14
	270	°	59	50.23	50.23	50.23

*See Figure 12-6 for overlay of main quadrant (above) and sector specific values.

12.3.6 Open Pit Optimisation Results

12.3.6.1 Option 1: Reconciliation

A representative block model was created for the pit optimisation study with the input parameters provided for Option 1. The following table provides the results created within the open pit model created to replicate the Resource Estimate optimised pit. The models reconciled to each other to within 1%, which is an acceptable variance for a PFS. No Net Percent Value (NPV) has been reported within the Resource Estimate. However, based on this study the Option 1 open pit optimisation provided an NPV of US$ 290.9 M and total rock tonnes of 60.7 Mt.

Table 12-3: Reconciliation Summary

Option	Reference	NPV	Profit	Revenue	Processing Cost	Mining Cost	Rock	Ore	Waste	TiO_2 Product	Garnet Product
		US$ M	US$ M	US$ M	US$ M	US$ M	Mt	Mt	Mt	Mt	Mt
RE	Run 3, RE	-	419.8	958.0	375.5	162.7	61.6	37.6	24.1	0.7	0.9
1	Pits_28	290.9	417.1	953.1	375.7	160.2	60.7	37.7	23.0	0.7	0.9
Comparison	Pits_28: Run 3		99%	99%	100%	98%	99%	100%	96%	100%	99%

The following figures provide plan, North-South and East-West section views for Option 1.

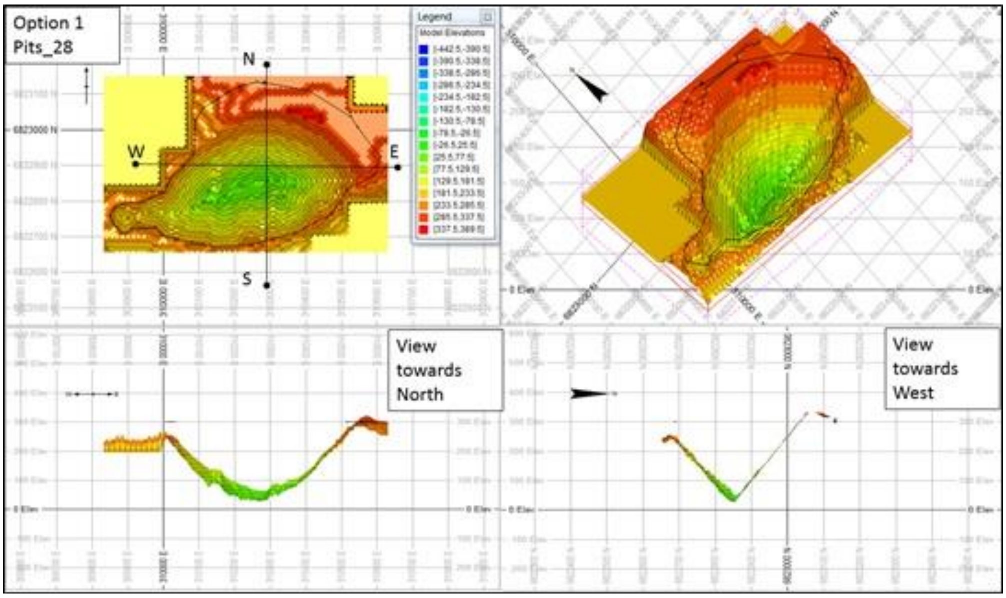


Figure 12-5: Open Pit Visuals - Options 1

12.3.6.2 *Option 2: Incorporation of Ramp Design into Option 1*

The ultimate pit in Option 1 was derived from the geotechnical recommendation for overall slope angles. These overall slope angles did not incorporate a ramp design. Two shortfalls of this approach are, firstly, that an optimistic business case is presented and secondly, that the detailed pit design normally goes outside of the optimised pit shell to accommodate the ramp design.

The following process steps were taken to obtain realistic overall pit slopes that comply to both the geotechnical design recommendations and detailed pit design requirements:

- Export of the ultimate pit shell to DESWIK to create a high-level pit design that included a ramp per haul truck size alternative
- At this stage of the PFS, the final haul truck size had not been selected; ramp designs for five haul truck sizes were constructed, therefore, to accommodate several possible truck sizes
- The overall slope angles for the major geotechnical sectors were measured and the NPV Scheduler slope settings were adjusted accordingly

- This process was repeated three times until minor adjustments were required when constructing the pit design.

Table 12-4 below provides the ramp design specifications based on the respective haul truck sizes:

Table 12-4: Open Pit Ramp Design Specification per Haul Truck

Truck	Capacity (t)	Width (m)	Dual Traffic Passing Width (m)	Tyre Size	Tyre Height Berm (m)	Drainage (m)	Barrier Width (Tyre Height Berm plus Drainage Berm (m))	Ramp Width (m)
ADT 740	40	4.2	12.6	29.5R25	1.9	2	3.9	17
CAT 775	60	5.8	17.4	24.00R35	2.2	2	4.2	22
CAT 777	90	6.1	18.3	27.11R49	2.6	2	4.6	23
CAT 785	130	6.7	20.1	33.00R51	3	2	5	26
CAT 789	180	7.7	23.1	37.00R57	3.4	2	5.4	29

Table 12-5 below provides the adjusted overall slope angles per major geotechnical sector for the 90 t haul truck type (CAT 777):

Table 12-5: Adjusted Overall Slope Angles

Sector	Orientation	Max Slope (degrees)	Bench Height (m)	Berm Width (m)	Bench Face Angle (degrees)	Ramp Width (m)	Inter Ramp Angle with-Out Ramp (degrees)	Inter Ramp Angle with Ramp (degrees)	Overall Slope Angle (degrees)	Overall Slope Angle (degrees)	Overall Slope Angle (degrees)
									Step 1	Step 2	Step 3
North	327.8-21.3	270	15	5	85	23	67.18	27.10	67.18	51.68	51.68
North East	21.3-88.6	260	15	5	68	23	53.60	23.77	53.60	47.77	47.77
South East	88.6-151.2	145	15	5	68	23	53.60	23.77	53.60	45.20	45.20
South	151.2-253.9	215	15	5	68	23	53.60	23.77	53.60	45.20	49.14
North West	253.9-327.8	180	15	5	74	23	58.20	24.91	58.20	48.79	48.79

Figure 12-6 below provides the major geotechnical sector definition as recommended by WAI. These sectors have been used for Options 3, 4 and 5. Overlaid in red are the four major sectors used for Options 1 and 2.

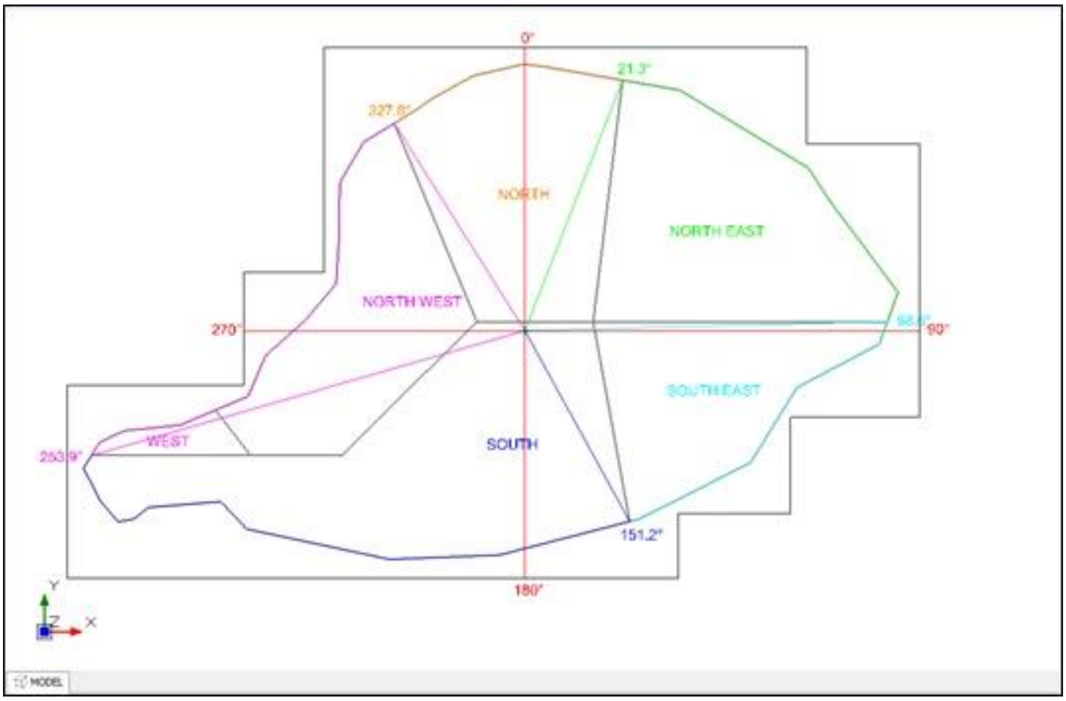


Figure 12-6: Geotechnical Slope Sector Design

Table 12-6 below shows the results generated within the open pit model created to replicate the Resource Estimate optimised pit incorporating a ramp design for the CAT 777 haul truck. The ramp design reduced the total rock tonnes mined to 49.7 Mt and generated an NPV of US$ 253.4 M.

Table 12-6: Impact of Ramp Design on Optimisation

Option	Reference	NPV	Profit	Revenue	Processing Cost	Mining Cost	Rock	Ore	Waste	TiO_2 Product	Garnet Product
		US$ M	US$ M	US$ M	US$ M	US$ M	Mt	Mt	Mt	Mt	Mt
1	Pits_28	290.9	417.1	953.1	375.7	160.2	60.7	37.7	23.0	0.7	0.9
2	Pits_79	253.4	341.3	778.0	305.4	131.3	49.7	30.6	19.1	0.6	0.7
Comparison	Pits_79Pits_28	87%	82%	82%	81%	82%	82%	81%	83%	82%	81%

Figure 12-7 below shows plan, North-South and East-West section views for Option 2.

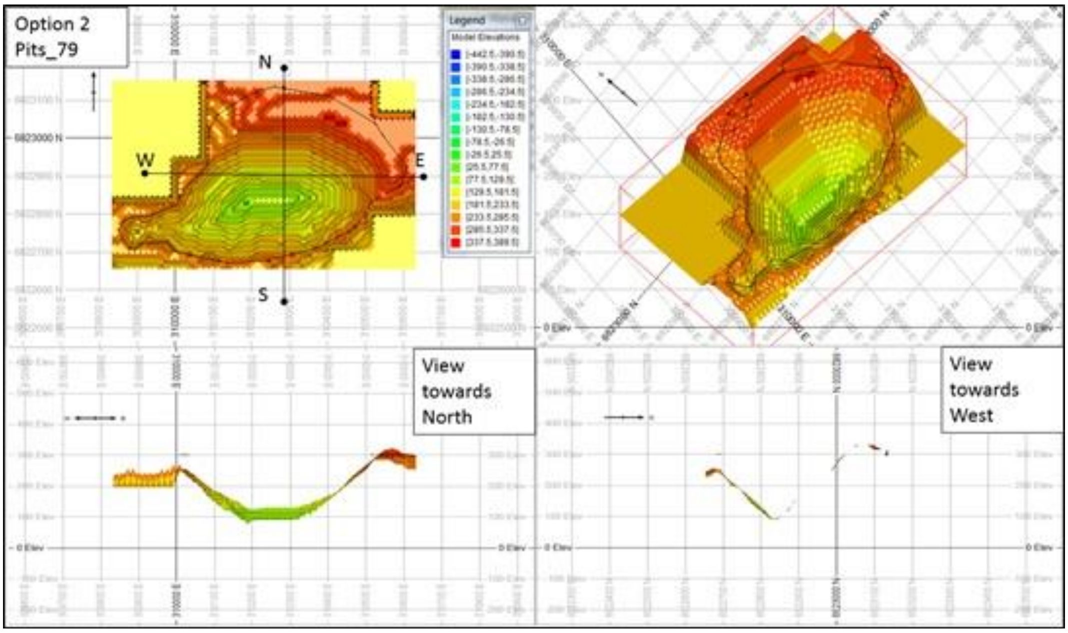


Figure 12-7: Open Pit Visuals - Options 2

12.3.6.3 *Option 3: Updated Information Generated During the PFS Phase*

During the optimisation phase of the PFS, the following updated information was received:

- Product prices: a rutile sales price US$ 1,050/t, a garnet sales price of US$ 220/t
- Garnet sales volume: the view on the garnet market volume increased from 100 ktpa to 300 ktpa at steady state from 2029 onwards. This implies that at the lower expected garnet recoveries, little to no garnet disposal will occur. The total market volume for the open pit period is estimated at 8 Mt
- Mining OPEX: a quotation was obtained from a large mining contractor with operations in Norway. The pricing obtained from the contractor was subsequently benchmarked from first principles for owner operation. Waste stripping costs are estimated at US$ 2.5/t and ore mining at US$ 2.1/t. Ore mining unit costs are lower due to the short and horizontal hauling to the RoM glory hole which will be accessible from each bench
- Product distribution: distribution unit cost for both products are estimated at US$ 3/t Free on Board based on bulk sales
- Garnet recovery: low confidence exists pertaining to garnet recovery of leuco-eclogite and transitional-eclogite. A recovery of 22.1% is expected for both rock types. Higher confidence exists for the garnet associated with ferro-eclogite for which a recovery of 33.5% is expected (note: these recoveries were subsequently amended later in the study once more testwork results became available)
- TiO_2 cut-off grade: moderate confidence existed for TiO_2 except for lower cut-off grade values. A 2% cut-off grade was, therefore, used as a lower limit.

Table 12-7 below shows the results for Option 3. The total rock mined increased to 59.6 Mt and the NPV to US$ 580.7 M.

Table 12-7: Optimisation Results - Option 3

Option	Reference	NPV	Profit	Revenue	Processing Cost	Mining Cost	Rock	Ore	Waste	TiO_2 Product	Garnet Product
		US$ M	US$ M	US$ M	US$ M	US$ M	Mt	Mt	Mt	Mt	Mt
1	Pits_28	290.9	417.1	953.1	375.7	160.2	60.7	37.7	23.0	0.7	0.9
3	Pits_40	580.7	984.0	1,440.7	329.2	127.5	59.6	31.7	27.9	0.6	3.8
Comparison	Pits_40: Pits_28	200%	236%	151%	88%	80%	98%	84%	121%	84%	417%

Figure 12-8 below shows plan, North-South and East-West section views for Option 3.

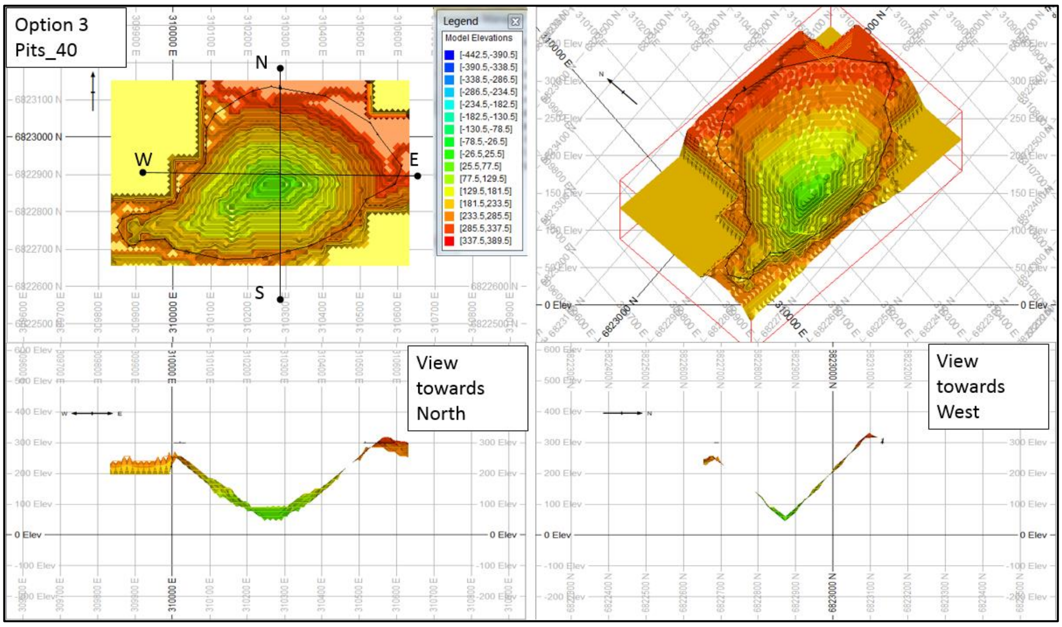


Figure 12-8: Open Pit Visuals - Option 3

12.3.6.4 *Option 4: Cut-off Grade Analysis*

Cut-off grade analysis based on TiO_2 was conducted for two scenarios. In both scenarios, the TiO_2 cut-off grades were calculated from 0% to 2.5% at increments of 0.5%.

- Option 4a: evaluate the impact of TiO_2 cut-off grade whilst keeping all other inputs the same as for Option 3
- Option 4b: evaluate the impact of TiO_2 cut-off grade whilst reducing garnet recovery for the respective ore rock types. A zero garnet recovery for leuco-eclogite was used. A trans-eclogite yield of 11% and a ferro-eclogite yield of 18.3% was used.

Table 12-8 below shows the results for Option 4a. Total rock mined remained at 59.7 Mt up to a TiO_2 cut-off grade of 1.5%, where after the NPV dropped for higher TiO_2 cut-off grades. The corresponding NPV was US$ 580.8 M.

Table 12-8: Optimisation Results - Cut-off Grade Sensitivity - Option 4a

Option	Reference	Cut-off Grade	NPV	Profit	Revenue	Processing Cost	Mining Cost	Rock	Ore	TiO_2 Product	Garnet Product
		%	US$ M	US$ M	US$ M	US$ M	US$ M	Mt	Mt	Mt	Mt
4a	Pit_49	0	580.8	1,003.1	1,471.5	340.8	127.7	59.7	32.8	0.6	3.9
4a	Pit_54	1	580.8	1,003.1	1,471.5	340.8	127.7	59.7	32.8	0.6	3.9
4a	Pit_59	1.5	580.8	1,003.1	1,471.5	340.8	127.7	59.7	32.8	0.6	3.9
4a	Pit_64	2	580.7	984.0	1,440.7	329.2	127.5	59.6	31.7	0.6	3.8
4a	Pit_104	2.5	564.2	897.3	1,308.0	284.9	125.8	58.8	27.4	0.5	3.5

Table 12-9 below shows the results for Option 4b. Total rock mined dropped to 54.8 Mt up to a TiO_2 cut-off grade of 2%. The corresponding NPV was US$ 361.6 M; the main reason for the drop in NPV in comparison to Option 4a was the reduced garnet recoveries (yields). There was a slight improvement, 0.1%, in NPV at a TiO_2 cut-off grade of 2%; however, for higher cut-off grades the NPV dropped.

Table 12-9: Optimisation Results - Cut-off Grade Sensitivity - Option 4b

Option	Reference	Cut-off Grade	NPV	Profit	Revenue	Processing Cost	Mining Cost	Rock	Ore	TIO_2 Product	Garnet Product
		%	US$ M	US$ M	US$ M	US$ M	US$ M	Mt	Mt	Mt	Mt
4b	Pit_84	0	361.6	609.6	1,056.5	329.9	117.0	54.8	32.3	0.6	2.1
4b	Pit_89	1	361.6	609.6	1,056.5	329.9	117.0	54.8	32.3	0.6	2.1
4b	Pit_94	1.5	361.6	609.6	1,056.5	329.9	117.0	54.8	32.3	0.6	2.1
4b	Pit_99	2	362.1	601.3	1,037.5	319.3	116.8	54.8	31.2	0.6	2.0
4b	Pit_109	2.5	354.9	554.6	945.8	276.7	114.5	53.7	27.0	0.5	1.9

Figure 12-9 below shows a plan view of the pit at varying cut-offs for Option 4a.

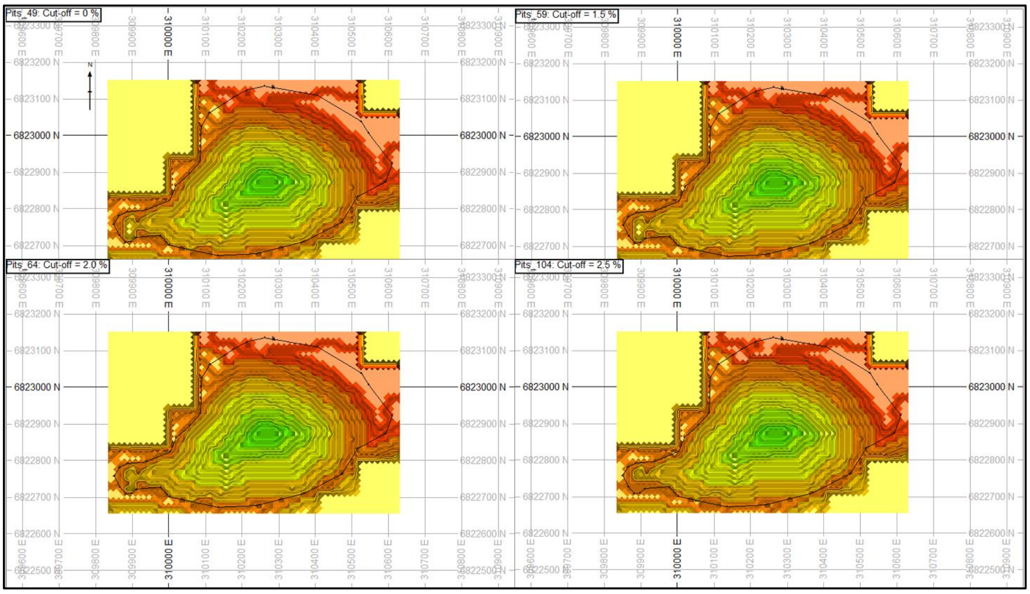


Figure 12-9: Open Pit Visuals - Option 4a

Figure 12-10 below shows a plan view of the pit at varying cut-offs for Option 4b.

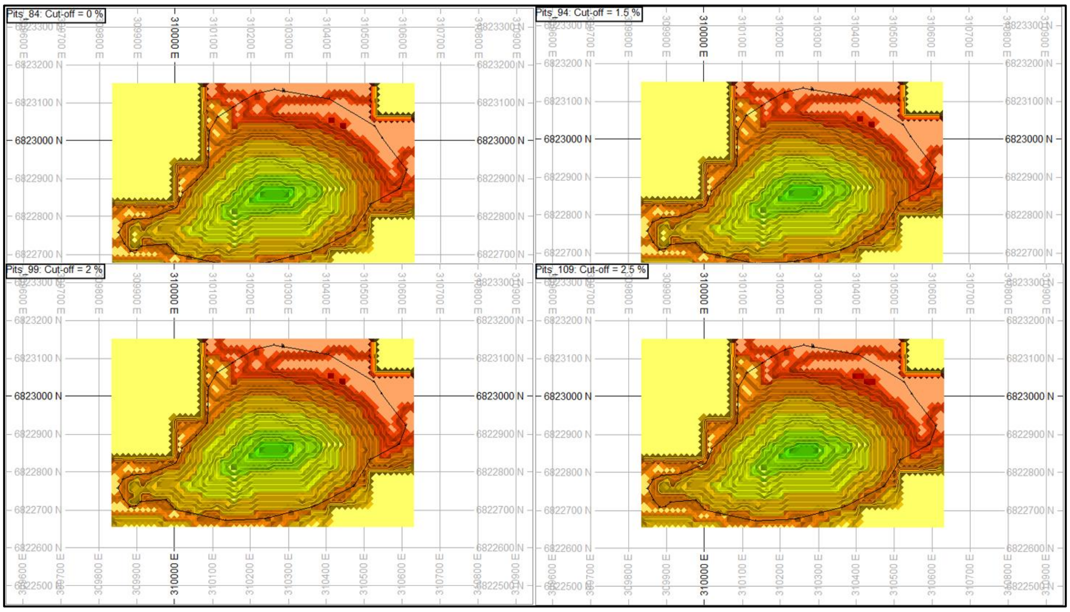


Figure 12-10: Open Pit Visuals - Option 4b

12.3.6.5 Option 5: Pushback Strategy

In Option 5, a pushback strategy was developed for Option 4b at a TiO_2 cut-off grade of 1.5%. Three pushbacks of similar size, with an average rock mined of 18 Mt per pushback, were created. The aim with the pushback strategy was to reduce the stripping ratio in the first few years. In practice, however, only two pushbacks were used with pushbacks 1 and 2 being combined to incorporate the isolated portion on the south-west side of the pit which was not considered to be practically mineable.

Figure 12-11 below shows a plan view and section view towards the North for the three pushbacks created. On the plan views shown on the left-hand side, the blue colour indicates the active mining area. On the section views shown on the right-hand side, the red colour indicates the active mining area.

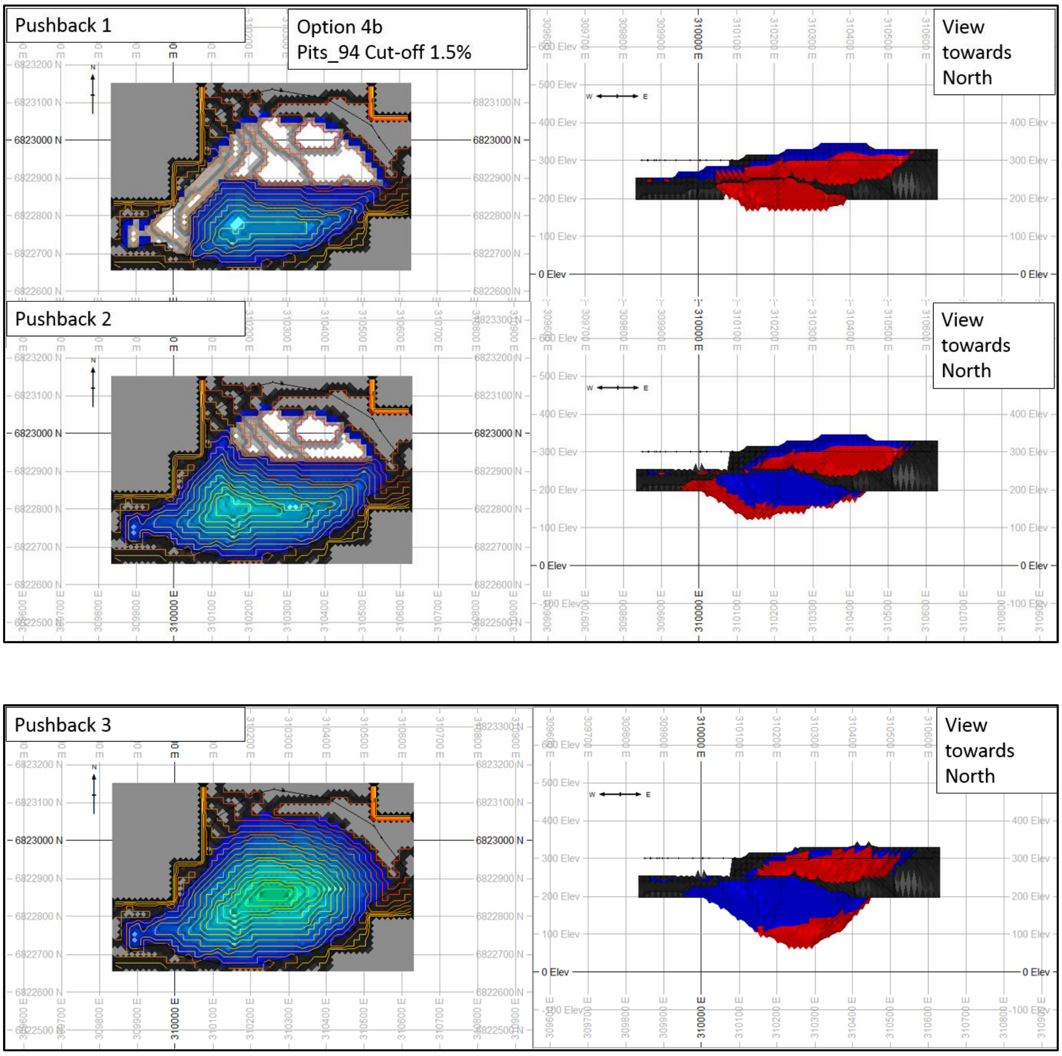


Figure 12-11: Pushback Visuals - Option 4b

12.3.6.6 *Option 6: Ferro and High-grade Trans Ore*

Towards the end of the PFS phase, metallurgical results were received that indicated low garnet yields, nominally less than 7%, could be expected when leuco-eclogite is processed. Consequently, a Project decision was taken to classify leuco-eclogite as waste and at the same time to evaluate options for a high-grade garnet and TiO_2 operation. The objectives of this evaluation were twofold; firstly, to identify options that would improve the IRR of the Project and secondly, to minimise the loss of value (in terms of NPV and the LoM) due to ore being classified as waste.

The following cut-off assumptions were, therefore, applied to Option 6:

- Leuco-eclogite was classified as waste, irrespective of its TiO_2 grade
- Trans-eclogite was classified as ore if its TiO_2 grade exceeded certain cut-offs, evaluated in increments of 0.5% from 0% to 3.0%
- For ferro-eclogite, no cut-off grade was applied to either TiO_2 or garnet.

Table 12-10 shows the results obtained from the NPV Scheduler runs.

Table 12-10: Optimisation Results - Cut-off Grade Sensitivity - Option 6

Option	Reference	Cut-off Grade	NPV	Profit	Revenue	Processing Cost	Mining Cost	Rock	Ore	TIO_2 Product	Garnet Product
		%	US$ M	US$ M	US$ M	US$ M	US$ M	Mt	Mt	Mt	Mt
6	Pit_134	0	279.3	557.6	995.8	324.1	114.1	53.5	31.7	0.6	1.7
6	Pit_129	1.5	279.3	557.6	995.8	324.1	114.1	53.5	31.7	0.6	1.7
6	Pit_119	2	294.2	567.9	995.3	313.3	114.0	53.5	30.6	0.6	1.7
6	Pit_124	2.5	318.1	548.4	898.0	236.2	113.4	53.2	23.0	0.5	1.7
6	Pit_139	3	318.7	548.9	897.0	235.9	112.1	52.6	22.9	0.5	1.7

Figure 12-12 below provides the cumulative NPV over the first five years for cut-offs varying from 0% to 3% for transitional-eclogite. The graph indicates potential impact of cut-off on IRR; since all options have the same capital (same capacity), the steeper angle options (2.5% and 3.0%) indicate potential improvements in IRR.

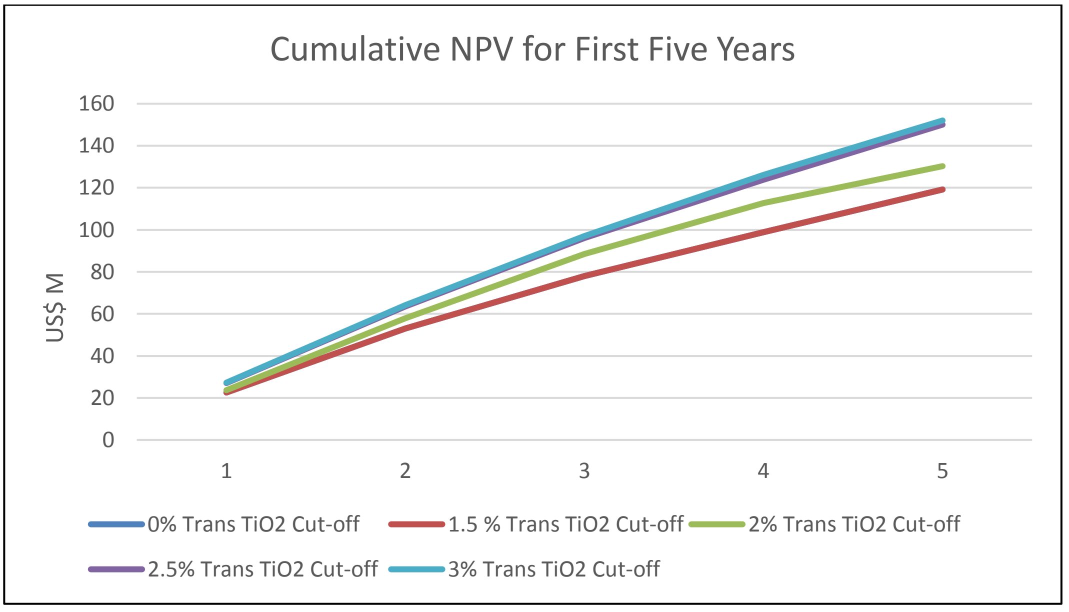


Figure 12-12: Cumulative NPV for Various Cut-off Grades – Option 6

It can be seen from Table 12-10 and Figure 12-12 above that selecting a 2.5% or 3% cut-off generates the best NPVs. It is recommended that a 2.5% cut-off is selected in this case to ensure that the ore tonnes mined and NPV are both maximised.

Figure 12-13 below indicates the ultimate pit shell for the transitional-eclogite for TiO_2 cut-off grades varying from 1.5% to 3.0%.

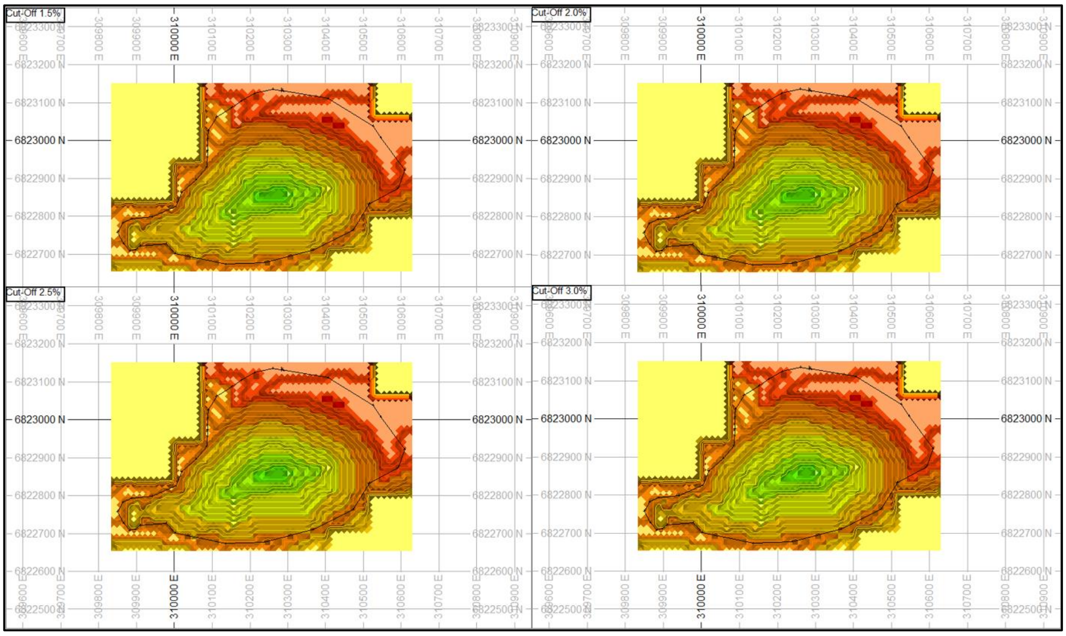


Figure 12-13: Option 6 - Pit Shells for TiO_2 Cut-offs of 1.5%, 2.0%, 2.5% and 3%

It can be seen from Figure 12-13 above that changes to the pit outline are minimal when moving from a 1.5% TiO_2 cut-off to 3% TiO_2 cut-off for trans-eclogite ore.

12.3.7 Results and Recommendation

The reconstructed model reconciles with the Resource Estimate report. Once the ramp was included in the open pit optimisation process to accommodate a CAT 777 truck, the total rock within the Resource Estimate pit dropped by approximately 10 Mt. During the next study phase, the economic impact of using smaller equipment or in-pit conveying alternatives should be investigated.

In all the options investigated, the pit boundaries to the southern, western and eastern sides are consistent and fully extend to the owner's boundary limit. The major contributors to the ultimate pit shell within these sectors are the owner's boundary limit, geotechnical recommendations and ramp dimension. The northern pit boundary, however, varies in each optimisation run as a result of metallurgical and economic considerations that result from unit cost and revenue related aspects. The northern sector consists mainly of leuco-eclogite ore. Uncertainty exists pertaining to the metallurgical properties of this ore and further test-work is required before the ore classification within this sector is improved.

Garnet contributes significantly to the economic value of the Project, with additional market volumes the major contributor. Any additional sales will have a significant positive impact on the Project. It should be noted that garnet market volumes do not alter the shape of the ultimate pit.

No significant impact was observed by increasing the TiO_2 cut-off grade from zero to 2%. However, it was noted that above a 2% TiO_2 cut-off grade the business cases are negatively impacted. This is due to the positive contribution of garnet.

It is recommended that the pit shell derived from Option 4b (TiO_2 cut-off grade 1%) is taken forward as the basis of the pit design; the pit shell for this option has been termed Pit 1. Within the Pit 1 shell, two major pushbacks have been designed (Option 5) to improve the project economics with pushback 2 containing predominantly lower grade ore.

Further to the completion of the initial pit optimisation as reported here, it was agreed by the Project Team to remove leuco-eclogite from consideration as an ore type due uncertainties around the ability of the process plant to recover products from this ore. This resulted in leuco-eclogite ore being classified as waste. The impact of this change on the pit shape is minimal as the pit optimisation continues to the bottom of the pit to extract high value ore. However, the economics of the Project is impacted negatively by the exclusion of leuco-eclogite due to the higher waste stripping requirements. Option 6 indicated that selecting a TiO_2 cut-off grade for trans-eclogite of 2.5% generates a higher NPV than lower cut-offs. It should be noted that the pit shell for Option 6 is similar, but not identical, to the Pit 1 shell used for earlier options.

In summary, the recommended option to be taken into detailed design and scheduling as the basis of the final PFS mine design is:

- Pit 1 ultimate pit shell (ultimate pit)
- A two-pushback strategy within Pit 1
- Ferro-eclogite with no cut-offs applied to either TiO_2 or garnet
- Trans-eclogite with a TiO_2 cut-off grade of 2.5%
- No consideration of leuco-eclogite as ore; this material should be classified as waste to be stripped and stockpiled for future use.

Subsequent to completion of the pit optimisation study, financial modelling of a number of mining options (see Section 12.9) using updated OPEX and CAPEX numbers indicated that the development of a high-grade option to mine and process ferro ore only at a capacity of 1.5 Mtpa provided a better IRR than options which mined both trans ore and ferro ore. For this reason, the final mine plan developed considered the mining and processing of ferro ore only, with trans ore being treated as waste. A Project decision was made, however, to continue to use the Pit 1 ultimate pit shell as the basis of the open pit mine plan and schedule. This decision is valid when considering the pit shells developed for Option 4b and Option 6 respectively (see Figure 12-10 and Figure 12-13 above). As can be seen from these options, the change in ultimate pit shell shape with increasing trans cut-off grades is insignificantly small. Typically, there is a 1 Mt difference in total tonnes mined (ore and waste, i.e. rock) between low cut-off grade and high cut-off grade options. The main change, therefore, is the way ore and waste is classified, with more ore being classified as waste for the high cut-off grade options.

The pit optimisation studies carried out in this phase will need to be reviewed and updated in the DFS to consider all changes to modifying factors which have occurred to date and which may occur.

12.4 Open Pit Mine Design

Based on the Pit 1 shell for Option 5 derived from the pit optimisation as reported above, the following steps describe the pit design process undertaken in the DESWIK mine design software package:

- Step 1: import of the Pit 1 NPV Optimisation shell derived from Option 5
- Step 2: contouring of the NPV Pit 1 shell and smoothing of the contours
- Step 3: generation of a Digital Terrain Model (DTM) surface based on smoothed contours
- Step 4: using the DESWIK open pit mine design tools, pit strings (for the top and bottom of benches and for ramps) were created based on the DTM surface, the mine boundary, the geotechnical parameters as defied by WAI (see Section 9.1), as well as the equipment parameters (truck size and turning radius). A bottom up design approach was taken. The design strings are shown in Figure 12-14 below.

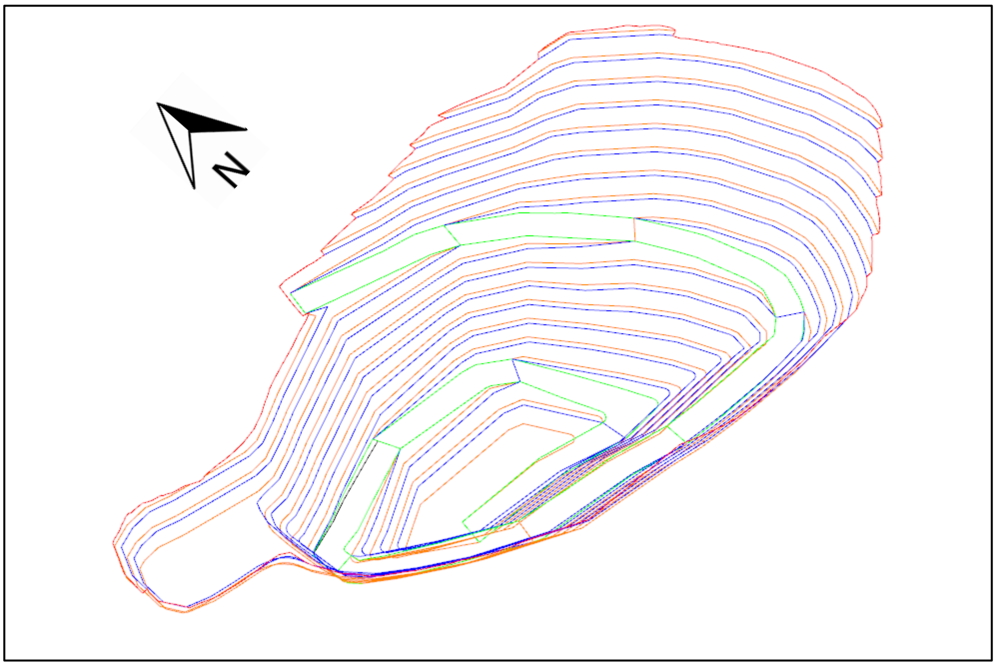


Figure 12-14: Ultimate Pit Design Strings

Key parameters for the pit design are summarised in Table 12-11 below.

Table 12-11: Key Pit Design Parameters

Parameters	Unit	Value
Bench height	m	15
Berm width	m	5
Bench face angle	º	Varies per pit sector
Ramp grade	%	10
Ramp width	m	23
Turning clearance diameter	m	29
Top bench elevation	m	330
Bottom bench elevation	m	90

- Step 5: using the ultimate design strings, a DTM was created which was then cut to the topography and made into a solid shape. This shape was used for interrogation of the block model
- Step 6: sequencing of the blocks within the shape.

12.4.1 *Pushbacks*

Before the sequencing of the pit was carried out, it was observed that significant amounts of Inferred material were to be found in the northern and eastern parts of the pit, which, in line with the guidelines of the JORC code, would be classified as waste in the mine plan. This Inferred material is shown in Figure 12-15 below, coloured in red. To minimise waste mining in the early years, therefore, pushback sequencing was introduced to defer the waste material stripping to later in the LoM with a view to improving the NPV of the Project.

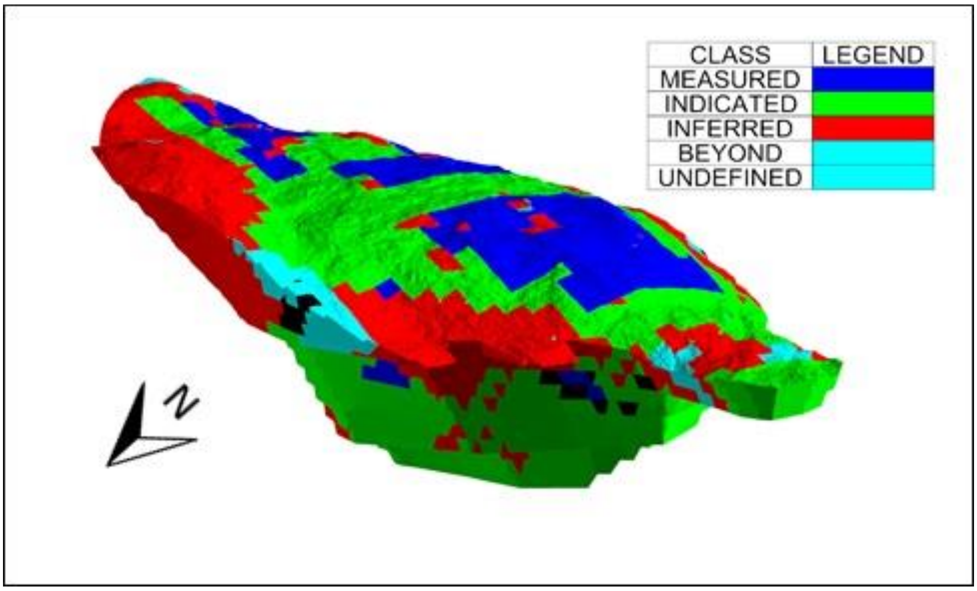


Figure 12-15: Categorisation of Ore in the Pit

The pushbacks were designed to be practical pushbacks from a production perspective. To achieve this, the bottom bench of pushback 1 (the initial cut) had to be large enough to be accessed by the selected equipment fleet. Furthermore, a minimum mining width of 40 m was employed for pushback 2. Consideration was given to the drop rate on pushback 2 to ensure that only a maximum of four benches were mined per year.

The two pushbacks used to sequence the pit are shown in Figure 12-16 below.

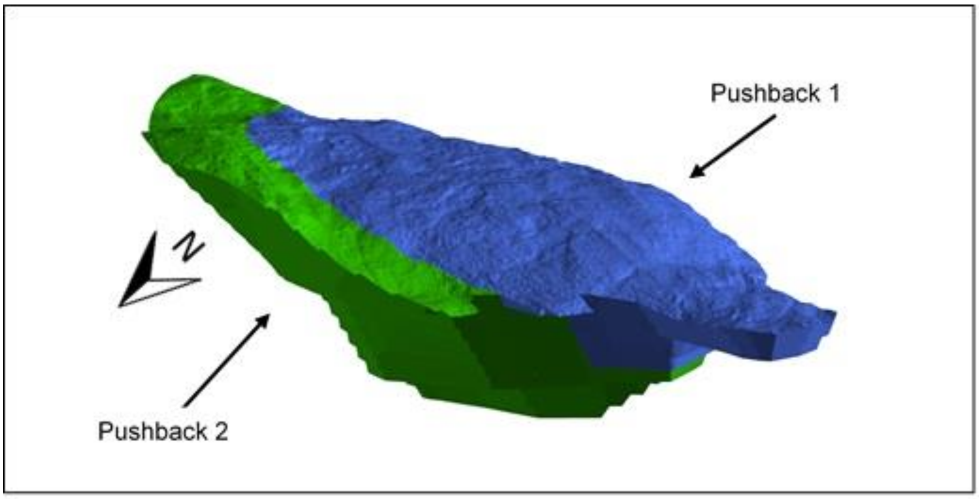


Figure 12-16: Pit Design Pushbacks

12.5 Underground Mining Block Model Development

12.5.1 Defining the Mining Model for the Underground

Development of the underground mining model started with the removal of blocks which protruded above surface (caused by regularisation of the model). Thereafter, a 50 m geotechnical boundary was applied around the pit to ensure that underground design did not encroach on the open pit, which could result in stability issues. This is in line with the recommendations of SINTEF, the geotechnical consultants employed to validate the underground design.

The result of applying geotechnical boundaries around the pit is shown in Figure 12-17.

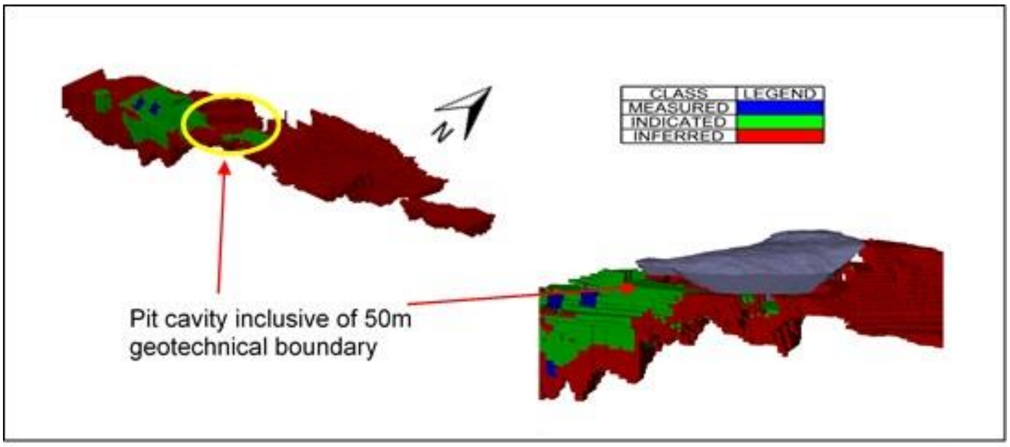


Figure 12-17: Pit Excavation including Geotechnical Boundary

Attention was now focussed on the underground orebody and, in particular, the extent of the Measured and Indicated Resources, which formed the basis for the underground mining method selection process.

12.6 Underground Mining Method Selection

The pit design studies, as summarised in Section 12.4 above, defined an economic pit which mined the Engebø orebody from 330 masl to 90 m masl elevation. Below this level, and laterally outside of the pit, all ore can be considered as mineable by underground mining methods only. A high-level study was undertaken to determine the underground mining methods applicable for Engebø. By doing this, the scope of the PFS was more clearly defined and a mine design and schedule, including associated mining operating and capital cost estimates, were drawn up as input to the business case financial model.

The following methodology was used to shortlist possible underground mining methods for Engebø.

