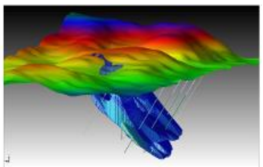
DMT Consulting Ltd
International Mining Consultants

Interim Technical Report

On the

BLÖTBERGET IRON ORE MINE PROJECT, SWEDEN









On behalf of

NORDIC IRON ORE

April 2015

Earth. Insight. Values.

DMT Consulting Ltd
International Mining Consultants

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Document Ref.: C22-R-124

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1 EXECUTIVE SUMMARY AND CONCLUSIONS

1.1 Introduction

1.2 General

Nordic Iron Ore AB (“NIO” or the “Company”) is a mining and exploration company formed in 2008 with the main aim of resuming mining operations at the currently closed Ludvika mines of Blötberget and Håksberg and developing the Väsman field into an integrated mining operation.

NIO owns 19 exploration permits and two mining concessions covering 9,423 hectares of historical mining land in the Västerbergslagen region, where iron ore mining dates back to the 1600s. Since 2010, NIO has been working on development of a number of brownfield and green-field iron ore sites including a preliminary economic assessment (“PEA”) and several internal scoping and trade-off studies.

The company was formed through the contribution in kind of 12 exploration assets by Kopperberg Minerals AB and IGE Nordic AB, all whom are current owners of NIO. NIO received funding during 2014 that the Company intended to use for a Feasibility Study (FS) for the development of its Blötberget and Håksberg mines located near Ludvika in the Västerbergslagen region of Sweden.

Following internal reviews, NIO has changed the company strategy in order to focus initially on restarting operations at Blötberget as a standalone project utilising, as far as possible, the existing mine infrastructure from the previous mining operations. As a result, NIO now intends to limit the FS to the Blötberget brownfield mine as a stand-alone project.

NIO commissioned DMT Consulting Limited (formerly IMC Group Consulting Limited or “DMT”) to undertake the role of Lead Consultant for the FS of the Blötberget Iron Ore Project in Ludvika, Sweden.

As a result of the current economic climate, and in particular the present depressed state of the global iron ore sector, NIO subsequently has decided to delay the completion of the full FS until later in 2015. Instead of completing the originally intended FS, DMT has been asked to compile a more limited technical report, the Draft Interim Technical Report (the “Report” or “Study”), that updates the 2010 PEA and provides the technical details and cost estimates of Phase 1 of NIO’s current development strategy for the Ludvika deposits. This report is an interim report as part of the ongoing FS process and provides the basis of the Project to be taken forward into the FS.

In parallel with the Study, DMT, assisted by NIO, has produced a new Mineral Resource Estimate, dated 10 April 2015. The MRE Report is attached at Appendix E.

The mineral inventory used as the basis of the life-of-mine plan (LOMP) and the process mass presented in this report is taken from an interim mineral resource estimate, which is different to the aforementioned latest MRE. As a result, it will be necessary to update both the LOMP and the mass balance to match the April 2015 MRE.

Phase 1 of NIO's development strategy is for the development and re-opening of the Blötberget brownfield iron ore mine (the "Project"). It is the Blötberget Phase 1 project that is the subject of this Report.

1.3 Project Background

In August 2011, the Mining Inspectorate of Sweden granted NIO the mining concessions for the brownfield mines of Blötberget and Håksberg. The environmental permit for the sites, having undergone a final environmental court hearing in early 2014, was granted later that year.

The Blötberget mine closed in 1979 but was in operation from the early part of the 1900s producing lump, sinter fines and concentrates.

At the location of the Blötberget mine area historical data exists involving magnetic anomaly surveys carried out during the mining operations, SGU Survey in the 1960's and, more recently, surveys conducted by NIO in 2010/11. These and other surveys, as well as historical mine data were used to focus the efforts on several drilling campaigns in 2012 and 2014.

During 2011/12 Berg och Gruvundersökningar AB ("BGU") was engaged by NIO to log and sample historical archive cores that were stored at the SGU repository in Malå. Cores from 13 holes, totalling 5077.21m were re-logged for geological and geotechnical data (for RQD).