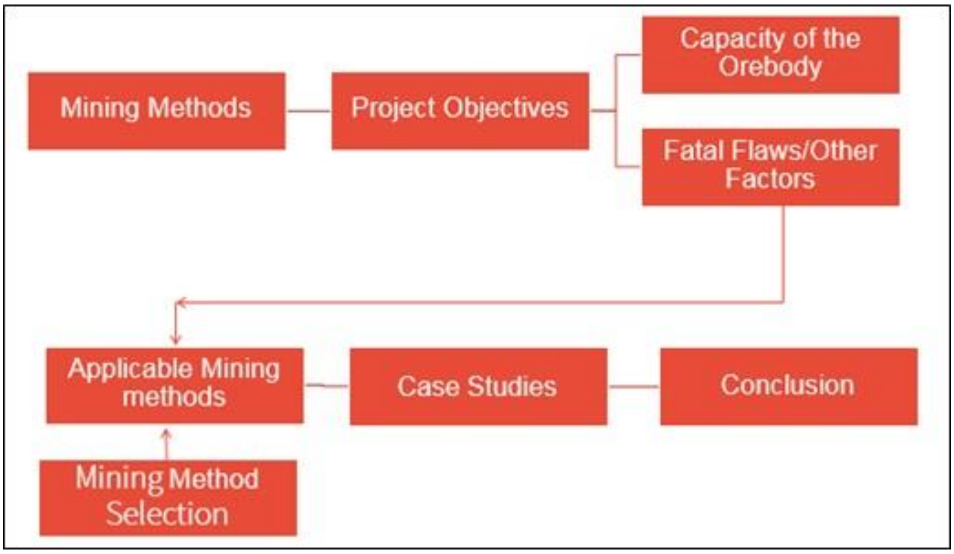


Figure 12-18: Underground Mining Method Selection Process

The process to select applicable mining methods for Engebø started with a consideration of all possible mining methods. Thereafter, the Project objectives were aligned with the potential mining methods, after which capacity considerations were brought into the analysis. Finally, fatal flaws applicable to the Engebø orebody were considered as well as other factors (such as grade control and dilution).

12.6.1 Major Underground Mining Methods in Use Globally

The first step in the high-level study to determine viable underground mining methods for Engebø was to consider all major mining methods in use globally. These are summarised in Figure 12-19 below.

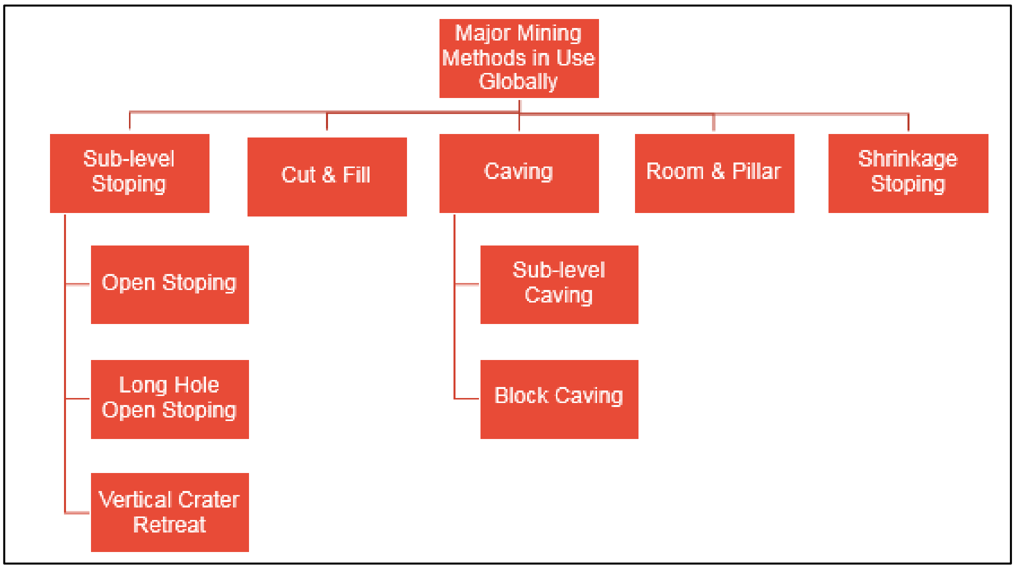


Figure 12-19: Major Mining Methods in Use Globally

A summary of the each of the above mining methods follows.

12.6.1.1 Sub-level Stoping

Sub-level stoping is a method of underground mining that involves vertical mining in a large, open stope that has been created inside an orebody. Drilling, blasting and mining are carried out at different elevations within the stope. The mining process starts with sublevel drifts being drilled into the orebody. The drilled sublevels are situated directly above a main haulage level. Drill rigs are used to drill a ring of holes around the drift or to drill holes below the sublevel; the holes are then filled with explosives. The explosives, once detonated, blast apart the drilled rock which falls to the bottom of the stope. Load Haul Dump trucks (LHDs) are then used to transport the muck to an ore pass from where it normally enters a crusher before being conveyed to surface. The excavation process is repeated until the stope is left completely empty.

Three variants of the sub-level mining method – sub-level open stoping, sub-level long hole open stoping and vertical crater retreat, are illustrated in Figure 12-20 to Figure 12-22 below.

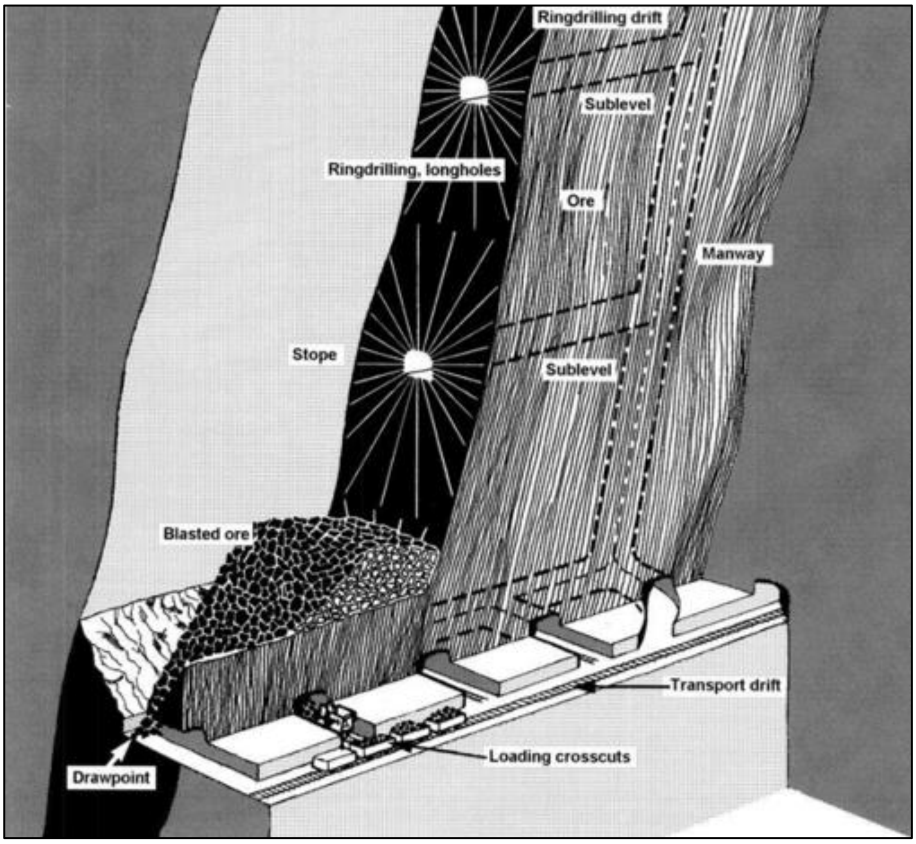


Figure 12-20: Sub-level Open Stoping

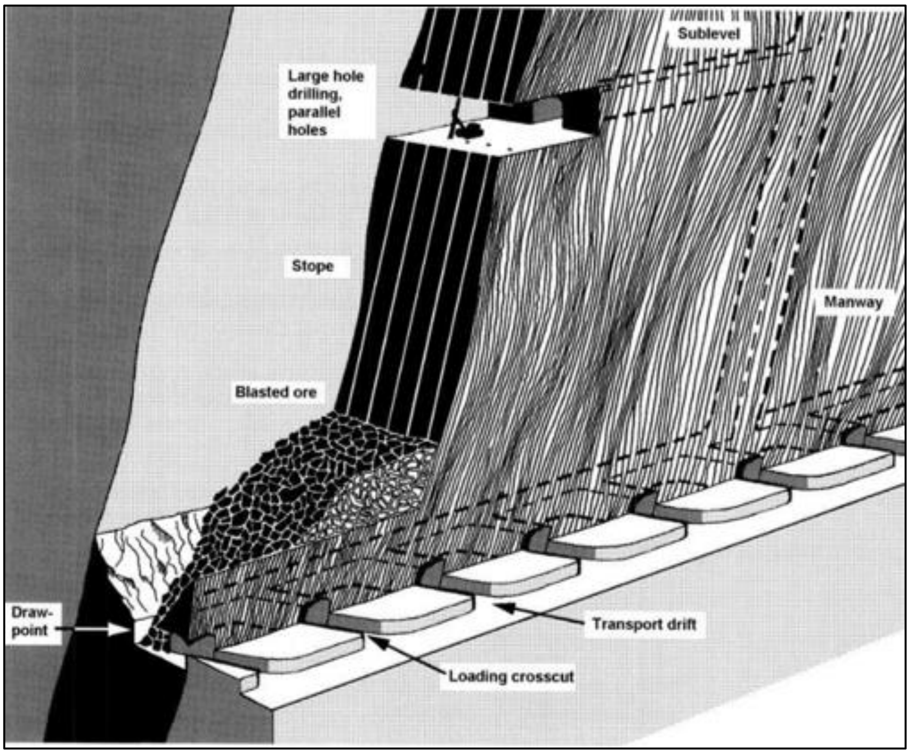


Figure 12-21: Sub-level Long Hole Open Stoping

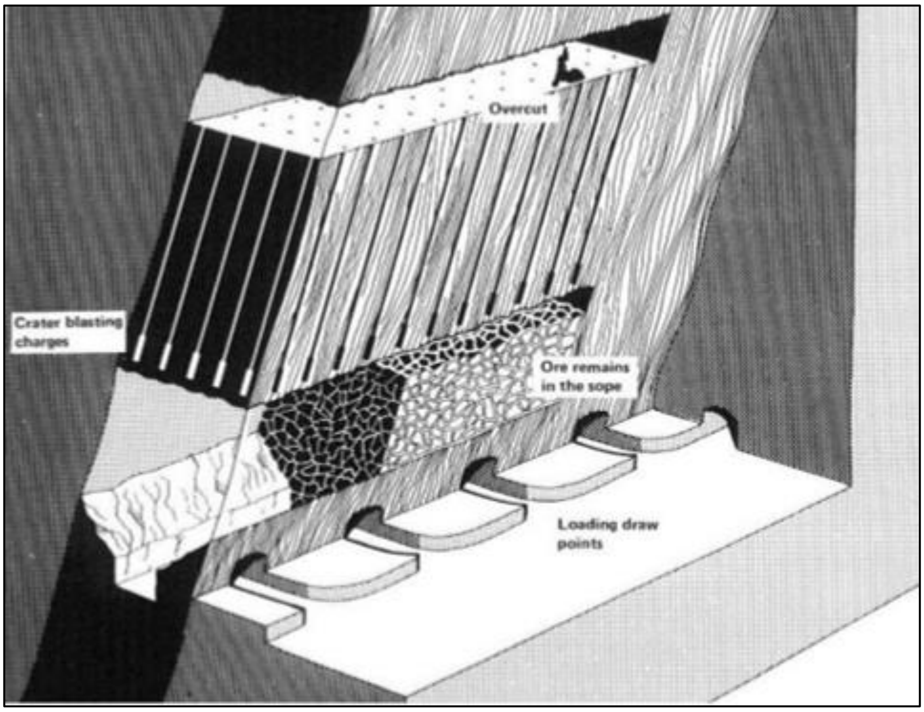


Figure 12-22: Vertical Crater Retreat

12.6.1.2 Cut and Fill

Cut and fill mining is a method of short-hole mining used in steeply dipping or irregular ore zones, in particular where the hanging wall limits the use of long-hole methods. The ore is mined in horizontal or slightly inclined slices, and then filled with waste rock, sand or tailings. Fill may be consolidated with concrete or left unconsolidated. Cut and fill mining is an expensive but selective method, with low ore loss and dilution. This mining method is illustrated in Figure 12-23 below.

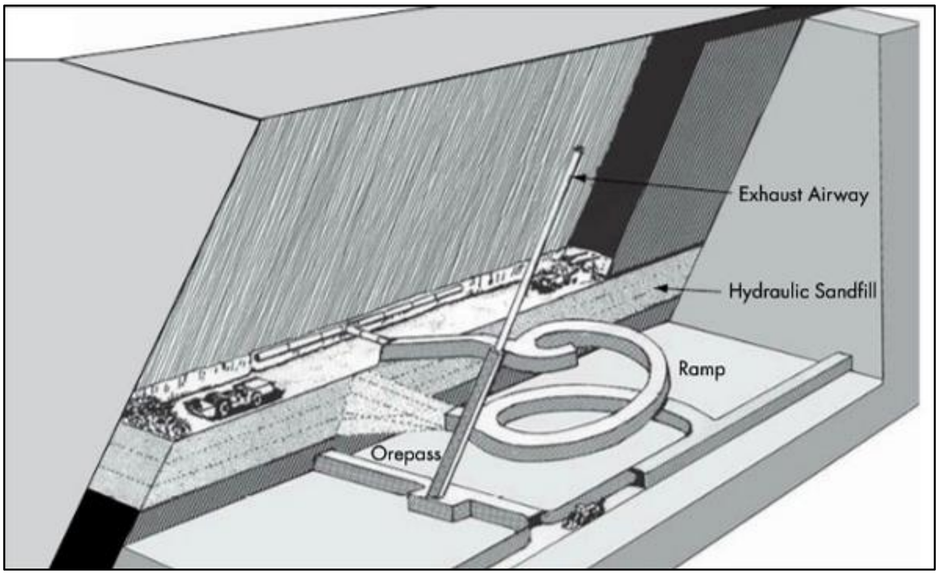


Figure 12-23: Cut and Fill

12.6.1.3 Caving

Cave mining is a mass mining method that allows for the bulk mining of large, relatively lower grade, orebodies. In this method, the orebody is undercut on a number of levels and selectively blasted above the undercut levels. Two key variants of the caving method are sub-level caving and block caving, as described below. In block caving, removal of ore at the undercut level induces the remainder of the orebody, which does not need to be blasted, to cave.

12.6.1.3.1 Sub-level Caving

This is a large-scale mining method suitable for large ore bodies with a steep dip and a rock mass with a host rock in the hanging wall which will fracture under controlled conditions. Infrastructure is always placed on the footwall side. Mining starts at the top of the orebody and progresses downwards in a safe sequence. All the ore is fragmented by blasting, causing the host rock in the hanging wall of the orebody to cave. Once the production drifts have been excavated and reinforced, long hole drilling is carried out. Rock is loaded from the cave front after each blast. To control dilution of waste rock in the cave, loading of a predetermined extraction percentage of rock is done. Ore is dumped into orepasses which connect to a haulage level. Caving will, sooner or later, also cause subsidence on the surface. The sub-level caving mining method is shown in Figure 12-24 below.

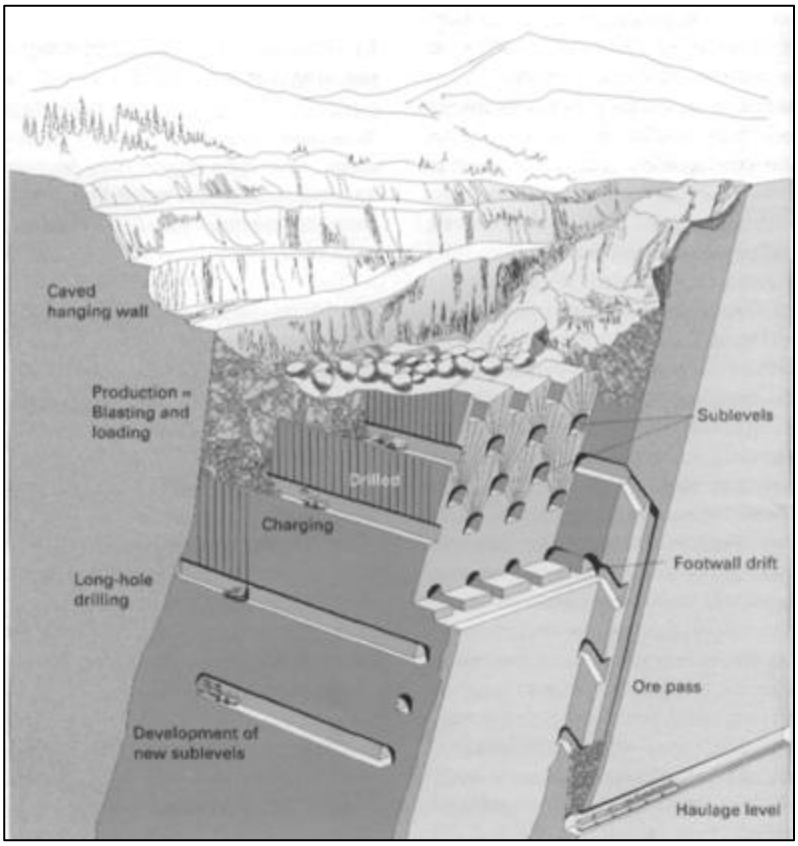


Figure 12-24: Sub-level Caving

12.6.1.3.2 Block Caving

Block cave mining is a mass mining method that allows for the bulk mining of large, relatively lower grade, orebodies. Underground tunnels lead to draw points where the overlying rock, broken by gravity, flows to the draw point, to be gathered and taken away for processing. Block cave mining is characterised by caving and extraction of a massive volume of rock which almost always translates into the formation of a surface depression. The block caving mining method is shown in Figure 12-25 below.

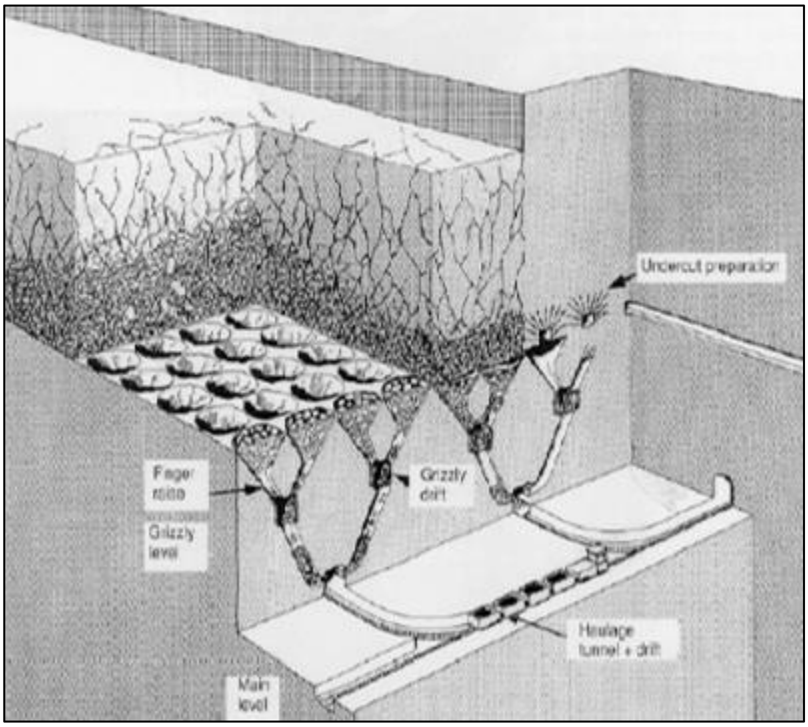


Figure 12-25: Block Caving

12.6.1.4 Room and Pillar

Room and pillar, also called bord and pillar, is a mining system in which the mined material is extracted across a horizontal plane, creating horizontal arrays of rooms and pillars. The ore is extracted in two phases. In the first, "pillars" of untouched material are left to support the roof overburden, and open areas or "rooms" are extracted underground; the pillars are then partially extracted in the same manner as in the "bord and pillar method". The technique is usually used for relatively flat-lying deposits, such as those that follow a particular stratum. It is used in the mining of coal, iron and base metals ores, particularly when found as tabular deposits, stone and aggregates, talc, soda ash and potash. The key to successful room and pillar mining lies in the selection of the optimum pillar size. In general practice, the size of both room and pillars are kept almost equal, while in bord and pillar, pillar size is much larger than bord (gallery). If the pillars are too small the mine will collapse, but if they are too large then significant quantities of valuable material will be left behind, reducing the profitability of the mine. The percentage of material mined varies depending on many factors, including the material mined, height of the pillar, and roof conditions; typical values are: stone and aggregates 75%, coal 60%, and potash 50%. The room and pillar mining method is illustrated in Figure 12-26 below.

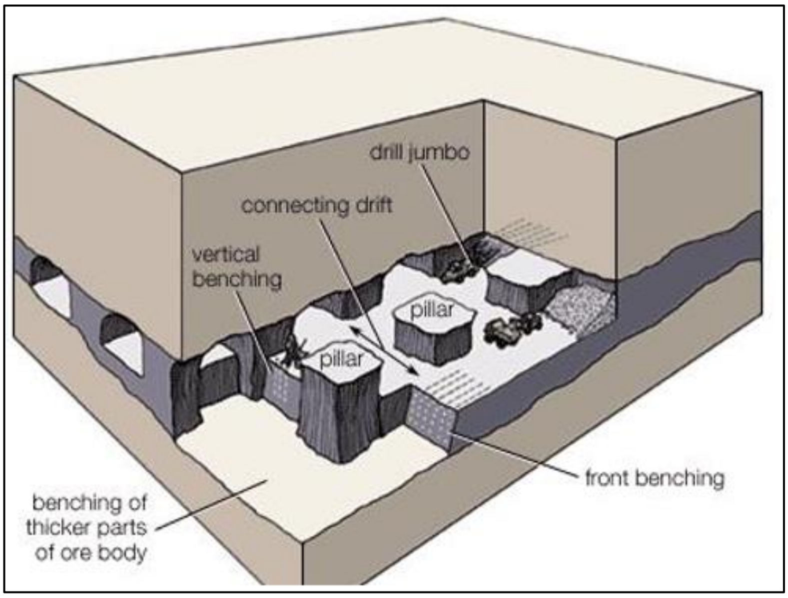


Figure 12-26: Room and Pillar Mining

Shrinkage Stoping

Shrinkage stoping is a mining method used for steeply dipping, narrower orebodies with self-supporting walls and ore. It is an overhand mining method that relies on broken ore being left in the stope to be used as the "working floor" and to support the walls. During the mining cycle, only 30% to 35% of the ore blasted is extracted being equivalent to the swell factor of *in-situ* ore to broken. When mining is complete to the next upper horizon, the ore is extracted. Although it is not necessary to fill the resulting voids, they are commonly filled with waste rock from development. Level intervals seldom exceed 40 m due to uneven muck draw. Recoverable pillars are left at each level. The shrinkage stoping mining method is shown in Figure 12-27 below.

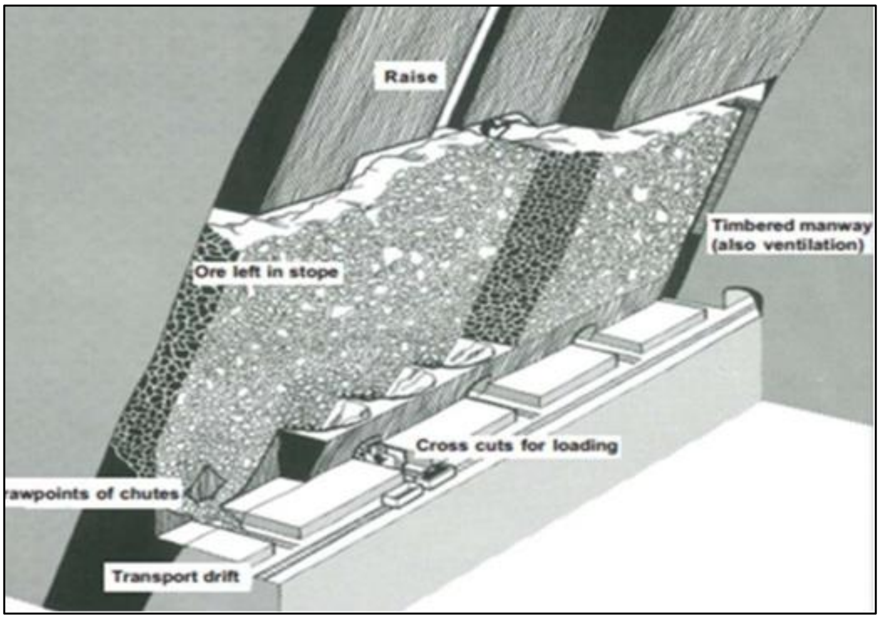


Figure 12-27: Shrinkage Stoping

12.6.2 Project Objectives and Capacity of the Orebody

All the above mining methods were then evaluated in terms of the project objectives and the match of the mining method to the size of the orebody and the mining rate. Key project objectives were defined as follows:

- To optimise underground extraction to improve the mine plan and Project economics
- To maximise underground extraction through the mining of reserves in line with the guidelines of the JORC Code
- To determine the practically achievable extraction – primary and secondary extraction
- To give due consideration to garnet income, which is key for the project economics
- To align the mining method to the size of the orebody and the mining rate.

12.6.3 Fatal Flaws/Other Considerations

A list of fatal flaws, against which the mining methods were considered, follows:

- In terms of the current permits and social licence to operate, subsidence of the ground surface at Engebø will not be allowed
- Will the orebody cave naturally or will caving need to be induced? There are cost and safety implications to induced caving
- Affordability of backfill to maximise extraction percentage.

Caving methods were discarded due to the fact that they inevitably lead to surface subsidence, as well as the fact that the competent nature of the orebody means that it is unlikely to cave naturally, thereby incurring additional cost to induce caving.

At this stage of the study high-level work indicated that backfilling of the orebody to enable secondary extraction to take place would not be economically viable, in which case a lower overall extraction percentage could be assumed. This precluded the use of cut and fill mining methods for Engebø.

Other factors which were taken into consideration when studying viable mining methods for Engebø were:

- Grade control – this is likely to be an important consideration to maximises plant recoveries by controlling the grade of ore fed to the plant. Grade control has an impact on the stope size, with large stopes (generally extracted at a low operating cost) having low grade certainty and smaller stopes (higher operating cost) providing much better control of grades fed to the plant. This may preclude the use of caving mining methods
- Dilution, which is likely to be lower for bulk mining methods
- Blending – the blending of ore may be critical to optimal operation of the process plant. If blending is required, then a number of stopes needs to be available for mining at any one time; this may preclude the use of caving mining methods

- Overall mining operating costs, both for primary extraction and for secondary extraction (post backfill). Lower operating costs will inevitably improve the business case
- Timing of the underground – with approximately 27 Mt of ore reserves in the open pit, it was assumed that underground mining would only take place after at least ten years of open pit mining.

12.6.4 Selection Methods

The common mining methods in use globally, as summarised in Section 12.6.1 above, were then ranked according to the three common selection methods, taking cognisance of the project objectives, capacity of the orebody, fatal flaws and other factors. The three selection methods used globally are as follows:

- UBC Modified Nicholas: this method involves summation and ranking of numerical values associated with orebody characteristics that reflect the suitability of a particular mining method. In this case, the Engebø orebody was considered for massive mining and tabular mining (room and pillar mining)
- Boshkov and Wright: this is a qualitative method of ranking mining methods where geometry, grade distribution and rock mechanical characteristics are ranked according to acceptability for ten common mining methods
- Hartman: this is a qualitative procedure oriented towards both open pit and underground mining. Once the selection of either open pit mining or underground mining has been made based on the depth of the orebody, ore rock strength and the geometry of the orebody are considered to determine suitable mining methods.

All three of the above methods were used to determine suitable mining methods for Engebø. The results of the analysis are summarised below.

12.6.4.1 UBC Modified Nicholas Ranking

As shown in Table 12-12 and Table 12-13 below, this analysis, which ranked the mining method in terms of orebody geometry and strength, showed that sub-level stoping, followed by sub-level caving and cut and fill, are the highest scoring massive mining methods; sub-level caving, however, can be ruled out in this case as this method is considered fatally flawed for Engebø. Likewise, cut and fill mining is considered to be fatally flawed due to the high cost of backfilling. For massive (bulk) mining, therefore, sub-level stoping is seen as the only viable mining method.

If the Engebø orebody is considered as an orebody suitable for tabular mining, the longwall and cut and fill methods score the highest, followed by room and pillar mining. Longwall mining cannot be applied at Engebø as the ore is too hard for shearers (which are typically used in softer rock environments such as coal mines) and there is a risk of surface subsidence. Cut and fill mining is considered to be fatally flawed on the grounds of the high backfill cost. As a result, only room and pillar mining can be considered for use at Engebø.

Table 12-12: UBC Modified Nicholas Scoring – Orebody defined as Massive

Ranking	Mining Method	Points
1	Sub-level Stoping	29
2	Sub-level Caving	27
3	Cut and Fill	25

Table 12-13: UBC Modified Nicholas Scoring – Orebody defined as Tabular

Ranking	Mining Method	Points
1	Longwall	31
2	Cut and Fill	29
3	Room and Pillar	25

12.6.4.2 *Boshkov and Wright Ranking*

The Boshkov and Wright ranking method identified ten applicable mining methods for Engebo, as shown in Figure 12-28 below.

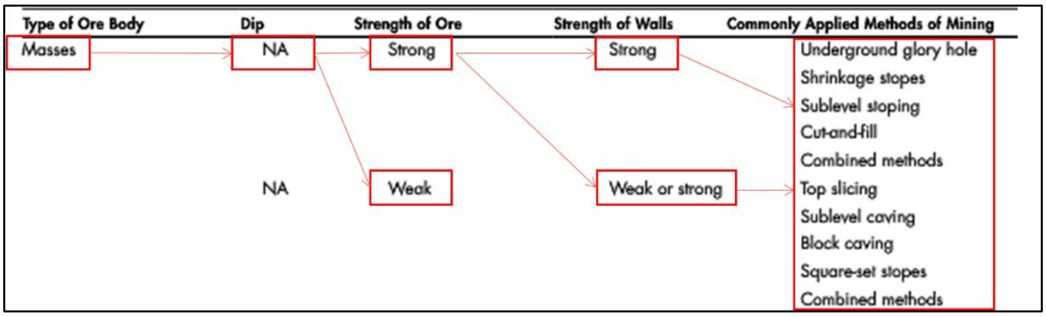


Figure 12-28: Boshkov and Wright Mining Methods Ranking

A number of the methods, such as square-set stoping (which is labour intensive and is high risk from a safety perspective) and the caving methods cannot be considered for use at Engebø. Out of the list of 10 methods identified, only underground glory hole, shrinkage stoping, sub-level stoping, and cut and fill are major mining methods worthy of consideration.

12.6.4.3 *Hartman Ranking*

Probably the most comprehensive of the three ranking methods considered in this study and one which is in common use in the industry, the Hartman method first divides mining methods into surface and underground, after which the competence (rock strength) of the orebody is considered. By dividing the potential underground mining methods into competent, incompetent and weak (caveable), the method identifies three major mining methods for Engebo, namely stope and pillar, shrinkage stoping and sub-level stoping. The Hartman ranking method applied to Engebø is shown in Figure 12-29 below. Shrinkage stoping can be ruled out on the grounds of safety concerns, leaving only stope and pillar mining and sub-level stoping.

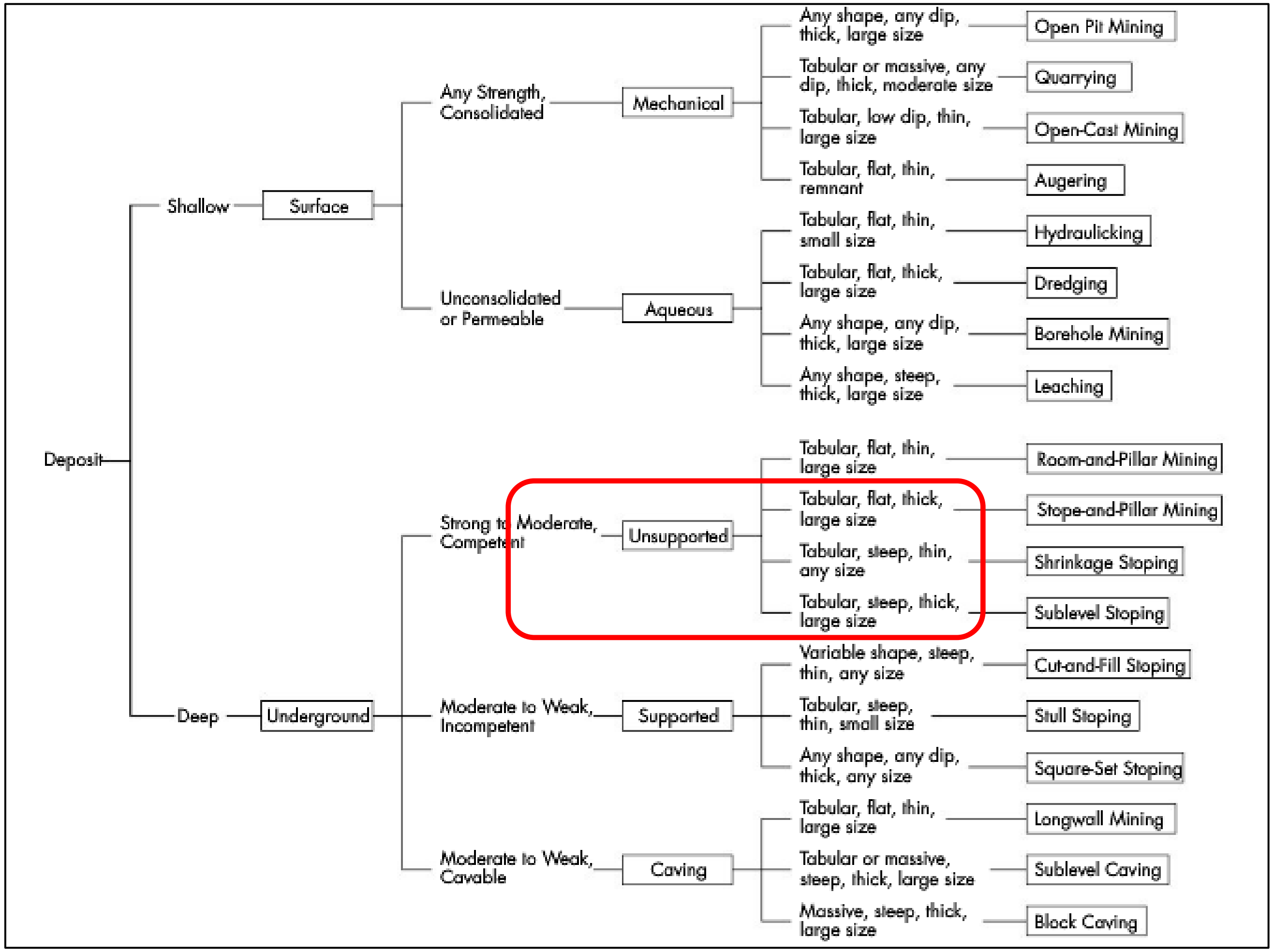


Figure 12-29: Hartman Ranking Method

Combining the results of the above ranking methods as well as taking due consideration of the qualitative nature of the ranking, the following selection of applicable mining methods was made.

Table 12-14: Mining Methods Shortlist for Engebø

Mining Method	Applicable to Engebø	Rationale for Applicability
Sub-level Open Stoping	Yes	Low cost mining and highly productive and flexible
Sub-level Long Hole Open Stoping	Yes	Low cost mining, highly productive and flexible; modern drilling technology will most likely make it cheaper than sub-level open stoping
Vertical Crater Retreat	No	Highly constrained by sequence, but can be used to minimise development in waste
Cut and Fill	No	High cost of mining with backfill
Sub-level Caving	No	No surface subsidence permitted, high upfront development capital
Block Caving	No	No surface subsidence permitted, high upfront capital and long development time
Room and Pillar	Yes	Flexible mining, highly mechanised, medium to high productivity
Shrinkage Stoping	No	Low productivity and unsafe

For the purposes of this study, long hole open stoping was selected as the mining method to be used, primarily for the reason that it uses the latest technology and, therefore, will have the lowest operating cost of the three potential methods identified in Table 12-14 above.

12.7 Underground Mine Design

The underground mine design comprises two key components, namely the stoping design and associated access tunnels design, as well as the design of the underground infrastructure to support open pit and underground mining. Both components are described below.

12.7.1 Stoping Design

Having selected long hole open stoping as the underground mining method design basis, the steps to define the underground mine design were as follows:

Step 1: based on initial mining layouts as reviewed by SINTEF (see Section 9.2), the final stope shape was determined and a grid with the appropriate stoping design initially overlaid on top of the mining blocks in areas with Measured and Indicated ore resources. Figure 12-30 below illustrates the grid used for the stoping design.

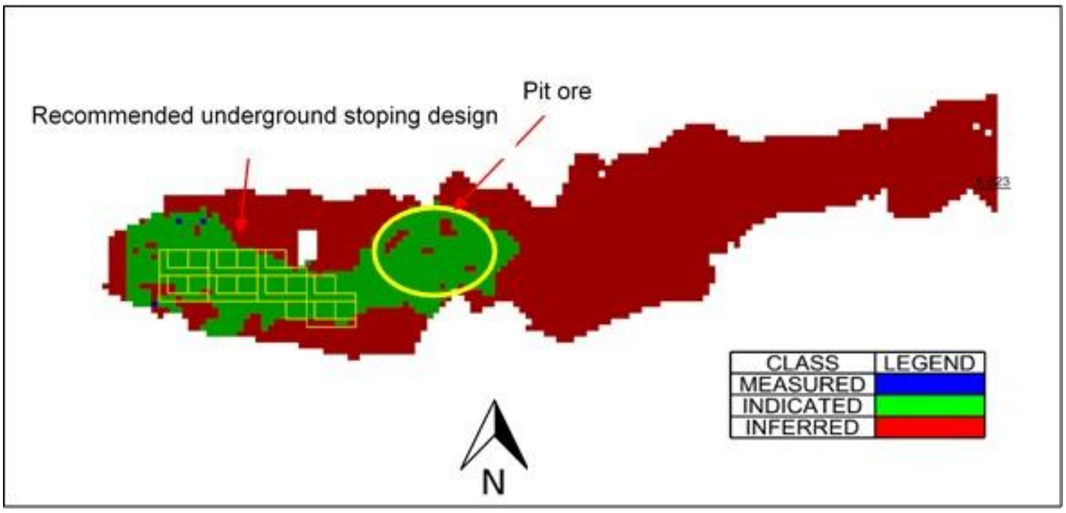


Figure 12-30: Stoping Block showing Stoping Design Grid Overlain

Step 2: the overlaid grid was then trimmed to meet additional constraining parameters, namely:

- Blocks that penetrated the topography
- Blocks within the geotechnical boundary of the open pit and the fjord.

The constrained mining zone is shown in Figure 12-31 below.

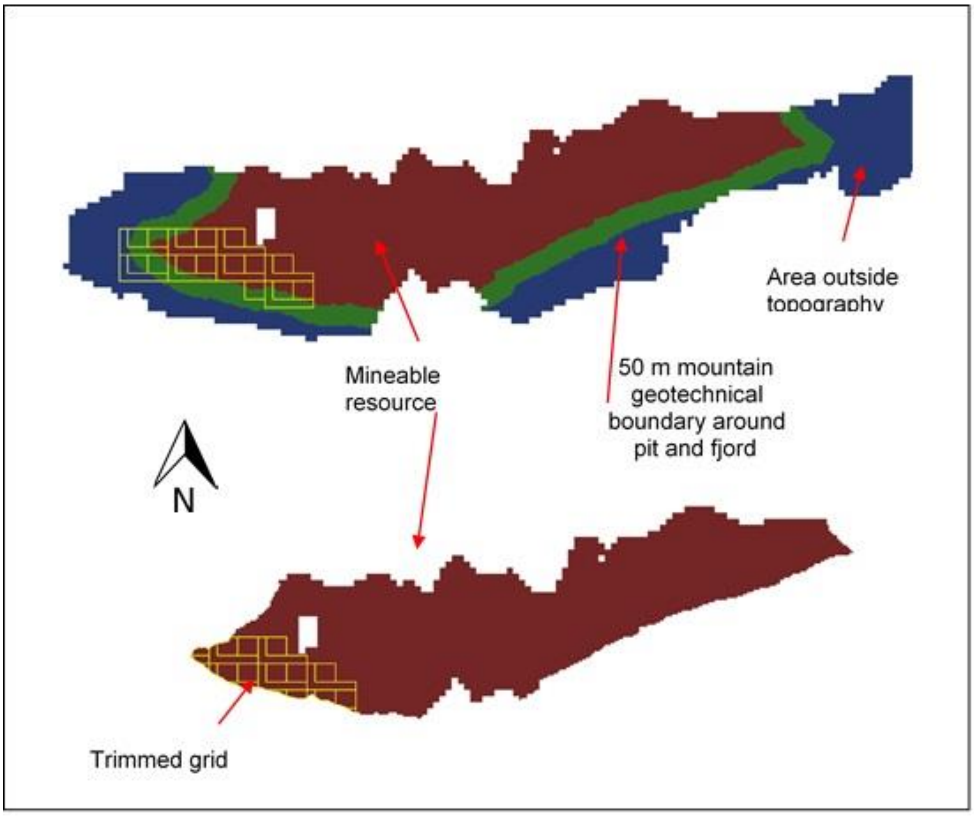


Figure 12-31: Final Mining Shape (shown in brown)

Step 3: this exercise was repeated for all mining blocks in the design. The resulting stope design in shown in Figure 12-32 below.

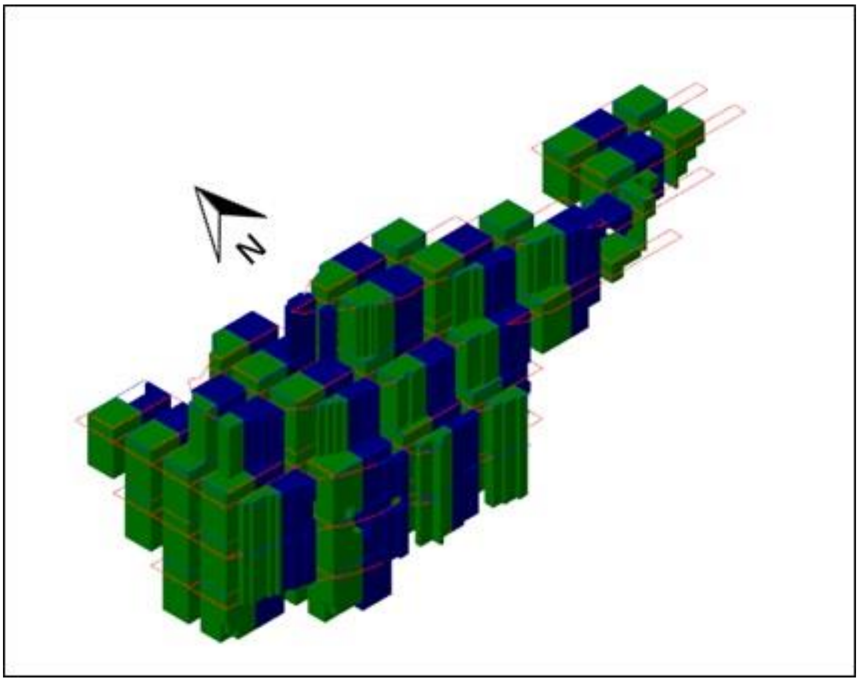


Figure 12-32: Stope Design

Step 4: further geotechnical, geological, physical and practical constraints were then applied. These constraints included:

- In terms of the SINTEF recommendations, the requirement of a 15 m sill pillar at the top and bottom of the 60 m high stopes
- The existence of a road tunnel through the orebody resulting in sterilisation of an area within a 50 m radius of the tunnel
- Removal from the layout of stopes that were not considered practical to mine
- All rock types within the layout which were not ferro ore being classified as waste.

The final design shape after consideration of the above constraints is shown in Figure 12-33 below.

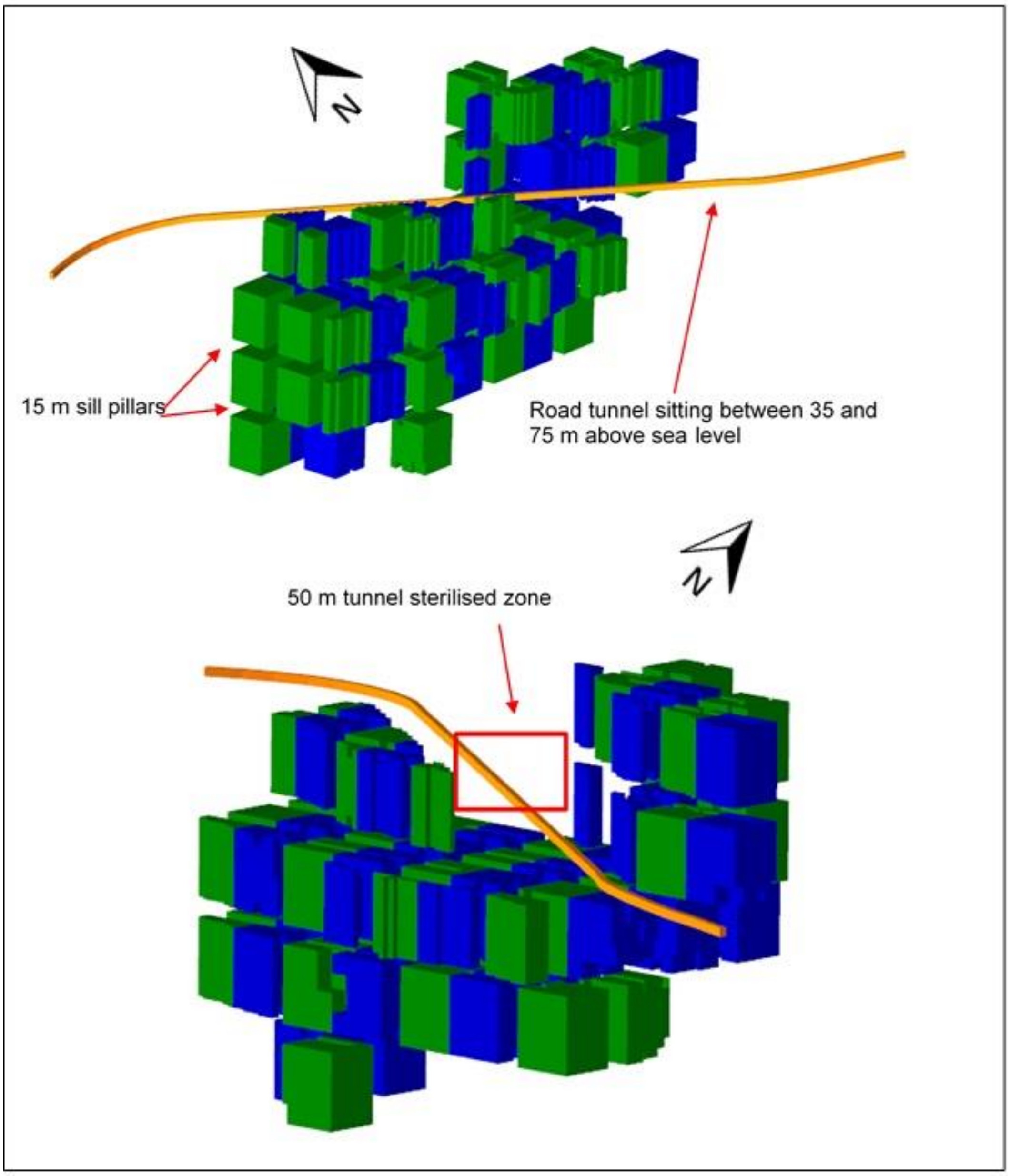


Figure 12-33: Final Underground Design Shape

Step 5: the final design component for the underground mine design was the development of an access system from the pit via a decline, as well as ore development so that each stope could be accessed from top and bottom to enable the mining method to be implemented. This development is illustrated in Figure 12-34 below.

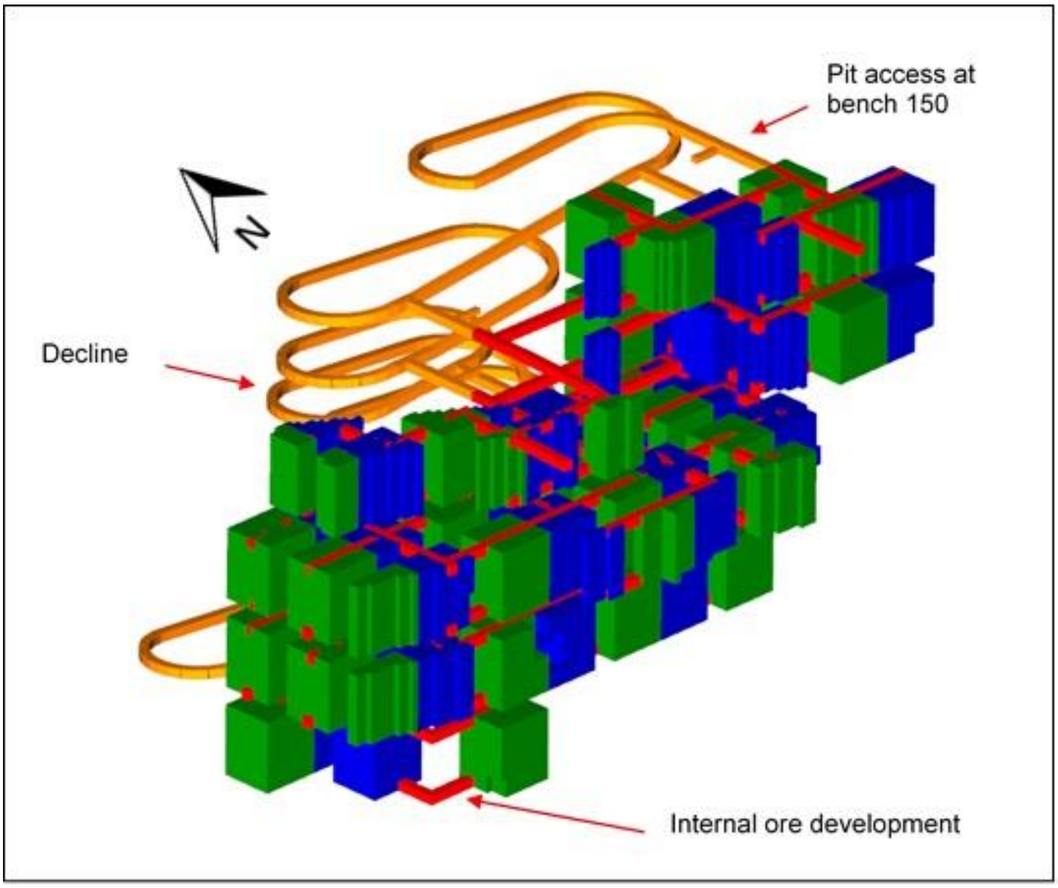


Figure 12-34: Decline Development and Ore Development

The stope design layout is based on top and bottom access to the stope by means of a single tunnel, which is then widened to create a slot covering the entire area of the stope. Thereafter, vertical slots are created to enable them to be drilled, blasted and loaded (mucked) from the bottom of the stope. This layout enables a long hole open stoping mine design to be scheduled. This is illustrated in Figure 12-35 below.

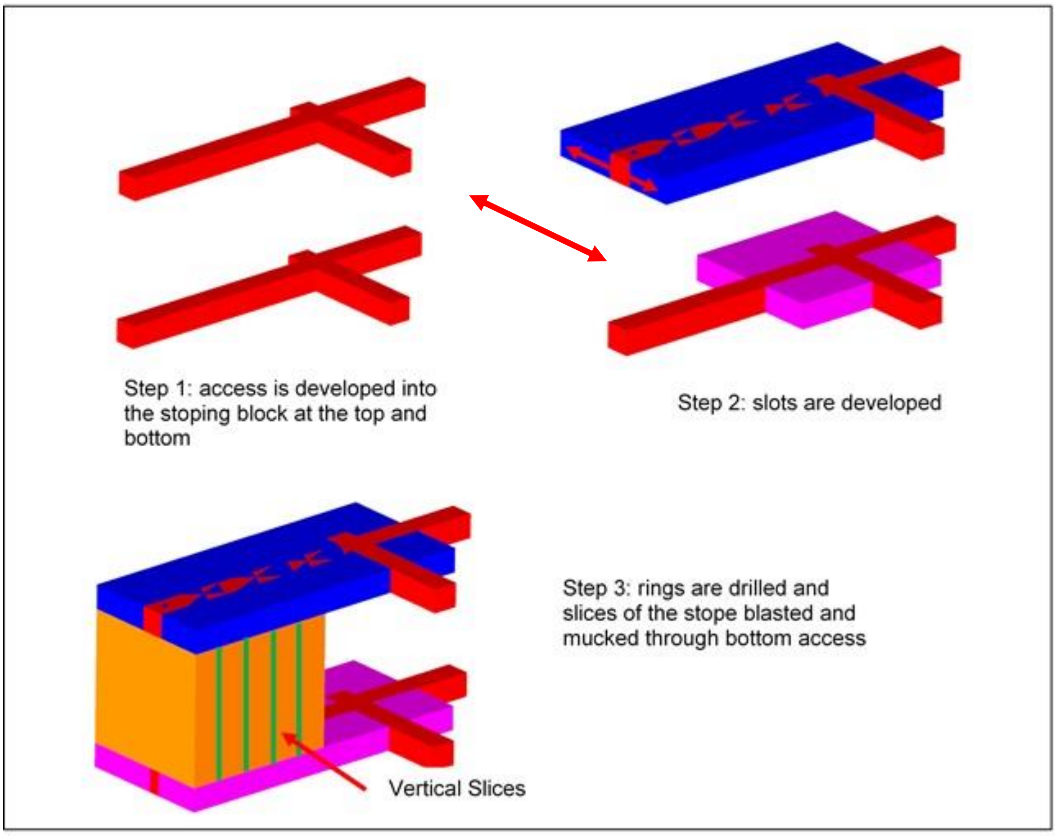


Figure 12-35: Stope Design and Mining Methodology

12.7.2 Underground Infrastructure Design

The underground infrastructure design to support open pit and underground mining is illustrated in Figure 12-36 below. To support open pit mining, underground excavations will be built, which include a glory hole plus grizzly arrangement in the pit, a primary crusher and crusher chamber, a silo and ore reclaim system, top and bottom access to the silo system, an ore conveyor belt from the silo reclaim system to the plant site, and a second egress from the top of the silo system to the plant site. For underground mining, a new ore pass (underground glory hole) and primary crusher chamber and crusher will be constructed to the east of the main underground mining areas. The crusher chamber will be connected to the existing silos and reclaim system by means of an underground conveyor belt system.

The design was reviewed by SINTEF, geotechnical consultants, who confirmed that the design was acceptable from a geotechnical perspective.

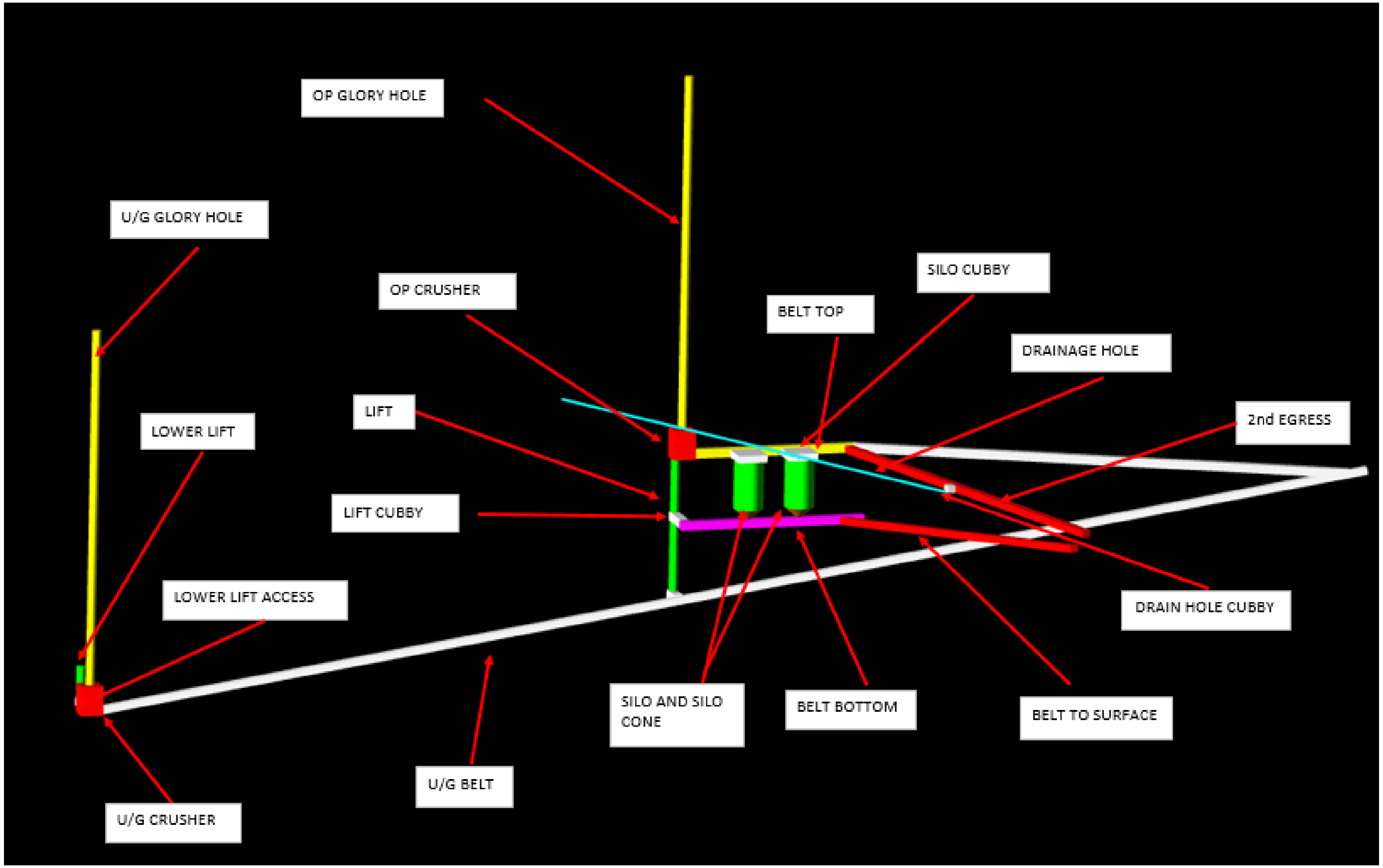


Figure 12-36: Underground Infrastructure Layout

Key dimensions of the underground excavations are summarised in Table 12-15 below.

Table 12-15: Summary of Excavation Dimensions

Description	Volume (m^3)	Length (m)	Diameter (m)	Width (m)	Height (m)
Belt Bottom	4,554	127		6	6
Belt Top	3,996	133		6	5
Drain Hole Cubby	166	6		5	5
Drain Hole	859	1,094	1		
Belt Surface	9,150	305		6	5
Egress	8,954	298		6	5
Lift	2,482	88	6		
Lift Cubby	2,069	59		6	6
Underground Crusher	4,500	15		15	20
Lower Lift	624	22	6		
Lower Lift Access	672	22		6	5
Open Pit Crusher	4,500	15		15	20
Open Pit Glory Hole	6,795	240	6		
Silos	20,697		20		40
Silo Cones	1,813	10	5-15		
Silo Cubby	4,304	43		20	5
Underground Glory Hole	6,816	241	6		
Underground Belt	38,834	1,294		6	5

12.8 Plant Design Capacity

12.8.1 Background

Capacity analysis forms an important part of the initial steps within the project development process. The capacity analysis was conducted within a matrix environment capable of simulating multiple variables simultaneously. The analysis included the total value chain from mining to sales. The capacity optimisation process incorporated:

- Mining block model attribute analysis for both the open pit and underground design
- Mine sequencing per level
- Multi-bench scheduling
- Processing alternatives
- Market volume alternatives.

12.8.2 *Approach*

The approach followed to define the capacity analysis is laid out in Figure 12-37 below. The objective of the simulation model was to combine design configurations and business sensitivity drivers within one simulation environment. Design aspects consist of two components: firstly, a cut-off grade sensitivity analysis, and secondly, a sequence and schedule optimisation functionality.

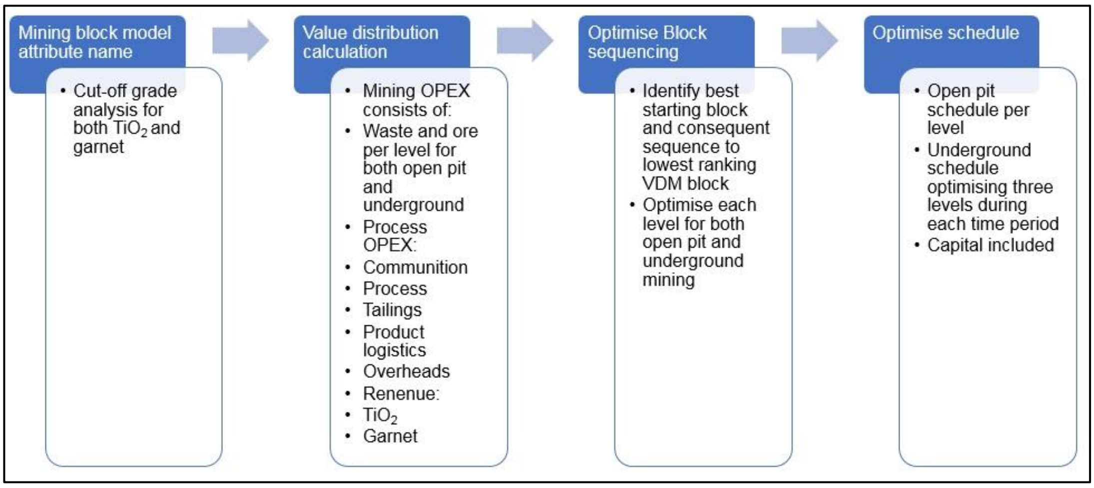


Figure 12-37: Capacity Analysis Process Steps

The sequence and schedule optimisation functionality ensured that an optimum mining schedule was obtained for each sensitivity seeded within the model. This functionality was not analysed further within this part of the study and was addressed as part of the mine schedule. However, it is important to note that this functionality was actively working in the background whilst the sensitivity runs were performed.

The design sensitivity analysis (cut-off grade) was combined with the business sensitivity analysis (price and sales volume) to obtain the most optimal project capacity. Plant (RoM) capacities from 1.5 Mtpa to 4 Mtpa in increments of 0.1 Mtpa were analysed.

The first step of the capacity analysis (mining block model attribute filtering) consisted of a cut-off grade seeding, where material containing less grade than the seeded grade was classified as waste and material with a higher grade than the seeded grade was classified as ore. The following scenarios were analysed:

- Ore based only on material with a grade higher than the TiO_2 cut-off grade (1.0% to 3.0% in increments of 0.1%)
- Ore based only on material with a grade higher than the garnet cut-off grade (1% to 35% in increments of 5%)
- Ore based on material where both TiO_2 and garnet grades were higher than the respective cut-off grades (using the same increments as indicated above).

The second step of the capacity model consisted of an economic value calculation per mining block, also known as a Value Distribution Model (VDM). The VDM values are closely related to the profit per mining block and were calculated by subtracting the direct related operating cost from the revenue. The total value chain, from mining to shipment, was included within the VDM values. The VDM values were then used to rank each mining block for both open pit and underground operations.

The third step consisted of optimising the block sequence per level. The outcome of this step was to select the most economical starting block and consequent mining sequence. The following aspects are of importance:

- VDM values were used as ranking criteria
- The mining sequence per level followed a practical path from a starting block, progressing to the next block with the highest VDM value, selected from all available blocks. Blocks became available only if the particular block could be mined from a mined-out block in a perpendicular direction
- Each starting block was linked to the level access by means of a development drive. The block sequence optimisation included the VDM value for each development drive per starting block.

The fourth step consisted of schedule optimisation. For the open pit, each level was mined out before progressing to the next level. Within the underground workings, up to three levels could be mined simultaneously; each level's starting time was dependent on completions of the decline and development drive per level. Start-up and up-grade capital was included during this step.

12.8.3 Evaluation Discussion

The objective of the capacity analysis was to determine the project capacity range that delivered the most robust business case. Various aspects that influence the outcome of the business case were simulated as variable inputs in Steps 1 and 2 of the capacity analysis process, as shown in Figure 12-37. However, to analyse the effect of each aspect, a step-by-step approach was followed, starting with cut-off grade. NPV and IRR were used as primary ranking criteria.

Using the assumption that ferro-eclogite, transitional-eclogite and leuco-eclogite were all considered as ore, the following scenarios were considered:

- Scenario 1 – varying cut-off grades:
  - Ore based only on material with a grade higher than the TiO_2 cut-off grade (1.0% to 3.0% in increments of 0.1%)
  - Ore based only on material with a grade higher than the garnet cut-off grade (25% to 40% in increments of 5%)
  - Ore based on material where both TiO_2 and garnet grades are higher than the respective cut-off grades (same increments as indicated above).
- Scenario 2 – 16% garnet yield with price sensitivity:
  - TiO_2 price range from US$ 700/t to US$ 1,500/t of product
  - Garnet price range from US$ 150/t to US$ 300/t of product.

12.8.4 *Scenario 1 – Cut-off Grade Analysis*

The following options were analysed:

- TiO_2 cut-off grade: ore based only on material with a grade higher than the TiO_2 cut-off grade (1.0% to 3.0% in increments of 0.1%), with a 16% garnet yield
- Garnet cut-off grade: ore based only on material with a grade higher than the garnet cut-off grade (25% to 40% in increments of 5%), with a 1% TiO_2 cut-off grade.

For the cut-off grade analysis, the following input assumptions were applied:

- TiO_2 price range from US$ 700/t to US$ 1,500/t of product
- A garnet price range from US$ 150/t to US$ 300/t of product
- A constrained garnet market (garnet sales profile is matched and not exceeded).

12.8.4.1 *TiO_2 Cut-off Grade*

The following graphs compare NPV and IRR with TiO_2 cut-off grade. NPV is indicated in US$ M and IRR in percentage. The TiO_2 cut-off grade (*10%) is indicated on the x-axis. The vertical scatter within the graphs relates to the TiO_2 and garnet price ranges seeded per TiO_2 cut-off grade. The red oval indicates the 90% confidence range. From the analysis, the lower range TiO_2 cut-off grades are preferred. Both NPV and IRR follows the same trend in the sense that at a cut-off grade of 3% almost half of the value is destroyed; however, between 1% and 2% the difference is only 15%.

At a 1% cut-off grade the net revenue from only TiO_2 is below its marginal cost. The effect that low TiO_2 cut-off grade contributes to NPV and IRR indicates that firstly, the combined revenue, from both TiO_2 and garnet is above the marginal cost; and secondly, that the rate (tempo) of revenue generation in the early years is not significantly diluted by lower value ore. In the event that lower grade ore impacts on the margin generated within the early years, one would see a downwards trend towards the lower cut-off grades, especially within the IRR graph.

To simplify the capacity analysis, a TiO_2 cut-off grade equal to 1% was used from here onwards.

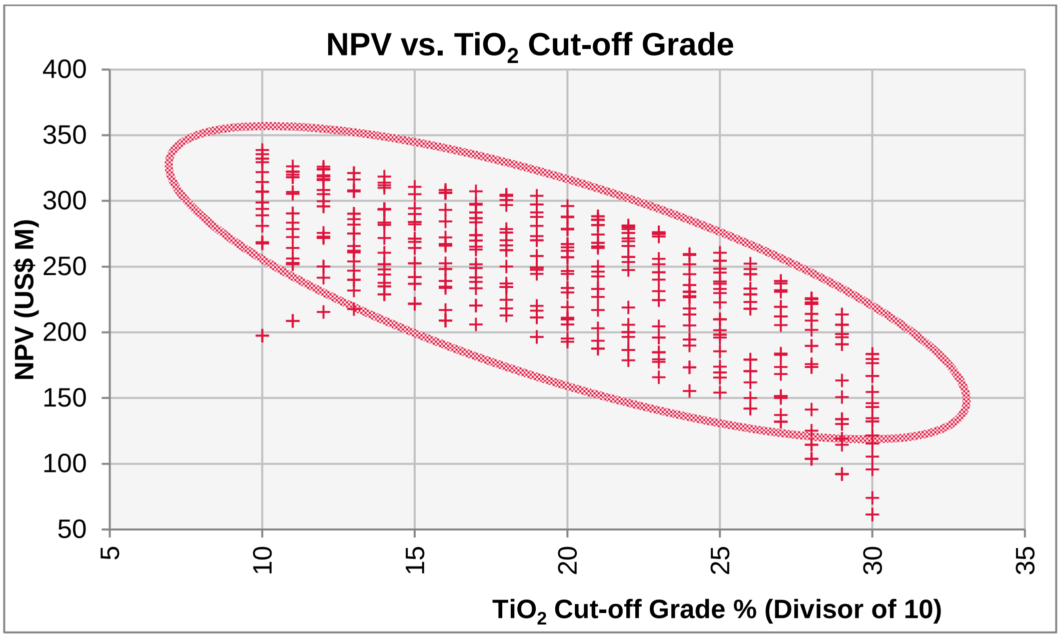


Figure 12-38: NPV vs. TiO_2 Cut-off Grade

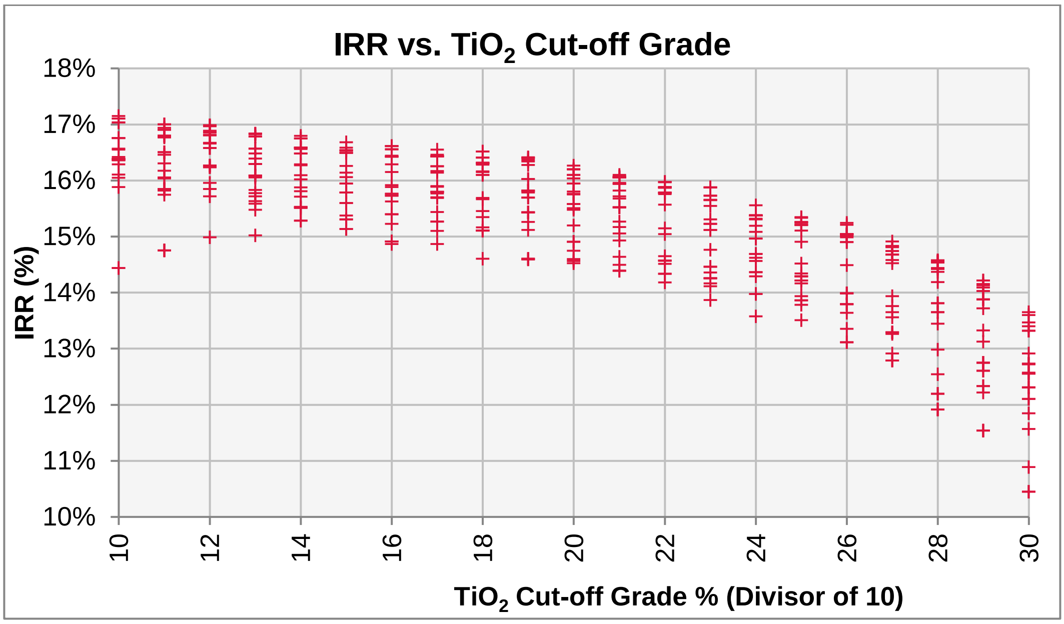


Figure 12-39: IRR vs. TiO_2 Cut-off Grade

12.8.4.2 Garnet Cut-off Grade

The following graphs compare NPV and IRR with garnet cut-off grade. Garnet cut-off grades from 25% to 40% in increments of 5% were evaluated. The garnet cut-off grade follows the same trend as the TiO_2 cut-off grade graphs, indicating that lower garnet cut-off grades are preferred.

To simplify the capacity analysis, a TiO_2 cut-off grade equal to 1% was used going forward. Based on Figure 12-38 and Figure 12-39, it was confirmed that up to a 1% TiO_2 cut-off, both the NPV and IRR improve.

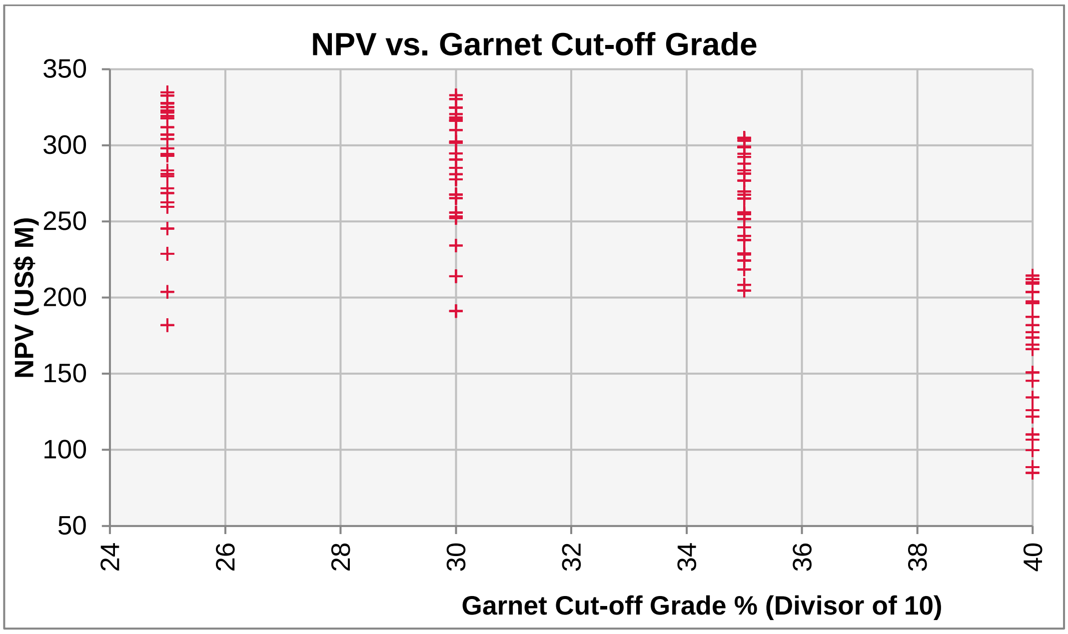


Figure 12-40: NPV vs. Garnet Cut-off Grade

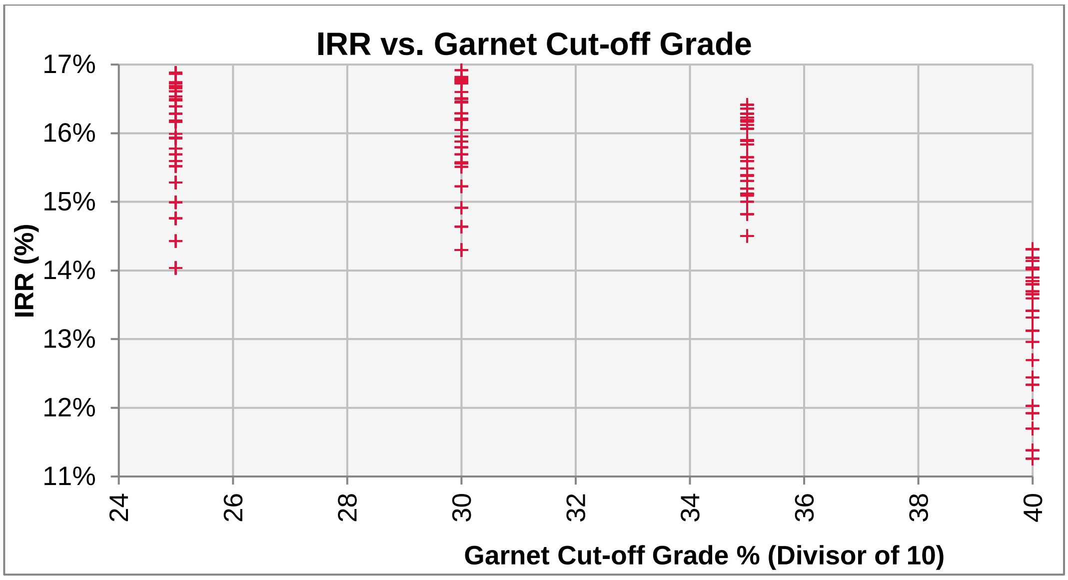


Figure 12-41: IRR vs. Garnet Cut-off Grade

12.8.5 Scenario 2 - Capacity Analysis

The capacity analysis is presented using a 1% cut-off grade for both TiO_2 and garnet together with a fixed 16% yield for garnet recoveries. It should be noted that although these fixed values are used to simplify the discussion, the simulation data contain results for all cut-off grade ranges as discussed in Section 12.8.4 above. To estimate the effect of sales price on capacity, the following price ranges were analysed:

- A TiO_2 price range from US$ 700/t to US$ 1,500/t of product (combined with a garnet base price of US$ 220/t of product)

- A garnet price range from US$ 150/t to US$ 300/t of product (combined with a TiO_2 base price of US$ 1,050/t of product).

Figure 12-42 below indicates NPV vs. capacity for TiO_2 price ranges. NPV is indicated on the y-axis, capacity on the x-axis and the respective TiO_2 price options are plotted.

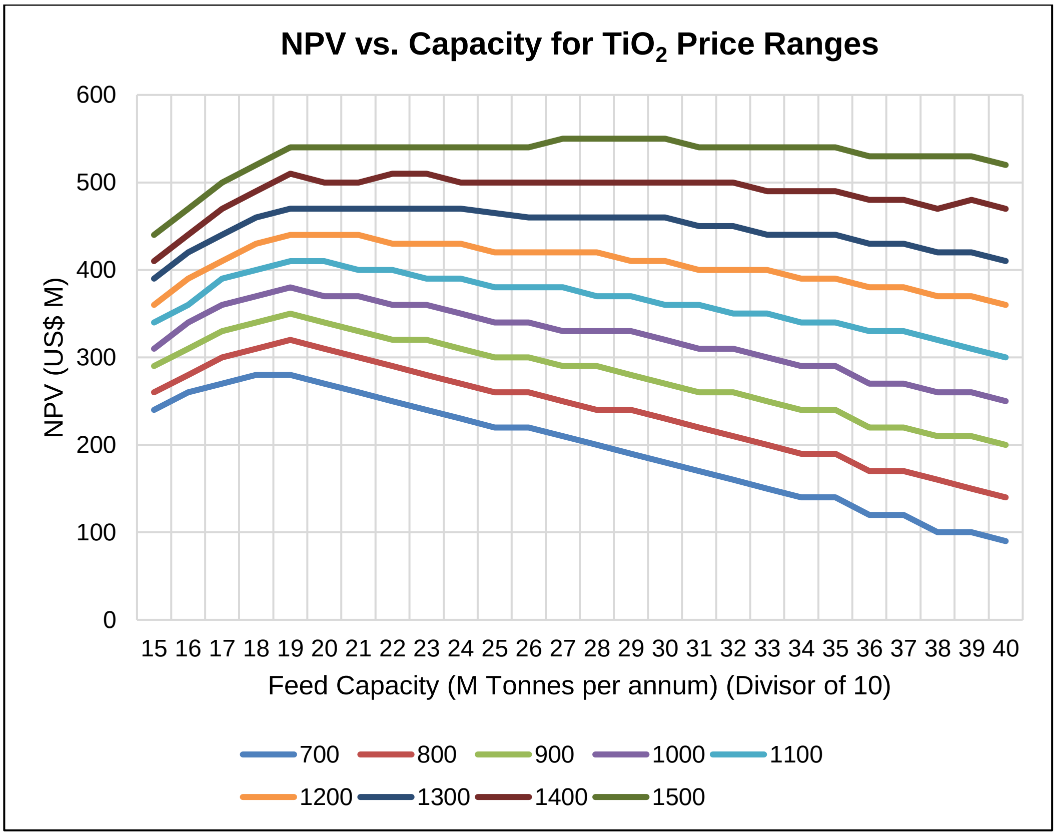


Figure 12-42: NPV vs. Capacity for TiO_2 Price Ranges

From the above graph, the following can be observed:

- There is no gain in NPV for capacities above 2.0 Mtpa, even at the highest TiO_2 price of US$ 1,500/t
- For the lower price options, the NPV drops off significantly above 2.0 Mtpa capacity
- For all prices, NPVs are consistently lower below 2.0 Mtpa; the effect is less for prices below US$1,100/t TiO_2, however.

Figure 12-43 below indicates the IRR vs. capacity for TiO_2 price ranges. IRR is indicated on the y-axis, capacity on the x-axis and the respective TiO_2 price options are plotted.

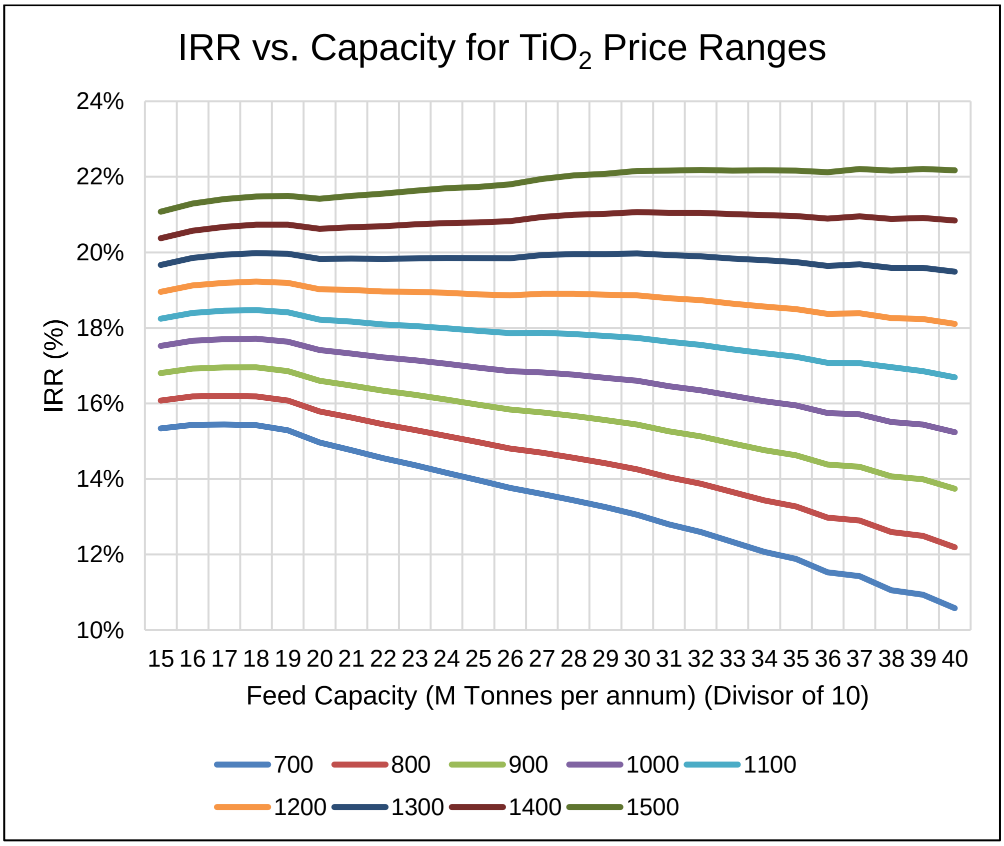


Figure 12-43: IRR vs. Capacity for TiO_2 Price Ranges

From the above graph, the following can be observed:

- For prices above US$ 1,300/t TiO_2 there is a slight advantage for capacities above 1.9 Mtpa
- For prices below US$ 1,100/t TiO_2 there is an increasing disadvantage for capacities below 1.9 Mtpa
- For all capacities below 1.9 Mtpa there is almost no change in the respective IRR for the total range of prices evaluated.

Figure 12-44 below indicates NPV vs. capacity for garnet price ranges. NPV is indicated on the y-axis, capacity on the x-axis and the respective garnet price options are plotted.

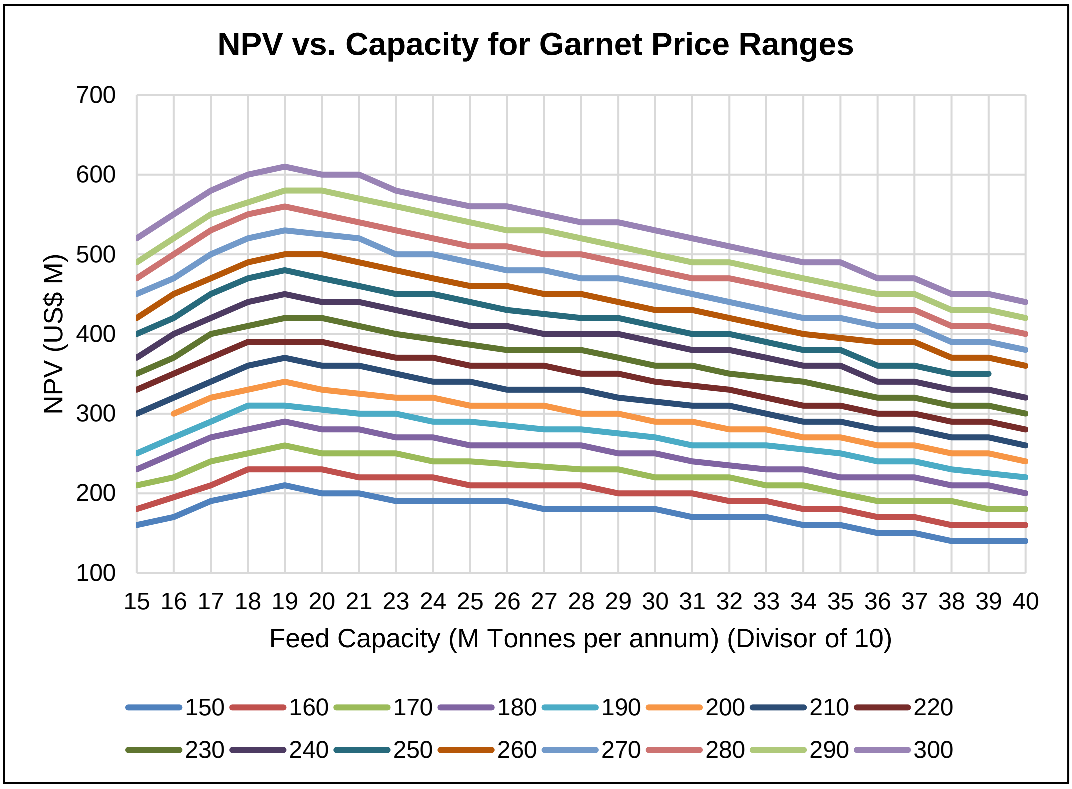


Figure 12-44: NPV vs. Capacity for Garnet Price Ranges

From the above graph, the following can be observed:

- All price scenarios indicate an optimum capacity of around 2.0 Mtpa
- For all the price scenarios, the higher capacity ranges have a lower NPV than the lower capacity ranges.

Figure 12-45 below indicates the IRR vs. capacity for garnet price ranges. IRR is indicated on the y-axis, capacity on the x-axis and the respective garnet price ranges are plotted.

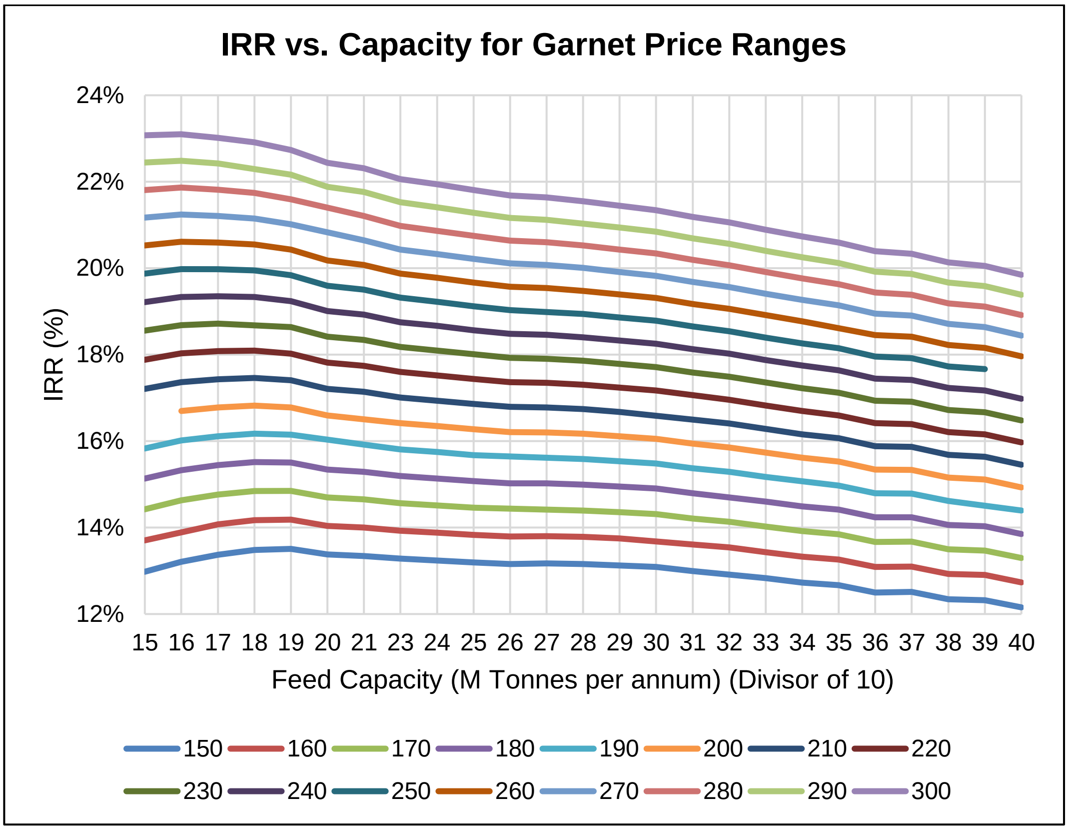


Figure 12-45: IRR vs. Capacity for Garnet Price Ranges

From the above graph, the following can be observed:

- For all price scenarios, there is a slight disadvantage for higher capacities
- For the lower capacity scenarios, there is almost no change in the IRR below 2.0 Mtpa.

At this stage of the Project, the base line price assumptions for TiO_2 and garnet were US$ 1,050/t and US$ 220/t of product respectively. Figure 12-46 and Figure 12-47 below show the indicative Project NPV and IRR compared to capacity. From these graphs, it can be concluded that the most suitable final installed capacity should be within a range between 1.8 Mtpa to 2.0 Mtpa. Based on these capacities, the pre-tax indicative NPV is US$ 383 M and the IRR is 18%.

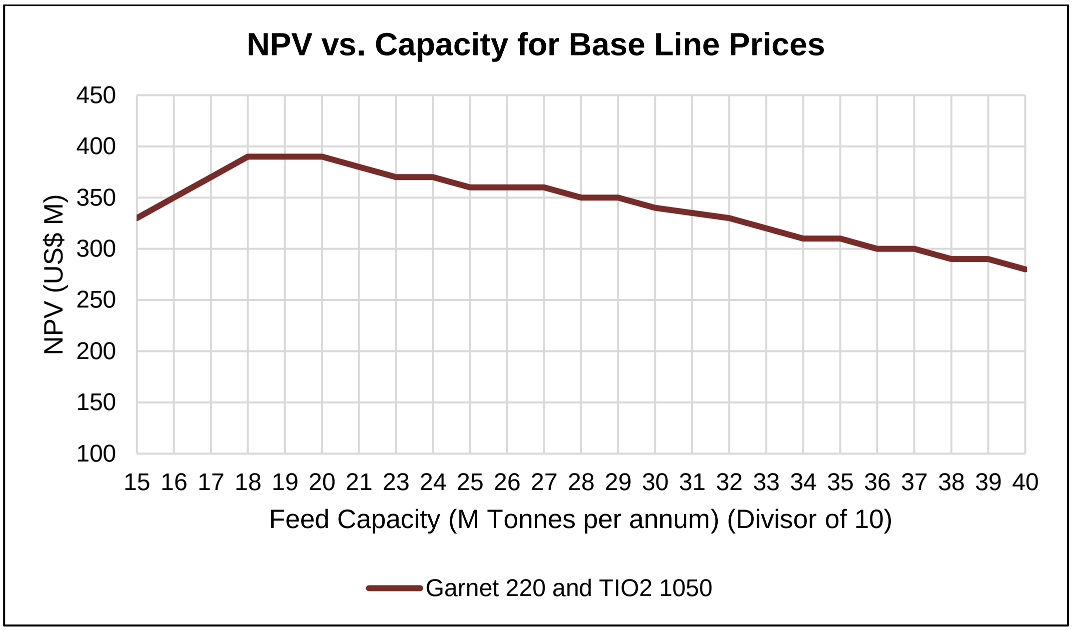


Figure 12-46: NPV vs. Capacity for Base Line Prices

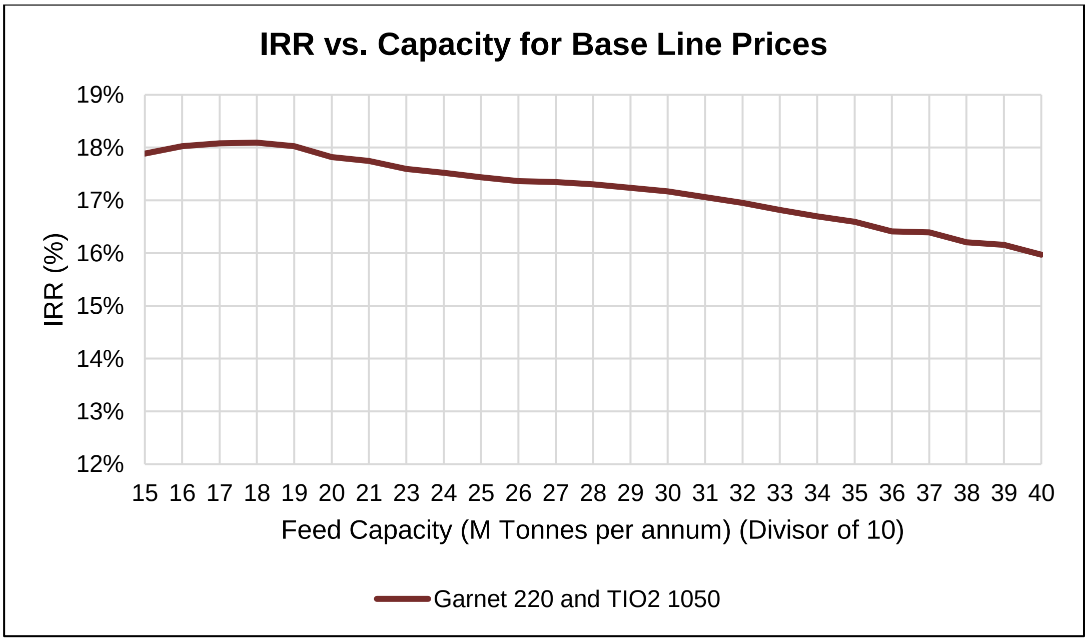


Figure 12-47: IRR vs. Capacity for Base Line Prices

It is important to note that if the IRR criterion alone is considered in the above analysis, there is minimal change in the IRR in the capacity range of 1.5 Mtpa to 2.0 Mtpa. Capacity modelling studies such as the one reported here are, by definition, high level and the results are not absolute. Subsequent definitive financial modelling (run for increased base line price assumptions for TiO_2 of US$ 1,070/t and US$ 250/t for garnet) indicated that on an IRR basis, a capacity of 1.5 Mtpa used for high grade ore (ferro only) showed a better return than 2.0 Mtpa options where lower grade ore was mined and processed. As a result of this work, the decision was made to select a 1.5 Mtpa capacity mine and plant as the preferred business case for the Project. It is recommended, however, that the capacity decision is reconsidered in the DFS phase, incorporating a revised set of input assumptions, as new information becomes available. This recommendation is made because the IRR for a 2.0 Mtpa plant capacity option was only marginally lower than the 1.5 Mtpa option.

12.9 Mining Options Evaluated

Before the final production schedule could be drawn up, a number of options were evaluated based on the mining and processing of two ore types, ferro-eclogite (ferro ore) and trans-eclogite (trans ore). The ferro ore has higher grades of rutile and garnet than trans ore. The ferro ore also has higher recoveries. The mining of these two ore types drives mining strategy as grade drives revenue, which in turn dominates profitability.

A number of options were investigated to evaluate the best financial option. The options fall into three main categories, namely:

- The mining and processing of ore at a rate of 1.5 Mtpa. In this option, a process plant is commissioned in 2021 to treat 1.0 Mtpa in that year (the ramp-up year), after which a maximum of 1.5 Mtpa is treated going forward

Figure 12-48 below illustrates the ore treatment profile associated with this option.

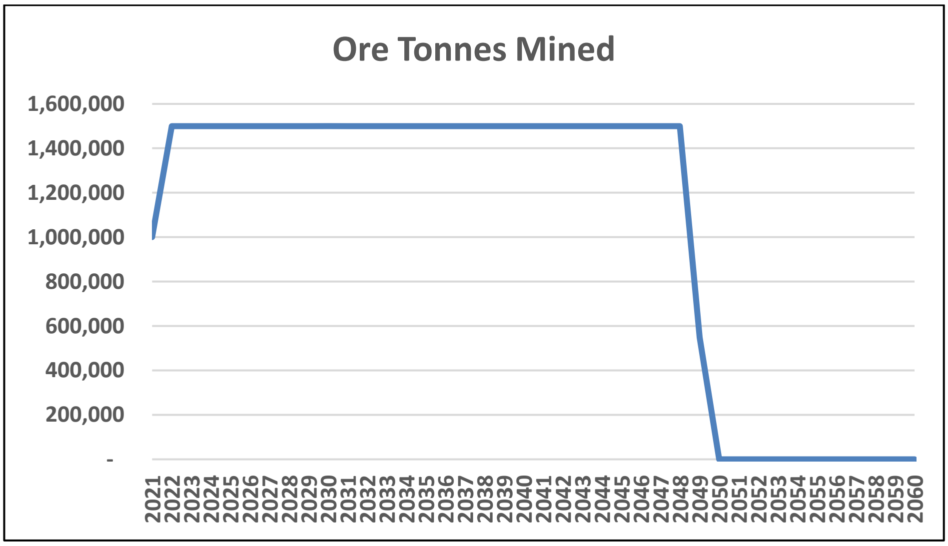


Figure 12-48: Ore Profile for 1.5 Mtpa Option

- After ramping up in 2021 to 1.0 Mtpa of ore, the process plant will operate at 1.5 Mtpa capacity until 2027 when it will be upgraded to run at 2.0 Mtpa capacity.

Figure 12-49 below illustrates the ore treatment profile associated with this option.