In 2012, a 16 hole drill programme which included twinned drilling to confirm the quality of historical drilling data, as well as infill and step-out drilling was completed by NIO. Drilling for this programme totalled 7,430 m

The 2014 NIO drilling programme was designed to investigate the area between Flygruvan/Kalvgruvan and Hugget and to infill the intermediate depth extension of Hugget in order to improve the confidence of the geological model.13 drillholes, totalling 7,093 m, were drilled

1.3.1 2011 Preliminary Economic Assessment

A Preliminary Economic Assessment ("PEA") was prepared by Ramböll, on behalf of NIO, in January 2012. This was based on a project that combined the two existing Ludvika mines, Blötberget and Håksberg.

This study resulted in positive economics (with the iron ore prices current at that time) and involved the upgrade of the existing declines to the old mines to gain access from surface. A new surface decline was planned for Blötberget, while at Håksberg the existing decline was to be upgraded and extended to the 300 m level. Other existing mine infrastructure was planned to be upgraded to allow the use of modern equipment and higher production rates than was the case historically. The mining method is planned to be a version of the previously used sublevel caving and open stoping methods.

The mined ore was to be crushed and transported to surface via vertical and inclined shafts at both mines. At Blötberget mine the ore would be fed directly to the new concentration plant in Skeppmora, while from Håksberg the ore would be transferred by rail to the concentration plant in Skeppmora.

At a combined production rate of 5.5 Mtpa (2.5 Mtpa at Blötberget and 3 Mtpa at Håksberg) the life of mine was estimated to be approximately 12 years, based on the Indicated + Inferred mineral resource, classified at the time.

1.3.2 Phase 1 – Blötberget Project

The Blötberget Project is part of the Ludvika mining area that includes Grängesberg and Håksberg. The Blötberget area extends 1.2 km, striking east-northeast at approximately 060^0 and includes several independent orebody units named from west to east:

- Kalvgruvan (apatite-rich magnetite mineralisation);
- Flygruvan (apatite-rich hematite dominated mineralisation with minor magnetite mineralisation);
- Hugget (apatite-rich magnetite-hematite mineralisation);
- Sandell (apatite-rich magnetite-hematite mineralisation).

The units dip towards the southeast at between 600 - 550 in the mined-out areas, near-surface and flatten at depth to ~250.

During mining operations, the orebodies were continually investigated by underground diamond core drilling, where it was often completed at around 25m spacing, particularly nearer the surface and close to the mine face operations with drilling tending to be less dense in the deeper parts of the mine.

The Ludvika orebodies contain both magnetite and hematite (martite) ores which require different processing routes.

Since 2010, when NIO acquired the assets, several independent Mineral Resource Estimates (MREs) have been carried out, MRE IV, was dated January 2014 and prepared for NIO by GeoVista AB.

During 2014, NIO undertook a further exploration and testwork programme and this has been incorporated into the geological database and will be used to produce an updated Mineral Resource Estimate, dated 10 April 2015.

The run of the mine (ROM) ore from Blötberget is planned to be 3.0 Mtpa (million tonnes per annum) when fully operational with an expected production of about 1.4 Mtpa of high quality finished iron ore products with an average iron content of approximately 69%.

1.4 DMT Draft Interim Technical Report

In 2013, NIO changed the company strategy in order to focus initially on restarting operations at Blötberget as a standalone project, utilising as far as possible, the existing mine infrastructure from the previous mining operations.

The primary purpose of this Draft Interim Technical Report (the Report) is to present the technical aspects of the standalone Blötberget Project at a preliminary level of engineering and cost estimation. No economic analysis of the Project is presented in this report.

DMT notes that certain elements of the Project have been developed to an advanced level of engineering definition, whilst others are at a preliminary level. The former includes the Skeppmora rail terminal, which has been developed to final engineering level by consultant company WSP, contracted by STA.