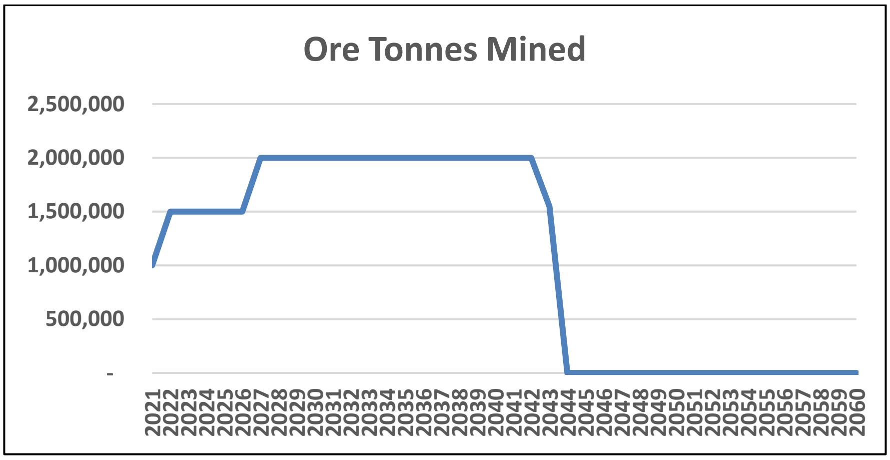


Figure 12-49: Ore Profile for 1.5 Mtpa with Stepped Upgrade to 2.0 Mtpa Option

- After ramping up in 2021 to 1.0 Mtpa of ore, the process plant will operate at incrementally increasing tonnages up to 1.5 Mtpa. Sufficient tonnes will be treated to ensure the garnet sales profile is achieved. Once the mine plan requires more than 1.5 Mtpa of ore to be mined and treated to achieve the garnet sales profile, the process plant will be upgraded to treat 2.0 Mtpa. It will then operate at incrementally increasing tonnages until the 2.0 Mpa steady state production rate is achieved.

Figure 12-50 below indicates the ore treatment profile associated with this option.

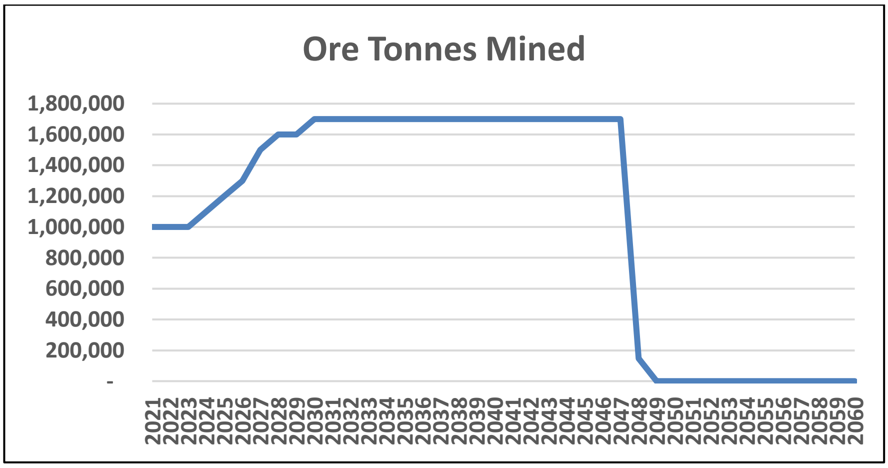


Figure 12-50: Ore Profile for 1.5 Mtpa with Smoothed Upgrade to 2.0 Mtpa Options

Besides plant capacity, the options evaluated considered three main ore categories, namely:

- The mining of ferro ore only, effectively a high grading strategy
- The mining of ferro ore plus trans ore above a 2.5% cut-off grade
- The mining of ferro plus trans with no cut-off grade applied.

Based on the above categories, nine options as follows were evaluated:

- Option 1 - mining of ferro ore only at a rate of 1.5 Mtpa
- Option 2 - mining of ferro ore only, starting at a rate of 1.5 Mtpa and assuming a stepped upgrade of plant capacity to reach a final nameplate capacity of 2.0 Mtpa
- Option 3 - mining of ferro ore only, having a smoothed upgrading of plant capacity to match the anticipated garnet sales profile
- Option 4 - mining of ferro ore and trans ore with a cut-off of 2.5% for rutile at a rate of 1.5 Mtpa
- Option 5 - mining of ferro ore and trans ore with a cut-off of 2.5% for rutile, having a stepped upgrade of plant capacity to reach a final nameplate capacity of 2.0 Mtpa
- Option 6 - mining of ferro ore and trans ore with a cut-off of 2.5% for rutile, having a smoothed upgrading of plant capacity to match the anticipated garnet sales profile
- Option 7 - mining of ferro ore and trans ore with no cut-off grade for the trans ore at a rate of 1.5 Mtpa
- Option 8 - mining of ferro ore and trans ore with no cut-off grade for the trans ore, having a stepped upgrade of plant capacity to reach a final nameplate capacity of 2.0 Mtpa
- Option 9 - mining of ferro ore and trans ore with no cut-off grade for the trans ore, having a smoothed upgrading of plant capacity to match the anticipated garnet sales profile.

For ease of understanding the above options are summarised in Table 12-16 below.

Table 12-16: Summary of Options Evaluated

Option Number	Option Description	1.5 Mtpa	1.5 Mtpa, Stepped Upgrade to 2.0 Mtpa	1.5 Mtpa, Smoothed Upgrade to 2.0 Mtpa
1	Ferro only	√		
2	Ferro only		√	
3	Ferro only			√
4	Ferro and Trans, Trans cut-off 2.5%	√		
5	Ferro and Trans, Trans cut-off 2.5%		√	
6	Ferro and Trans, Trans cut-off 2.5%			√
7	Ferro and Trans, no cut-off	√		
8	Ferro and Trans, no cut-off		√	
9	Ferro and Trans, no cut-off			√

Based primarily on IRR, Option 1, the mining and processing of ferro-eclogite only at a rate of 1.5 Mtpa, was selected as the preferred business case. A mine plan and production schedule was developed for Option 1, therefore, as reported below. Refer to Section 23 for more details on the financial analysis of the nine options.

12.10 Mine Plan and Production Schedule

A mine plan and production schedule for the LoM for Option 1 were developed using the DESWIK scheduling package. Key elements of the schedule are summarised below.

12.10.1 Mining Process

Mining will commence from an open pit which will give access from the highwall to an underground mine at a later stage in the LoM. It is planned to mine and process only one ore type, ferro-eclogite (which generally contains >16% Fe_2O_3 and >3% TiO_2). Trans-eclogite, which generally contains 14% to 16% Fe_2O_3 and 2% to 3% TiO_2, and Leuco-eclogite (which generally contains <14% Fe_2O_3 and <2% TiO_2), are considered future potential additions to the reserves.

Ore in the open pit will be drilled, blasted, loaded and hauled to a glory hole (ore pass) in the pit, after which it will undergo primary, secondary and tertiary crushing, followed by processing. Waste will be drilled, blasted, loaded and hauled to a waste rock disposal area to the north-east of the pit. To prevent blockages, there will be a grizzly (coarse screen) on top of the glory hole to ensure that large rocks do not enter the glory hole system. Once ore from the open pit has been dumped into the glory hole, primary crushing will take place underground by means of a jaw crusher, after which the ore will be transported via conveyor to one of two 20,000 t silos for storage. Ore from the ore silos will be reclaimed and conveyed to the secondary (cone) and tertiary (impact) crusher on surface at the process plant site. Secondary egress to the underground facilities as required by law is catered for by means of a second tunnel to the process plant site from the top of the silos. Processing of garnet will be achieved by means of milling, gravity concentration and dry magnetic separation to generate a fine and coarse product stream; rutile recovery is primarily achieved by means of flotation and spirals. Final products from the rutile, fine garnet and coarse garnet streams will be stored in product silos prior to shipping as bulk products from the port adjacent to the process plant site. All tailings from production will be safely placed in a deep sea fjord disposal area near the process plant site.

Towards the end of the life of the open pit, access to the underground orebody lying to the west of the pit will be made via a roadway driven through the highwall of one of the lower benches of the pit (bench 150 masl). This will enable decline development and access to open up five levels of underground mining. As noted, the underground mine plan is based on the long hole open stoping mining method, a bulk mining method whereby slots are opened at the top and bottom of a stope to enable vertical drilling and blasting of slices of ore within the stope. Ore is loaded out from the bottom level of the stopes and hauled to a crosscut where it is dumped into a glory hole. The glory hole enables the ore to be fed into a primary crusher, after which a conveyor will transport the ore to the silo system situated below the open pit. Both the glory hole and primary crusher for underground operations will be a duplicate of the open pit system established before the start of the open pit mining operation.

The Project is being developed in line with the guidelines of the JORC Code. The mine plan makes use, therefore, of Measured and Indicated resources only, which have been converted to Probable and Proven reserves by means of modifying factors. No Inferred ore resources have been used in the mine plan to determine the reserves.

12.10.2 Key Production Statistics

Key production statistics for the mine plan are summarised in Table 12-18 below.

Table 12-17: Mine Plan Key Production Statistics

Activity	Units	Total – Life of Mine	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Open Pit Mining																
Waste Mined	kt	30,446	2,139	1,866	1,377	1,202	1,670	1,787	1,151	893	3,238	3,608	1,895	3,090	3,512	1,599
Ferro Ore Mined (before losses and dilution)	kt	22,616	1,000	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,410
Underground Mining																
Waste Mined	kt	10,020	-	-	-	-	-	-	-	-	-	-	-	-	-	226
Ferro Ore Mined (before losses and dilution)	kt	19,432	-	-	-	-	-	-	-	-	-	-	-	-	-	90
All Mining																
Total Waste Mined	kt	40,466	2,139	1,866	1,377	1,202	1,670	1,787	1,151	893	3,238	3,608	1,895	3,090	3,512	1,826
Total Ferro Ore Mined (before losses and dilution)	kt	42,048	1,000	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500
Total Ferro Ore Mined Rutile Grade (before losses and dilution)	%	3.72	3.98	3.69	3.69	3.95	3.73	3.57	3.53	3.76	3.41	3.77	3.87	3.86	3.81	3.80
Total Ferro Ore Mined Garnet Grade (before losses and dilution)	%	40.3	39.6	39.7	39.9	41.6	40.1	39.7	40.6	41.2	35.4	35.2	35.0	35.8	36.6	40.7
Feed to Plant																
Ferro Ore Feed to Plant	kt	41,896	988	1,482	1,482	1,482	1,482	1,482	1,482	1,482	1,482	1,482	1,482	1,482	1,482	1,482
Ferro Ore Grade - Rutile	%	3.46	3.83	3.55	3.55	3.80	3.59	3.43	3.39	3.61	3.27	3.62	3.72	3.71	3.66	3.65
Ferro Ore Yield - Garnet	%	17.47	16.72	17.63	17.63	17.63	17.63	17.63	17.63	17.63	17.63	17.63	17.63	17.63	17.63	17.63
Plant Production and Sales	kt															
Rutile Produced	kt	910.4	20.2	32.4	32.3	34.6	32.7	31.3	30.9	32.9	29.9	33.0	34.0	33.8	33.4	33.3
Rutile Sales	kt	910.4	20.2	32.4	32.3	34.6	32.7	31.3	30.9	32.9	29.9	33.0	34.0	33.8	33.4	33.3
Garnet Produced	kt	7,318.1	165.2	261.3	261.3	261.3	261.3	261.3	261.3	261.3	261.3	261.3	261.3	261.3	261.3	261.3
Garnet Sales	kt	6,956.2	140.0	158.0	175.0	194.0	213.0	234.0	257.0	261.3	261.3	261.3	261.3	261.3	261.3	261.3

Table 12-18: Key Schedule Production Statistics

Activity	Units	Total – Life of Mine	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049
Open Pit Mining																	
Waste Mined	kt	30,446	862	556	-	-	-	-	-	-	-	-	-	-	-	-	-
Ferro Ore Mined	kt	22,616	1,004	1,201	-	-	-	-	-	-	-	-	-	-	-	-	-
Underground Mining																	
Waste Mined	kt	10,020	180	131	498	603	880	781	1,312	163	78	265	1,188	548	405	1,962	802
Ferro Ore Mined	kt	19,432	496	299	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	548
All Mining																	
Total Waste Mined	kt	40,466	1,042	687	498	603	880	781	1,312	163	78	265	1,188	548	405	1,962	802
Total Ferro Ore Mined (before losses and dilution)	kt	42,048	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	548
Total Ferro Ore Mined Rutile Grade (before losses and dilution)	%	3.72	3.89	3.73	3.52	3.74	3.89	3.78	3.99	3.61	3.51	3.78	3.56	3.40	3.74	3.64	3.57
Total Ferro Ore Mined Garnet Grade (before losses and dilution)	%	40.30	41.0	41.4	42.2	41.8	41.1	41.0	43.6	42.4	44.5	43.9	41.4	39.7	44.3	37.3	41.9
Feed to Plant																	
Ferro Ore Feed to Plant	kt	41,896	1,482	1,482	1,511	1,511	1,511	1,511	1,511	1,511	1,511	1,511	1,511	1,511	1,511	1,511	552
Ferro Ore Grade - Rutile	%	3.46	3.74	3.59	3.32	3.53	3.67	3.57	3.76	3.40	3.31	3.56	3.36	3.20	3.53	3.43	3.37
Ferro Ore Yield - Garnet	%	17.47	17.63	17.63	17.31	17.31	17.31	17.31	17.31	17.31	17.31	17.31	17.31	17.31	17.31	17.31	17.31
Plant Production and Sales																	
Rutile Produced	kt	910.4	34.1	32.7	30.9	32.8	34.1	33.2	35.0	31.6	30.8	33.1	31.2	29.8	32.8	31.9	11.4
Rutile Sales	kt	910.4	34.1	32.7	30.9	32.8	34.1	33.2	35.0	31.6	30.8	33.1	31.2	29.8	32.8	31.9	11.4
Garnet Produced	kt	7,318.1	261.3	261.3	261.5	261.5	261.5	261.5	261.5	261.5	261.5	261.5	261.5	261.5	261.5	261.5	95.5
Garnet Sales	kt	6,956.2	261.3	261.3	261.5	261.5	261.5	261.5	261.5	261.5	261.5	261.5	261.5	261.5	261.5	261.5	95.5

12.10.3 Methodology and Key Assumptions

The methodology and key assumptions used to develop the mine plan and schedule are summarised below:

- In line with the guidelines of the JORC Code, only Measured and Indicated Resources have been used to determine reserves. All ore tonnages and grades reported in the mine plan are for Probable and Proven reserves
- The mine plan targets an ore mining rate of 1.5 Mtpa, with the exception of the ramp-up year (2021). In total, 22.6 Mt of ore is mined from the open pit and 30.4 Mt of waste, giving a stripping ratio of 1.35 (t: t). In the underground 19.4 Mt of ore is mined and 10.0 Mt of waste, comprising development waste and ore in stopes which are not in the Measured or Indicated Resource classifications. In total, 41.8 Mt of ore and 40.5 Mt of waste is mined over the LoM
- Only ferro ore was mined and processed in the mine plan. Other ore types present in the Engebø orebody – trans ore and leuco ore – are treated as waste and are hauled to the waste rock disposal facility. Some of the waste material may be disposed of underground once mined-out stopes are available. The trans ore and leuco ore will be stockpiled separately to pure waste as consideration may be given in future to processing this material if it can be shown to improve the project economics
- The LoM is 29 years, with the open pit producing for 16 years, followed by bulk underground mining (long hole open stoping)
- Production is assumed to start in January 2021. A ramp-up in Year 1 has been incorporated. The ramp-up assumes that 30 kt of ore is processed through the plant in January 2021, with production building up to steady-state production of 125 kt in December 2021. The total ore to be processed in Year 1 is 1.0 Mt. The plant efficiency (a percentage of the expected steady-state recovery) is assumed to increase from 60% to 100% in six months for the garnet recovery part of the plant; due to the higher complexity of the rutile recovery part of the plant, it is assumed that it will take 12 months for the rutile plant to achieve 100% recovery efficiency
- Assuming a mining contractor is used upfront to establish the mine, it is estimated that it will take two years to construct and equip the underground infrastructure required to start production from the open pit (access tunnels from the process plant site, a crusher chamber; silos and reclaim system and a glory hole); this construction period is in line, therefore, with the time required to construct the process plant and associated infrastructure including product loadout facilities and bulk services. Construction of the underground infrastructure and the process plant is expected to run in parallel. For the open pit, all ore is mined to a glory hole in the pit; all waste is mined to a waste rock disposal facility to the north-east of the pit
- For underground mining, tonnages have been split into six material types, namely:

- Decline development in waste: a 15% allowance was added to the DESWIK production statistics to cater for cubbies and passing bays. This allowance is included in the numbers reported in Table 12-18 above
- Decline development in ore
- Horizontal development in waste: in the main, this represents development between stopes
- Horizontal development in ore. In the main, this represents development along the top of a stope to develop the slot from which bulk mining of the stope can take place
- Stoping in waste – this is ore which needs to be mined as part of the stope design but which is not Proven and Probable reserves of ferro ore
- Stoping in ore.

- The targeted production rate in the mine plan is 1.5 Mtpa of ferro ore mined; this equates to a feed to plant production rate of 1.482 Mtpa of ferro ore once ore losses and dilution have been included. The methodology used for applying ore losses and dilution to the mined ore tonnes is as follows:
  - Throughout the mine plan, ore losses of 5% were applied to the 1.5 Mtpa ore production rate, resulting in an ore feed to plant tonnage of 1.425 Mtpa
  - Dilution was then applied to the ore feed. A percentage dilution of 4% was assumed for the open pit and 6% for the underground. This increased the ore feed to plant tonnage to 1.482 Mtpa for open pit production and to 1.510 Mtpa for underground production
  - The final grade of the ore fed to the plant represents a diluted ore grade, where the grade of dilution was assumed to be nil for both open pit and underground dilution.
- Except for the ramp-up year (2021) when lower plant recoveries have been assumed, a recovery of 58.4% for pure rutile from ferro ore over the LoM has been assumed at a product grade of 94.9%. The diluted grade of ferro ore mined has been used to determine the production of final product. The recovery and product grade factors are based on testwork programme results. In line with the pricing strategy as outlined in Section 17, final rutile production tonnes reflect the selling of a 94.9% rutile product, so the plant output volumes of pure rutile have been increased by a factor of 100/94.9 to cater for product grade
- To determine garnet production volumes, a different approach was taken due to the bulk nature of the product. Irrespective of the garnet grade in ore a garnet yield of 18.3% was assumed for ferro ore throughout the Life of Mine (except for the ramp-up year when lower plant recoveries were assumed. The yield factor is based on testwork programme results. The yield factor was applied to the ferro ore tonnage sent to the Plant (ore mined less ore losses plus dilution) to determine the production of final

product. It should be noted that not all garnet produced is sold. In the early years of the Project, more garnet is produced than can be sold, so some garnet is disposed of with tailings by means of sea disposal. At steady state, the targeted garnet sales volume of 300 ktpa is not achieved, with garnet production only reaching 261.3 ktpa

- The underground stoping area is accessed via a decline from the open pit (from bench 150 masl). To act as a second egress, additional level drive extensions connecting to surface on the western side of the Engebø mountain have been planned at the 150 masl level and at the 90 masl level. These holings will also assist with the ventilation and drainage of the underground workings
- The following advance rates were used in the production schedule:
  - Access tunnels from plant site and decline advance rate = 4.5 m/day (135 m/month)
  - Glory holes development rate (including piloting and reaming) = 4.5m/day
  - Top and bottom stope slot development rate = 1,000 t/day
  - Stope mining rate – typically 5,000 t/day to 6,000 t/day.

12.10.4 Pit Development Sequence

Figure 12-51 to Figure 12-54 below illustrate the extent of open pit mining at various points in the LoM.

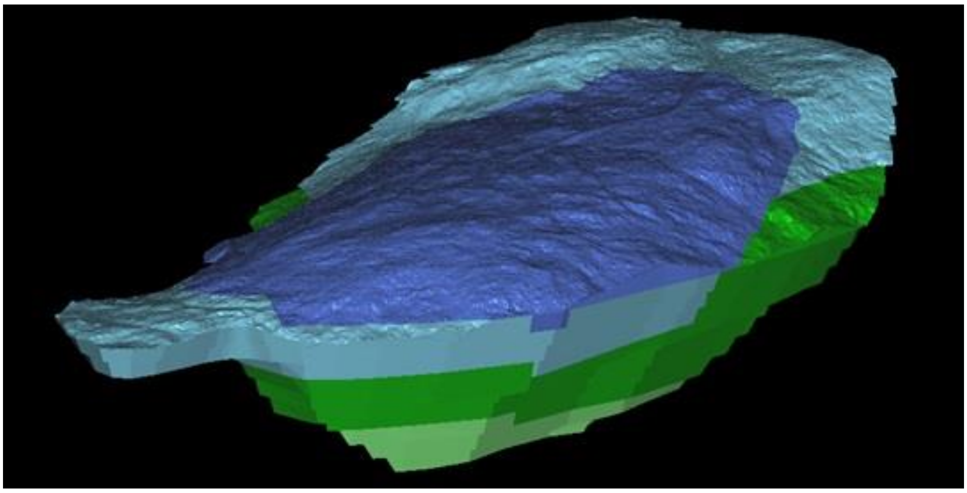


Figure 12-51: Pit Shape before Mining (looking North-East), Coloured in Years

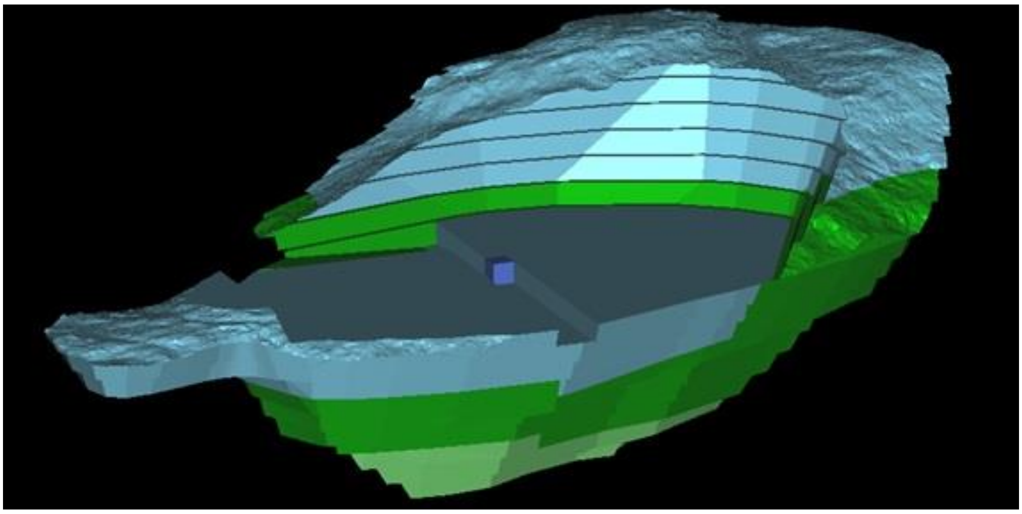


Figure 12-52: Extent of Mining after 5 Years (January 2026) (looking North-East)

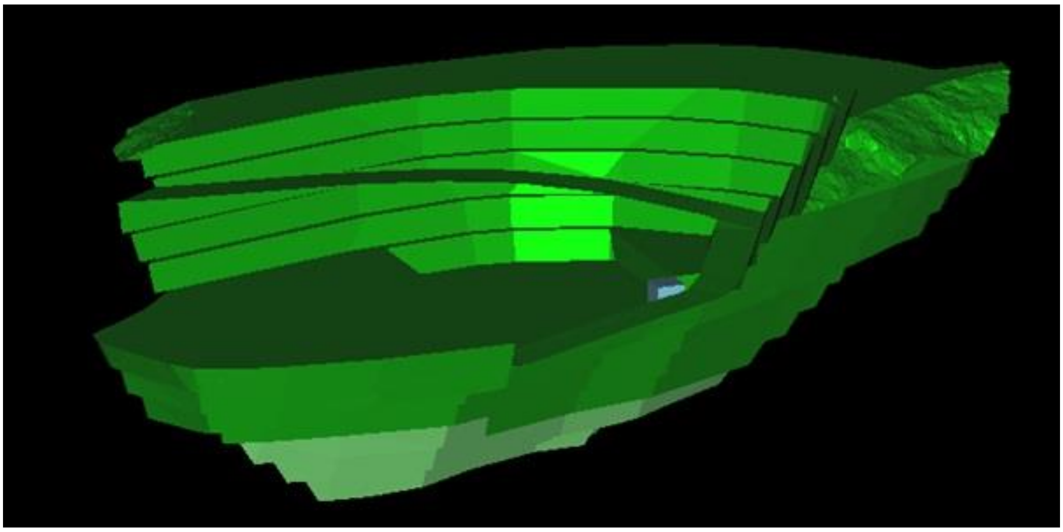


Figure 12-53: Extent of Mining after 10 Years (January 2031) (looking North-East)

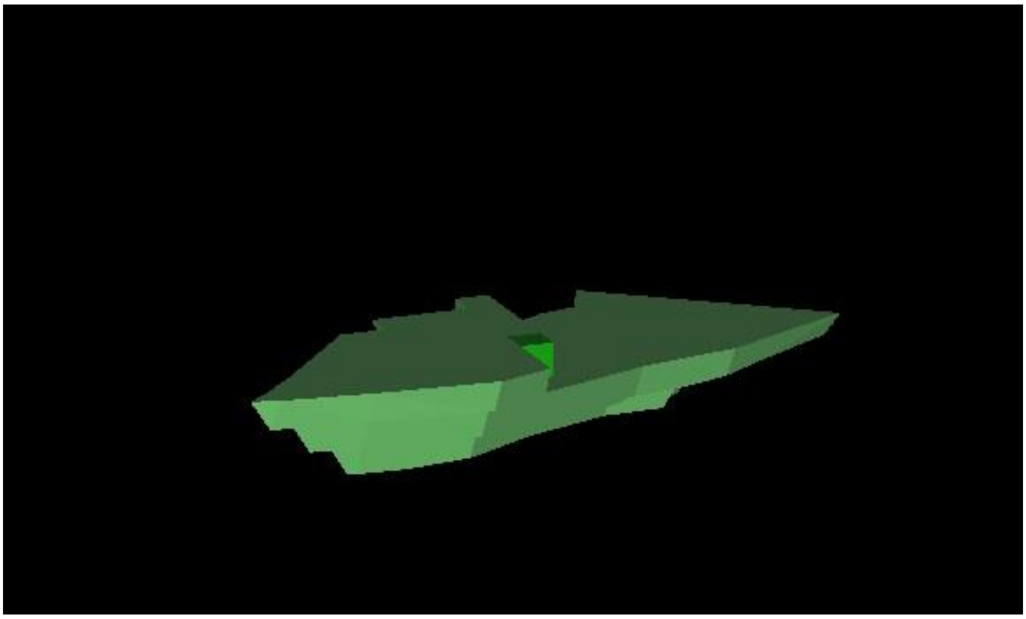


Figure 12-54: Extent of Mining after 15 Years (January 2036) (looking North-East)

12.10.5 Underground Development Sequence

Figure 12-55 to Figure 12-58 below illustrate the extent of underground mining at various points in the LoM.

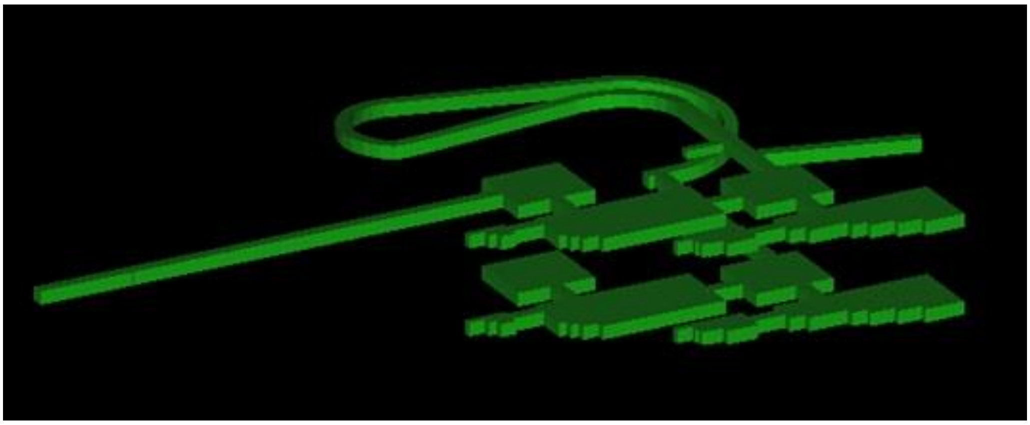


Figure 12-55: Extent of Underground Mining in January 2036

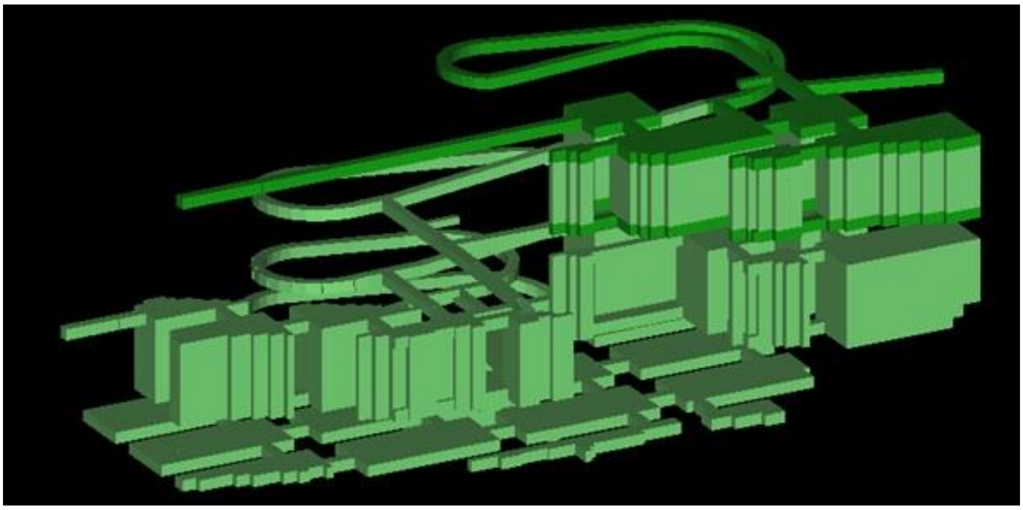


Figure 12-56: Extent of Underground Mining in January 2041

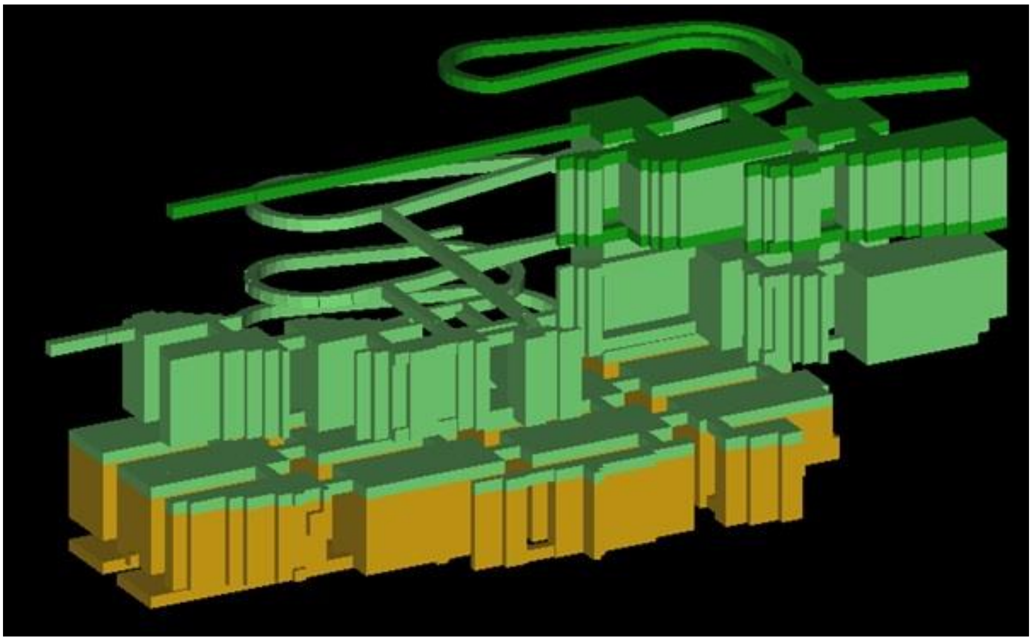


Figure 12-57: Extent of Underground Mining in January 2046

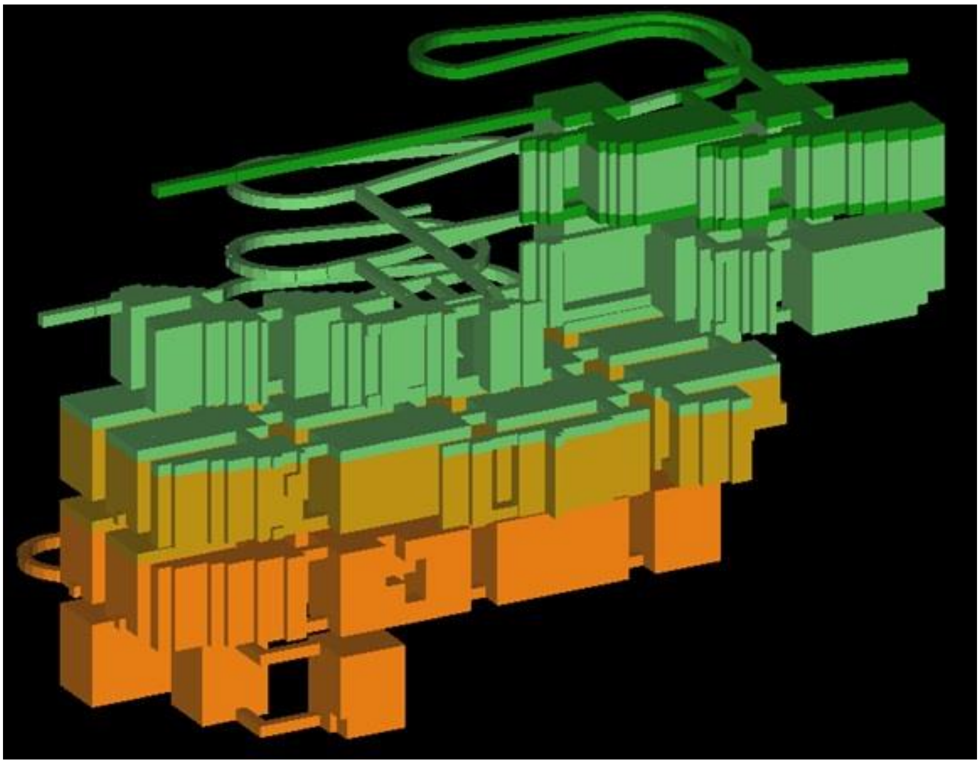


Figure 12-58: Extent of Underground Mining in 2049 (end of Life of Mine)

The full design used to derive the open pit and underground mine plans is shown in Figure 12-59 below.

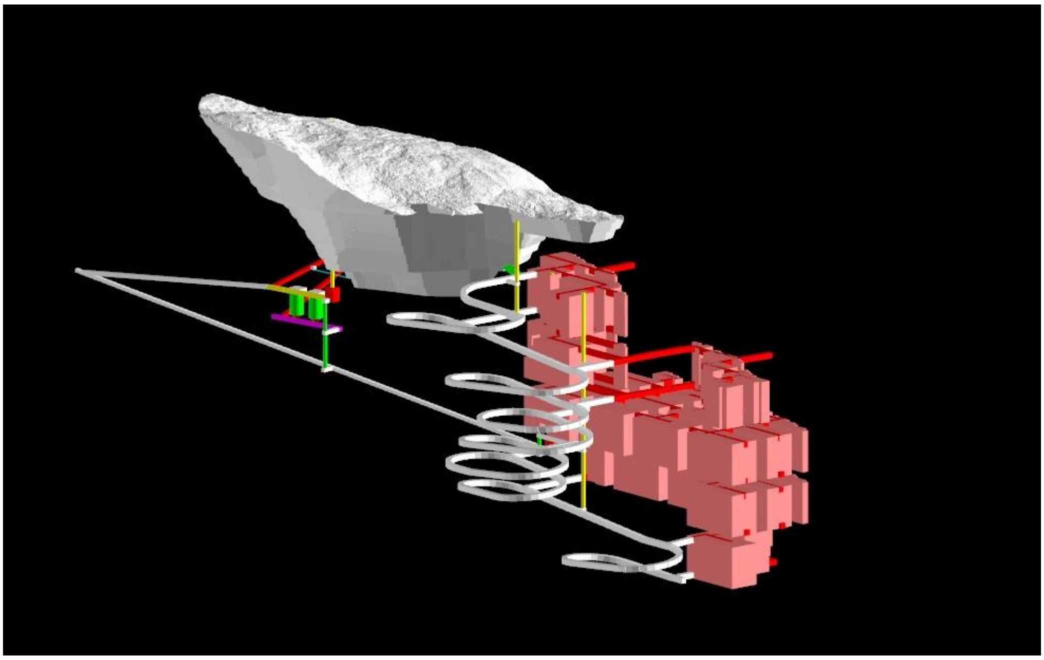


Figure 12-59: Open Pit and Underground Mine Plan (looking south-east)

12.10.6 Waste Rock Disposal Facility

A waste rock disposal facility (landfill deposition site) has been designed to fit in the valley to the north-east of the open pit, as shown in Figure 12-60 below.

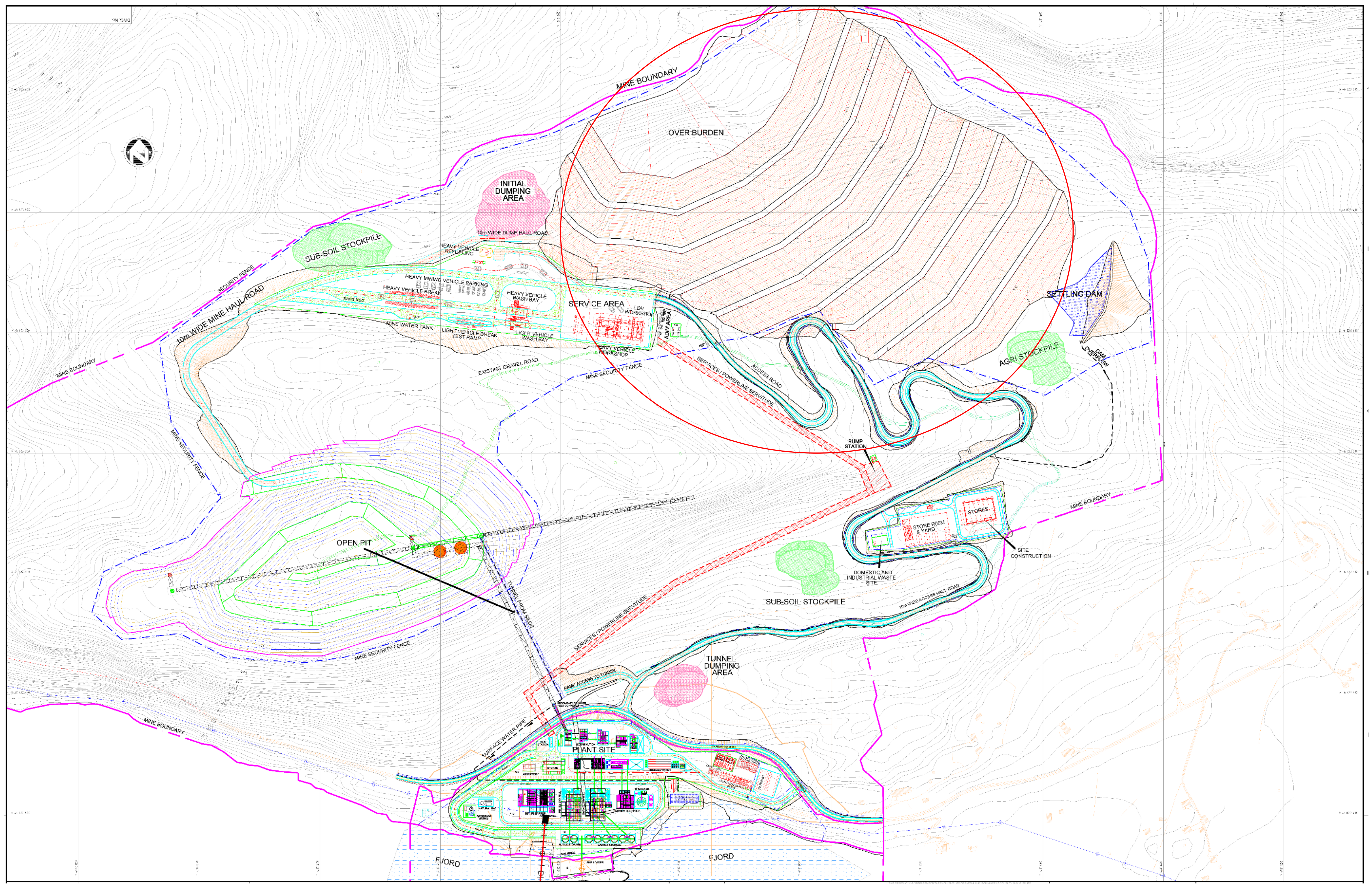


Figure 12-60: Plan Showing Waste Rock Disposal Facility (circled)

The disposal facility has been designed in nine steps (benches), each 20 m high, to provide stability to the rock face. Disposal will start from the lowest bench in the valley to provide a containment face, behind which the facility will be extended up the valley in line with the mining rate. The total volume of the disposal facility is approximately 14 Mm^3. Assuming an SG of 3.3 for rock and a loose rock factor of 1.6, the capacity of the facility is approximately 29 Mt. The discharge permit states that a maximum of 15 Mm^3 of waste rock can be disposed of in the landfill deposition site, or approximately 30 Mt. The permitted disposal tonnage is slightly higher than the disposal capacity, therefore.

The total tonnage of waste rock to be disposed of according to the mine plan is 40.5 Mt, consisting of 30.5 Mt of open pit waste and 10.0 Mt of underground waste. Therefore, additional disposal areas and/or solutions will be required such as:

- Disposal of underground waste in mined-out stopes underground – this will allow approximately 10 Mt of waste to be stored underground
- Optimisation of the mine plan in the DFS phase to reduce the amount of waste mining from underground
- The mining and processing of trans-eclogite so that it is treated as ore and not as waste
- Disposal of underground waste rock in the bottom of the pit once it is mined out
- Selling of waste rock as armour stone.

Once the mine plan has been optimised in the DFS phase, a waste rock storage solution will be drawn up to cater for the total volume as dictated by the mine plan.

12.10.7 Mining Labour

The mining labour requirement is summarised in Table 12-19 below, based on two mining shifts operating for five days per week, which is in line with the permitting restrictions of the Project. It should be noted that the labour count shown reflects typical values for the life of the open pit; in practice, the labour count may vary in some years, depending on the ore and waste stripping requirements and associated equipment numbers.

Once mining operations move underground, similar labour requirements are envisaged with the addition of one additional labourer per shift.

Table 12-19: Mining Labour Requirements – Open Pit

Mining (Two Shifts per Day, 5 Days per Week)	Head Count
Foreman	2
Primary Open Pit Equipment	
Haul Truck	4
Wheel Loader	2
Surface Drill Rig	2
General Labourer	2
Secondary Open Pit Equipment	
Wheel Dozer	2
Track Dozer	2
Track Dozer	2
Grader	2
Water Truck	2
Service Truck	2
Water Bowser	4
Total	**28**

12.10.8 Capital Cost Estimate

For a summary of the mining related capital cost estimate, refer to Section 21.

12.10.9 Operating Cost Estimate

For a summary of the mining related operating costs, refer to Section 22.

13. Ore Reserve Estimate

13.1 Overview

The approach taken to estimate the Ore Reserves included the following steps:

- The Mineral Resource block model was prepared for pit optimisation and underground design purposes
- Pit optimisation was carried out to produce a series of optimised pit shells
- The optimised pit shell was used as a guide in the design of an ultimate pit design, with haul road access, bench and berm configurations. This refined ultimate pit was used as the basis for Mineral Reserve evaluation
- The ultimate pit design was split into two pushbacks to improve the Project economics by deferring waste stripping. Practical mining constraints such as pit room for equipment and the annual drop rate of the pit were considered when defining the pushbacks. All leuco-eclogite and trans-eclogite was included as waste
- Based on the selected mining method (long hole open stoping) for underground mining, a stoping grid was overlain over the Measured and Indicated Resources
- The stoping layout was trimmed to cater for topographical constraints and constraints such as the pit and fjord geotechnical stability boundaries
- Stopes, pillars and sills in line with the geotechnical consultant's recommendations were designed within the remaining Measured and Indicated Resources
- The ultimate pit design with pushbacks, together with the underground design, was scheduled using the DESWIK software package to target an annual RoM capacity of 1.5 Mtpa at steady-state production.

13.2 Ore Reserve Statement

The ore reserve for the mine plan is presented in Table 13-1 below.

Table 13-1: Ore Reserve Estimate

Ore Type	Proven Reserves			Probable Reserves		
	M Tonnes	TiO_2 %	Garnet %	M Tonnes	TiO_2 %	Garnet %
Ferro Ore - Open Pit	8.519	3.87	43.8	13.826	3.54	41.8
Ferro Ore - Underground	1.675	3.49	37.8	17.876	3.21	37.8
Ferro Ore - Total	10.194	3.81	43.4	31.702	3.35	39.5

The basis of conversion of Mineral Resources to Ore Reserves is as follows:

- Ore reserve estimate is as of 30 September 2017
- Only Measured and Indicated Resources have been used to determine reserves; all Inferred Resources within the mineable envelope have been classified as waste

- Open pit mining is carried out for the first 16 years; thereafter the mining method is bulk underground mining (long hole open stoping)
- The open pit mine design is based on the recommendations of the geotechnical consultants for all pit design parameters
- The underground mine design is based on recommendations of the geotechnical consultants, assuming 100 m-long stopes, 45 m wide and 60 m high, with continuous pillars 20 m wide between stopes and sills 15 m thick above and below the stopes
- The garnet grades as reported above were not used to determine the final product volumes for garnet. Instead, a yield approach was used, which was considered to be more applicable for determining recoveries of a bulk mineral such as garnet. The yield approach assumed a yield of 17.6% garnet for ferro ore (18.3% before dilution was applied – see Table 11-1 for garnet recovery results)
- A rutile recovery of 58.5% was assumed, based on an operational factor of 97%
- A cut-off of 3% on TiO_2 has been applied to ferro ore
- Ore losses of 5% have been assumed throughout the mine plan
- Dilution of 4% for open pit and 6% for underground has been applied for all ore with a dilution grade of 0% for rutile and garnet.

13.3 Competent Person Sign-off

The mine plan and associated ore reserve statement for Engebø was compiled by an independent mining consultant, Mr. Adam Wheeler, who has been working on the Engebø Project since 2008. He has received full access to all available data and information connected with the deposit and project development, and has received unlimited assistance from all Nordic Mining personnel connected with the Project. Mr. Wheeler has visited the site several times, including three times during 2016, in connection with the recent drilling campaign.

As noted in Section 2.2, Mr. Wheeler qualifies as a Competent Person in terms of the JORC Code.

14. Project Infrastructure

The project infrastructure facilities, shown in Figure 14-1 below, can be broken down into the following main areas:

- Open pit mine facilities
- Haul road
- Underground mine facilities
- Process plant site facilities.

The facilities in each of the above main areas are described in more detail below.

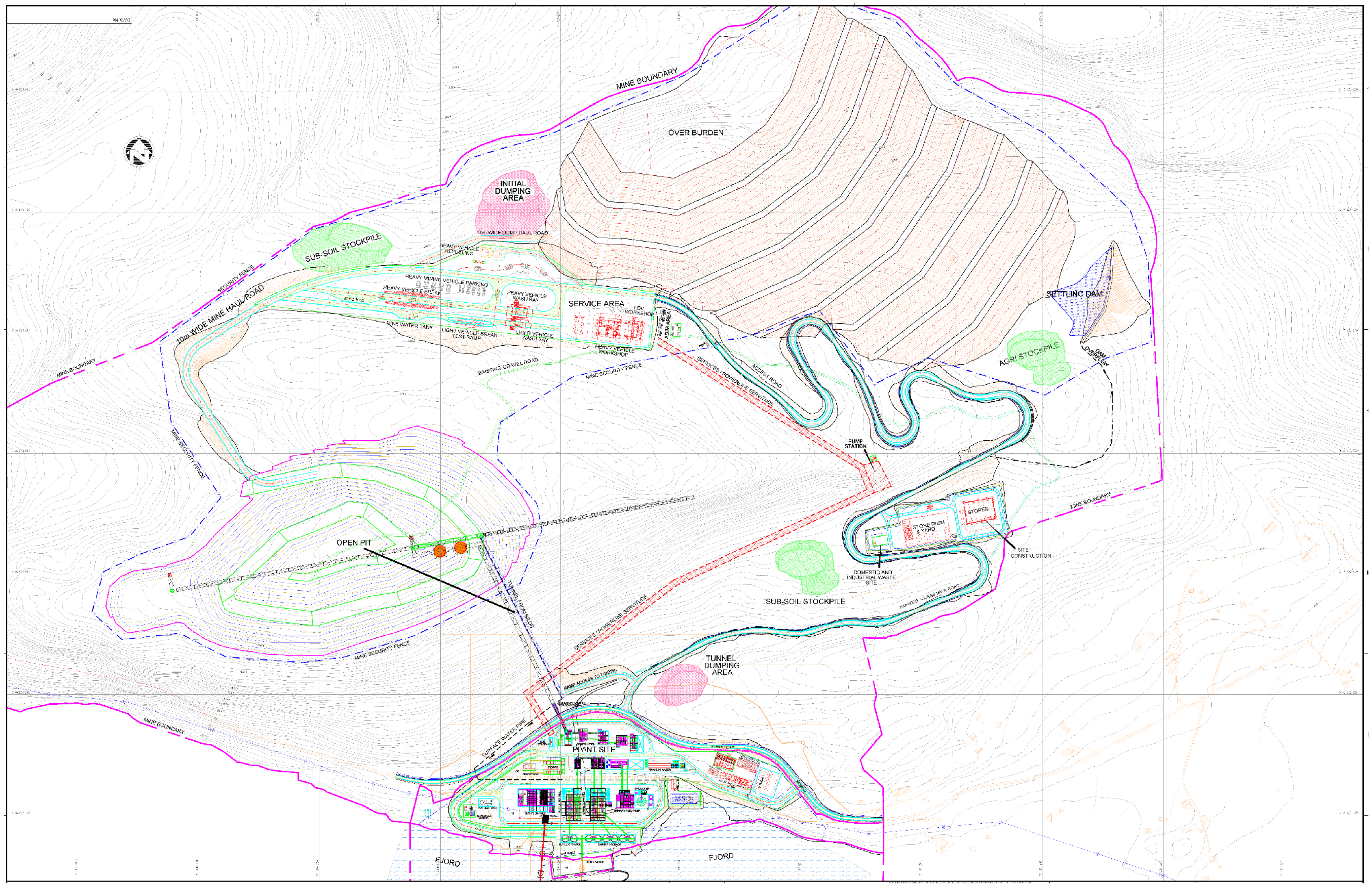


Figure 14-1: Site Layout

14.1 Open Pit Mine Facilities

The open pit facilities consist of the following:

- Separate stockpiles for agri-topsoil (the top seed-bearing layer of the topsoil) and subsoil stockpiles
- A waste rock disposal facility in the valley to the north-east of the pit to dispose of all open pit waste rock and a portion of the underground waste
- A settling dam for all runoff water from the waste rock disposal facility
- Open pit dewatering pipes and pumps
- Earthworks for the construction of workshops and a tyre bay to maintain the open pit equipment fleet, including heavy mine vehicle parking, a wash bay and brake testing ramps. It is assumed that the actual facilities will be built by the open pit equipment supplier
- Earthworks to establish an open pit explosives plant and magazine
- Open pit office administration buildings (to be used by Management, Geology, Mine Planning, Survey etc.) and an ablution block
- Security fencing around the waste rock disposal facility and open pit area.

14.2 Haul Road

A 10 m wide haul road will be constructed from near the plant site at the Fv 611 county road to the top of the open pit area. The haul road will be a new construction, following the route of the existing gravel access road to the top of Engebø for part of its route.

14.3 Underground Mine Facilities

Underground mine facilities will be built in two phases as follows to support open pit and underground mining:

- To support open pit mining, underground excavations will be built (see Section 12.7.2) which include a glory hole plus grizzly arrangement in the pit, a primary crusher and crusher chamber, a silo and ore reclaim system, top and bottom access to the silo system, an ore conveyor belt from the silo reclaim system to the plant site and a second egress from the top of the silo system to the plant site
- For underground mining, it has been assumed that a new ore pass (underground glory hole) and primary crusher chamber and crusher will be constructed to the east of the main underground mining areas. The crusher chamber will be connected to the existing silos and reclaim system by means of an underground conveyor belt system
- Electrical installations to support underground mining, including switchgear, substations, cabling and transformers

14.4 Process Plant Site Facilities

The general infrastructure and services required for the process plant facility include:

- Water storage and process water reticulation as described in Section 15.2 below
- Natural gas as described in Section 15.3 below
- Compressed air systems (instrumentation and plant air) as described in Section 11.6.3
- Dust and off-gas handling of the process plant dust extraction systems as described in Section 11.6.3
- Potable water and sewage treatment plants sized and provided for the operational personnel
- Communication as described in Section 15.5 below
- A cost allowance for fire protection and HVAC (Heating, Ventilation and Air Conditioning)
- Security fencing of the open pit mining facility as described in Section 14.1 above
- A cost allowance for the upgrade and rejuvenation of the existing port/quay at the Engebø site has been made.

The following administration and support buildings have been included:

- Administration office building
- Process plant control room building
- Change house, ablution and canteen buildings for operational and administrative personnel
- First aid building
- Process plant laboratory and stores buildings.

14.4.1 Tailings Disposal

The tailings disposal system for Engebø has been designed by COWI AS, an international consulting group based in Norway specialising in engineering, environmental science and economics. In their study COWI considered a number of designs and they recommend a system, termed the outfall design, which has been used successfully at Island Copper Mine, Vancouver Island, Canada.

COWI's design consists of a mixing tank, where seawater is added to tailings from the process plant, and an outfall line which transports the seawater/tailings mix to the bottom of the fjord. The system is self-propelling due to the difference in density between the diluted tailings and the seawater. Continuous monitoring of the discharge system is a vital element of the design. A schematic of the outfall design is shown in Figure 14-2 below.

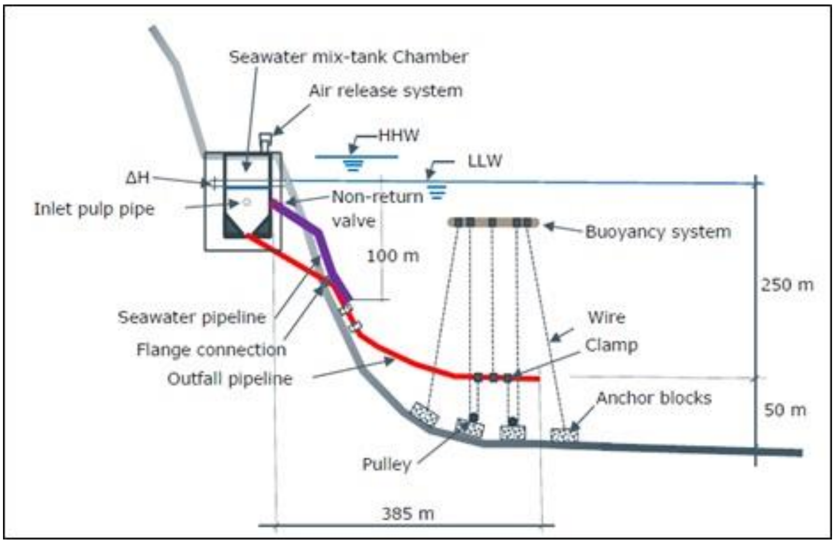


Figure 14-2: Schematic of the COWI Design for Sea Disposal of Tailings

15. General Infrastructure

15.1 Power

Bulk power for the Project will be supplied by SFE, the regional power supply company. By upgrading the existing 22 kV grid from Øyravatnet transformer station to Engebø in combination with a new 22 kV crossing of the fjord (most likely a subsea cable) and with additional grid reinforcements from the crossing-point to Engebø, SFE will have sufficient grid capacity and reliability for the Project. The power intake transformers positioned at the main incoming substation at Engebø will be supplied by SFE.

15.2 Water

Bulk water supply for the Project is planned to be sourced from the Skorven water system, situated at the southern side of the fjord. The power company SFE controls the waterfall, and it is possible to obtain water next to their power plant as part of their permit.

A new pump station will be constructed adjacent to SFE's power plant.

Four alternative water pipeline routes (as indicated in Figure 15-1 below) have been considered by Asplan Viak, who carried out a bulk water supply study for the PFS. Alternative 1 option, which has the most favourable routing and the lowest capital cost, has been selected as the basis for this study.

The distance for sourcing bulk water from Skorven to Engebø is approximately 9 km.

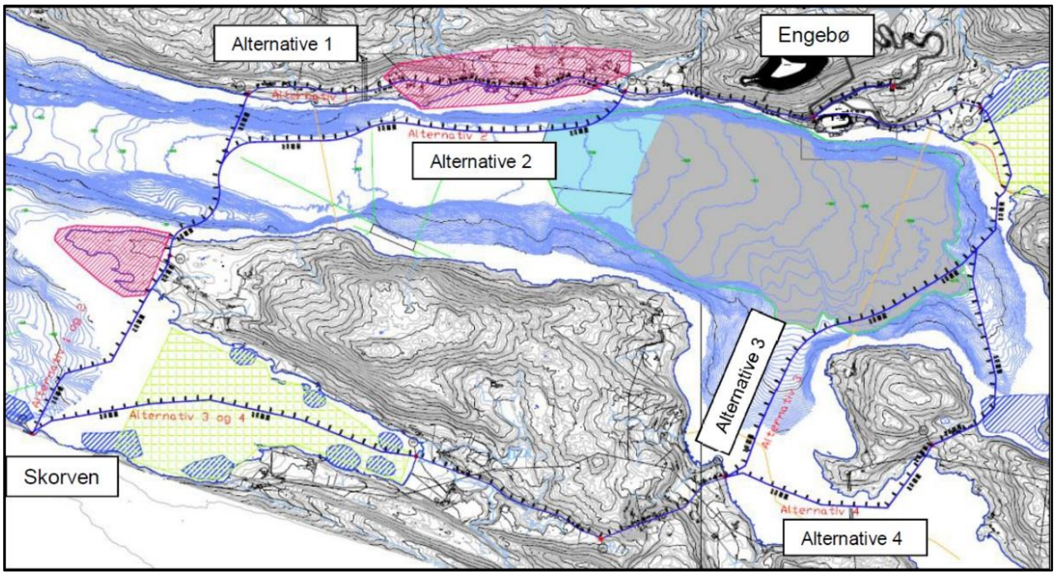


Figure 15-1: Bulk Water Supply – Alternative Routes considered from Skorven to Engebø

The bulk water pumped to Engebø will be discharged into the fire and raw water storage tanks at the process plant facility.

15.3 Natural Gas

Natural gas for the Project will be supplied by a local gas supplier. An appropriate area for gas storage tanks has been located on site, and the supplier will transport gas to site by road on a weekly basis.

15.4 Access Roads

The main county road (Fv 611) providing access to the Engebø site will be diverted around the process plant site. The re-routing design and cost estimate of this road have been completed by Asplan Viak for this study.

15.5 Communications

A cost allowance for communications on site has been made for this study. At the time of project implementation, it is envisaged that a 5G communication link will be available at the Engebø site.

16. Engineering Design

16.1 Civil and Earthworks Design

The bulk earthworks consist of a raised terrace area at three distinct levels (as described in Section 11.8) for the process plant facility. A storm water sand trap has been included to facilitate the storm water control around the process plant site.

Security fencing is positioned around the full perimeter of the open pit mining area and the Process Plant area.

The terraces and bulk earthworks were modelling in a 3D CAD environment and quantities based on the LIDAR survey and the 3D model. Fencing quantities were established by measurements of the overall site layout plan.

The route and alignment of the mining haul and access haul roads (as described in Section 14.2) have been indicated on the overall site layout plan. In-plant roads for the process plant have also been included.

16.2 Structural Design and Engineering

The structural steel and concrete quantities are based on the mechanical 3D model of the plant. Quantitative metrics, based on similar projects, were used to determine the overall steel tonnages and concrete cubic meters.

No basic engineering analysis and design was done for the steel or concrete scope; this will be developed further in the DFS phase of the Project.

The quantitative metrics consider the local environmental factors such as wind and temperature. These metrics have assumed no pre-assembly or modularisation of the plant and are based on the plant being constructed as “stick built”. This will be investigated further in the DFS phase of the Project.

The metrics also consider the use of the structure, for example heavy cranes or equipment, as well as whether the structures are open or enclosed with cladding.

No detailed geotechnical information regarding founding conditions was available and it was assumed that standard pad footings, strip footings and, in selected cases, raft foundations will be used. It is recommended that a geotechnical investigation is conducted in the DFS phase of the Project.

16.3 Mechanical Design and Engineering

The mechanical equipment required for the process plant is prescribed by the process design and site layout requirements, as described in Section 11.8. All equipment indicated on the PFDs has been sized and selected in accordance with these requirements. The mechanical equipment requirements (such as maintenance and access) as indicated in the preliminary Mechanical Design Criteria have also been considered.

Preliminary vendor equipment sizes and information has been used to develop the mechanical 3D model of the plant. A mechanical equipment list for the 1.5 Mtpa plant with preliminary vendor information has been generated. The plant mechanical layout and general arrangement drawings, as well as preliminary design calculations for equipment dependent on the layout (pumps, sumps/bins and conveyors) have been generated for this study.

Mechanical equipment supply costs have been obtained from budget quotations received from the market. About 90% of the mechanical equipment costs have been obtained from the market, while 10% of the costs have been obtained from database costs from similar recently-completed projects. Database rates have been used for conveyors and platework.

16.4 Instrumentation Design

No instrumentation design was completed for this study. This will be developed further in the DFS phase.

16.5 Electrical Design

The electrical engineering and design has been based on the electrical design criteria with the following deliverables being generated for the study:

- Electrical load list: the electrical load list had been based on the mechanical equipment list with a total estimated load of 10.5 MVA
- Single line diagram for the overall medium voltage reticulation based on the reticulation network simulation study
- Cable schedule: this was based on the electrical load list and site general arrangement drawings. Cable lengths had been estimated and scaled from the site general arrangement drawings
- Medium voltage reticulation network load flow and fault current simulation study: this study had been carried out using the software package ETAP (Electrical Transient Analysis Program). The short circuit power had been based on an estimated 10.5 MVA at 66 kV. This number needs to be confirmed in the DFS phase prior to further network simulation studies. Electrical plant item data have been based on the ETAP standard library. With a total power factor correction facility of 4 MVAr, no undesirable voltage regulations were recorded
- Typical schematic diagrams: these have been based on standard typical schematic diagrams as well as the load list requirements. These were used for pricing of the proposed motor control centers (MCCs).

The intake power battery limit will be the secondary terminations on the two intake power transformers. These transformers will be supplied by the Power Supply Utility/Company (SFE). Electrical plant item battery limits for the mechanical packages will be the termination of power onto electrical motors, hydraulic power packs etc. Lighting and small power has been excluded from the engineering and design. A factored cost allowance for these items has been included.

17. Market Information

A summary of the market studies completed in this phase is given below. The market study for rutile was carried out by TZMI, a leading global independent consulting and publishing company which specialises in technical, strategic and commercial analyses of the opaque minerals, chemical and metals sector. TZMI has a worldwide presence with offices in Australia, China, Africa and the USA. The market study for garnet was carried out by TAK Industrial Mineral Consultancy, a UK based minerals marketing company.

Nordic Mining has signed a Memorandum of Understanding (MoU) with a leading, international producer of industrial minerals. The parties intend to establish long-term cooperation within development, sales, marketing and distribution of garnet products from Engebø. This may include off-take agreements, joint marketing, and sales and distribution arrangements for garnet products to be sold in international markets.

Further market information in relation to garnet production, assumptions of market penetration, product qualities and sales has been guided by Nordic Mining's MoU partner.

17.1 Rutile Market

17.1.1 Introduction to Titanium Feedstocks

The mineral sands industry is orientated primarily towards the supply of titanium raw materials to produce titanium dioxide (TiO_2) pigments and titanium metal. The term "mineral sands" refers to concentrations of minerals commonly found in sand deposits, which include the titanium minerals ilmenite and rutile. The other mineral of significance usually found in these deposits is zircon, which most producers consider a co-product of their titanium mineral products.

Ilmenite is the most abundant titanium mineral and typically has a TiO_2 content ranging from 44% to 65% depending on its geological history. TZMI has classified ilmenite feedstock with TiO_2 content between 58% and 65% as chloride ilmenite and ilmenite with TiO_2 content between 44% and 57% as sulphate ilmenite.

Leucoxene is a natural alteration product of ilmenite, having a TiO_2 content ranging from 65% to more than 90%. The weathering process responsible for the alteration of ilmenite to leucoxene results in the removal of iron and hence, upgrading of the TiO_2 content. Circulating groundwater can also redeposit impurity elements within and around the weathered ilmenite grain. Commercial leucoxene products in the marketplace have a very wide range of compositions with varying levels of impurities.

Rutile is composed essentially of crystalline titanium dioxide and, in its pure state, would contain close to 100% TiO_2. Naturally occurring rutile exhibits minor impurities and commercial concentrates of the mineral typically contain 94% to 96% TiO_2.

Naturally-occurring high-level TiO_2 minerals suitable for the chloride process for production of TiO_2 pigments are limited in supply. This situation has prompted the mineral sands industry to develop beneficiated products that can be used as substitutes for, or in conjunction with, natural rutile. Two processes have been developed commercially, one for synthetic rutile manufacture and the other to produce titanium slag. Both processes use ilmenite as raw material and are essentially processes for the removal of iron oxides.

- Synthetic rutile: this is a product that is made by pyro-metallurgical processing of an ilmenite, typically having a 58% to 62% TiO_2 content, to remove the iron. The final product typically has a TiO_2 content of 90% to 95%
- Sulphate and chloride slag: a slag is created when an ilmenite source is reduced in an electric furnace to produce two products: metallic iron and a TiO_2 slag. The slag is then crushed and, depending on the impurity content, is either used in the chloride or the sulphate process. Commercial sulphate slag products generally have a TiO_2 content of 75% to 80%, while chloride slag products have a TiO_2 content of 85% to 90%. The fines which are generated when crushing chloride slag are mostly used in the sulphate process under the name “chloride fines”
- Upgraded slag or UGS**:** UGS has a TiO_2 content of more than 94%. It is only produced by Rio Tinto Fer et Titane in Canada.

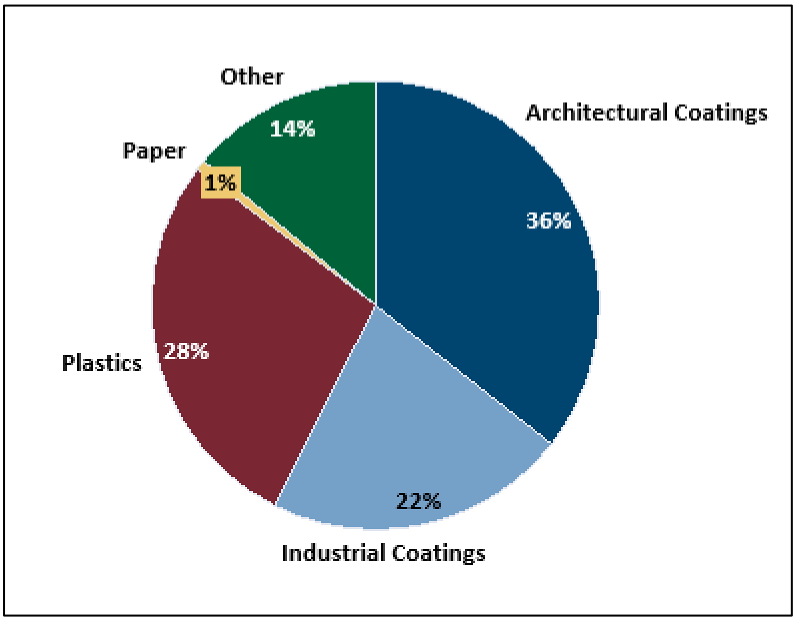
17.1.2 TiO_2 Pigment Market

The global TiO_2 pigment market accounts for approximately 90% of all titanium feedstock demand, and is, therefore, the dominant driver of offtake. The following analysis is based on TZMI’s latest supply/demand update completed in August 2017.

TiO_2 pigment is used predominantly in the production of high-quality surface finishes, and is essentially a lifestyle product. Historically, its use has developed strongly in the most economically developed countries of the world where it is an essential component of basic consumer products such as housing, motor vehicles and plastic products.

TiO_2 consumption generally increases as disposable income increases, and thus there is a close link between GDP growth, urbanisation and TiO_2 pigment consumption. The evolution of the demand historically was driven by a significant shift in urbanisation in emerging economies (most notably China) and the shift of the industrial base from the west to the east as free trade agreements were put into place. Future evolution will be driven more by changing consumption patterns of the newly industrialised countries in the east and growth in infrastructure. Movements away from free trade agreements – both in the US and in Europe – may shift these growth patterns.

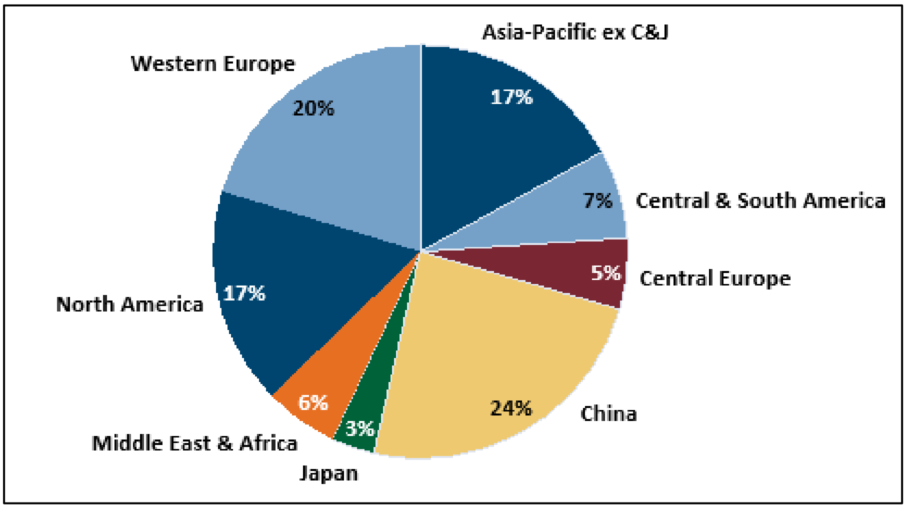
Figure 17-1 below shows the segmentation for TiO_2 pigment demand in 2016, while Figure 17-2 below indicates regional demand for the same period.



Source: TZMI ©

Figure 17-1: Global TiO_2 Pigment Demand by End-use Segment in 2016

Total demand for 2016 is estimated at approximately 5.99 Mt globally, with China and Europe dominating demand, accounting for nearly 50% of global TiO_2 pigment consumption.



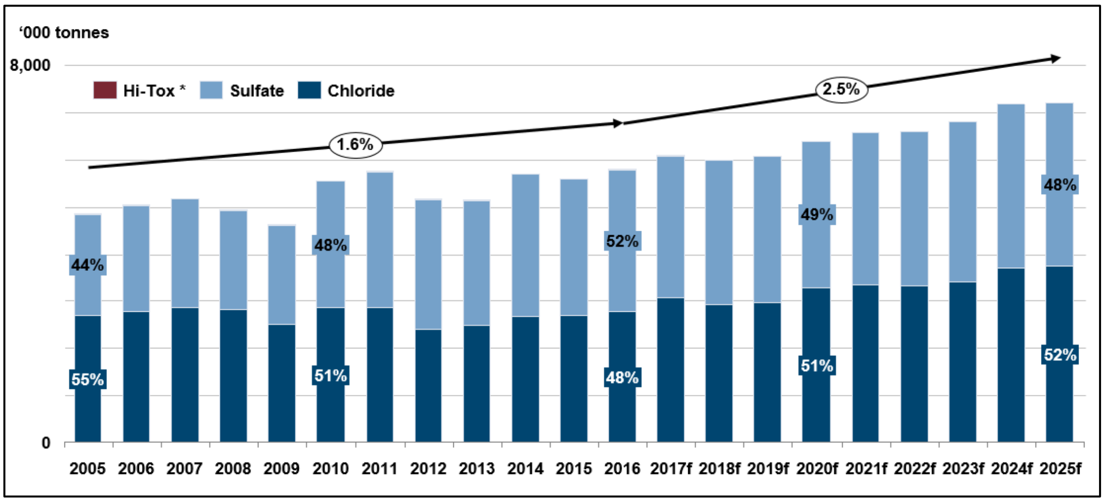
Source: TZMI ©

Figure 17-2: Regional TiO_2 Pigment Demand for 2016

Global pigment production capacity grew from 5.4 Mt in 2005 to 6.7 Mt in 2010 and 7.2 Mt in 2016. The industry is dominated by six producers of which five operate in multiple regions. These six producers account for approximately 61% of global capacity.

Figure 17-3 below shows TZMI's forecast for global TiO_2 pigment production between 2010 to 2025. In terms of supply, global TiO_2 production is estimated at 5.80 Mt for 2016, up 3.4% from 2015 levels and recapturing any losses seen in 2014. TZMI is forecasting global TiO_2 pigment supply to reach close to 6.4 Mt by 2020 and 7.2 Mt by 2025, a growth rate of approximately 2.5% per annum. It is expected that Chinese pigment production will display the highest growth rates globally.

Sulphate pigment accounted for 52% of total production in 2016, but this is expected to reverse by 2025, with chloride pigment accounting for a slightly larger proportion of the production share.



Source: TZMI ©

** Hi-Tox is a beige coloured rutile TiO_2 pigment produced by TOR Minerals, designed for use in non-white paints or coatings*

Figure 17-3: Global TiO_2 Pigment Production by Technology: 2010 to 2025

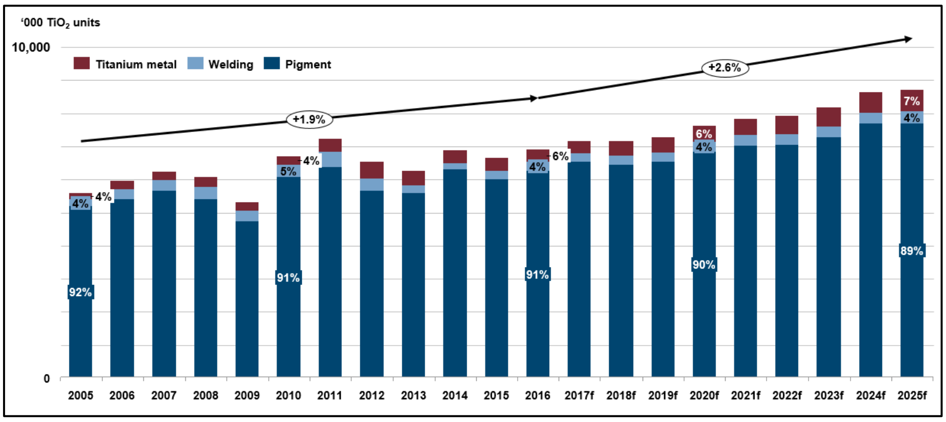
The TiO_2 pigment industry has been plagued with supply overhang during the past few years, which did nothing to bolster demand for titanium feedstocks. As a result, the global titanium feedstock market has, up until recently, been in a state of oversupply, with high inventory levels throughout the supply chain. Most pigment producers are understood to be holding normal to low inventory currently, based on recent public releases by companies.

17.1.3 Titanium Feedstock Market

Global demand for titanium feedstock is dominated by the TiO_2 pigment end-use. TZMI estimated that pigment end-use accounted for approximately 91% of total demand in 2016, or 6.26 M TiO_2 units.

For the purposes of the titanium feedstock supply/demand analysis, TiO_2 unit is used as the common denominator to account for the varying TiO_2 content in feedstock products and to address the issue of combining consumption of products with a range of TiO_2 levels. One TiO_2 unit is equal to one ton of contained TiO_2.

Figure 17-4 below shows the TZMI forecast for TiO_2 feedstock demand by end-use segment between 2005 and 2025.



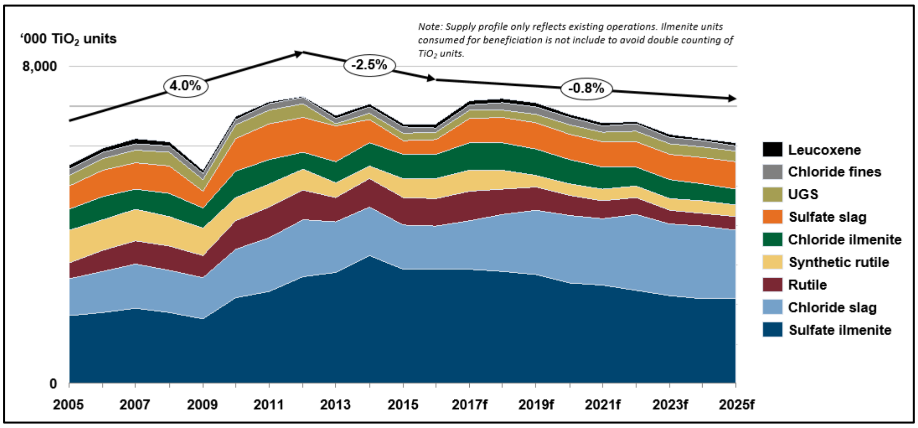
Source: TZMI ©

Figure 17-4: TiO_2 Feedstock Demand by End-use: 2005 to 2025

Global demand for titanium feedstocks is estimated to reach 8.7 M TiO_2 units by 2025, a growth of 2.6% Compound Annual Growth Rate (CAGR). Pigment is expected to lead the growth in volume terms, adding 1.44 M TiO_2 units during the next nine years, with a CAGR of 2.3%. Both titanium metal and other uses are also expected to show strong growth, estimated at 5.6% and 3.8% CAGR respectively, albeit from a lower base.

In terms of supply, global titanium feedstock production is estimated at 6.55 M TiO_2 units in 2016. To avoid double counting, total feedstock supply reflects net production excluding production of ilmenite used in the manufacture of titanium slag and synthetic rutile.

Figure 17-5 below shows the global titanium feedstock supply between 2005 and 2025.



Source: TZMI ©

Figure 17-5: Global Titanium Feedstock Supply by Product: 2005 to 2025

Following a period of continuous growth between 2005 and 2012, global supply fell modestly during the past few years, partially reflecting the decision by several major producers to cut back on production to meet demand, to avoid the build-up of feedstock inventory.

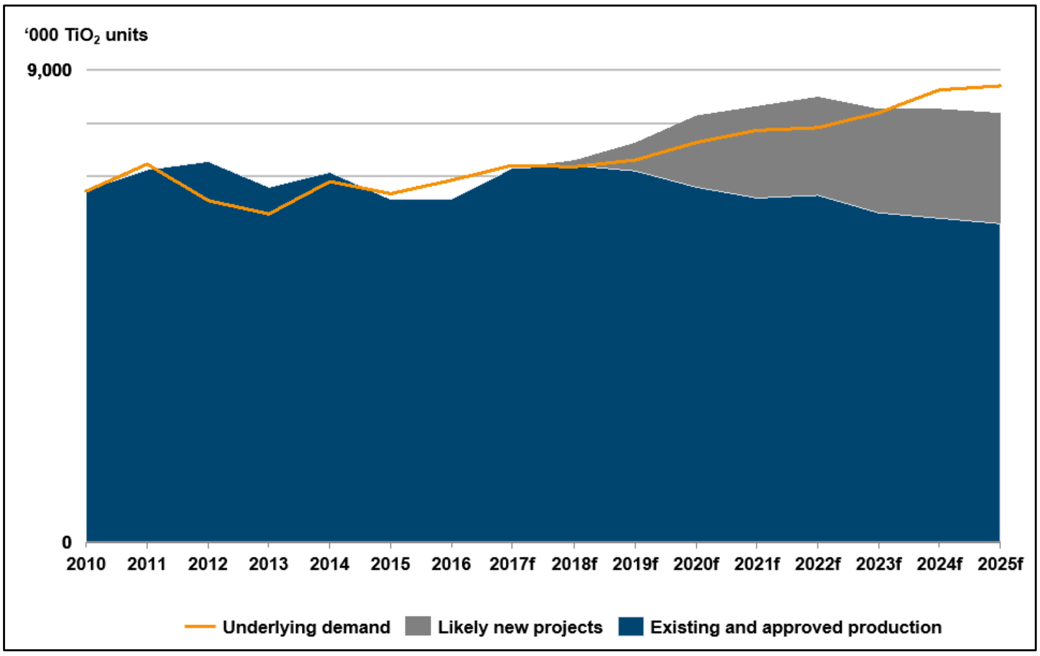
TZMI's current forecasts suggest a supply recovery during 2017 and 2018 as market conditions improve, with output of most feedstocks projected to move higher. In particular, output of chloride slag is forecast to rise considerably, underpinned by the production ramp up at TiZir Tyssedal, improved utilisation rates in South Africa, and higher output at Cristal Jazan.

In addition, higher sulphate ilmenite output from China is anticipated during the forecast period as progressively higher sulphate ilmenite prices are being achieved, prompting some vanadium titano-magnetite (VTM) producers and processing plants to increase production. Higher furnace utilisation rates in Canada and South Africa are also expected to contribute to supply growth in the medium term.

For 2016, the global feedstock market was in considerable supply deficit, estimated at 356 k TiO_2 units, predominantly reflecting the 4% increase in global demand, while supply remained flat year-on-year.

Figure 17-6 below shows the supply/demand balance and outlook to 2025 for titanium feedstocks.

TZMI expects overall feedstock supply and demand to be in balance in 2017 and 2018, before new projects bring additional supply into the market from 2019. The probabilistic estimate of new supply could see oversupply occur through to 2023 before demand catches up. However, in all likelihood, only the initial projects to be financed for construction in 2017 and early 2018 will come on stream in this period. Projects that do not obtain financing are likely to be pushed out into the early/mid 2020s to align with the next supply deficit that is forecast.

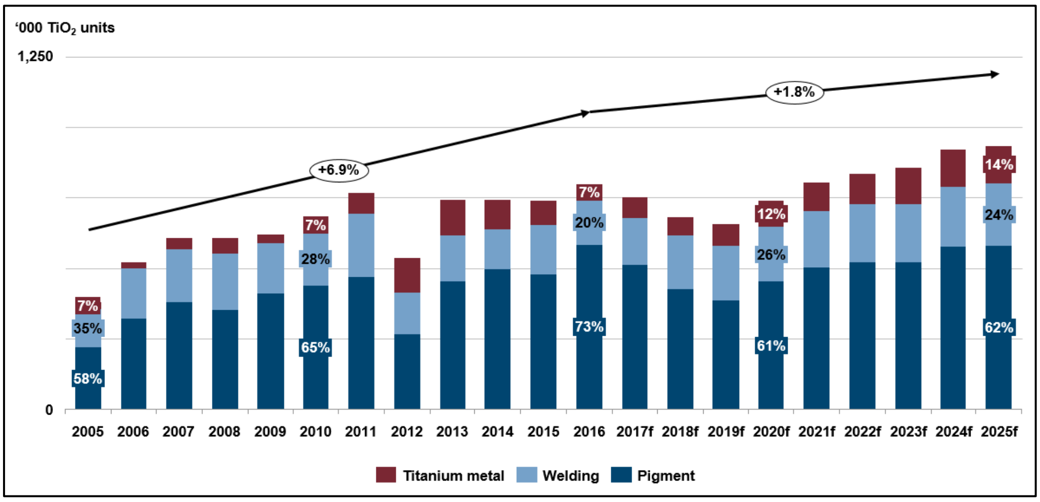


Source: TZMI ©

Figure 17-6: Global Titanium Feedstock Supply/Demand Balance

17.1.4 Rutile Supply/Demand and Outlook to 2025

As with other TiO_2 feedstocks, rutile is consumed largely for pigment manufacture with consumption by TiO_2 pigment producers accounting for approximately 73% of global rutile demand in 2016 or 548 k TiO_2 units. Rutile consumption in titanium sponge manufacture and other uses account for the remaining 7% and 20% respectively. In the other uses segment, rutile is predominantly consumed for the manufacture of welding electrode fluxes. Figure 17-7 below depicts the global demand for rutile for the period 2005 to 2025.



Source: TZMI ©

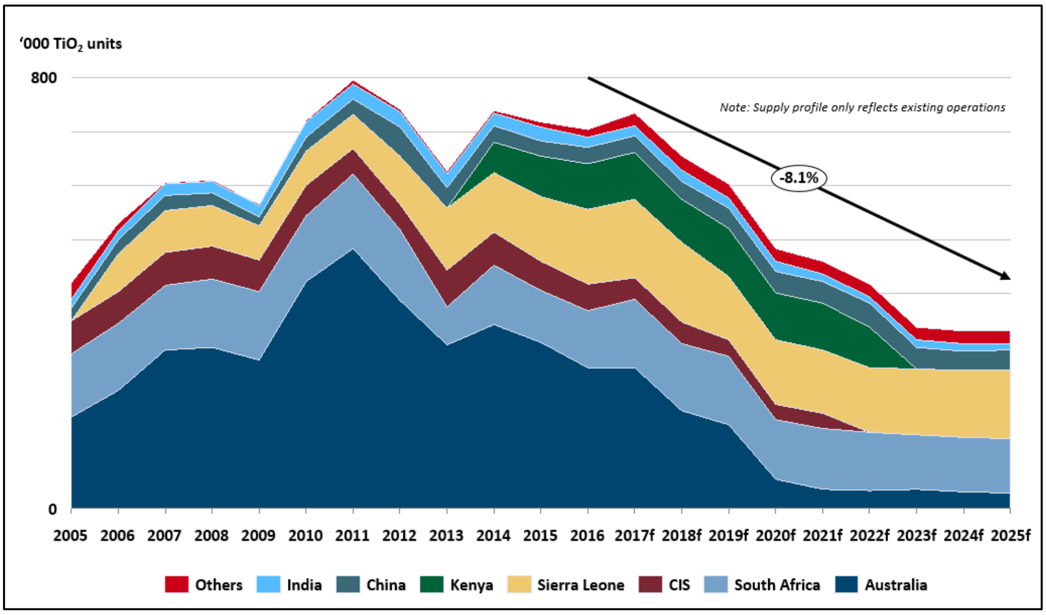
Figure 17-7: Global Rutile Demand: 2005 to 2025

Global rutile demand growth for all end-use over the next nine years (2016 to 2025) is expected to average 1.8% CAGR, reaching 930 k TiO_2 units by 2025, or an increase of approximately 140 k TiO_2 units on 2016 levels.

Consumption for titanium metal end-use is expected to lead demand growth, adding 77 k TiO_2 units during the forecast period. Strong demand growth from this end-use is projected due to demand pull from commercial aerospace flowing through to sponge demand with the rollout of next generation aircraft, particularly the B787 and A350, which consume more titanium by weight compared to the older models.

Following the decline in prices initiated by the destocking by pigment producers in late 2012, there has been an increase in rutile consumption by pigment producers to take advantage of the low rutile prices (on a relative economic value basis) compared to other high-grade feedstocks, but this trend is expected to reverse in the short term as prices begin to trend up during the period 2017 to 2020. The share of rutile consumption for TiO_2 pigment use is estimated at 62% by 2025.

As far as rutile supply is concerned, global rutile production was down 2% year-on-year in 2016 to approximately 773 kt (704 k TiO_2 units). The majority of this decline can be attributed to reduced output from the Australian operations, offset by higher output in other regions. TZMI's global rutile supply forecast is shown in Figure 17-8 below.



Source: TZMI ©

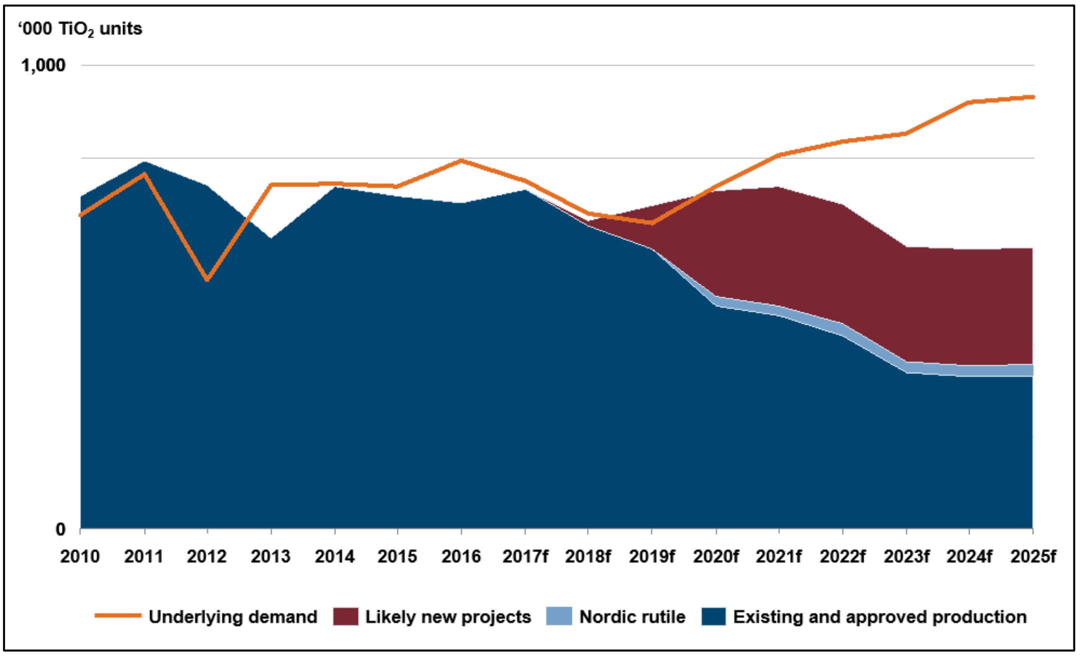
Figure 17-8: Global Rutile Supply from Existing Operations: 2005 to 2025

Global supply of rutile is set to decline considerably during the forecast period, with output in 2025 expected to be 50% lower than 2016 levels.

Australia's position as the global rutile powerhouse is slowly diminishing as mine grade declines and with the closure of several mines. Mining at Iluka Murray Basin was completed in early 2015, with rutile currently produced from heavy minerals concentrate stockpiles. There will be no rutile output from this location following the scheduled closure of the Hamilton Mineral Separation Plant in October 2017. Sibelco, another major rutile producer, is also expected to close its North Stradbroke mine in 2019.

Even with the onset of likely new supply from new projects, global rutile output is unlikely to exceed 800 k TiO_2 units.

Figure 17-9 below shows the supply/demand balance for rutile for the period to 2025.



Source: TZMI ©

Figure 17-9: Global Rutile Supply/Demand Balance to 2025

The global rutile market was in significant surplus in 2012 as chloride pigment producers embarked on a de-stocking cycle and curtailed pigment production, resulting in reduced consumption of high-grade feedstocks. The resultant inventory overhang is progressively being worked through and by end-2016 should have trended back below normal levels.

General consensus is that the global rutile market will experience tight market conditions in 2017, underpinned by improving market conditions in the chloride pigment sector.

The longer-term outlook indicates that a significant supply deficit will develop if no new projects are commissioned. TZMI has adjusted its demand outlook for rutile in the short to medium term to take into consideration the interchangeability of feedstock blend among some pigment producers given the declining rutile supply profile and the availability of other high-grade feedstocks. However, the trend should reverse beyond 2020 once the existing supply of other high-grade feedstocks is exhausted.

17.1.5 *Rutile Prices*

The global rutile market was stable in 2016, with identified imports estimated at 574 kt, down 2% year-on-year, with a weighted average price of US$ 725/t FOB.

Global rutile prices have been relatively stable throughout the first half of 2017, but it is worth noting that rutile prices in the domestic Chinese market have witnessed a strong recovery in recent months. Domestic Chinese rutile prices grew slowly during the first three months of 2017, from RMB 5,500/t inclusive of VAT in early January to RMB 6,500/t inclusive of VAT by the end of March. Prices then jumped considerably during April and May, up nearly RMB 1,200/t from prices at the end of March. Current spot prices for rutile 95% TiO_2 in Hainan are quoted at RMB 6,700/t inclusive of VAT, down approximately RMB1,000/t from the peak at the end of May.

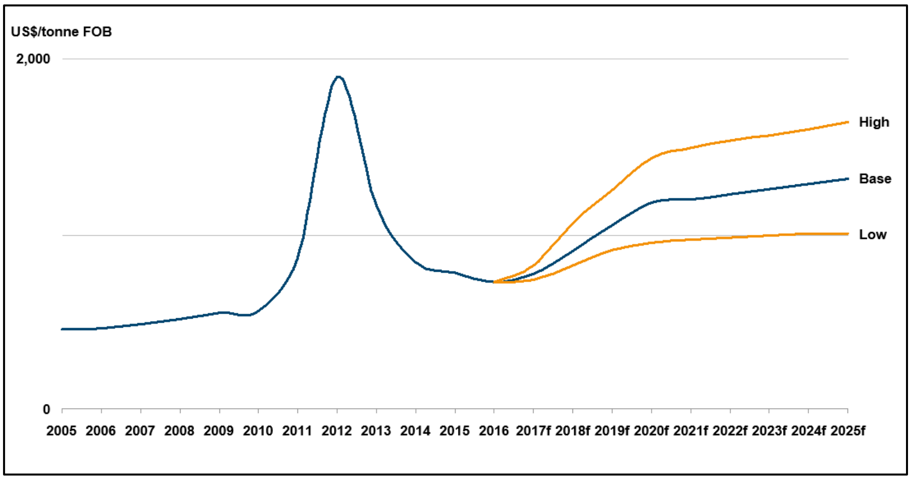
Prices of rutile shipments into western markets are generally in the range US$ 690/t to US$ 800/t FOB during the first half of 2017.

Higher prices are anticipated in the second half 2017, with increases in the order of US$ 60/t expected, taking the weighted average price in the second half of 2017 to just under US$ 800/t FOB. The weighted average price for 2017 is estimated at US$ 765/t FOB.

While the current expectation is for rutile to lead the price growth among high-grade chloride feedstocks in 2018, the downside risks remains if chloride slag prices were to stay low (in the mid to high US$ 600s/t range), particularly if the idled capacity within the supply chain is restarted, which would result in an increase in supply in the near term.

Over the longer term, TZMI expects prices to trend towards the inducement price level (US$ 1,070/t real 2016 dollars) by 2020/2021, along with other high-grade chloride feedstocks, to ensure there is sufficient inducement for new supply to meet demand growth.

TZMI's price forecast for rutile is shown in Figure 17-10 below.



Source: TZMI ©

Figure 17-10: Nominal Rutile Prices to 2021

17.1.6 Product Quality Considerations

There are many feedstock options available to chloride route pigment manufacturers ranging from ilmenite with TiO_2 contents above 58%, through to natural or beneficiated products with TiO_2 contents above 95%. The feedstock selected for a given chloride pigment plant is primarily a function of processing cost of the feedstock, the technical expertise of the operator and the emissions and waste disposal options available. Additional consequences of the feedstock choice are the plant capital requirements, the ongoing maintenance requirements and the importance of logistics management, particularly for very large chloride plants that are located a long distance from the feedstock source.

When evaluating a feedstock for the chloride process there are two main qualities that must be examined. These are the feedstock physical characteristics and the presence and quantity of specific impurities.

For chloride route pigment production, an important quality consideration is low levels of elements that form high boiling point chlorides, particularly for calcium, magnesium and to a lesser extent manganese. Chlorides of these elements tend to liquefy at fluid bed operating temperatures, causing bed "stickiness" and ultimately de-fluidisation.

Elevated levels of vanadium in the feedstock could result in significant decolourisation of the base pigment if not removed fully.

Radionuclides (U+Th) are important for both the sulphate and chloride processes, but more so for the chloride process as they concentrate in the waste stream and result in environmental constraints on the disposal of this waste.

Particle size is also an important quality criterion. Finer particles tend to blow-over from the chlorinator, thus increasing the extent of material to be recycled, resulting in higher TiO_2 losses. Particle density is also a factor that affects the overall transportation and storage of the feedstock, including the delivery method into the chlorinator and the fluidisation dynamics.

If the feedstock is targeted at titanium metal manufacture, it should have low levels of tin dioxide, as the presence of tin tends to make the metal more brittle. For welding electrodes application, the feedstock should have low levels of phosphorus and sulphur so that the integrity of the weld strength is not compromised.

17.1.7 Engebø Rutile

TZMI has used the preliminary product specifications generated during the PFS testwork campaign as a basis for assessing the Engebø rutile product quality. The indicative quality of the rutile compares favourably to most other competing products as shown in Table 17-1 below.

Table 17-1: Engebø Indicative Rutile Assay

Chemical Analysis	TiO_2	Fe_2O_3	Al_2O_3	CaO	Cr_2O_3	V_2O_5	MgO	MnO
Engebø	94.9	1.63	0.31	0.35	0.01	0.41	0.03	0.02
Typical Market Specifications	95.0	<1.0	<1.5	≤0.8/0.15*	-	<0.65	<1.0	<1.0

Chemical Analysis	Nb_2O_5	S	P_2O_5	SiO_2	SnO_2	ZrO_2	U+Th (ppm)	D_{50} (µm)
Engebø	n/d	0.17	0.01	1.53	<0.02	0.06	<10	147/106
Typical Market Specifications	<0.25-0.5	<0.03**	<0.03**	<2.5	<0.05***	<1.0	-	>120

**Non-sieve plate and sieve plate specification*

***Welding rod specification*

****Molten salt, titanium metal specification*

The following comments are applicable to the Engebø planned rutile product:

- The industry standard for a "premium" grade rutile classification requires a TiO_2 content >95%, Fe_2O_3 <1%, SiO_2 <2.5% and ZrO_2 <1%. The Engebø rutile product appears to meet these criteria except for the elevated Fe_2O_3 content at 1.63%
- While the Fe_2O_3 level in the Engebø rutile is higher than other commercial rutile products in the marketplace, it is still an acceptable product for chloride pigment production. The level of Fe_2O_3 does not impact on final pigment product quality, as there are other chloride grade feedstocks such as synthetic rutile and chloride ilmenite with a much higher presence of iron which are still being used in chloride pigment production. However, high iron content in the feedstock will result in high iron chloride production and can lead to downstream duct and heat exchanger deposition issues if the system is not operated correctly. This could result in higher pigment production costs and on this basis, pigment producers could ask for a price discount on the Engebø product; the counter argument should be that normal operation of the chlorinator and exit cyclone should not create issues
- The calcium oxide content at 0.35% is higher than other commercial rutile products. A value of 0.21% has been achieved for a rutile product at an earlier stage. Reduction in the level of CaO will be further investigated as part of the DFS. The magnesium oxide level at 0.03% is acceptable.
- All other reported impurities appear to be in line with other commercial rutile products in the market
- The D_{50} of the Engebø rutile product at 147 µm is suitable for use in western chlorinators. However, the latest test results gave a finer rutile product with a D_{50} of 106 µm with about 15% material below 75 µm, compared with the earlier results showing approximately 5% below 75 µm. The finer product can be removed from the main product and sold into the molten sand market. The Engebø output of approximately 3.5 ktpa of this material should be readily absorbed in this market. The main pigment grade rutile will then be within the current grain size specifications. Considering the market opportunity to supply rutile to the molten sand titanium

production industry, no recovery or price penalty has been applied in the financial model to compensate for the high fine particle content. Testwork in the DFS will be carried out with the aim of making a coarser grained rutile, as has been achieved in parts of the testwork programme in this phase

- For titanium metal application, the tin dioxide level needs to be less than 0.05%. Feedstocks with tin dioxide levels greater than 0.05% tend to make the final titanium metal products brittle. Some sponge producers specify a maximum threshold as low as 0.03% tin dioxide. Engebø rutile has a tin dioxide level of less than 0.02% and should therefore not impact negatively on final sponge quality.
- For welding applications, phosphorus and sulphur contents of <0.03% are preferred. Thus, the sulphur at 0.17% will be too high and hence preclude Engebø rutile for this end-use.