For this Study, a process weight recovery of 45%, is based on the results from the 2015 MRE and the mining plan based on SLOS, which produces an average ROM feed grade to the process plant of 36.1% Fe. %[1]. Optimisation of the mining schedule in the next stage of development using SLOS as the mining method should allow an NIO to improve the process yield through this more selective mining method resulting in a higher headgrade.

It should be noted that the mineral inventory used as the basis for the life-of-mine plan (LOMP) and the process mass presented in this report is taken from an earlier interim mineral resource estimate, which is different to the aforementioned latest MRE. As a result, it will be necessary to update both the LOMP, the mass balance and the cost estimate to match the April 2015 MRE.

The engineering and cost estimation has been carried out by various consultants (including DMT) directly appointed by NIO. DMT has reviewed the work undertaken by the other consultants, including NIO itself, and has adopted it for the purposes of this Report.

Below are the Consultants that have contributed to the preparation of the Report:

- DMT
- Ramböll
- Golder Associates
- Tata Steel Consulting (TSC)
- WSP
- Nordic Iron Ore (NIO)
- Oxport

1.5 Geology & Mineral Resource estimate

The Blötberget apatite-iron oxide deposit is located in the western part of the intensely mineralised Paleoproterozoic Bergslagen Province (“**the Province**”) in south central Sweden.

[1] It should be noted that this is higher than stated in Section 8 of the report, for which results based on a pilot test bulk sample of 33.6% are reported. This difference in ROM feed grade of 2.5% supports using 45% for this Study. Updates of this report will include for a revised mining method to SLOS and improved headgrade.

The deposits in the neighbouring area occur along a ~40 km long, broad zone This zone of mineralisation is the third largest iron ore deposit in Sweden by production, only outnumbered by the giant Kirunavaara and Malmberget iron ores in Norrbotten, northern Sweden.

The mineralised zone at Blötberget appears as a set of vertically narrow, elongated lenses dipping 50°–70° to the SE. Airborne geophysical surveys and historical drillholes indicate that mineralisation extends to a depth of at least 900 m below surface.

The Blötberget field consists of five mineralised bodies, from west to east, these are:

Table 1-1: Mineralisation zones

Mineralised Body	Short Form Name	Mineralisation
Kalvgruvan	KALV	apatite-rich magnetite
Flygruvan	-FLY	apatite-rich, hematite-dominated, minor magnetite
Hugget & Betstamalmen	‘Betsta’ or ‘the Wedge’	Apatite-rich magnetite-hematite
Sandellmalmen	‘Sandell’- SAND	Apatite-rich magnetite

The Hugget and Kalvgruvan/Flygruvan zones had previously been mined down from near-surface to the 200 m and 240 m levels respectively. The units dip towards the southeast at between 50° and 55° in the near-surface mined-out areas, and flatten at depth to ~25°.

The area, previously known as ‘the Wedge’ or Betsta, was an unknown area between the two former mining concessions, Vulcanus and Blötberget. The Wedge was successfully explored during the 2014 drilling programme.

1.5.1 Exploration and Drilling Data

Since the formation of NIO, several surface sampling campaigns have taken place. The majority of these have been within the mining concession areas but some have extended to include the surrounding exploration concessions in order to allow a better understanding of the geochemical relationship between the satellite deposits and the main Blötberget mineralised zone. This work assisted with the realisation of potential sites for a bulk sample/test mining site. A handheld magnetic susceptibility (KT-10) device and a Thermo Niton x-ray fluorescence (“**XRF**”) XL3 were used to ascertain iron and magnetite percentages of outcrop samples. Rock samples were then sent to ALS for chemical assay.

During 2009, Kopparberg Mineral AB carried out a more detailed magnetometry survey over a limited part of the Blötberget area on behalf of NIO

A drilling programme was undertaken during the summer and winter of 2012 and was completed in November 2012. This 16 hole programme included twinned drilling to confirm the quality of historical drilling data, as well as infill and step-out drilling.

NIO completed 16 drillholes totalling 7,430 m of drilling. The NIO drilling in 2012 was carried out by the Swedish contractor, Drillcon Core AB, or by their Finnish subsidiary Suomen Malmi Oy (“**SMOY**”) using Onram 1000 and Onram 1500 drill rigs and wireline 56 methodology, the programme recovered 39 mm diameter drill core.