Based on the indicative product quality and particle size distribution, the Engebø rutile would be a suitable feedstock for chloride pigment and titanium metal applications.

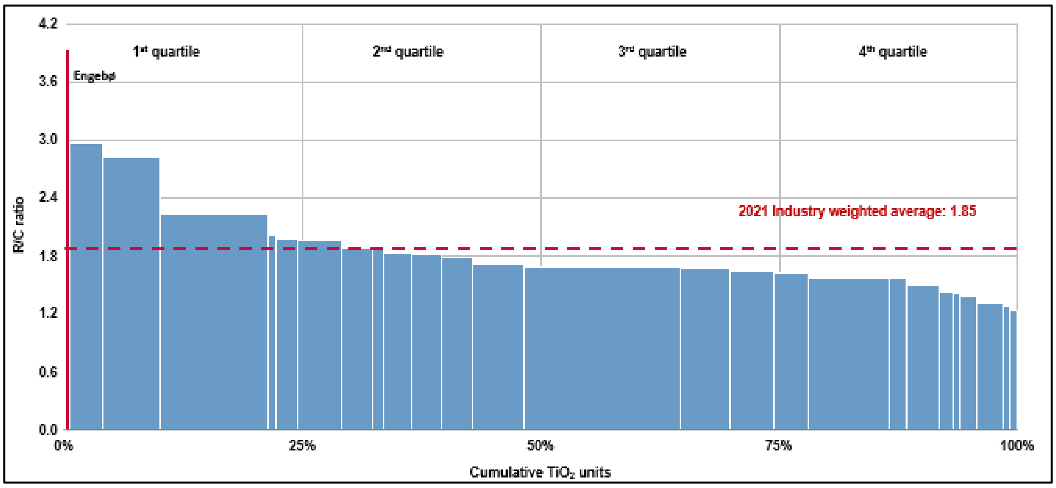
Global demand for rutile for pigment and titanium metal end-use is estimated to reach 540 k TiO_2 units by 2020 and 710 k TiO_2 units by 2025, although higher demand is a possibility if there is a shortage of chloride ilmenite and other high-grade feedstocks such as chloride slag and synthetic rutile. TZMI's current forecast indicates that supply deficit of the global rutile market could reach more than 250 k TiO_2 units by 2020 and 600 k TiO_2 units by 2025. As such, the planned output of approximately 30 ktpa should easily be absorbed by the market by the time the Project comes on stream.

From a pricing perspective, TZMI estimates that the planned rutile product should be able to achieve the long-term price of a standard rutile (US$1,070/t FOB real 2016 dollars) if targeted at chloride pigment or as a feed for titanium sponge manufacture.

17.1.8 Revenue to Cost and Competitor Analysis

Due to the multi-product nature of individual feedstock producers the cash cost of production cannot be directly compared across the industry. Similarly, due to the wide difference in product values and varying product revenue streams, revenue also needs to be taken into account when comparing the relative competitiveness of TiO_2 feedstock operations across the sector. TZMI uses the ratio of revenue to cash costs (R/C) as its primary measure of competitiveness for individual operations in the industry.

The industry R/C ratio curve for 2021 is shown in Figure 17-11 below. The industry weighted average R/C is estimated at 1.85, with individual R/C ratio ranging between 1.23 and 3.46. The Project is positioned towards the top end of the curve, with an average R/C ratio of 3.90. This R/C ratio is estimated based on the first ten years of operations. However, it should be noted that the high R/C ratio for the Project is not typical of mineral sands projects. The Project benefits from having high value products such as rutile and garnet in the product mix, and relatively low mining costs.



Source: TZMI ©

Figure 17-11: 2021 Revenue to Cash Cost Curve

Note: The Project revenue has been estimated using TZMI's long-term inducement price for rutile (US$ 1,070/t FOB) while the garnet price is assumed at US$ 250/t FOB. Operating cost estimates were provided by Nordic Mining.

17.1.9 Contractual Structure

Prior to 2010, the titanium feedstock market had been dominated by long-term contracts (five- to ten-year duration) that typically had an annual price negotiation, based on a base price determined at the start of the contract that was escalated at the USA PPI or CPI. Many contracts without price escalation formulas had pricing "cap and collars", such that irrespective of the supply demand dynamics of the market, annual price moves were constrained, often to a maximum of 5% per annum. These pricing mechanisms ensured that there was little cyclicality in prices for titanium feedstocks, and that in real terms prices remained flat or declined.

However, the contractual regime in the sector has changed considerably in the past few years, partially underpinned by the supply/demand dynamics that evolved during 2010 to 2011, which saw contract durations and frequency of price negotiation being cut short to six monthly or quarterly as opposed to the historic norm of annual price negotiation. The concept is to remove any supply certainty from consumers to extract maximum pricing upside and ensure considerable competition amongst customers to position for what product is available.

Most of the large supply contracts (five- to ten-years duration with cap and collar pricing constraints) between major feedstock supplier and TiO_2 pigment producers had expired by the end of 2014, and prices in new contracts were reset to reflect market prices at the time.

There were also some suppliers (amongst new projects that were commissioned after 2012), who adopted the concept of providing supply under a longer-term contract (five years) with guaranteed minimum price level protection over the contract period but with market pricing to apply above the minimum.

17.1.10 Conclusion

The Engebø rutile product is suitable as a feed for chloride pigment production or as a feedstock for titanium sponge manufacture.

With global supply from existing operations expected to decline rapidly over the period to 2025, and a lack of new rutile projects being introduced to the pipeline, TZMI believes there will be no impediment to selling the Engebø rutile at prices close to the market average. The projected rutile output of approximately 30 ktpa is minor on a global scale and should be readily absorbed by the market by the time the Project is commissioned.

17.2 Garnet Market

17.2.1 Introduction

Garnet is a family of minerals. The main garnet used commercially is almandine, which is the type found in the Engebø ore. Almandine has a hardness normally quoted as 7.5 to 8 on the Mohs scale and a density of 3.9 to 4.2 SG.

Garnet is a general name for a family of six complex silicate minerals based on the same general chemical formula and with similar crystal structures and therefore physical properties. The general formula for the garnet group is:

$$R{+}{+}3R{+}{+}{+}2(SiO4)3$$

Where:

R++ is calcium, magnesium, iron or manganese; R+++ is iron, aluminium, chromium, or titanium.

In nature, garnets rarely approach the theoretical compositions due to substitution and solid solution which in turn affects characteristics such as specific gravity and hardness.

Garnet can be derived from heavy mineral sands or from hard rock sources. All the Indian and Australian-sourced material is currently produced from heavy mineral sands, generally in conjunction with titanium minerals. Hard rock sources like the one at Engebø are exploited in China and the USA.

The primary markets for garnet are in abrasive blasting and waterjet cutting, although for some coarse grades there is also a market in water filtration, which is unlikely to be a significant market for garnet from Engebø because of the size requirements. There is also a market in abrasion resistant materials such as in flooring, but that seems to be restricted mainly to China at the moment.

In the abrasive blasting sector, a wide range of materials is used and garnet has currently a relatively small share of the total market compared with lower priced materials such as coal slag, copper or nickel slag, and other abrasives such as crushed glass, olivine, and staurolite. Garnet tends to be the preferred abrasive where a good profile is required on steel being prepared for painting, especially in harsh environments such as the offshore oil industry, although the specification normally requires a coarse-grained product, certainly a 30/60 mesh size and frequently a 20/40 grade. (Mesh is a commonly used unit in the abrasive industry; a 30 mesh product has been passed over a screen with 30 openings per square inch; a 60 mesh product has passed over a screen with 60 openings per square inch; as the number indicating the mesh size increases, the size of the openings and thus the size of particles captured decreases).

In the waterjet cutting industry, garnet is the dominant abrasive used. Other materials such as olivine have been tried in this relatively young industry and staurolite has been proposed as an alternative. However, garnet has proved to be the most efficient and economical material. The primary grade used in waterjet cutting is an 80 mesh product which makes up 90% or more of the material used, although 120 mesh or 240 mesh grades can be used for some specialised cutting applications and there is a small portion of 60 mesh used. Larger grain sizes are not used because of the apertures of the nozzles commonly used in waterjet equipment.

Garnet used in water filtration is generally a coarse-grained material. A support bed of garnet is used as the base of a water filtration bed with a grainsize generally around 1.4 to 2 mm. A relatively thin finer grained filtration layer is put on top of this normally with a grainsize of about 30 mesh and strict limits on any grains below 50 mesh. Some 30/60 mesh material may be specified as well as 20/40 but also sizes up to almost 5.0 mm if available.

Compared with rutile, garnet is a relatively “young” mineral in industrial applications. Whilst titanium feedstock production including rutile has developed basically in line with global economic growth over many decades, garnet production has primarily developed over the last 20 to 25 years. This is illustrated in Figure 17-12 below.

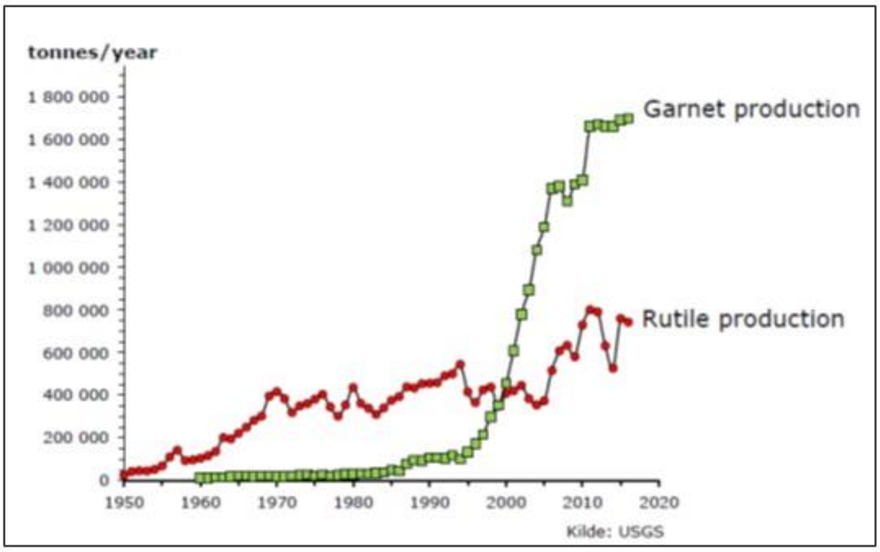


Figure 17-12: Annual Garnet Production

17.2.2 Key Producers

Current world production of garnet is estimated to be about 1.4 Mtpa. India is the largest producer, with estimated production of 450 ktpa to 500 ktpa; however, the installed capacity is estimated to be around 800 ktpa. Indian garnet production has dropped significantly over the last two years as a result of claimed illegal mining of mineral sands with elevated levels of radioactive elements.

Australia is the next largest producer essentially from a single company, GMA, with production in 2015 of over 280 kt. China is the other major producer. Whilst it is difficult to obtain accurate estimates of total production from China, USGS estimates that actual production is in the range of 200 ktpa to 300 ktpa, most of which is used domestically.

The USA is a relatively minor producer, producing at about 34 ktpa, with even lower production from Canada and Mexico. Currently there are no garnet producers in Europe.

17.2.3 Demand Forecast

The size of the three main markets for garnet is as follows:

- The current European market for garnet is estimated at about 160 ktpa. This market is split roughly evenly between abrasive blasting and waterjet cutting. As economies improve the European demand is expected to grow to above 200 ktpa within five years and possibly 250 ktpa within ten years. Additional potential lies in more distant markets in North America and the Persian Gulf
- The US market (net imports and domestic production) is estimated to be 290 ktpa
- The third major consuming region is the Persian Gulf, with imports of almost 180 kt in 2015.

Total apparent global consumption of garnet excluding Chinese domestic consumption is estimated to be almost 890 ktpa (see Figure 17-13 below). If historic growth rates over the last 20 years are extrapolated linearly, apparent consumption excluding China could grow to over 1.4 Mtpa in the next ten years.

In the waterjet sector, there is still growth in what is a relatively young industry. The largest users tended to be in the aerospace, automobile and high-tech industries, but there have been more recent developments in stone cutting, paper and other industries. Growth rates are expected to be of the order of 6% per annum in terms of garnet volumes, possibly even higher as the economy recovers.

In addition to the current market, if a very competitively priced garnet was offered to the abrasive blasting market there is the possibility of gaining market share from lower priced but lesser performing materials such as coal or copper slags. The European market alone for these materials is estimated to be of the order of 1.0 Mtpa. It may be possible to sell a product with a lower level of garnet than current grades to compete with slags, although this will need comparative testwork to demonstrate its effectiveness combined with a market development programme.

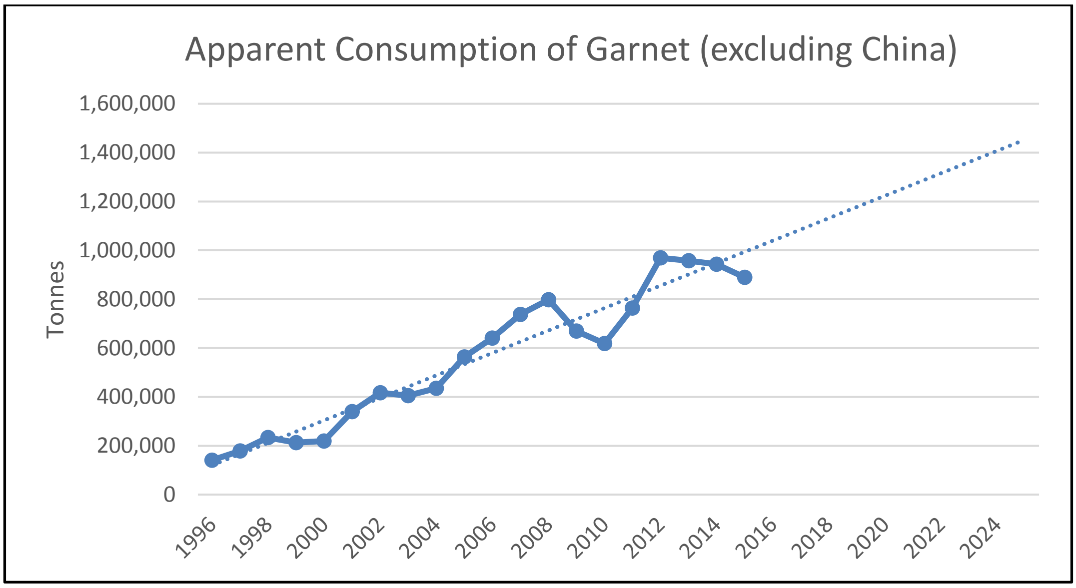


Figure 17-13: Garnet Consumption Development (TAK Industrial Mineral Consultancy)

17.2.4 Supply Forecast

The current world production of garnet is estimated at 1 Mtpa; India is the largest producer (estimated production is 450 ktpa to 500 ktpa); with Australia being the next largest producer at an estimated 280 ktpa production level. China is the third significant producer at an estimated 200 ktpa to 300 ktpa output.

Whilst supply from India comes from a number of companies, output from Australia essentially comes from a single company, namely GMA.

In line with the above production statistics India and Australia are the primary exporters to world markets at estimated levels of 478 ktpa and 293 ktpa respectively.

17.2.5 Competitor Analysis

Through its long-standing MoU partner for garnet, Nordic Mining has established relevant insight into the garnet industry, although no detailed competitor analysis has been carried out. It is unclear at this stage how competitors will respond to this Project; however, Nordic Mining will be the first industrial producer of garnet in Europe. The overall strategy of Nordic Mining is to be a consistent high-quality garnet producer and to establish a long-term market position in the European and other markets.

17.2.6 Marketing

Logistics is an important element of garnet marketing, with deliveries often expected within a few days of order. In this regard Engebø is very well placed with its direct access to the North Sea and thereafter, major European waterways, resulting in lower transport costs and reduced time to market relative to the key global producers in India, Australia and China. Bagging and distribution costs can add significant costs to the delivered price, especially for smaller quantities.

The ultimate consumers in Europe tend to be relatively small, but there are a number of companies that trade and distribute abrasive media including garnet. Some are closely linked to producers in India and Australia with GMA having its own extensive warehousing and distribution network. Suppliers of abrasive blasting and waterjet cutting machinery may also supply garnet and other abrasives to their customers as part of their service.

17.2.7 Pricing

Prices for garnet are normally quoted as:

- Cost, Insurance and Freight (CIF), requiring the seller to arrange for the carriage of goods by sea to a port of destination at his cost, and provide the buyer with the documents necessary to obtain the goods from the carrier; or
- Free on Board (FOB), meaning that the seller pays for transportation of the goods to the port of shipment, plus loading costs. The buyer pays the cost of marine freight transport, insurance, unloading and transportation from the arrival port to the final destination. The passing of risks occurs when the goods are loaded on board at the port of shipment.

There are no terminal markets for garnet and no reliable published prices for products. Products are sold through negotiations between buyer and seller. In the case of abrasives for both blasting and waterjet cutting there are different levels of sale -- from the producer with bulk quantities through to a primary distributor, who may, in some cases, be a subsidiary or division of the producing company, and on through secondary distributors and even down to local and retail sellers. At every stage, a handling and profit margin is added. Delivered prices can be significantly higher than CIF bulk prices especially for less than truckload quantities. Larger international shipments may be in bulk containers of breakbulk transport, but equally can be in 1.0 t or 2.0 t big bags or 2.0 kg bags, depending on where the added value of bagging takes place and the relative economy of shipping bagged vs bulk material. Currently, container shipping rates from India or Australia to Europe, North America or the Middle East are at very low levels, potentially reducing CIF costs, although at some stage the costs should increase as freight markets normalise.

While care must be taken regarding the accuracy of trade statistics, some general trends can be derived from average prices (it must be stressed that these are average prices and do not account for the differences in price of the various grades on the market). In 2015, average prices FOB Australia were US$ 212/t, followed by China at US$ 208/t and India at US$ 196/t. Australian prices had been generally rising up to 2012, but have fallen since that time. Indian prices had been very stable in the period 2008 to 2012 but rose to virtually equal Australian prices before falling in 2015. USA average prices have been omitted as they involve considerably higher priced grades averaging US$ 704/t in 2015 and in some years over US$ 1000/t; these prices apply for low volume specialist products.

Figure 17-14 below shows average FOB garnet export prices for the period 1996 to 2015.

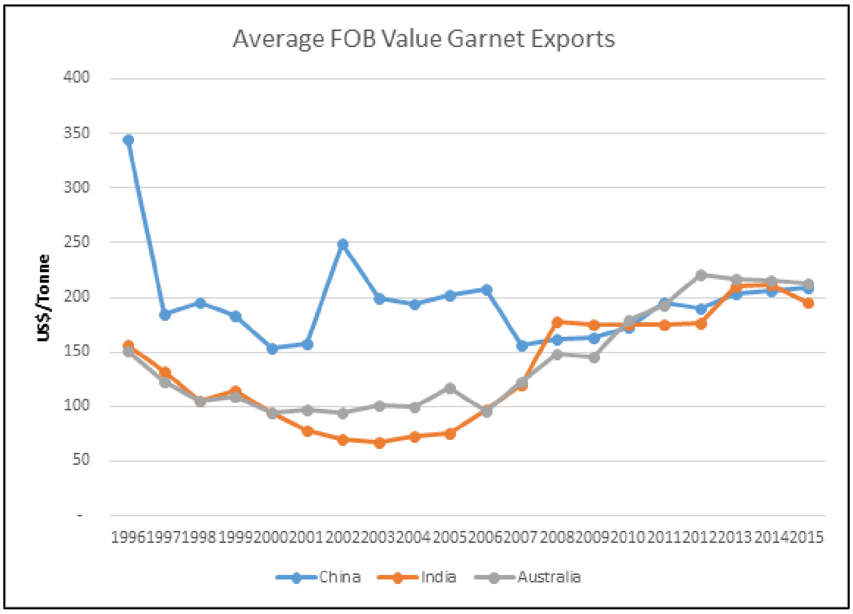


Figure 17-14: Average FOB Garnet Export Prices
(TAK Industrial Mineral Consultancy)

Increased freight rates alone are likely to have added 10% to those prices and these are expected to remain in place. If there continues to be shortages from India, prices are likely to remain significantly higher but even if the supply situation stabilises either from India or alternative sources, including Engebø, many in the industry feel that while the price increases being pushed through due to lack of supply may moderate they will remain at above historic prices by a further 10%. The new plant opened by GMA in the USA to process South African garnet may ease the situation in North America but the cost of freight from South Africa and then shipping on to European markets should they decide to do that adds to the cost from that source.

Once the situation stabilises it is reasonable to expect prices in Europe to be in the € 240/t to € 260/t ton CIF range for an 80 or a 100 mesh material and € 260/t to € 280/t for 30/60 mesh grades; longer term increases will probably be following general inflation trends, unless there is any further disruption to supplies.

For the purposes of financial evaluation in this study, an FOB garnet basket price of US$ 250/t has been assumed. This number is in line with expected export prices as shown above, but assumes that some recovery in the growth rates of global economies will occur. The US$ 250/t price is based on an average price for three products, – 80 mesh waterjet, 100 mesh waterjet and 30/60 mesh blast market. It is anticipated that Engebø will produce approximately equal volumes of each of the above products.

Once the European economy makes a reasonable recovery it is possible that total demand for garnet will exceed 200 ktpa within the next five years, and as the growth in waterjet cutting progresses, could reach 250 ktpa tonnes within 10 years. This is a significant increase above the current estimate demand of 160 ktpa, which is expected to have a positive impact on prices. These numbers do not take into account the gaining of significant market share from other abrasives if Engebø can produce a price-competitive garnet with grades coarser than 30/60 mesh.

17.2.8 Contractual Structure

The contract regime for garnet is to a large extent dominated by spot trading where cargos are purchased individually. At the same time, part of the garnet market has been dominated by a few larger players who have been integrated down to end-user level in certain markets. As the garnet market grows it is expected to be more structured with a contract regime which is more similar to other industrial minerals. In the bigger markets, there is a tendency to operate on a forward stocking basis where the distributors take positions in product in storage hubs for further sales and distribution. There are also examples of lifting contracts on an exclusive basis, however with dynamic pricing.

17.2.9 Specific Downstream Treatment and Upgrading Requirements

The assumed FOB garnet basket price of US$ 250/t used for revenue estimation in this study is based on production of final end-user products according to established market specifications; hence, the products will not need any further treatment or upgrading. Testwork has shown that three various garnet products, 30/60 mesh, 80 mesh and 100 mesh, can be produced according to market specifications with respect to grade and particle distribution. Most of the garnet production is assumed to be sold in bulk, i.e. some bagging may be carried out in the market place at the centre of distribution or by customers, which normally will increase the unit prices.

18. Health and Safety

18.1 Health and Safety Standards to be Followed by the Project

Health and safety are important aspects of everyday life in Norway, as well as in the work place. To meet these expectations, Nordic Mining has adopted the following health and safety goals:

- No injuries or serious incidents during construction and operation
- All employees and subcontractors shall follow planned health and safety procedures
- All health and safety procedures to be in compliance with the Working Environment Act
- Comply with ISO 9001 to ensure systematic quality control.

18.2 Health and Safety Plan

A detailed Health and Safety Plan will be developed during the DFS phase to ensure:

- Health and safety standards meet statutory requirements and industrial best practice
- That every employee and subcontractor knows and abides by all health and safety procedures through training in, amongst others, documentation, routines and checklists
- Ongoing reviews to improve performance.

19. Environmental and Social Responsibility

19.1 Introduction

The overarching principle which will be adhered to when operating Engebø is that Nordic Mining will adopt a good citizen approach and demonstrate that it can plan, build and operate Engebø as follows:

- In a manner which demonstrates environmental responsibility within the environmental terms of permitting and approvals requirements
- With a commitment to a long-term life of the operation by building a solid long-term mining company that will benefit the community
- In a way that introduces factors that can positively influence the operation's neighbours
- With established routines to continuously improve the environmental track record.

The following sections provide more detail of how Nordic Mining will apply the above principle in practice.

19.2 Environmental Setting

The environmental setting for the Project is driven by two key legislative requirements for Nordic Mining to construct and operate a mining and processing operation at Engebø. The two legislative requirements, the discharge permit and the zoning plan, have been fully met and mean that Engebø is fully compliant according to Norwegian environmental legislation, as discussed in more detail below.

19.2.1 Discharge Permit

The final discharge permit for Engebø was issued on 29 September 2016 after minor adjustments were made to the permit by the Order in Council on 19 February 2016, resulting from consideration of complaints on the matter. Changes to the permit may be made by the Norwegian Environment Agency if all or substantial parts of the permit have not been exercised within four years of the date of issue (i.e. by 29 September 2020). The discharge permit allows for an annual tailings disposal volume corresponding to a RoM of approximately 4 Mtpa, i.e. a significantly higher tonnage than the PFS business case.

The permit covers both an environmental and social license to operate; in the process of granting the permit, the Norwegian Environment Agency focuses on pollution-related issues and their mitigation.

19.2.2 Zoning Plan (Planning Permit)

The Engebø deposit and the planned mining and processing plant areas are located adjacent to the Fv 611 county road and a deep-water harbour facility. Nordic Mining has access to the mining area both by land and sea by means of an approved zoning plan for the planned mining area and processing plant. The zoning plan was adopted by the local Municipalities in 2011 and finally approved by the Ministry of Local Government and Modernisation on 17 April 2015. The zoning plan allows for, and provides guidelines on, the operation of the following activities:

- The processing site at Engebø
- The extraction of rock mass in open pit production and underground mining
- The service area at Engebø
- The gangue deposition site in Engjabødalen
- A subsea area for tailings disposal on the sea floor of the Førde Fjord
- The works road running between the Engebø mining operation and the process plant
- The rerouting of county road Fv 611
- The rerouting of a 22 kV power line and the stringing of a new cable between the process plant and the top of the Engebø ridge.

The zoning plan also ensures that measures are put in place to reduce the consequences of the above activities on the landscape.

Nordic Mining will need to apply for a regular operating license upon commencement of operations at Engebø.

19.3 Environmental Studies

To obtain the Discharge Permit and Zoning Plan (Planning Permit) discussed above, numerous environmental studies were carried out for the Project. Some 44 environmental and social responsibility documents have been developed to date over the life of the Project to demonstrate Nordic Mining's commitment to environmental and social responsibility.

The tailings discharge will be conditioned by seawater and de-aired in a sea disposal system and transported in a pipeline down to the Førde Fjord seafloor at a depth of approximately 320 m. The fjord basin is a sedimentation environment confined by thresholds to the inner part of the fjord and by a glaciation sill to open sea. A 4,4 km^2 area of the fjord has been regulated for tailings deposition.

A comprehensive EIA programme has been carried out in several campaigns in the period 2009 to 2013. Detailed baseline studies were done to map the biodiversity in the fjord; this included test fishing, grab sampling and remotely operated vehicle (ROV) investigations. Currents, salinity, turbidity and temperature was measured in the fjord throughout a twelve-month period to document the fjord environment. Figure 19-1 below shows the planned disposal system with the pipe transporting the tailings to the fjord basin.

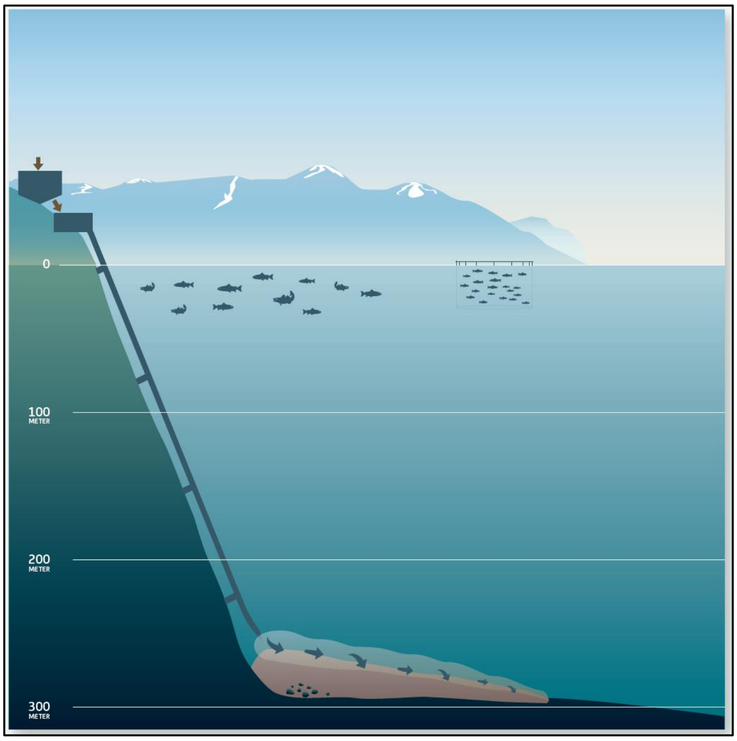


Figure 19-1: Schematic of the Planned Sea Disposal System for Tailings

The main conclusions regarding the fjord disposal solution from the EIA studies were:

- The tailings will mainly sediment within the area regulated for tailings disposal which comprises 5% of the total fjord area
- The currents in the tailings area are moderate and there is limited risk for erosion currents that could potentially transport tailings outside the regulated area
- Limited effects are expected outside the regulated area and in the water column above the tailings outlet
- The tailings are benign, meaning they consist of non-harmful naturally occurring minerals with no heavy metal contamination
- The chemical additives that follow the tailings from flotation and thickening are biodegradable and in non-harmful concentrations
- The tailings consist of mainly sand and silt fractions and a little clay; they are somewhat coarser than the sediments constituting the fjord bottom today
- The baseline studies showed that the fjord habitat has biodiversity that is typical for western Norwegian fjords
- There are no corals found in the tailings area
- The tailings deposits pose little threat to cod which have their breeding grounds in shallow fjord areas

- The tailings solution poses little threat to endangered fish that dwell in the fjord
- The tailings will affect bottom living organisms within the regulated area where the sedimentation rate is high. Mobile species such as fish will avoid areas with high turbidity
- The tailings will likely be recolonised within a few years after the tailings deposition ends. The biodiversity is expected to return to as good a state as it was before the depositing took place
- The fjord has no commercial fishing, but some recreational fishing. The tailings will not affect recreational fishing in the fjord
- The tailings will not affect fish farms that are operated in the fjord.

Norway has long-term experience with deposition of tailings into fjords. Currently there are five active tailings deposits in Norway and two (including Engebø) have recently been permitted. Experience with fjord deposition in Norway is, for the most part, positive. Advanced systems for monitoring exist and there are guidelines for tailings deposition to limit the environmental footprint as far as possible.

19.4 Ongoing Environmental Initiatives

Current initiatives include consideration of the following measures to reduce environmental impacts in the designs:

- Cladding on buildings to reduce heating requirements
- Recycling of water on site and minimisation of water discharge from the property
- Power optimisation
- Green mining (e.g. autonomous mining, electric hauling). Ratification of the Paris Accord restricting future use of fossil fuels means that mining companies need to consider ways in which they can reduce dependence on fossil fuels.

19.5 Waste Management Plan

Nordic Mining's discharge permit states that the company shall establish a waste management plan in place before production starts. The plan will describe all waste generated from mining and processing, how it is to be disposed of, and if there are alternative uses. It will also cover environmental impacts and mitigating actions.

A complete waste management plan will be developed as a part of the DFS and will be in accordance with the discharge permit and legislation.

19.6 Social Setting

The Naustdal municipality is located in the Sogn og Fjordane County and has its northern border towards the Gloppen municipality; to the east it has a border with Jølster, to the west against the Flora municipality and towards the south-east against the city municipality of Førde. Across the Førde Fjord to the south is the Askvoll municipality.

The land area of Naustdal is 370 km² and the total population is approximately 2,700.

Naustdal is an agricultural area with milk production and sheep herding as the main agricultural activities on small- to medium-sized farms. In addition, the municipality has several smaller businesses in housing and earthworks, as well as handcrafts of various sorts. Almost 60% of the active workforce in Naustdal have work outside the municipality, mainly in the city municipality of Førde. Førde is located in the inner eastern part of the Førde Fjord and is the main centre in the Sogn and Fjordane County.

Naustdal has entered into an agreement with three other municipalities, Førde, Gaular and Jølster to merge into one larger municipality called Sunnfjord kommune in 2020.

Naustdal has four hydroelectric power plants with a total capacity of approximately 20 GWh. Three months before the liberation in 1945, Naustdal and the Førde Fjord was the scene of the biggest air battle in Norway during the Second World War. The battle is displayed at the Air Battle Museum located close to the Engebø deposit.

19.7 Socio-economic Studies

As a part of the EIA, the Norwegian research institution "*Norwegian Institute for Urban and Regional Research*" carried out a study on the social consequences of the planned mining activities at Engebø. The study concluded that the Project will have substantial positive effects on the local settlements as well as the local business community. The location of the Project, close to the city centres of Førde and Florø, makes it attractive for the recruitment of local labour. Based on assumptions regarding local settlement for future employees, the study concluded that that the Project will have a significant positive consequence for the economy of the Naustdal Municipality.

In 2013, the Norwegian research institution SINTEF was engaged by the regional development organisation "*Sunnfjord Næringsutvikling*" to carry out a study on regional economic consequences from mining activities at Engebø. The study included model simulations based on updated data for demographics, labour markets and regional economy. The Project is situated in a region consisting of nine municipalities. The study concluded that the regional contributions to the capital investments may represent up to 17%. Further, it was estimated that during local operation the local region will represent approximately 17% of regular supplies and services, based on statistics from other mining operations in Norway. On direct and indirect employments as a result from the Project, the study estimates an indirect employment factor of 2.9, i.e. if the Project needs 110 direct employees, the total number of jobs generated will be approximately 340, of which 60 to 90 will be indirect employees in the local region. Due to historic in-balances in the local labour market, there have been significant relocations and commuting of the workforce in the local region. The study indicated that the Project will have a significant impact on local settlement and the commuting trend.

19.8 Closure Planning

No consideration of closure planning has been made in this phase of the Project and, consequently, no capital cost allowance has been made in the financial model for closure planning costs in light of the fact that the current LoM of 29 years is likely to be extended significantly by means of a drilling programme to be undertaken in the DFS phase. It is considered likely, therefore, that these resources will be shown to be technically and economically mineable, in which case they will be converted to reserves. This implies a significant increase in the LoM.

It should be noted that although no capital provision for closure has been made, an annual contribution of 0.9 M kr (US$ 0.108 M) to a rehabilitation fund has been made in the financial model. This provision has been made every year over the LoM.

Detailed closure planning studies will be carried out in the DFS phase.

20. Human Resources

20.1 Introduction

The labour complement for the Project was developed from individual estimates for each Work Breakdown Structure (WBS) area, based on first principles. An owner operating philosophy will be adopted for the Project, driven primarily by the fact that the long potential LoM dictates that a contracting operation will become expensive over such a long period of time; the potential to use contractors for various activities, particularly mining, will be considered in the next phase based on economic analysis. It has been assumed that full time employees will be used for all production activities on site. The following working hours and shift schedule are assumed:

- Mining – in line with the requirements of the zoning plan, mining will operate five days a week from 07h00 am to 23h00 (16 hours per day) from Monday to Friday, amounting to an 80 hour working week. The mining team will work two, eight-hour shifts
- Processing and Product Dispatch – these areas will operate on a continuous basis, 24 hours per day, seven days a week. This equates to 168 hours per week
- Norwegian working hours are strongly regulated. A five-shift system has been assumed for Engebø, which is an accepted system for operating a plant continuously 365 days a year where the working hours and free time are averaged out over the year. This system means that there are always four shift teams at work while the fifth shift is on leave. The mining operation will only operate on a two, eight-hour shift basis, however.

20.2 Labour Costs

Labour costs per hour on a basic salary basis were determined for five main categories of workers – unskilled, skilled, foreman, middle manager and senior manager – to reflect the pay scales of workers who will be employed at Engebø. Cost-to-company annual salary packages were then calculated for each of these categories, taking cognisance of the requirements of the local trade union agreement Hustadmarmor AS 2014-2016, which stipulates minimum hourly rates, add-ons for workers on shifts and age increments to reflect the experience of workers, overtime allowances and social costs.

Table 20-1 below summarises the cost-to-company salaries used as part of the calculations to determine the operating costs for Engebø.

Table 20-1: Summary of Cost-to-Company Salary Calculations

	Basic Salary kr/hr	Age Increment (estimated at kr 5/hr)	Annual Salary (1,800hrs)	Shift Allowance	Salary including Shift Allowance	Overtime Allowance (10% @ 1.75 hourly rate)	Salary including Overtime	Social Costs*	Cost-to-Company Salary (kr)	Cost-to-Company Salary (US$)
Unskilled										
Surface Labourer – Day Shift	151.00	156.00	271,800	0%	271,800	47,565	319,365	38%	440,724	52,819
Surface Labourer – Two Shifts	151.00	156.00	271,800	10%	298,980	52,322	351,302	38%	484,796	58,101
Surface Labourer – Three Shifts	151.00	156.00	271,800	16%	315,288	55,175	370,463	38%	511,239	61,270
Underground Labourer – Day Shift	168.52	173.52	303,336	0%	303,336	53,084	356,420	38%	491,859	58,948
Underground Labourer – Two Shifts	168.52	173.52	303,336	10%	333,670	58,392	392,062	38%	541,045	64,842
Underground Labourer – Three Shifts	168.52	173.52	303,336	16%	351,870	61,577	413,447	38%	570,557	68,379
Weekend Shifts	80% add on to basic salary									
Skilled (with certificate)										
Surface Labourer – Day Shift	161.00	166.00	289,800	0%	289,800	50,715	340,515	38%	469,911	56,317
Surface Labourer – Two Shifts	161.00	166.00	289,800	10%	318,780	55,787	374,567	38%	516,902	61,949
Surface Labourer – Three Shifts	161.00	166.00	289,800	16%	336,168	58,829	394,997	38%	545,096	65,328
Underground Labourer – Day Shift	178.52	183.52	321,336	0%	321,336	56,234	377,570	38%	521,046	62,446
Underground Labourer – Two Shifts	178.52	183.52	321,336	10%	353,470	61,857	415,327	38%	573,151	68,690
Underground Labourer – Three Shifts	178.52	183.52	321,336	16%	372,750	65,231	437,981	38%	604,414	72,437
Weekend Shifts	80% add on to the day shift salary									

	Basic Salary kr/hr	Age Increment (estimated at kr 5/hr)	Annual Salary (1,800hrs)	Shift Allowance	Salary including Shift Allowance	Overtime Allowance (10% @ 1.75 hourly rate)	Salary including Overtime	Social Costs*	Cost-to-Company Salary (kr)	Cost-to-Company Salary (US$)
Foreman (assumes 30% above skilled labour rate)	1.30									
Surface Labourer – Day Shift	209.30	214.30	376,740	0%	376,740	65,930	442,670	38%	610,884	73,212
Surface Labourer – Two Shifts	209.30	214.30	376,740	10%	414,414	72,522	486,936	38%	671,972	80,534
Surface Labourer – Three Shifts	209.30	214.30	376,740	16%	437,018	76,478	513,497	38%	708,625	84,926
Underground Labourer – Day Shift	232.08	237.08	417,737	0%	417,737	73,104	490,841	38%	677,360	81,179
Underground Labourer – Two Shifts	232.08	237.08	417,737	10%	459,510	80,414	539,925	38%	745,096	89,297
Underground Labourer – Three Shifts	232.08	237.08	417,737	16%	484,575	84,801	569,375	38%	785,738	94,168
Weekend Shifts	80% add on to the day shift salary									
Middle Manager (assumes 50% above skilled labour rate)	1.75									
Working on day shift only	281.75	286.75	507,150	0%	507,150	88,751	595,901	38%	822,344	98,555
Senior Manager (assumes 100% above skilled labour rate)	2.75									
Working on day shift only	442.75	447.75	796,950	0%	796,950	139,466	936,416	38%	1,292,254	154,872

Notes: Above costs are based on trade union agreement Hustadmarmor AS 2014-2016

Cost-to-company estimates above make no allowance for annual leave, sick leave and absenteeism, which is catered for in the labour complement

* Social costs cover health benefits, pension, 13th cheque (leave bonus) etc.

20.3 Labour Complement

The steady-state labour complement and associated annual labour costs for the 1.5 Mtpa feed to plant business case are summarised in Table 20-2 below. The total staffing requirement is estimated at 106 people. The mining complement of 28 people given below represents the typical complement for the 1.5 Mtpa RoM mine plan for the open pit operation; in practice, the complement will vary per year depending on the production profile of ore and waste, as well as the mining method employed (open pit or underground).

Table 20-2: Steady-state Labour Complement Summary

Summary	Head Count	Annual Cost-to-Company (US$)
Management and Administration	18	1,768,360
Technical Services	4	270,323
Mining	28	1,764,044
Comminution & Process	40	2,500,274
Product Dispatch	1	56,317
Engineering	15	1,000,015
Total	**106**	**7,359,332**

The labour complement per discipline by job description is shown in Table 20-3 to Table 20-8 below.

Table 20-3: Management and Administration Labour Complement

Management and Administration	Head Count	Annual Cost-to-Company (US$)
General Manager	1	154,872
Plant Manager	1	154,872
Mine Manager	1	154,872
Maintenance Manager (Engineering)	1	154,872
Finance Manager	1	154,872
Marketing Manager	1	98,555
Environmental Manager, SHEQ and Training Manager	1	98,555
HR Manager	1	98,555
Marketing Admin Officer (loading coordinator	1	73,212
Marketing Assistant	1	73,212
Accountant	1	98,555
IT Professional	1	98,555
Accounts Clerical	1	56,317
General Clerical and others	1	56,317
SHEQ and Training Officers	1	56,317
HR Officers	1	56,317
Buyers	1	73,212
Issuing Clerks	1	56,317
Total	**18**	**1,768,360**

Table 20-4: Technical Services Labour Complement

Technical Services (Mining, Geology, Survey, Environmental, Laboratory)	Head Count	Annual Cost-to-Company (US$)
Mine Planner	1	98,555
Geologist (modelling etc.)	1	98,555
Surveyor	1	73,212
Laboratory Assistant (incl. Environmental Monitoring)	1	73,212
Total	**4**	**270,323**

Table 20-5: Mining Labour Complement

Mining (Two Shifts Per Day, 5 Days Per Week)	Head Count	Annual Cost-to-Company (US$)
Foreman	2	161,067
Primary Open Pit Equipment		
Haul Truck	4	247,796
Wheel Loader	2	123,898
Surface Drill Rig	2	123,898
General Labourer	2	116,202
Secondary Open Pit Equipment		
Wheel Dozer	2	123,898
Track Dozer	2	123,898
Track Dozer	2	123,898
Grader	2	123,898
Water Truck	2	123,898
Service Truck	2	123,898
Water Bowser	4	247,796
Total	**28**	**1,764,044**

Table 20-6: Comminution and Process Labour Complement

Comminution and Process – All Areas	Head Count	Annual Cost - to-Company (US$)
Production Area Manager	1	98,555
Process Engineering/Metallurgist	1	98,555
Control Room Operators	10	653,279
Production Shift Operators	20	1,306,559
Production Day Labourers	5	264,096
Trainees (training rates apply – 50% of surface labourer rate)	3	79,229
Total	**40**	**2,500,274**

Table 20-7: Product Dispatch Labour Complement

Product Dispatch	Head Count	Annual Cost-to-Company (US$)
Product Logistics Operator	1	56,317
Total	**1**	**56,317**

Table 20-8: Engineering Labour Complement

Engineering	Head Count	Annual Cost-to-Company (US$)
Mechanical Fitter	2	146,425
Electrician	2	146,425
Automation Electrician/IT Assistants	3	219,637
Assistants	2	112,634
Maintenance Engineer (1 x Mechanical, 1 x Electrician, 1 x Automation)	3	295,665
Trainees (training rates apply – 50% of surface labourer rate)	3	79,229
Total	**15**	**1,000,015**

20.4 Training/Skills Requirement

It is anticipated that the majority of human resources will be sourced locally and regionally, and trained as required. Senior management and key technical skills will, however, most likely be sourced nationally from within Norway as they are unlikely to be available in the local region.

21. Capital Cost Estimate

The capital cost estimate for the 1.5 Mtpa RoM business case was developed to PFS estimating standards. The estimate covers all capital spent during construction and the ramp-up period for establishment of an open pit mining operation and associated process plant, as well as for establishment of the underground mining operation after approximately 15 years of open pit life. A new primary crusher, glory hole and conveyor to feed crushed ore to the existing silo system under the open pit have been included in the estimate to support underground mining.

With the exception of the underground mine capital, any capital spent after nameplate capacity is reached has been defined as sustaining and working capital; this has been included in the financial model developed for the project business case.

21.1 Capital Estimate Summary

The upfront capital estimate to establish the open pit mining operation and the process plant is US$ 207.176 M. The estimated cost of establishing the underground mine after 15 years of open pit operation is US$ 16.931 M (current money terms). The capital estimates per main WBS areas are as follows:

Table 21-1: Upfront Capital Estimate

Upfront Capital Estimate	US$ M
Open Pit Mining	10.027
Comminution	16.802
Mineral Processing	61.573
Tailings Handling and Disposal	7.045
Product Storage and Loadout Facilities	13.108
Infrastructure	22.565
Indirects (excluding contingency)	41.834
Contingency	34.222
Total	**207.176**

Table 21-2: Deferred Capital Estimate for Underground Mining

Deferred Capital Estimate – Underground Mining	US$ M
Underground Mining	7.833
Comminution	2.970
Indirects (excluding contingency)	2.747
Contingency	3.381
Total	**16.931**

21.2 Basis of Estimate

The estimates were prepared on the following basis:

- Software used - the final estimate was produced using Construction Computer Software's (CCS) Candy application, an analytical, resource-based estimating system utilising both simple and complex resources and worksheets to create unit rates. The unit rates are stored as libraries and used for generating estimates
- Estimate base date – 30 September 2017
- Estimate currency – United States Dollars (US$)
- Exchange rates used:
  - 1 US$ = 8.30 Norwegian Krone (kr)
  - 1 US$ = 0.90 Euro (€)
  - 1 US$ = 13.40 South African Rand (R)
  - 1 kr (Swedish Krone) = 1.070 kr (Norwegian Krone).
- Target accuracy of -20% to +30%
- Budget quotes were sourced for selected major equipment and combined with historical data for minor packages
- Labour rates were calculated based on recent Norwegian contractor rates and compared against data received from contractors for other recent studies in Northern Europe
- Equipment lists and Material Take-Offs (MTOs)
- A defined WBS was used
- Metric units of measure were used
- Contingency was included based on a Quantitative Risk Analysis.

21.3 Exclusions and Clarifications

Allowances for the following items were not included in the estimates:

- General sales tax, fringe benefits tax, sales tax, and any government levies and taxes
- Working capital, sustaining capital and Stay in Business (SIB) capital
- Foreign exchange currency fluctuations
- Forward escalation
- Schedule acceleration costs
- Schedule delays and associated costs, such as those caused by:
  - Unexpected site conditions
  - Unidentified ground conditions

- Labour disputes
- Force majeure
- Permit applications.

21.4 Estimating Methodology

The approach used for preparation of the estimate was as follows:

- Set-up:
  - An Estimate Plan was prepared
  - Review of Scope of Work for the Project
  - Review of available drawings and other project information
  - Review of available procurement plans and strategies
  - Development of the WBS.
- Inputs:
  - Quantification of the work in accordance with standard commodities, Equipment lists and MTOs developed by discipline engineers
  - Budget quotations for major equipment
  - Models and drawings as provided by discipline engineers to assist estimating
  - Project Execution Schedule and plan assumptions
  - Determined purchase cost of the installed equipment and materials. Major equipment supply pricing was generally based on budget quotations and historical data for minor equipment and bulk commodities
  - Construction costs were generally included based on first principles estimates and compared with data received in September 2017 from contractors for other current studies in Northern Europe
  - Project Indirects were included based on direct cost factors
  - Contingency was included based on Quantitative Risk Analysis.
- Estimating
  - Set-up the estimate in iPas CE according to the established WBS
  - Calculate direct labour rates and contractor distributable
  - Establish, and allow for, items excluded from budget quotes e.g. freight, vendor representatives, spares etc. on a factor basis of the material and equipment supply cost
  - Determine the purchase cost of equipment and commodity bulk materials.

- Reviews (scope, quantities and rates)
  - Project Team review including Owner's Team.

21.5 Estimating Methodology for Direct Costs

Direct costs are generally quantity based and include all the permanent equipment, materials and labour associated with the physical construction of the permanent process facility, and include:

- Purchase and installation of permanent plant, equipment and materials
- Freight
- Construction labour
- Contractor's supplied construction facilities
- General construction plant and equipment
- Contractor's preambles overheads and profit.

21.5.1 Permanent Equipment

A mechanical equipment list was developed and used as an input to the estimate.

Major mechanical equipment was priced based on budget quotations and supplemented with historical data for minor equipment. A bid evaluation process took place for all vendor-sourced pricing; during this process, the vendor submissions were evaluated for technical and commercial compliance. A recommendation was made by discipline engineers as to which vendor's submission should be included in the final capital estimate.

All equipment pricing in the estimate was reviewed to ensure the following criteria were addressed, and considered, where necessary:

- Allowance for attendance by vendor/supplier technical support for installation and pre-commissioning support, including transport to site and accommodation
- Freight and packaging
- Duty and taxes
- Exchange rate variations (i.e. between quoted rates and agreed rates adopted in the estimate)
- Site access, crane selection and scaffolding requirements were determined for special items of equipment.

Equipment installation was estimated using a combination of historical data and first principles estimating.

21.5.2 Bulk Materials

Bulk materials estimates were developed from MTOs and applied to unit rates based on first principles estimating and budget quotes.

21.6 Estimating Methodology for Indirect Costs

Indirect costs include items that are necessary for the completion of the Project, but are not directly related to the direct construction costs, and included the following items.

21.6.1 Temporary Construction Facilities and Services

Temporary construction facilities for the Engineering, Procurement, Construction Management (EPCM) and Owner's Team were included in the estimate based on a factor of the direct project costs.

21.6.2 Engineering and Project Construction Management

The execution strategy for the Project and the associated EPCM costs are based on a factor of the direct project costs.

21.6.3 Owner's Costs

Owner's costs were included based on a factor of the direct project costs.

21.6.4 Contingency/Risk Allowance

Contingency is a provision for known project costs which will occur, but which cannot be defined in sufficient detail for estimating purposes due to the lack of complete, accurate and detailed information, as well as limited engineering which has been performed. The addition of contingency is required in order to determine the most likely cost of the Project.

Contingency is not intended to cover scope changes and Project exclusions.

To assess the uncertainty associated with the project cost and schedule, a Quantitative Risk Analysis (QRA) was conducted which integrated project risks, schedule and estimate uncertainty to provide:

- An overall project cost risk profile
- A schedule risk profile
- A project risk profile that quantifies the identified project risk events.

The QRA also allowed the team to make risk based decisions and integrate with value management processes.

The final contingency provision was calculated at 19.8%, or US$ 34.222 M, based on Monte Carlo analysis using @Risk and Pertmaster. This represents the P80 contingency value derived from the QRA process. This means that, with the above contingency added into the capital estimate, there is an 80% probability that the final capital cost estimate will be at or below the total estimate cost of US$ 207.176 M.

Contingency was excluded on the upfront payments for bulk water and electricity supply, rehabilitation fund payments, and royalties.

21.7 Escalation

No allowance for forward escalation was included/calculated in the estimate.

22. Operating Cost Estimate

22.1 Introduction

Operating costs have been developed per WBS area from first principles, based on the following cost categories:

- Labour
- Reagents
- Spares and consumables
- Power and water
- Hourly costs for capital recovery for the mining fleet.

Key input cost assumptions used by all disciplines to derive operating costs are as follows:

Table 22-1: Key Input Cost Assumptions

Item	Unit	Cost (US$)
Power	kWh	0.056
Diesel	l	0.84
Natural Gas	tonne	590
Water	m^3	0.07

22.2 Summary of Operating Cost Estimate

Operating costs for the 1.5 Mtpa RoM production case are summarised in Table 22-2 below. The total average operating cost per RoM ton and per sales ton are US$ 16.28/RoM t and US$ 86.92/product t FOB (rutile and garnet combined) respectively.

Table 22-2: Operating Cost Summary

Item	Unit	1.5 Mtpa RoM – Cost/t (US$)	Source
Open Pit - Waste Mining	Waste tonne	1.89	Estimate
Open Pit - Ore Mining	Ore tonne	1.82	Estimate
Underground Decline Development – Waste Mining	Waste tonne	5.03	Estimate
Underground Decline Development – Ore Mining	Ore tonne	3.28	Estimate
Underground Horizontal Development – Waste Mining	Waste tonne	5.27	Estimate
Underground Horizontal Development – Ore Mining	Ore tonne	4.59	Estimate
Underground Stoping – Waste Mining	Waste tonne	3.67	Estimate
Underground Stoping – Ore Mining	Ore tonne	2.93	Estimate
Comminution	ROM tonne	3.93	Estimate
Process	ROM tonne	5.39	Estimate
Tailings Disposal	ROM tonne	0.17	COWI and DNVGL estimate (sea disposal and continuous monitoring)
Product Dispatch	ROM tonne	0.33	Estimate
Overheads	ROM tonne	1.36	Estimate
Rehabilitation	US$ per annum	108,434	NOK 0.9 M per year contribution to a rehabilitation fund
Cost per RoM Tonne	US$	16.28	
Cost per Sales Tonne*	US$	86.92	

** Cost per Sales Tonne reflects cost for all sales tonnes (garnet and rutile combined)*

22.3 Operating Cost Estimates per Discipline

22.3.1 Mining

Mining operating costs were calculated as a function of the production schedule for ore and waste, and equipment operating and consumption factors. The production schedule was drawn up using the DESWIK mining software package. The following process was followed to derive the costs:

- A primary mining fleet of Caterpillar (CAT) 777 trucks (100 t capacity) and CAT 992 front end loaders (FELs) was assumed. This is considered to be the best practical size of equipment for the Engebø pit; the same equipment was used for underground mining
- For underground mining, the use of a long hole open stoping mining method was assumed, including horizontal development, opening of a top and bottom horizontal access slot for each stope, as well as vertical slot to provide space to drill and blast into for each stope. For waste ore underground, the assumption was made that this material would be loaded, hauled and deposited on the surface waste rock disposal facility to be used for open pit waste disposal
- For blasting the assumption was made that down-the-hole slurry would be used with a powder factor of 0.91 kg/m^3 for the open pit and 0.9 kg/m^3 for underground
- The cycle time for ore and waste was calculated for the queuing, loading, hauling and deposition time per block, taking cognisance of the position of the block in the pit and efficiencies. Ore loading and hauling to a glory hole and underground primary crusher system in the pit and underground was assumed. In this way, Direct Operating Hours (DOH) for the primary fleet were generated
- For drill and blast costs the drilling time in minutes per metre was calculated for open pit and underground drilling, assuming an Atlas Copco ROC L8 drill rig was used in the open pit and a Sandvik DT1331i drilling jumbo for horizontal drilling underground. In this way, Direct Operating hours for drilling were generated
- Costs for a secondary fleet of equipment (CAT 824 wheel dozer, CAT D8 and D10 wheel dozers, a CAT 16M grader and a CAT 777W water bowser, as well as a service truck and diesel bowser) were calculated based on an assumed 2,000 operating hours per year. In this way DOHs for the secondary fleet were generated
- Mining labour costs to operate each piece of equipment on a two-shift basis, together with supervision (one foreman) and a labourer, were calculated using standard labour costs as summarised in Section 20 of this report
- Assuming an owner mining strategy, operating costs for each piece of equipment per hour were calculated for:
  - Capital recovery (capital costs divided by typical replacement hours, which vary from 40,000 hours for the primary mining fleet to 20,000 hours for drills)
  - Overhaul costs (parts and labour)

- Maintenance costs (parts and labour)
- Fuel consumption, lubes, tyres and wear parts.

The final open pit operating costs (as summarised in Table 22-2 above) for steady-state production of US$ 1.82/RoM t for ore and US$ 1.89/waste t were benchmarked against similar operations in Scandinavia; the costs compare favourably and are in line with a budget quotation obtained from a Norwegian mining contractor. The contractor quoted an open pit drill, blast, load and haul cost for ore of US$ 3.73/RoM t for ore and US$ 3.65/ton for waste. Once a contactor margin of 20% is taken off these figures, the average contractor cost amounts to US$ 2.95/ton (ore and waste) in comparison to the Project estimate of US$ 2.35/ton (ore and waste). It is important to note, however, that the contractor quoted a higher cost for ore mining than waste mining; this indicates that the contractor did not fully account for the short hauls to the glory hole for ore in the mining schedule. The ore load and haul cost should, therefore, be significantly lower than the waste mining cost where waste will be hauled out of the pit to a waste rock disposal facility.

22.3.2 Comminution

Comminution (crushing and milling) for Engebø consists of a primary jaw crusher, a secondary cone crusher, a tertiary impact crusher circuit as well as screening. Comminution operating costs were calculated from first principles as per the following categories:

- Labour
- Spares and consumables
- Power and water.

Due to the interrelated nature of the comminution and process plants, the comminution labour was estimated together with process in combination. Labour costs were calculated on a cost to company basis for five main categories of workers; the cost amounts to US$ 1.17/RoM t for 28 personnel. For more details of the comminution labour count and costs, refer to Section 20 of this report.

Spares and consumables requirements were estimated from a similar project in South Africa; the total estimated cost for the 1.5 Mtpa RoM production case is US$ 2.38/RoM ton, with the main spares costs being associated with the tertiary crusher circuit and rod mills.

The power consumption for comminution assumes a total installed power requirement of 2.4 MW used for 5,840 hours per year (16 hours per day as per mining plus an additional four hours to crush and mill ore in the underground silos). The power cost amounts to US$ 0.38/RoM t. Water consumption has been assumed to be nil.

Adding the above incremental costs gives an overall estimated comminution operating cost for a 1.5 Mtpa RoM production rate of US$ 3.93/RoM t.

22.3.3 Processing

The mineral process for Engebø consists primarily of magnetic separation and gravity concentration technologies applied across two mineral streams, rutile and garnet. The operating costs were calculated from first principles for the rutile and garnet circuits as per the following categories:

- Labour
- Reagents
- Spares and consumables
- Power and water
- Product drying and reheating.

Due to the inter-related nature of the comminution and process plants, the comminution labour was estimated together with process in combination. Labour costs were calculated on a cost to company basis for five main categories of workers; the cost amounts to US$ 1.17/RoM t for 28 personnel for the 1.5 Mtpa RoM production case. For more details of the comminution labour count and costs, refer to Section 20 of this report.

Reagents costs of US$ 0.06/RoM t have been estimated for rutile processing (Xanthate and SIBX) for the rutile circuit; no reagents will be required for garnet processing. The flocculation reagent Magnafloc will be used for dewatering and conditioning of tailings in the thickener.

Spares and consumables costs have been estimated at 3% per year of the mechanical installation (direct) capital cost for each circuit. For example, the mechanical installation cost for the rutile circuit is estimated at US$ 30.786 M, so the spares and consumables operating cost is estimated at US$ 923.589 M, or US$ 0.62/RoM t for a 1.5 Mtpa RoM production rate. The same cost has been used for the garnet circuit since the interconnected nature of the two process streams makes it difficult to separate out the costs per process stream. The overall cost for spares and consumables equates, therefore, to US$ 0.62/RoM t for the 1.5 Mtpa RoM production rate.

Power costs have been based on the estimated installed power requirement for feed preparation, rutile processing, garnet processing, water systems, and general and services. The total estimated power requirement for the above areas is 5.5 MW. Based on 7,008 operating hours per year, the cost equates to US$ 0.56/RoM t for both the 1.5 Mtpa production rate.

For product drying and reheating, a liquid natural gas (propane) has been assumed as the drying medium. At US$ 1.80/RoM t for the 1.5 Mtpa production case, this is the single biggest contributor to the overall process cost.

22.3.4 Tailings Disposal

All tailings will be disposed via a deep sea disposal system to a dedicated seafloor area in the Førde Fjord adjacent to the process plan and port facility. The following costs have been allowed for in the operating cost estimate:

- Daily monitoring of sea disposal of tailings
- Annual inspection of sea pipes
- Moving of the discharge point to a new position every fifth year
- Annual maintenance of valves and monitors
- Power and test pumping of the seawater pump
- Sampling and analysis of recipient water.

In addition to the tailings deposition activities, for permitting purposes allowance has been made for continuous monitoring of the sea disposal of tailings as follows:

- Mobilisation, fieldwork and overhaul for third-party personnel
- Maintenance of a vessel
- Equipment maintenance, data control and sediment sampling
- Quality control, data handling and data presentation.

The cost estimation input for the above has been provided by COWI for the tailings deposition system and DNV GL for the monitoring of the sea disposal area. Combining the COWI and DNV GL costs gives a total estimated operating cost for tailings disposal for a 1.5 Mtpa RoM operation of US$ 0.17/RoM t.

22.3.5 Product Dispatch

Product dispatch for Engebø consists of storage silos and a shiploading system to enable the rutile product and three garnet products to be loaded onto ships at the port. Product dispatch operating costs were calculated from first principles as per the following categories:

- Labour
- Spares and consumables
- Power and water.

A labour cost for one operator for the shiploading system has been allocated, equating to a cost of US$ 0.03/RoM t.

In a similar manner to Processing, spares and consumables costs for product dispatch have been estimated at 3% per year of the mechanical installation (direct) capital cost for the product dispatch equipment. For a mechanical installation cost of US$ 13.108 M, the spares and consumables operating cost is estimated at US$ 393.249 M, or US$ 0.26/RoM t for a 1.5 Mtpa RoM production rate.

Power costs have been based on the estimated installed power requirement for product dispatch, which is 314 kW operating for approximately 1,000 hours per year. The cost equates to US$ 0.03/RoM t. No water cost has been estimated for the product dispatch facilities as this cost will be minimal.

22.3.6 Overheads

The overheads for the operation consist of a labour cost for the management, administration and technical services team. This team is estimated to have a head count of 18, equating to US$1.18/RoM t. For further details of the make-up of the overheads cost, refer to Section 20.

22.3.7 Rehabilitation

The cost of rehabilitation of the open pit has been estimated at NOK 0.9 M per year in the form of a contribution to a rehabilitation fund. This cost has been allowed for in the financial model every year throughout the LoM. On a RoM ton basis, the cost equates to US$ 0.07/RoM t at steady state production.

23. Financial Analysis

To carry out financial analysis of the Project, a financial model was built using the FAST (Flexible, Appropriate, Structured and Transparent) standard of financial modelling.

Figure 23-1 below shows the make-up of the main cash flows for the financial model. The main cash flow types are revenue, capital expenditure (CAPEX), operating expenditure (OPEX) and stay-in-business (SIB) capital. The different cash flow streams have different drivers. These different drivers were identified and quantified when constructing the financial model.

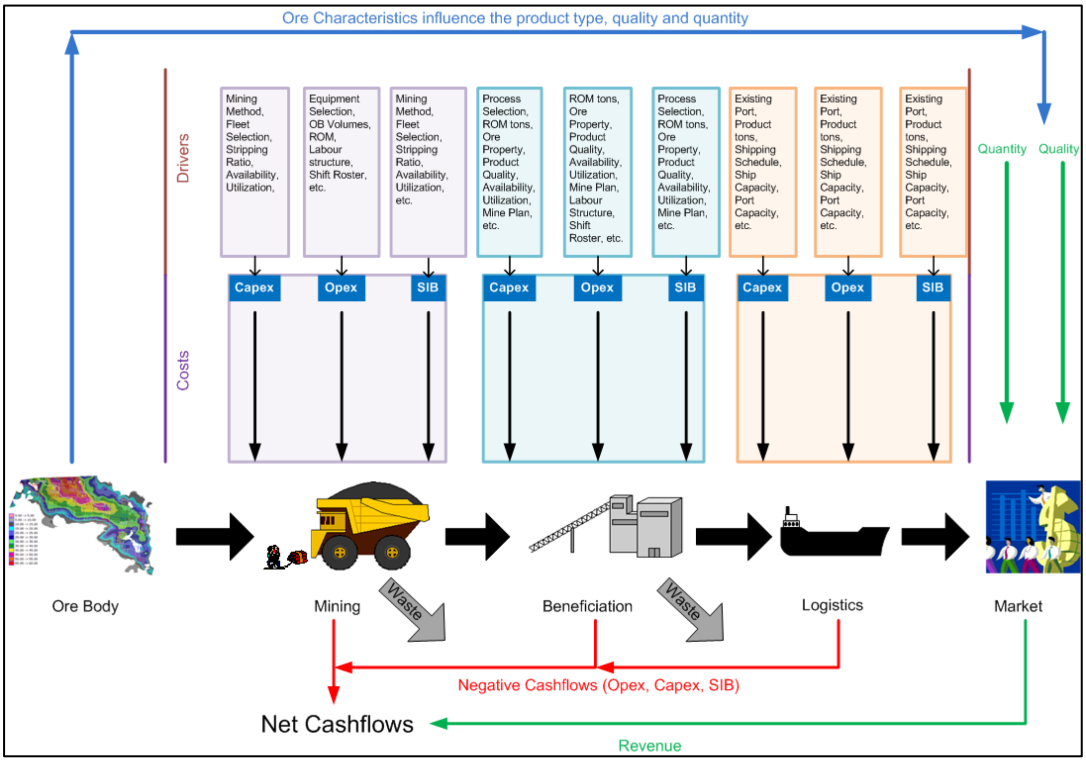


Figure 23-1: Cashflow Drivers for Project Activities

23.1 Key Assumptions

The financial model assumed the following as its basis:

- The base date of the model is September 2017
- The model calculates the cashflows on an annual basis
- The design end date is 1 January 2019
- The construction end date is 1 January 2021
- The dilution during the open pit operation is based on a half drilling burden of 1.75 m
- The dilution during the underground operation is a half burden around the stope of 1.75 m

- The product sales prices used for garnet is US$ 250/t (basket price) and US$ 1,070/t for rutile
- Working capital assumptions:
  - Accounts receivable = 30 days
  - Accounts payable = 30 days
  - Inventory days = 60 days.
- A royalty of 0.5% of the revenue from sales was applied
- A corporate tax rate of 23% was assumed
- A Weighted Average Cost of Capital (WACC) of 8.0% was assumed with a corresponding post-tax WACC of 6.4%
- The SIB capital is an annual expenditure which is 3% of the initial capital investment.

23.2 Options Evaluated

The orebody has two ore types, ferro-eclogite (ferro ore) and trans-eclogite (trans ore), considered for mining. The ferro ore has higher grades of rutile and garnet than trans ore. The ferro ore also has higher recoveries. The mining of these two ore types drives mining strategy as grade drives revenue, which in turn dominates profitability.

A number of options were investigated to evaluate the best financial option, as discussed in Section 12.9. The options are summarised in Table 23-1 below.

Table 23-1: Summary of Options Evaluated

Option Number	Option Description	1.5 Mtpa	1.5 Mtpa, Stepped Upgrade to 2.0 Mtpa	1.5 Mtpa, Smoothed Upgrade to 2.0 Mtpa
1	Ferro only	√		
2	Ferro only		√	
3	Ferro only			√
4	Ferro and Trans, Trans cut-off 2.5%	√		
5	Ferro and Trans, Trans cut-off 2.5%		√	
6	Ferro and Trans, Trans cut-off 2.5%			√
7	Ferro and Trans, no cut-off	√		
8	Ferro and Trans, no cut-off		√	
9	Ferro and Trans, no cut-off			√

Table 23-2 below summarises the key metrics for the options evaluated. The payback period is the time to payback the initial capital invested. The internal rate of return (IRR) calculates the return for a discounted present value for the Project of zero. The net present value (NPV) is the discounted present value for the Project at a pre-tax WACC of 8.0%.

The operating years is the number of production years of the Project for the current declared resource.

Table 23-2: Financial Metrics for Options Evaluated

Metric	IRR (Pre-tax)	NPV (Pre-tax)	Payback Period	Life of Mine
Unit	%	US$ M	Years	Years
Options 1	23.8%	332	4.1	29
Options 2	23.4%	324	4.1	23
Options 3	22.2%	326	4.4	28
Options 4	22.9%	326	4.2	34
Options 5	23.0%	348	4.2	27
Options 6	22.3%	343	4.6	29
Options 7	21.5%	298	4.4	39
Options 8	22.2%	349	4.4	31
Options 9	21.6%	344	4.6	32

Based on its superior IRR, the preferred option is Option 1. This option forms the basis of the discussion in the rest of this section.

23.3 Inputs

23.3.1 Production

The mined tonnage and their grade of rutile and garnet is based on the mine plan inputted into the financial model. Figure 23-2 below shows the ore and waste production profiles for Option 1 over the life of the Project.

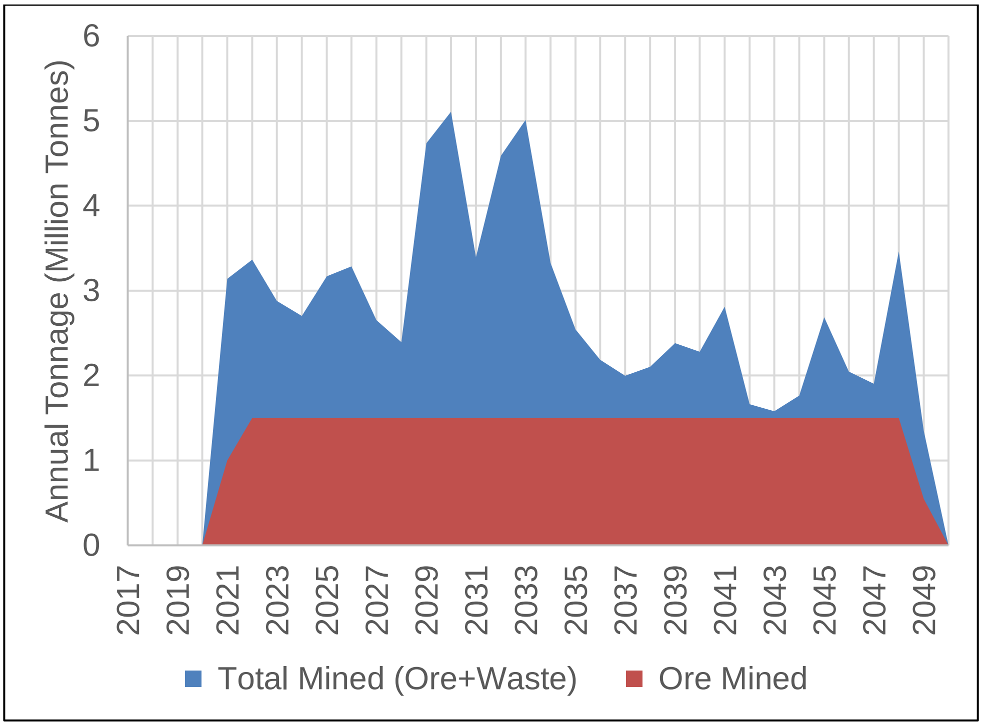


Figure 23-2: Ore and Waste Production Profiles

23.3.2 OPEX

Key assumptions used to derive the OPEX are as follows:

- The mine unit cost for each year was provided by the mining discipline and is based on haulage distances, direct operating hours on equipment, drilling and blasting costs, and labour
- The non-mining operating costs were split into a fixed and variable component to be used in the financial model. These are summarised in Table 23-3 below.

Table 23-3: Fixed and Variable Operating Costs for Non-Mining Activities

Description	Fixed Annual Cost (US$)	Variable Cost (US$/RoM t)
Comminution	164,046	3.82
Garnet and Rutile Beneficiation	-	5.43
Tailings Disposal	240,000	0.01
Product Dispatch	60,000	0.29
Rehabilitation	108,434	
Overheads	1,700,779	0.23

Figure 23-3 below shows the OPEX over the life of the Project. The total OPEX and the OPEX per activity is shown. There is an open pit and underground mining phase with OPEX shown by the red and green lines respectively. The red area shows the total mining cost, open pit plus underground.

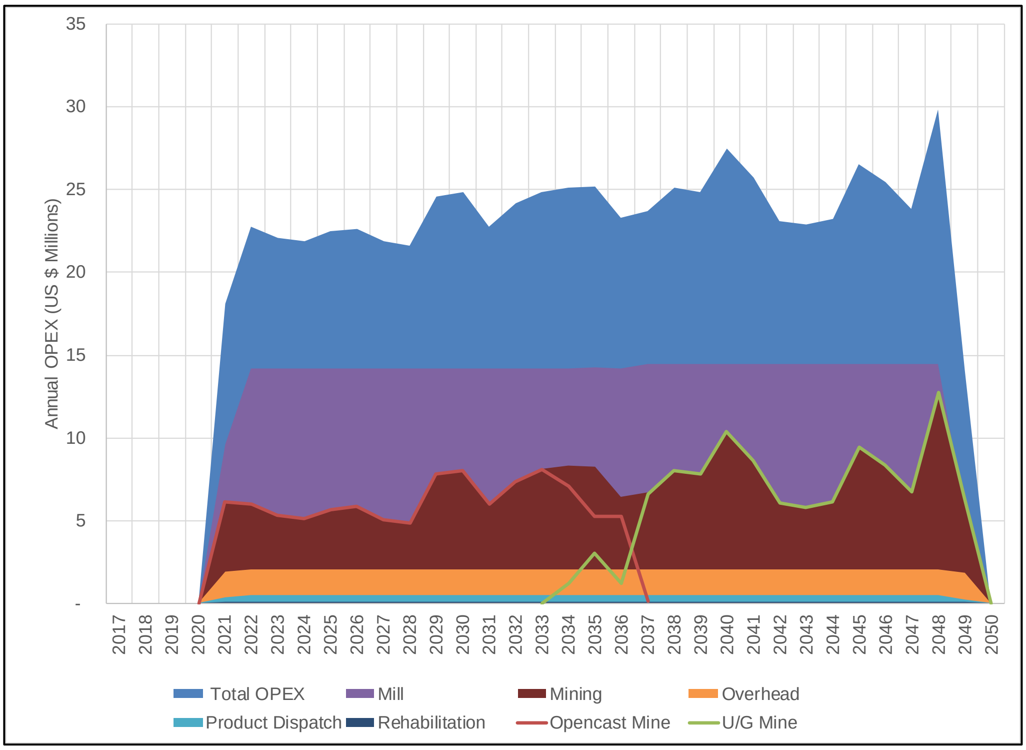


Figure 23-3: OPEX over the Life of the Project

The processing of the ore (mill) costs is the largest contributor to the OPEX over the project life, followed by mining, overheads, product dispatch and rehabilitation.

23.3.3 CAPEX

Figure 23-4 shows the CAPEX over the life of the Project.

The initial CAPEX investment of US$ 207.2 M is profiled to be spent during initial construction (2019 to 2021), with the last payment of US$ 23.5 M being spent in 2021 when the production is ramping up.

There is CAPEX spending of US$ 16.9 M in 2033 prior to mining going underground.

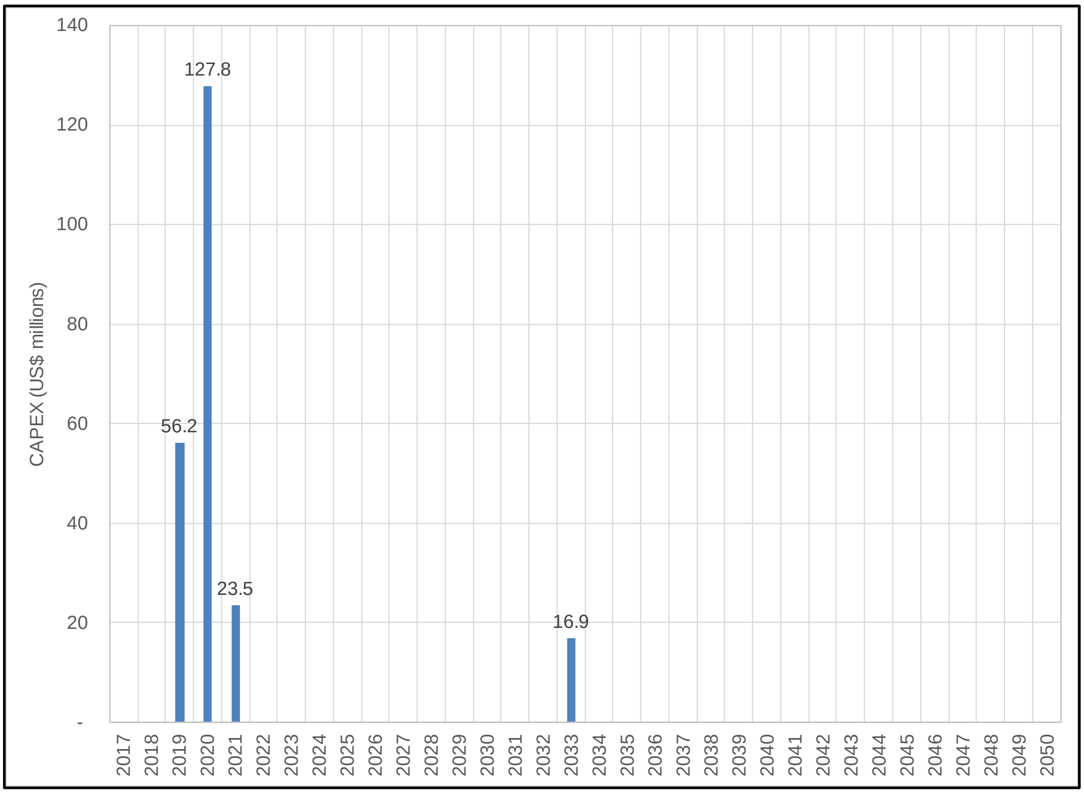


Figure 23-4: CAPEX Over the Life of the Project

23.3.4 Cashflows

Figure 23-5 below shows the cashflows over the life of the Project. The net cash flow, CAPEX, OPEX, SIB, Royalty and change in working capital are shown as vertical bars. The red area is the total Revenue and the green and purple lines are the garnet and rutile Revenues respectively.

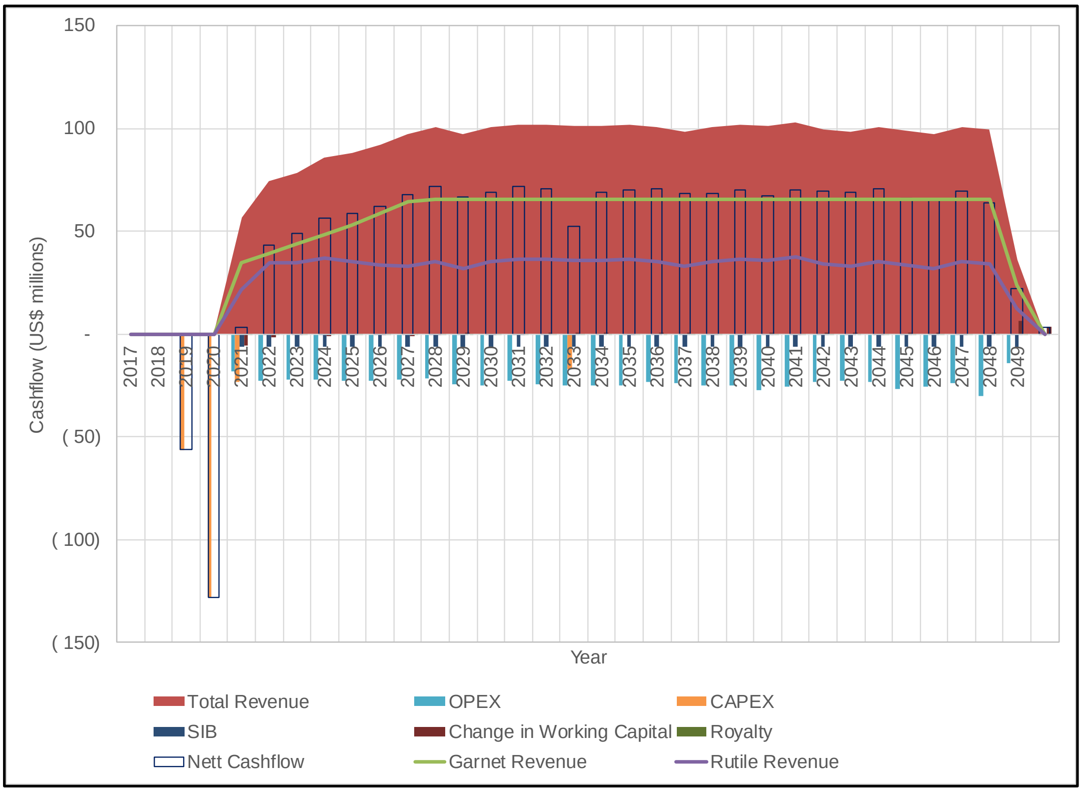


Figure 23-5: Cashflows Over the Life of the Project

23.3.5 *Key Financials*

The key financials are summarised in Table 23-4 below. The NPV is a real pre-tax value discounted by 8%, which is the WACC for the Project. The IRR is real with no escalations applied. The payback period is the number of periods once operations start that generate positive cashflow equal to the capital invested. The Life of Mine is the number of operating years for the reserve derived in line with the guidelines of the JORC Code. The profitability index is a ratio of the NPV divided by the capital discounted to a present value using a WACC value of 8%.

Table 23-4: Key Financials for Base Case Option

Parameter	Unit	Value
NPV @ 8%	US$ M	332
IRR	%	23.8%
Payback Period	years	4.1
Life of Mine	years	29
Profitability Index	ratio	3.1

The Project has a positive NPV and an IRR above 20%. Viewing Table 23-2, the base case option (Option 1) has the highest IRR, although the NPV is not the highest. The capital investment is lower, however, than the high NPV options. The low CAPEX requirement relative to the return is shown by the payback period (refer also to Table 23-2). The base case option has a profitability index of 3.1, implying more than three times the return than the amount invested accounting for the time value of money in the WACC used.

The base case option was optimised to lower the initial CAPEX investment; this lowers overall project risk and allows flexibility to change the operation should business conditions change.

The base case produces excess garnet, more than the expected sales profile (see Figure 23-2), in the early years. This creates an opportunity that could be exploited in a number of ways, namely:

- The extra garnet could be sold to generate revenue if the market can absorb the garnet
- The garnet could be stored for later sales. The cost of storing the garnet should be considered. Garnet does not degrade, making storage relatively easy
- The extra garnet could be sold at a discount to increase market penetration.

23.3.5.1 Post-tax financials

Table 23-5 shows the post-tax financials for the business case. The post-tax calculations have been made for illustrative purposes only and the taxation needs to be verified by an auditing company specialising in such services.

The corporate tax rate is assumed to be 23%. A Norwegian government budget proposal in 2017 is for a reduction of the corporate tax rate from 24% to 23% as from 2018. At the time of writing, there was broad political consensus for this change.

The post-tax NPV WACC is reduced to 6.4% from 8% to account for the tax effect of an assumed Project finance structure of 60% debt and 40% equity.

The depreciation was calculated based on the allocation of the CAPEX to different depreciation categories with different depreciation rules. The categories were:

- Plant and Buildings, which for tax purposes is 32% of the capital investment. This was depreciated using a 4% amortisation rule, where 4% of the outstanding capital can be considered a depreciation expense
- Machinery, which for tax purposes is 28% of the capital investment. This was depreciated using a 20% amortisation rule, where 20% of the outstanding capital can be considered a depreciation expense
- The intangible capital was depreciated using a straight-line method for a period of 30 years.

The remainder of the capital that could not depreciated was subtracted from the cashflow to calculate the taxable income. If it was more than the taxable income, the capital was carried forward as a tax loss until all the capital was written off against taxable income.

Table 23-5: Post-tax* Financials for Base Case Option

Parameter	Unit	Value
Post-Tax NPV @ 6.4%	US$ M	305
Post-Tax IRR	%	20.8%

* The post-tax calculations have been made for illustrative purposes only. Such calculations need to be verified by an auditing company specialising in such services

23.4 Sensitivity Analysis

23.4.1 NPV Sensitivity to WACC

Figure 23-6 shows the sensitivity of the NPV to the WACC used.

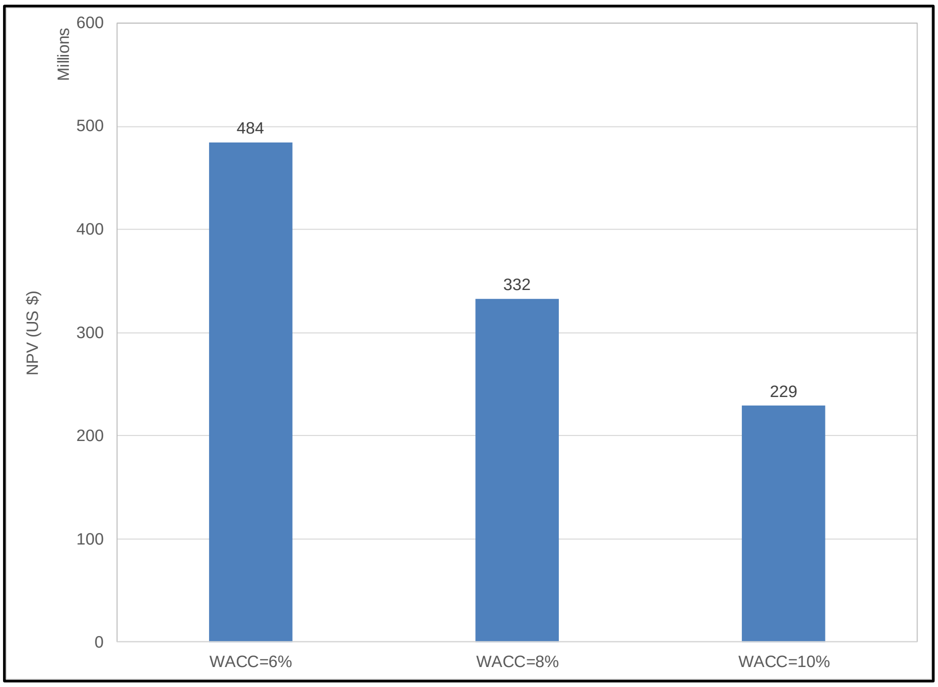


Figure 23-6: NPV Sensitivity to WACC for Option 1

Lowering the WACC has a substantial positive influence on NPV. WACC is an indicator of the cost of capital; if a project can negotiate a low interest rate with a bank, a low WACC is justified.

23.4.2 NPV Sensitivity

Figure 23-7 below shows the NPV sensitivity to OPEX, CAPEX and Revenue. Revenue sensitivity is split into the contributions of garnet and rutile.

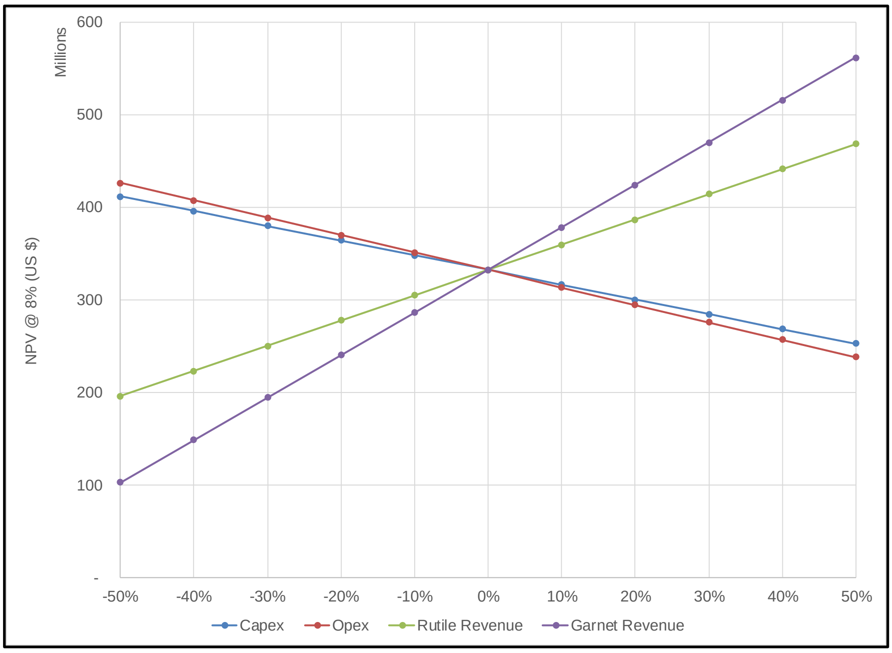


Figure 23-7: NPV Sensitivity to OPEX, CAPEX and Revenue

The NPV is positively correlated to rutile Revenue and garnet Revenue, and negatively to CAPEX and OPEX. Garnet Revenue has a larger influence than rutile, showing their relative Revenue contributions. OPEX has a slightly larger influence on NPV than CAPEX.

23.4.3 IRR Sensitivity

Figure 23-8 below shows IRR sensitivity to OPEX, CAPEX and Revenue. Revenue sensitivity is split into the contributions of garnet and rutile.

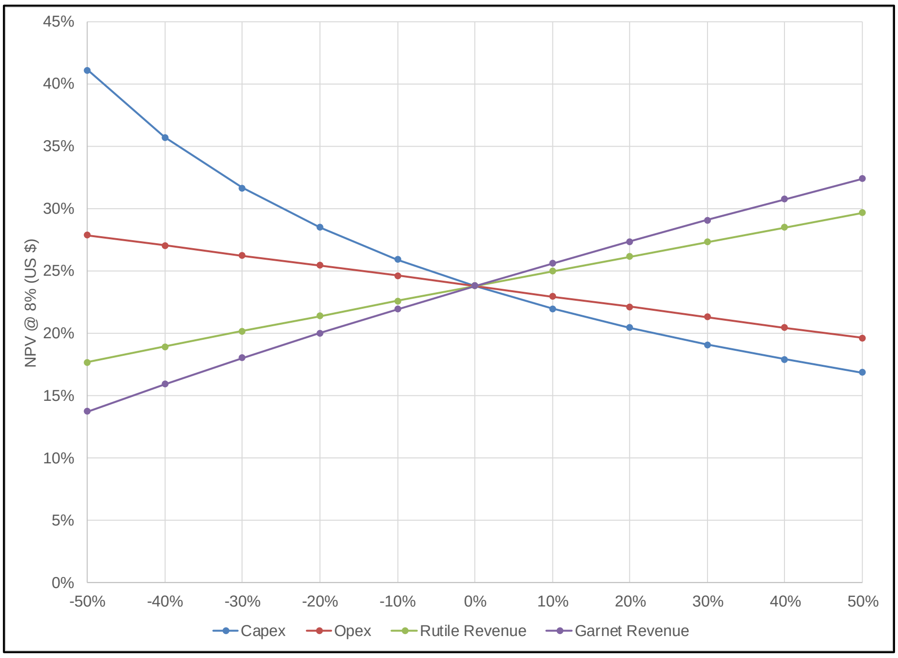


Figure 23-8: IRR Sensitivity for Option 1

The IRR is positively correlated to rutile Revenue and garnet Revenue, and negatively correlated to CAPEX and OPEX. Garnet Revenue has a larger influence than rutile, showing their relative Revenue contributions. CAPEX has a larger influence on IRR than OPEX. This is different to the NPV sensitivity as shown in Figure 23-8 above due to CAPEX being an early negative cashflow.

23.5 Upside Potential to the Business Model

Figure 23-9 below shows the upside potential of the base case (Option 1).

Three upside cases as follows are shown:

- Option 1A - explores increasing the production of garnet in the seventh year of production to meet the expected garnet sales profile, which is an annual amount of 300 kt. The invested amount is estimated to be US$ 36.3 million for an increase in RoM capacity of 240 ktpa
- Option 1B - builds on Option 1A where the Project life is extended to 40 years by including Inferred Resources. A rutile grade of 3.4% was assumed, based on the grade of the Inferred Resources in the mine schedule (this ore was excluded from the base case mine plan as it is not in line with the guidelines of the JORC Code)
- Option 1C - builds on Option 1B, where the garnet sales match the garnet production. This option assumes that the excess production of garnet during the initial ten year sales build-up period is sold.

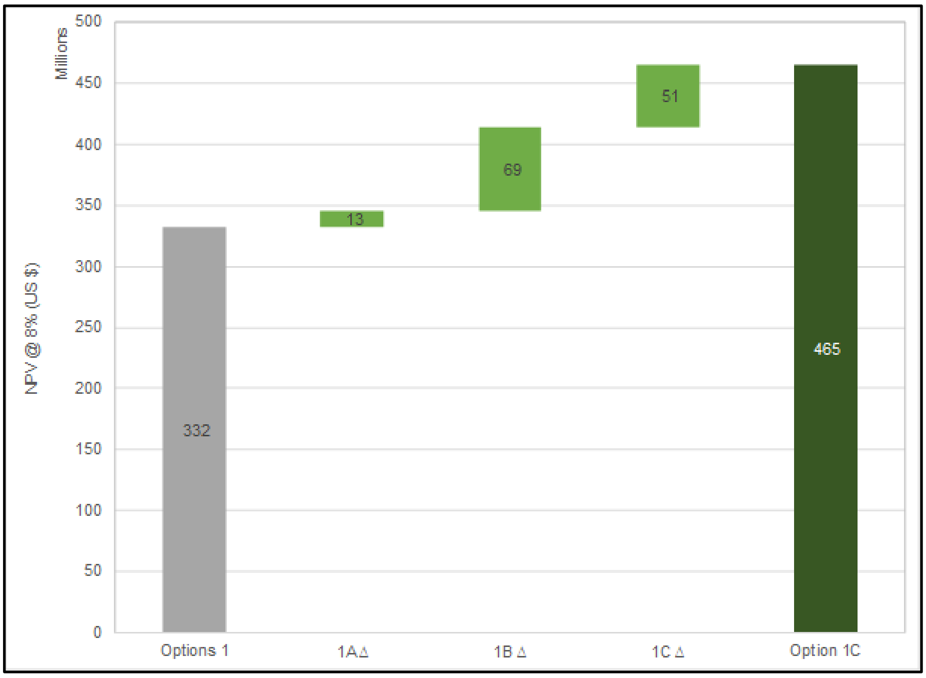


Figure 23-9: Upside Potential of Business Case

As can be seen from Figure 23-9 above, there is significant upside potential in the Project if increased production, Inferred Resources and additional garnet sales are considered. In this scenario, there is a 40% increase in NPV from US$ 332 M to US$ 465 M.

24. Definitive Feasibility Study Planning

Based on the favourable results achieved in this phase, it is planned for the Project to move into the DFS phase during Q4 2017. The DFS will be undertaken in two broad phases; the intent of this approach is to ensure that key studies, testwork, trade-offs and set-up have been completed before mobilisation of the discipline engineering and procurement personnel.

24.1 DFS Objectives

The primary objectives of the FS are to:

- Optimise a single Project concept arising from the PFS
- Mature the understanding of the risks, both for construction and performance of the mine
- Mature the engineering to an extent that supports the estimates and facilitates an understanding of the risks to performance of the mine
- Develop an AACE Class 3 CAPEX and OPEX cost estimate
- Develop a firm and detailed Project Execution Plan (PEP) that defines the way the Project will be delivered, and sets the baselines for monitoring project performance during implementation/execution.

Secondary objectives within specific work packages include:

- Business Case:
  - Confirm the cut-off grade
  - CAPEX and OPEX definition accuracy to -10 to +15%
  - Develop a financial model that gives direction to trade-offs and financial sensitivities
  - Maintain or improve the PFS financial metrics.
- Marketing:
  - Develop take-off agreements
  - Update market understanding and pricing of rutile and garnet.
- Mining:
  - Optimise open pit design
  - Finalise underground infrastructure requirements
  - Carry out an underground mining methods study to determine the preferred mining method
  - Cut-off grade determination
  - Update the mining production schedule.

- Resource / Reserve:
  - Update resource model with updated drilling results
  - Update reserve with outcome of the feasibility study.
- Process / Metallurgical:
  - Carr out variability testwork to confirm/validate the current flowsheets
  - Update the process design criteria
  - Update the mass and energy balance and flow sheets.
- Engineering:
  - Complete Arena modelling to show the material flow
  - Finalise the bulk water requirements and storage capacities
  - Optimise the bulk power requirements
  - Mature the 3D model to suit the execution strategy
  - Complete enough engineering to support an AACE Class 3 estimate.
- Modularisation:
  - Carry out trade-off studies to confirm whether modularisation is to be adopted as part of the Project development strategy
  - Visit potential module yards if the strategy confirms the adoption of modularisation.
- Project Planning:
  - Tailings disposal
  - Mature engineering to support an AACE Class 3 estimate.

24.2 Scope of Work

The high-level activities envisaged for the two study phases include:

- Phase 1 – Project set-up, key studies and trade-offs, including:
  - Update resource model
  - Mining geotechnical studies, including open pit, underground and seismic risk
  - Mining trade-off studies, including mine design and schedule, and mine planning
  - Mineral processing testwork to provide input to the process design criteria and process flow sheets. The phase 1 testwork will be used to develop the flow sheets for the DFS. Additional testwork will run in parallel and will be considered confirmatory
  - Project set-up for Phase 2, including development of key documentation for the DFS, finalisation of project design standards and criteria

- Modularisation and logistics studies – the strategy for the process plant fabrication, assembly and construction will be evaluated and approved before commencement of Phase 2 design development work. This study will encompass engineering, logistics and construction
- Determination of a procurement strategy and a procurement operating plan for the DFS
- Jetty assessment.

- Phase 2 – FEL-3 Design Development
  - Multi-disciplinary engineering design development
  - Procurement engagement with the market to support development of capital cost estimate
  - To protect the schedule, the construction enquiries will be issued to the market with indicative bills. Once the tenders have been received, the proposals will be updated with the latest bills from the engineering team
  - Development of DFS CAPEX and OPEX
  - Other studies and support for external consultants
  - Development of the Project Execution Plan (PEP) and Construction Schedule
  - QRA.

24.3 Schedule

A high-level DFS schedule as shown in Figure 24-1 below has been developed. Based on a start date of 1 November 2017, it is planned to complete the DFS by 22 November 2018. Thereafter, it is likely that the Project will proceed directly into the FEED (Front End Engineering Design) phase, where critical path engineering and procurement work will be continued to expedite the start of construction and to minimise the construction period.

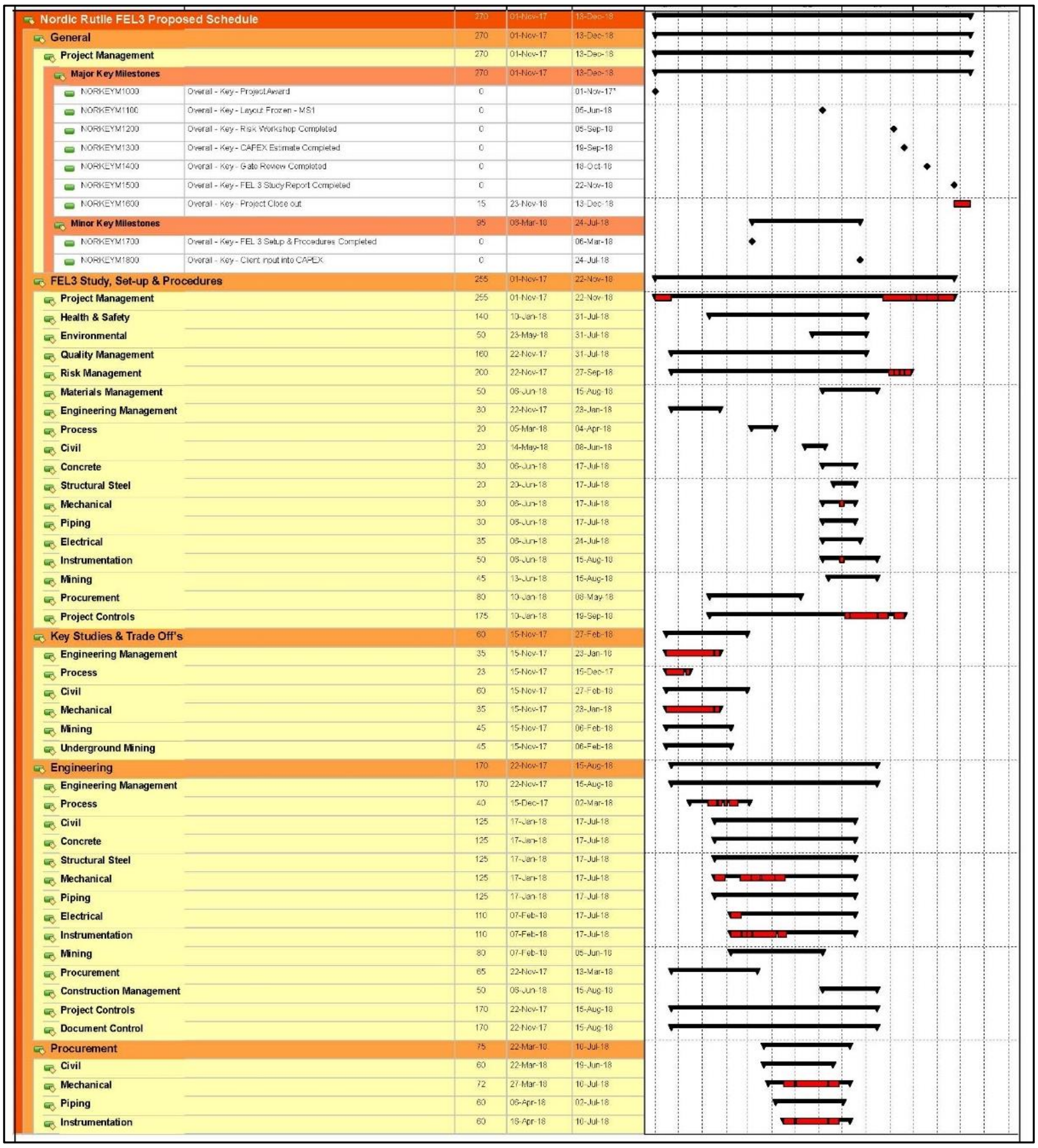


Figure 24-1: DFS Schedule

24.4 Project Team Location/Coordination

The DFS Project team will be based in Johannesburg, South Africa, with significant input from the Nordic Mining team based in Oslo, Norway. Coordination trips between the Johannesburg and Oslo teams, typically once every four weeks, have been allowed for in the Project schedule, together with weekly meetings, monthly steering committee meetings and monthly reports.

25. Execution Planning

25.1 Introduction

A preliminary Project Execution Plan (PEP) has been developed for the Project, based on the following key dates:

- Start of the DFS – Q4 2017
- Completion of the DFS – Q4 2018
- Start of FEED – Q4 2018
- Completion of FEED – Q2 2019
- Start of construction – Q2 2019
- End of construction – Q2 2021
- Start of commissioning and production ramp-up – Q2 2021
- End of commissioning – Q3 2021
- End of production ramp-up – Q4 2021.

The above dates assume that timeous availability of funding for each phase to enable the phases to run continuously. Key aspects of the PEP are summarised below.

25.2 Key Programme Drivers

Key programme drivers are summarised below.

25.2.1 Process Plant Design and Build Schedule

To de-risk the schedule, certified vendor data will be procured during the FEED phase. This will enable the engineering to mature to allow contractors to mobilise the moment funding is secured.

A two-year construction period is currently forecasted; this period may be optimised with a modularisation strategy that will be finalised during the DFS.

25.2.2 Construction and Contracting Strategy

Even with a low experience base in terms of construction of processing plants for hard rock mines, the manufacturing and construction industry in Norway is considered to be mature due to its support of the hydrocarbons industry and large infrastructure projects.

During the DFS, the project team will engage with the construction industry in Europe to assess their capacity to develop the infrastructure. This will be significantly influenced by the modularisation strategy that will decided on at the start of the DFS. A modularisation study will be undertaken to assess module yards in both Europe and the Far East. It is anticipated that due to the plant position and the ability to dock right next to the plant site, modularisation will be cost and safety effective.

25.2.3 Mine Ramp-up

The ramp-up of the open pit mine to 1.5 Mtpa Run of Mine (ROM) is estimated to take one year, including commissioning of the process plant.

At this stage, it is planned to mine and treat only ferro-eclogite ore, which will eliminate the need to ore blending.

25.2.4 Logistics

With the availability of the existing deep-water quay adjacent to the plant area, logistics is not seen as a major risk to the Project. Some work will be required in the DFS phase to confirm the condition of the quay and to take remedial action (if required) to enable it to load ships of up to 30 kt capacity. Such a capacity may be required to ship garnet to potential markets in the USA or Europe.

25.2.5 Operational Readiness

This will be a key focus area for Nordic Mining as Engebø will be the company's first operating mine; Nordic Mining will be required to rapidly transition from an exploration company to a project development company and then to an operating company.

25.2.6 Health and Safety

A project specific health and safety policy and associated standards will be developed. These will be practical application in line with Norwegian regulations and industry best practice standards.

25.2.7 Environmental and Social Responsibility

Environmental and social responsibility will be a key focus area in DFS. With daily restrictions on mining hours (7 am to 11 pm only, weekdays only) and annual restrictions on mining during the salmon spawning season in June, the extent to which such restrictions will apply during the construction period will need to be determined. Construction is anticipated to run for two years without any restrictions.

25.2.8 Expansion Opportunities

There may be an opportunity to increase the throughput of the mine and plant at a later stage; in particular, the mining of trans-eclogite ore will be considered should process testwork in the next phase indicate economic recovery of this ore. The current design caters for expansion of the mining and process facilities at a future date.

25.2.9 Cost Sensitivity

With Norway being a high cost country in which to construct and operate a Project (albeit Norway is also a highly productive country), minimising operating costs and capital costs will be key to the long-term success of the Project. As such, the Project design needs to adopt a 'mean and lean' approach as early as possible. Value engineering throughout the DFS and FEED phases will be essential to ensure that capital costs are minimised and that reduced capital costs do not result in increased operating costs.

25.3 Guiding Principles

25.3.1 Health and Safety

- Nordic Mining will need the Project to be designed and constructed using a 'best practice' safety approach in a country with stringent health and safety regulations

- Safety will be carried out according to policies and systems. On the ground supervision will be mobilised to site once construction starts, reporting to the Nordic Mining Safety Manager; all contractors will supply safety supervision to manage their own staff
- The Project Team will finalise the construction infrastructure in the DFS, but due to the proximity of medical facilities, it is not foreseen that the Project will develop extensive medical facilities.

25.3.2 Project and Construction Organisation

- The primary consultant appointed for the DFS and subsequent phases will co-ordinate all other consultants and contractors
- The primary consultant will report to all other consultants on the work done during the DFS, FEED and construction phases
- The primary consultant will compile all standard project-wide technical and commercial specifications and issue and receive all enquiries for the studies on behalf of all consultants.

25.3.3 Contracting Models

- Key contract packages will be identified during the DFS phase
- The contracting model will be EPCM for consultants (rates reimbursable)
- Contractor versus owner mining will be traded off during DFS to ensure the best strategy is selected for execution; if applicable, a number of contractor mining strategies will be considered to determine the best fit for the Project
- The mine equipment supplies and maintenance philosophy will be finalised in DFS
- Contracting models will be finalised in DFS
- During the DFS, it will be determined if it is feasible to place a low cost fixed price construction contract (SMEIP – Structural, Mechanical, Electrical, Instrumentation and Piping)
- There is a possibility of appointing a separate contractor from processing for all surface and underground conveyors
- Final project budget will be released when the construction contractors are appointed
- For major packages, the site installation shall be done by one SMEIP contractor. Package suppliers to provide installation supervision support (where required)
- A logistics contractor or contractors will be selected during the DFS. It is recommended that this contractor be used by all companies transporting goods to the Project site
- An in-country customs clearing agent may need to be appointed to expedite import of all materials required for construction

- A construction housing strategy will be finalised during DFS. Depending on the planned number of construction personnel, a temporary construction camp may be built on site to house construction workers with catering facilities on site; alternatively, housing of construction staff in the local area (including Naustdal) will be considered if the numbers are low.
- The mine contracting model for underground development and operations will be finalised in DFS
- Insurance needs and requirements will be developed during the DFS
- Three separate allowances will be estimated and managed:
  - Growth allowance (managed by EPCM)
  - Contingency (managed by owner's team)
  - Executive fund (managed by Executives).

25.3.4 Project Controls

- During the DFS, the system configuration for project controls will be finalised; the system will be managed by the primary consultant
- The primary consultant will complete cost reports for all areas they are managing, including all sub-consultants
- The primary consultant will compile a monthly report for review and edit by Nordic Mining before distribution. The report will cover all activities relevant to the Project and not just the areas under the primary consultant's control
- A master schedule in Primavera will be managed by the primary consultant
- Nordic Mining will be responsible for consolidating owner's costs. Where appropriate, information will be shared with the primary consultant to ensure correct incorporation into reporting
- A suitably qualified QS company will be appointed to provide the project quantity surveying services during construction
- All consultants will certify and approve their respective estimates
- The QS company appointed will provide contract administrators to the Project team during execution. These administrators will be managed by the primary consultant's commercial manager
- A document management system will be set up based on the primary consultant's system for central filing of all incoming and outgoing project documentation. Company standards will be set for document and asset numbering.

25.3.5 Construction Services, Procurement and Logistics

- During the DFS, the optimum procurement strategy will be investigated and finalised

- Commercial templates to support the selected contracting strategy will be developed by the primary consultant and to be used for all contracts
- Logistics:
  - To be managed by the primary consultant with the primary consultant's resources in Norway
  - The Project will appoint a loadmaster to be present at the logistics supplier's warehouse
  - The primary consultant will ensure that the proper administration is carried out for all the logistics. A detailed RACI matrix will be developed in DFS for the entire supply chain
  - A warehouse will be established on site and managed by the primary consultant for all project related equipment and materials.
- A low-cost procurement strategy for low-risk items (certain equipment, bulk materials and construction) will be identified and investigated during DFS
- Site fabrication for low-tech bulky items will be investigated during DFS
- All bulks will be procured by the contractors (cement, piping, steel, cable, etc.). Contracts for fabrication and installation will be re-measurable from drawings. It remains the contractor's responsibility to order and manage the bulk material to fabricate as per design
- The primary consultant will develop a quality and inspection plan for manufacturing at works (before shipping) if applicable
- A temporary fuel station will be established on site by a fuel supplier with the supplier selling fuel to contractors at negotiated rates
- Finance auditing will be managed by Nordic Mining's Chief Financial Officer.

25.3.6 Design Principles

- Engineering Standards will be finalised in DFS
- Construction facilities (offices, stores, accommodation, workshops etc.), where possible, will be used for permanent operational use to save CAPEX
- All documentation will be in English. Final copies of documentation to be used for training and operations (e.g. installation procedures, operating and maintenance manuals) may need to be supplied both in English and Norwegian
- All safety related signs and documents will be in English and Norwegian.

25.3.7 Information Technology

A 5G cellular network (to replace the current 4G network) at site is being established by cellphone companies at the current time and is likely to be available in 2018 before the start of construction facilities. Such a system will be able to handle all the data requirements of the Project.

25.3.8 *Security*

A security plan will be developed during the DFS phase and will be amended as required by the main contracting companies once they have been appointed.

25.4 Organisation and Responsibilities

As a minimum, the owner's team should consist of the following key roles:

- CEO, Nordic Mining
- CFO, Nordic Mining
- Project Manager/COO, Nordic Mining

  (The above three personnel will constitute the Project Steering Committee, together with Project Engineering Team personnel)

- Process and Engineering Manager, Nordic Mining
- Mining Manager, Nordic Mining
- Geology Manager, Nordic Mining
- Site Manager, Nordic Mining
- Procurement Manager, Nordic Mining
- Operational Readiness Manager, Nordic Mining
- Environmental and Social Manager
- Marketing Manager
- Chief Accountant.

Some of the above roles may be combined as applicable. As an example, marketing could fall under Process or Geology; the Chief Accountant role could be combined with the CFO role. The Environmental and Social Manager or the Operational Readiness Manager may take on the role of Safety Manager.

25.5 Local and Government Relations

All local and government relations will be handled by Nordic Mining, with input from consultants and the EPCM company as required. It is recommended that Nordic Mining appoint an Environmental Manager in the DFS phase to manage environmental and permitting issues.

26. Risks and Opportunities

Risks and opportunities for the Project were assessed by means of four exercises, namely:

- A "Hazard 2" study (PFD risk review)
- An overall risk assessment, resulting in the generation of a risk register
- Opportunities assessment
- Capital cost risks associated with the capital cost estimate, schedule to first production and project risks.

The results of the above studies are summarised below.

26.1 "Hazard 2" Study

A "Hazard 2" study was undertaken to review potential risks related to the process flow, mass balances, operational interface concerns, equipment locations, maintenance access and general occupational safety hazards. The scope of the study covered the open pit mine and plant. The study included the identification of appropriate remedial measures which can be taken to mitigate risks and hazards. No consideration was given to the hazards associated with underground mining at this stage since underground mining is only expected to start after 15 years of open pit operations at the earliest.

A total of 265 potential hazards were identified. All potential deviations and hazard scenarios with a high or extreme risk rating were assigned additional mitigation measures, resulting in 27 risk reduction and mitigation actions being noted. One of the 27 risk reduction measures was addressed through the mine design in this phase and the remaining 26 areas will be addressed in the DFS phase.

26.2 Risk

Project risk is defined as the effect of uncertainty on Project objectives (ISO31000:2009). Risks to the Project were identified, described and treatment plans proposed using the Hatch Project Risk Management framework and process. A risk assessment workshop was held on 9 August 2017 and the risks identified were prioritised in terms of their consequences and likelihood according to the Hatch Risk Matrix. Material risks were assigned to risk champions who proposed risk response plans (treatment actions). Response plans have been described and need to be considered in the overall project scope for the DFS phase of the Project.

Twenty-one project risks were identified and initially rated according to their expected consequence(s) and likelihood of occurrence. Of the 21 project risks, two were opportunities, and four were retired or closed as any uncertainty around their occurrence was eliminated. Fourteen risks described in the register are threats, of which seven are considered high risks (with a risk score above 20), five are moderate risks (with a risk score between 10 and 20) and two are low risks (with a score of less than 10). Table 26-1 below describes the threats from the risk register and the response plans which need to be considered when the scope for the DFS phase is defined.

Table 26-1: Threats

ID	Risk Name & Description	Cause	Impact	Timing	C	L	Score	Risk Response
001	Variability in Ore Variability in ore types reduces recoveries and increases RoM production requirements	1. Variability in ore types 2. Blending will be required to reduce variability in feed to process plant	Lower recoveries & revenue; increased ore losses due to need to blend unmatched volumes of different ore types	Operations	E Major	C Possible	29	Identify suitable sighter tests to estimate recovery of rutile from different ore types, potentially at NTNU.
004	Sea Disposal Permitting Compliance Non-compliance with permitting requirements of sea disposal site	Various - technical or operational	Operational shutdown; major fines & prosecution; major litigation	Operations	E Major	D Unlikely	25	Conceptual design advanced to a PFS standard, including measures to ensure compliance with permitting requirements
006	Rutile Product Price Lower than anticipated product prices for rutile	Natural chemical and physical properties; no rutile marketing sample presented for customer evaluation	Project economics	PFS	D Significant	C Possible	24	Marketing sample planned to be produced early in DFS for customer evaluation
007	Garnet Product Price Lower than anticipated product prices for garnet	Potential impact of Engebø production on the European market is unknown at this stage	Project economics	PFS	D Significant	C Possible	24	Marketing sample planned to be produced early in DFS for customer evaluation
022	Laboratory Capacities Large amount of test work required, laboratories may be bottleneck	Laboratory capacity relative to test work required	Schedule impact during FS	DFS	D Significant	C Possible	24	Ascertain current workloads and see about "booking slots" as soon as project decision is made
021	Highly Abrasive Material Highly abrasive material may require expensive equipment and maintenance levels and could lead to unacceptable down time, compromising revenue	Abrasive material to process	Production downtime compromises revenue. High maintenance costs increases OPEX and reduces net revenue	Operations	C Moderate	B Likely	22	Import lessons from other rutile and garnet sites, investigate and design for abrasion resistance. Added 5% to high and max ranges in QRA (4 September 2017)

ID	Risk Name & Description	Cause	Impact	Timing	C	L	Score	Risk Response
023	Quay Access during Construction The road from the site to the quay, which may need to be used to deliver heavy plant & equipment for construction, may need to be upgraded	Road between quay and site not adequate for heavy deliveries, steep gradient (due to short distance) between key (sea level) and site (30- to 60 m elevation)	Temporary access road may need to be developed to cater for heavy and large deliveries (cranes, plant & equipment, vendor packages etc.)	Construction	C Moderate	B Likely	22	Determine requirement to construct a temporary road from the key and update the contingency fund accordingly
017	Xanthate in Waste Stream Xanthate, a chemical required in the process, will be present in the waste stream. An environmental assessment of this and other chemicals that follow the tailings stream will be done in order to apply for a discharge permit to include the chemicals	Xanthate, the chemical required in the flotation circuit for rutile recoveries, is potentially toxic at certain levels and in certain environments. The risk through deposition in a fjord environment needs to be evaluated	Toxicity levels could compromise permitting, or require additional OPEX to remediate	DFS	D Significant	D Unlikely	19	Identify opportunities to minimise waste stream; Quantify volumes reporting to co-disposal and determine impact on eco-systems; Identify contingency waste disposal of pyrite; Test work indicates lower consumption requirement approaching the legal threshold
016	Variability in Comminution Performance Comminution results may vary compromising liberation	Variability in the ore and inherent to the comminution process (hard rock)	Lower recoveries & revenue	DFS	C Moderate	C Possible	18	Adjust the flow sheet and re-estimate based on new PFD
009	Garnet Recoveries Lower than anticipated process recoveries for coarse garnet	Recovery of coarse garnet requires additional testwork	Reduced revenue; increased OPEX & CAPEX	PFS	C Moderate	D Unlikely	13	Coarse sample will be sent to the laboratory; should results be below expectations, additional test work will be required to determine at which next finer PSD an acceptable recovery is achieved. Work will be scoped and scheduled as soon as preliminary results are received

ID	Risk Name & Description	Cause	Impact	Timing	C	L	Score	Risk Response
019	Uncertainty in Pricing Parameters There are many specifications that determine pricing and discount levels that may be difficult to meet on a sustained basis	Customer specifications are stringent and thresholds are tight	Lower prices, increased discounts compromises revenue	Operations	C Moderate	D Unlikely	13	Ensure product quality (impurities and PSD) complies with market specifications through the market analysis and process optimisation. Ensure Nordic Mining understands exactly what customers use the product for
015	PFD Updates Test work is not complete, bulk sample results are outstanding for variability test. PFDs may need to be re-developed to ensure minimum rutile recovery of 94.9%	Test work not completed, bulk sample results outstanding for variability test (rutile recovery below 94.9%)	Process design requires adjustment before the end of the PFS phase/ at the start of the FS	PFS	B Minor	C Possible	12	Understand trans ore characteristics before the QRA is conducted; cost and time implications to be catered for in contingencies for the first stage of DFS
008	Permitting Relevant permits not obtained in time	Several permits are required to construct and operate, e.g. planning, building, production, pollution, permitting for water supply etc. Key permits (zoning plan and discharge permit) are already in place	Schedule and production impact - production may be delayed	Ramp Up	B Minor	D Unlikely	8	Permitting checklist & schedule to be drawn up in early DFS. Management has a knowledge of requirements and will track permitting documentation timeously from PFS onwards
018	Construction Restrictions Noise levels may lead to objections to construction activities after hours and over weekends	Construction activities will be noisy	Inability to recover schedule during weekends or after hours	Construction	B Minor	D Unlikely	8	Ensure communication is timely and comprehensive

26.3 Opportunities

Two risks were considered opportunities (i.e. uncertainty that could have a positive effect on project objectives). Table 26-2 below describes the opportunities captured in the risk register.

Table 26-2: Opportunities

ID	Risk Name & Description	Cause	Impact	Timing	C	L	Priority	Risk Response
012	**Eastern Ore Resource Definition** Ore resource definition in Eastern extension of orebody may support additional flexibility in 15+ years of the mine plan	Eastern extension exploration drilling to be conducted	Additional flexibility regarding ore variability and blending requirements	DFS	C Moderate	C Possible	18	Decision to be made on the need for additional drilling after construction
020	**Time to Market** There may be significant upside to early production given current price trends	Garnet and rutile prices trending upwards	Increased revenue, payback acceleration	Ramp Up	C Moderate	C Possible	18	Compile an alternative, fast track project schedule; ensure risks to acceleration are considered

26.4 Capital Cost Risks

In accordance with PFS standards and requirements, a QRA was carried out on the Project to determine the Project capital risk profile.

Taking into account the Project's context (greenfields mine and plant), and the fact that Nordic Mining has a limited portfolio of projects, contingency is proposed at 80% confidence (i.e. the P80 amount) and described at 50% confidence (i.e. the P50 amount).

Table 26-3: Project Contingency Summary

	P50	%*	P80	%
Contingency	US$ 19.234 M	11.1%	US$ 34.195 M	19.8%

* Percentage of the base estimate, i.e. US$ 172.953 M before contingency

As the Project is in the PFS phase, the project execution and construction schedule is not yet at a detailed stage of development, therefore this aspect of the Project was not modelled. Assuming the DFS is completed and approved by the end of 2018, the project execution phase would commence in Q4 2018 or Q1 2019 and commissioning will finish by the end of Q1 2021. This QRA assumes a risk allowance of three months for this timeline, which would need to be catered for in a detailed execution plan.

When benchmarked to the AACEI guidelines, the accuracy of the capital estimate for a Class 3 to 4 estimate is as shown in Table 26-4 below:

Table 26-4: Project Capital Estimate Accuracy

Accuracy	AACEI	Project
Low	-5% to -20%	-9%
High	+10% to +30%	14%

Table 26-5 below summarises the P Value profile of the total capital risk analysis, taking into account the estimate and schedule risk allowance, as well as discrete project risk events.

Table 26-5: Confidence Levels and Contingency Results

Confidence	Value (US$ M)	Contingency (US$)	Percentage
Base Estimate	172.953	0	0%
10%	178.683	5.729	3.3%
Mean	194.532	21.578	12.5%
50%	192.188	19.234	11.1%
80%	207.149	34.195	19.8%
90%	215.496	42.542	24.6%

27. Value Improving Practices

Project guidelines for value improvement practices were used to focus on identifying opportunities to improve the business case of the Project, as well as capital estimate opportunities to be addressed in the DFS phase of the Project. The following aspects of value improvement were considered:

- Technology selection - to ensure that the technology chosen is the most competitive available technology; focuses on evaluation and selection of technology that is appropriate for the Project and is a viable solution for the business need
- Process simplification - to reduce capital and/or operating costs by reduction of process steps/ process complexity; an examination of the Project's overall manufacturing process and facilities to identify non-revenue producing and non-value adding processes or process steps
- Energy optimisation - to identify the facility, process and equipment options that achieve the most economical use of energy; employ technologies or materials of construction to optimise energy usage; make use of thermal or fuel waste streams to generate energy or reduce thermal or fuel requirement via recycling
- Sustainable development - to address sustainability objectives during the study phases in order to meet Nordic Mining's corporate sustainability objectives
- Designing for safety - to consistently produce facility designs that are safe to construct, operate and maintain
- Standardisation - to apply proven engineering and process standards and specifications that will facilitate a broad spectrum of proven manufacturing methods and vendors, negating the need for managing changes on-the-run
- Customised standards and specifications - to ensure that the cost of a facility is not increased by the application of codes, standards and specifications that exceed the actual needs of the specific facility.

The above elements were considered in the application of engineering and design deliverables, which resulted in the opportunities listed below in Table 27-1 being identified.

Table 27-1: Value Improvement Opportunities

#	Idea Generation			1st Pass Evaluation			2nd Pass	Risk	Rating	Decision
	Idea	Description	Benefit	Feasible?	Reduce CAPEX ?	Business Case	Discard / Proceed			
1	By incorporating recovery processes, materials handling facilities and catering for stockpiles in the design of the plant layout, produce armour rock from mined waste rock	Armour rock can be produced from hard waste rock mined. The opportunity requires test- and design- work to determine recovery processes required, as well as determination of product specifications that can be achieved which needs to be compared to market requirements	High level returns and possible NPV improvements are promising and warrant additional investigation of the opportunity	Yes	No	Yes	Proceed	Space, due to topographical conditions, may be limited, and stockpiling will require some space. Product and material handling will need to be carefully considered to prevent Armour rock collections from disrupting rutile and garnet collections at the quay.	Medium	Study
2	By moving the comminution circuit underground, save CAPEX by eliminating the need for waffle cladding of the comminution structure	Moving the grinding, crushing and milling circuits underground will eliminate the need for waffle cladding for reducing noise levels	Potential difference between cost of cladding versus cost of underground accommodation to be investigated	Yes	No	No	Discard	Initial consideration of the opportunity highlighted the impracticalities as well as the high costs of this idea. The costs are unlikely to be less than the cost of waffle cladding the comminution structure.	High	No Go

#	Idea Generation			1st Pass Evaluation			2nd Pass	Risk	Rating	Decision
	Idea	**Description**	**Benefit**	**Feasible?**	**Reduce CAPEX ?**	**Business Case**	**Discard / Proceed**			
3	By incorporating recovery and potentially drying processes, as well as designing product stockpiling/ binning in the design and layout of the plant, coarse garnet waste streams may be converted into a saleable sandblasting product	Garnet is a very hard material and may be suitable as a sandblasting material, or as an additive to sandblasting material	High-level returns and possible NPV improvements may warrant additional investigation of this opportunity, depending on the extent of process and product storage requirements	Maybe	No	Maybe	Proceed	Space, due to topographical conditions, may be limited, and product storage will require some space. Product and material handling will need to be carefully considered to prevent collections from disrupting rutile and garnet collections at the quay.	Medium	Study
4	By reducing primary crusher bins capacity to 4 or 5 days, maintain operability levels and reduce CAPEX	Reduce primary crusher bin capacity from 8 to 4 to 5 days, and make an allowance for the balance of 3 to 4 days to be constructed later (CAPEX to be deferred)	Deferment of CAPEX may increase the business case and assist in reaching the Project financial hurdle rates	Yes	Yes	Yes	Proceed	Bin capacity will need to be 8 days for sustaining operations in the event of significant disruptions in mining, therefore the CAPEX will need to be spent at some stage.	Medium	Study
5	Reduce CAPEX by removing the fine and coarse buffer silos ahead of the dry mill circuit; to be replaced with conventional outdoor stockpiles	Replace buffer silos, together with conveyance mechanisms with a conventional stockpile combined with front-end loading into a loading bin	Conventional stockpiles can be established for significantly lower CAPEX requirements, although there will be operational implications for the reclamation of stockpiles through an FEL into a loading bin	Yes	Yes	Maybe	Proceed	Conventional loading will require a front-end loader, a loading bin, as well as the employment of an operator for loading, although loading will not be continuous and should only be required for about 10% of operating time.	High	Study

#	Idea Generation			1st Pass Evaluation			2nd Pass	Risk	Rating	Decision
	Idea	Description	Benefit	Feasible?	Reduce CAPEX ?	Business Case	Discard / Proceed			
6	Defer fine garnet circuit by 2 to 4 years (remove WHIMS)	The fine garnet product has a long sales ramp-up period; deferring the production of the product may save CAPEX thereby enhancing the business case	CAPEX can be deferred by deferring the construction of the fine garnet circuit; will comprise 37.5% of garnet revenue for the deferment period	Yes	Yes	No	Discard	The net revenue discarded by discarding fine garnet product in the first 2 years is greater than the CAPEX likely to be deferred	Very High	No Go
7	Investigate fine garnet recovery through the dry rutile circuit only (i.e. not through WHIMS)	Remove fine garnet wet gravity circuit and all dewatering and drying components of the dry garnet circuit. Remove the tails dewatering required for the fines garnet circuit.	Removal of dewatering and drying components of the dry garnet circuit, as well as the tails dewatering required for the fines garnet circuit resulting in CAPEX reduction	Yes	Yes	Yes	Proceed	The WHIMS circuit was incorporated for the process effectiveness, and the alternative needs to have comparable effectiveness	Medium	Study
8	By adding saline to tails, reduce and/or eliminate the water recovery process and reduce CAPEX	Adding saline to the tails will allow tailings to report directly to fjord disposal without water recovery requirements	Reduction in CAPEX	Maybe	Yes	Maybe	Proceed	Fresh water recovery is central to the water and environmental permits; the opportunity would mean higher water consumption (due to no recovery) which could compromise permitting	Very High	Study
9	Reduce CAPEX by reducing fresh water tank capacity from 24 to 12 hours	Reduce the capacity in the fresh water reservoir tank to 12 hours	Reduction in CAPEX	Yes	Yes	Maybe	Proceed	Reduced water capacity to 12 hours; any disruption to supply beyond 12 hours will cause the plant to shut down	High	Go

#	Idea Generation			1st Pass Evaluation			2nd Pass	Risk	Rating	Decision
	Idea	Description	Benefit	Feasible?	Reduce CAPEX ?	Business Case	Discard / Proceed			
10	Consider using tailings and mine waste for land reclamation	Waste from mining for land reclamation	Backfill material may be provided by mine and tailings waste streams	Maybe	No	Maybe	Proceed	Additional materials handling capacity	Low	Study
11	Explore the potential for recovering a +550 µm coarse garnet from the process	Observations during the comminution test work suggested there may be a +550 µm coarse garnet product available for recovery	Additional revenue, but it will require additional process circuitry	Maybe	No	Maybe	Proceed	Additional circuit complexity	Low	Study

28. Reference List

Land and Mineral Tenure

- Kvale Advokatfirma – Land Tenure Memo.

Geology, Drilling and Sampling, Mineral Resource Estimate

- A Wheeler - Engebø Resource Estimation - April 2008
- A Wheeler - Engebø Resource Report - August 2016 .pdf
- SRK - Engebø Resource Review Report.pdf.

Mining Geotechnical

- Wardell Armstrong International – Open Pit Mining Geotechnical Slope Design Report
- SINTEF – Underground Mining Geotechnical Report
- SINTEF – Report on Rock Mass Quality at Engebø.

Hydrogeology, Hydrology and Geochemistry

- SINTEF - Hydrogeological Report
- ALS Global – Geochemistry Core Sample Analyses.

Mineral Processing

- Bulk Process Testwork Scoping (H352410-00000-210-248-0001 Rev A)
- IHC Robbins Proposal – Programme 1231 (1231-PM-QU-0000-8001 Rev C)
- IHC Robbins Report – Programme 1231 (1231-PM-REP-0000-8002 Rev 0)
- Core Metallurgy Report No. 1090A-001 – Flotation Testwork Programme on Spirals Concentrate
- IHC Robbins Proposal – Programme 1245 (1245-PM-QU-0000-8001 Rev B)
- IHC Robbins Report – Programme 1245 (1245-PM-REP-0000-8002 Rev 0)
- IHC Robbins Proposal – Programme 1308 (1308-PM-QU-0000-8001 Rev B)
- IHC Robbins Report – Programme 1308 (1308-PM-REP-0000-8002 Rev 4)
- IHC Robbins Proposal – Programme 1234 (1234-PM-QU-0000-8001 Rev D)
- IHC Robbins Report – Programme 1234 (TBC)
- Flotation Trade-off Study Memo (H352410-4000-210-030-0001)
- Testwork Results of Programme 1286-003
- Testwork Results of Programme 1286-004
- Mineral Technologies 83207 Item A report

- JKTech Report (17059 Pyrite Flotation Report rev02)
- IHC Robbins Report – Programme 1293 (1293-PM-REP-0000-8002 Rev 0)
- Plant Operating Factor (H352410-0000-210-230-0001 Rev A)
- Mass and Mineral Balance Insights (H352410-00000-210-030-0004)
- Use of Flotation Reagents in the Engebø Process (H352410-00000-210-030-0007)
- Garnet and Rutile Grade and Recovery Factors for PFS Financial Modelling (H352410-00000-210-030-0008)
- Block Flow Diagram (H352410-0000-210-252-0001)
- Process Flow Diagrams (H352410-3100-210-282-0001-0001 to H352410-7600-210-282-0003-0001).

Mining and Ore Reserve Estimate

- Pit Optimisation Results - H352410-00000-280-202-0001
- PFS Mine Plan - H352410-1000-280-076-0001.

Project Infrastructure

- Cowi AS – Tailings Study
- DNV GL – Monitoring of Sea Disposal of Tailings
- Asplan Viak – Bulk Water Supply Report.

Engineering Design

- Plant Wide - Mechanical - Design Criteria - H352410-00000-240-210-0001
- Plant Wide - Electrical - Design Criteria - H352410-00000-260-210-0001
- Plant Wide - Mechanical - Equipment List – 1.5 Mtpa Plant - H352410-00000-240-216-0001
- Process Plant – Site Plan - H352410-00000-240-270-0001-0001
- Comminution – Plan and Elevation – General Arrangement - H352410-00000-240-270-0002-0001
- Comminution – Side Elevations – General Arrangement - H352410-00000-240-270-0003-0001
- Primary Feed Preparation – Plan and Elevation – General Arrangement - H352410-00000-240-270-0004-0001
- Garnet Processing – Plan and Elevation – General Arrangement - H352410-00000-240-270-0005-0001
- Rutile Processing – Plan and Elevation – General Arrangement - H352410-00000-240-270-0006-0001

- Secondary Feed Preparation – Plan and Elevation – General Arrangement - H352410-00000-240-270-0007-0001
- Secondary Feed Preparation – Elevation – General Arrangement - H352410-00000-240-270-0008-0001
- Primary Crusher – Plan and Elevations – General Arrangement - H352410-00000-240-270-0009-0001
- Primary Crusher – Elevation – General Arrangement - H352410-00000-240-270-0010-0001.

Market Information

- TZMI - Global Rutile Market Study (August 2017)
- TAK Industrial Mineral Consultancy – Garnet Market Analysis (October 2016)
- TAK Industrial Mineral Consultancy – Garnet Market Analysis Update (July 2017).

Environmental and Social Responsibility

- Nordic Mining - Summary of Environmental and Social Studies
- Norwegian Environment Agency - Pollution Permit
- Municipal Council for Naustdal Municipality and Municipal Council for Askvoll Municipality - Zoning Plan.

Human Resources

- Labour summary - H352410-00000-210-202-0002.

Capital Cost Estimate

- Capital cost estimate - H352410-00000-200-016-0003.

Operating Cost Estimate

- Operating cost estimate (Comminution, Processing, Tailings, Product Dispatch, Rehabilitation and Overheads) - H352410-00000-210-202-0002
- Operating cost estimate (Mining) - H352410-1000-280-016-0001.

Financial Analysis

- Financial model - H352410-00000-200-016-0004.

Risks and Opportunities

- Hazard 2 Study – H352410-00000-142-066-00001
- Risk management and value improvement - H352410-00000-140-066-00001
- Quantitative Risk Assessment - H352410-00000-140-066-00002.

Appendix A:
JORC Code 2012 Supporting Information

JORC Code, 2012 Edition – Table 1

Section 4. Estimation and Reporting of Ore Reserves

Criteria	JORC Explanation	Information Required
MRE conversion to Ore Reserves (OR)	Description of Mineral Resource Estimate (MRE)	Stated in Table 8-1.
	MRE - inclusive of, or additional to, OR	The Mineral Resources are reported inclusive of the Ore Reserves.
Site Visit	Comment re CP visit	Mr. A. Wheeler visited the Engebø site and core processing facilities in Førde, from 8 to 10 February 2016, 7 to 8 March 2016 and 12 to 14 June 2016.
	If not, why not.	Not relevant.
Study status	Type and level of study undertaken	This current report is at a Prefeasibility (PFS) level. The reserve estimation work has been done by Hatch Africa (Pty) Ltd ("Hatch"). This work has subsequently been reviewed by Mr. A. Wheeler.
	At least PFS level required; has a viable mine plan been determined	This PFS study has determined that a mine plan is technically achievable and economically viable. Reasonable Modifying Factors have been applied.
Cut-off parameters	Basis and parameters	For Ore Reserve reporting, all ferro-eclogite material within the mine design has been included as ore. The ferro-eclogite is classified as such greater than 3% TiO_2. No ore reserves stem from transitional- or leuco-eclogite rock type. This ore definition approach was determined by economic analysis of the overall project, with respect to potential revenues from both rutile and garnet products. In this analysis assumed long-term prices of US$ 1,070/t rutile and US$ 250/t garnet were applied, with an average processing recovery of 60.2% for rutile and an 18.3% yield for garnet.
Mining Factors/Assumptions	Methods and assumptions	Ore Reserves were determined from the Measured and Indicated Mineral Resources contained within designed open pit and underground layouts.
	Choice of mining methods and parameters	For the open pit, a conventional drill and blast operation has been developed, using 15 m benches. For the underground part of the mine, long hole open stopes have been laid out. The mining methods and parameters are described in Section 12. The maximum open stope size used was 45 m by 150 m by 60 m.
	Assumptions for geotechnics, grade control and pre-production drilling	For the open pit, an independent geotechnical review was completed by Wardell Armstrong International (WAI). For the underground mine, geotechnical analysis was completed by Sintef Rock and Soil Mechanics (SINTEF). This analysis involved numerical modelling for different stoping methods and parameters. These analyses, and the design recommendations resulting, are shown in Section 10. Hatch has followed the recommendations of both studies in developing the open put and underground mine designs.
	Major assumptions used in pit/stope optimisation	The principal open pit optimisation parameters are summarised in Table 12-2. All the optimisation runs were limited by a hard boundary on all sides, reflecting the zoning planning for the Engebø project. Overall slopes angles were applied according to different sectors, varying from 55º to 59º, as per the geotechnical recommendations.
	Mining dilution factors	After regularisation of the resource block model to 15 m by 15 m by 15 m blocks, additional unplanned dilution factors of 4% and 6% were applied to open pit and underground reserves respectively, at zero grades. These factors were derived from approximately half the planned production drill spacing.
	Mining recovery factors	A mining recovery factor of 95% was applied (5% losses) in both open pit and underground reserve calculations.
	Minimum mining widths	A minimum mining width of 15 m has been applied implicitly, stemming from the regularisation of the resource block model.

Criteria	JORC Explanation	Information Required
	No use of inferred in MRE in mining studies	Any Inferred resource material within the planned open pit or underground designs was treated as waste, with zero grades assigned.
	Infrastructure requirements	Described in Section 14.
Metallurgical factors/assumptions	Met. Process proposed: appropriateness for mineralisation	Described in Section 13.
	Whether met. process is well tested.	The mineral processing envisaged is using well-tested technology.
	Nature of met test work	Described in Section 11
	Assumptions made for deleterious elements	While the Fe_2O_3 level in the Engebø rutile (1.7% from indicative tests) is higher than other commercial rutile products in the marketplace, it is still an acceptable product for chloride pigment production. The CaO content at 0.21% is slightly higher than other commercial rutile products but is nevertheless still acceptable for chloride pigment production. The MgO levels at 0.09% are also acceptable. All other reported impurities appear to be in line with other commercial rutile products in the market. No deleterious elements are anticipated in the garnet product.
	Bulk sample/pilot test > orebody representivity	Six bulk samples were drilled and blasted from the surface of the Engebø mountain, spread over the length of the eclogite outcrop, so as to provide representative samples of the ore in the first years of open pit mining of the deposit. In total, 110 tons were obtained.
	OR based on appropriate mineralogy	The Ore reserve estimation used assumed product recoveries based on proposed mineral processing that is appropriate for the expected mineralogy.
Environmental	EI studies. Waste rock characterisation. Residue storage, waste dumps.	The environmental setting for the Project is driven by two key legislative requirements, both of which have been fully met for the Engebø project. These requirements are the pollution permit and the zoning plan (planning permit), as described in Section 19.2.
Infrastructure	Existing land for development, power, water, transportation, labour.	Described in Section 14.
Costs	Derivation of capital costs.	Capital costs have been derived from first principles. For all mining works, all costs have been estimated on an owner-operated basis.
	Methodology for operating costs.	Mine and plant operating costs have also been derived from first principles, with a detailed cost model connected to all physical data obtained from the mine designs.
	Allowances for deleterious elements.	No penalties associated with deleterious elements are anticipated, so no additional costs have been for this.
	Source of exchange rates.	The source of exchange rate used in this report is prevailing exchange rates at the time of the study derived from Oanda.com website
	Derivation of transport charges.	Transportation costs are part of the mining cost estimates.
	Basis for treatment and refining charges, penalties.	Treatment and refining costs are not applicable. No penalties should be applied, prices applied are FOB.
	Allowances for royalties.	A royalty of 0.5% was calculated on the revenue of the Project.
Revenue Factors	Assumptions for head grades, exchange rates and other charges.	Feed grades assumed for process test work were 3.73% rutile and 44.6% garnet. Assumed exchange rates were 1 US$ = 8.30 Norwegian Krone (NOK); 1 US$ = 0.90 Euro (€); 1 US$ = 13.40 South African Rand (ZAR) and 1 SEK (Swedish Krone) = 1.070 NOK.
	Assumptions for metal prices.	In this analysis assumed long-term prices of US$ 1,070/t rutile and US$ 250/t garnet were applied.
Market Assessment	Demand, supply and stock situation	Global rutile demand growth for all end-use over the next 9 years (2016 to 2025) is expected to average 1.8% CAGR, reaching 930 kt TiO_2 units by 2025, or an increase of approximately 140 kt TiO_2 units on

Criteria	JORC Explanation	Information Required
		2016 levels. Global supply of rutile is set to decline considerably or the next decade, with output in 2025 expected to be 50% lower than 2016 levels. Total apparent annual global consumption of garnet (excluding Chinese domestic consumption) is estimated to be almost 890 kt. If historic growth rates over the last 20 years are extrapolated linearly, apparent consumption (excluding) China could grow to over 1.4 Mt in the next 10 years. The current world production of garnet is estimated at 1.1 Mtpa. India and Australia are the primary exporters to world markets at estimated levels of 478 ktpa and 293 ktpa respectively.
	Customer analysis, market windows.	Based on the indicative product quality and particle size distribution, the Engebø rutile would be a suitable feedstock for chloride pigment and titanium metal applications. Nordic Mining would be the first producer of garnet in Europe. The overall strategy of Nordic Mining is to be a consistent high-quality producer, establishing a long-term stronghold in the European and other markets
	Price and volume forecasts.	From a pricing perspective, TZMI estimates that the planned rutile product should be able to achieve the long-term price of a standard rutile (US$1,070 per ton FOB real 2016 dollars) if targeted at chloride pigment or as a feed for titanium sponge manufacture. The production schedule has been built up to produce a maximum of 35 ktpa of saleable rutile product. The assumed FOB garnet basket price of US$ 250/t used for revenue estimation in this study is based on production of final end-user products according to established market specifications; hence the products do not need any further treatment or upgrading. The production schedule has been built up to produce a maximum of 262 kt saleable garnet product.
	For industrial minerals: specification, testing and acceptance requirements.	With an expected Fe_2O_3 level of approximately 1.7% in the final rutile product, this is acceptable for the chloride pigment market. The D_{50} of the Engebø rutile product has been between 106 µm and147 µm. In general, the pigment producers would like to see the amount of material below 75 µm less than 5%. For the latest test results the level of <75 µm was 15%. The <75µm material is however a suitable feed stock for the molten salt industry. Surplus material can therefore be placed into this market which could take between 5 ktpa and 10 ktpa per year to plants in Russia, Ukraine and Kazakhstan. For titanium metal application, the SnO_2 levels need to be less than 0.05%. Analysis of the latest rutile concentrate shows that the SnO_2 content is below detection limit of 0.02% and should therefore be suitable for metal production. The assumed garnet US$ 250/t price is based on an average price for three products – 80 mesh waterjet, 100 mesh waterjet and 30/60 mesh blast market. It is anticipated that Engebø will produce approximately equal volumes of each of the above products.
Economic	Inputs for NPV in study. Source and confidence of economic inputs.	The discounted cashflow analysis has been based on 2017 Constant US$ values. The derivation of the project NPV is described in Section 23
	NPV ranges and sensitivities.	The NPV results are summarised in Section 23, with key results in Table 23-2.
Social	Status of agreements with key stakeholders, for social license to operate.	The social setting for the Project is driven by the same two key legislative requirements as for the Environmental Requirements, both of which have been fully met for the Engebø project. These requirements are the pollution permit and the zoning plan (planning permit), as described in Section 19.2.
Other	Impact of identified material risks.	The project risk analysis has been described in Section 26.
	Status of legal agreements and marketing arrangements.	There are no formal Marketing arrangements in place. However, it is considered that with global supply from existing rutile operations expected to decline rapidly over the period to 2025, and a lack of new rutile projects being introduced to the pipeline, TZMI believes there will be no impediment to selling the Engebø rutile at prices close to the market average. In terms of garnet, Nordic Mining has signed an MOU with a leading, international, producer of industrial minerals. The intention of the parties is to develop the relation further including off-take agreements etc.

Criteria	JORC Explanation	Information Required
	Status of government agreements and approvals.	The pollution permit for Engebø was issued on 5 June 2015 and minor adjustments were made to the permit by the Order in Council on 19 February 2016. The zoning plan, which was adopted by the Municipal Council for Naustdal Municipality in business item no. 022/11 on 11 May 2011 and the Municipal Council for Askvoll Municipality in business item no. 018/11 on 12 May 2011, allows for and provides guidelines on the operation of the following activities: • The processing site at Engebø • The extraction of rock mass in open pit production and underground mining • The service area at Engebøfjellet • The gangue deposition site in Engjabødalen • Subsea area, sea deposition in Førdefjorden • The works road running between Engebø and Engebøfjellet • The rerouting of county road Fv 611 • The rerouting of a 22kv power line and the stringing of a new cable between Engebø and Engebøfjellet.
Classification	Basis for classification of OR, into varying confidence categories.	The Ore Reserve estimate considers only Measured and Indicated Resources, which inside the designed open pit have been converted into Proven and Probable Ore Reserves. Any Inferred resource material within the designed pit was treated as waste.
	Whether results reflect CP's view.	The results do reflect the Competent Person's view of the deposit.
	Proportion of Probable Ore Reserves derived from Measured Mineral Resources.	There are no Measured Mineral Resources.
Audits or reviews	Results of any audits/reviews on OR estimates.	No audit or review of the Ore Reserve estimates has been completed by an independent external individual or company.
Discussion of relative accuracy/confidence	Statement of relative accuracy in the OR estimates, or if not appropriate, qualitative discussion.	Carried over from the resource model, the principal control on the resource categories, and consequently reserve categories, are the spacings of diamond drillhole intersections. Key drillhole section spacing limits have been established which are used as a guide in the assignment of resource categories, as described in Section 7.2
	Relation to global or local estimates.	The Ore Reserve statement relates to global estimates of tonnes and grade.
	Accuracy discussion with respect to any Modifying Factors.	All the Measured and Indicated Resources within the designed open pit, above the applied cut-off grade, have been converted into Proven and Probable Reserves. The Ore Reserve calculations are described in Section 12.3 and Section 12.4
	Where possible, comparison with production data.	Not applicable.

[Letterhead of Competent Person or Competent Person's employer]

Competent Person's Consent Form

Pursuant to the requirements of ASX Listing Rules 5.6, 5.22 and 5.24 and Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

Report name

Engebø Rutile and Garnet Project - Prefeasibility Study

(Insert name or heading of Report to be publicly released) ('Report')

Nordic Mining

(Insert name of company releasing the Report)

Engebø

(Insert name of the deposit to which the Report refers)

If there is insufficient space, complete the following sheet and sign it in the same manner as this original sheet.

30th October, 2017

(Date of Report)

Statement

I/~~We~~,

Adam Julian Wheeler

(Insert full name(s))

confirm that I am the Competent Person for the Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code 2012 Edition, having five years experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member or Fellow of The Australasian Institute of Mining and Metallurgy or the Australian Institute of Geoscientists or a 'Recognised Professional Organisation' (RPO) included in a list promulgated by ASX from time to time.
- I have reviewed the Report to which this Consent Statement applies.

I/We am a full time employee of

(Insert company name)

Or

I am

an Independent Mining Consultant.

(Insert company name)

and have been engaged by

Nordic Mining

(Insert company name)

to prepare the documentation for

Engebo

(Insert deposit name)

on which the Report is based, for the period ended

30th October, 2017

(Insert date of Resource/Reserve statement)

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Exploration Targets, Exploration Results, Mineral Resources and/or Ore Reserves (*select as appropriate*).

Consent

I consent to the release of the Report and this Consent Statement by the directors of:

Nordic Mining

(Insert reporting company name)

GWheeler	30th October, 2017
Signature of Competent Person	Date:
The Institute of Materials, Minerals and Mining	45501
Professional Membership: (Insert organisation name)	Membership Number:
	JEAN FITTON.
Signature of Witness:	Print Witness Name and Residence: (eg town/suburb) Cambrore T. P. TR16 4HT