The 2014 drilling programme was designed to investigate the area between Flygruvan/Kalvgruvan and Hugget (formally known as “**the Wedge**” or Betsta area) and to infill the intermediate depth extension of Hugget, (-320 m to -660 m; measured from surface depth and relating to mining blocks) in order to improve the confidence of the geological model.

13 drillholes, totalling 7,093 m, were drilled by the Finnish contractor Kati using a Sandvik DE 140 and Onram 1000 and NQ2 drilling methodology, recovering 50.6 mm diameter drill core and producing 75.5 mm diameter drillholes. Kati used a hexagonal reamer which helped ensure that the drillholes had minimum deviation.

One of the drillholes (BB_14-011) was drilled down-dip for geotechnical purposes.

As part of the verification program, NIO has re-logged and re-assayed many of the located historical cores. In total, 45 drillholes from Blötberget were found in Malå and 15 at the former mine storage facility in Håksberg.

There has been re-logging of 31 of these cores (6036 m), 950 m of mineralisation has been re-sampled and re-assayed according to current industry practice and standards. This included mineralised core that had not been sampled historically as it fell below the (visual) historic cut-off grade of 35 %. Approximately 5-10 m of mineralised core was sampled beyond the boundaries of the historical sampled sections.

All data of historical and recent drilling programs has been stored in an industry standard database software

The database has been rigorously checked for completeness and error for each drillhole, cross checked with the core photographs. All data has been exported into and modelled using industry standard modelling software (Surpac).

1.5.2 Wireframe Modelling

The interpretation followed the geological concept of a laterally continuous seam-like geometry, which is flexured along the dip direction of 145^o with dips ranging from 50^o at the surface to 35^o at a depth of 800 m below surface.

Three main iron rich zones lie as narrow mineralised envelopes, from the upper (hanging wall) zone to lower (footwall) zone these three zones are referred to as Sandell (SAND), Hugget-Flygruvan (HUGFLY) and Kalvgruvan (KALV). Each of these zones required wireframe modelling for grade estimation purposes.

Individual hanging and footwall triangulated surfaces were produced based on drillholes intersecting the mineralized zones and a set of underground maps of historic mining. The surfaces were extended with half the distance to nearest drillhole as lateral limit of mineralization. A fully enclosed 3D triangulated solid of each zone was achieved by cross-linking the boundary strings. A 15 % Fe cut-off grade has been applied to model the contacts of the mineralised zones. Some intersections did not show a composite grade above 15 % Fe. These low grade intersections were also included in the mineralised zone in order to honour the lateral continuity of the seam-like lava flow model.

The three solid models are representing the most optimistic envelopes which also consider waste material and low grade ore.

1.5.3 Mineral Resource Estimation

DMT has prepared a Mineral Resource estimate for the Blötberget Project with a drillhole database cut-off date of 1st January, 2015.

The Mineral Resource estimate has an effective date of 30th January, 2015 and has an issue date of 10th April 2015.

DMT applied preliminary mining and economic parameters, including commodity price and wireframe assumptions, to estimate a cut-off grade for resource estimation of 25 % Fe (Total).

The total Measured and Indicated Resource estimated for the Blötberget Project, at a preliminary economic cut-off Grade of 25 % Fe, is 47.8 Mt at a grade of 41.5 % Fe (Total) and 0.5 % P.

Of the total estimated contained Fe, the magnetite proportion is estimated at 62% and the hematite at 38%.

DMT considers all of the material reported as Measured and Indicated Resources to have ‘reasonable prospect of economic extraction’ given appropriate economic and technical considerations.

Table 1-2: Blötberget Estimated Measured and Indicated Resource - March 2015

Fe Cut-off % Fe	Resource Category	Volume Mm^3	Tonnage Mt	Density t/m^3	Fe %	Magnetite %	Hematite %	Magnetite proportion %	Hematite proportion %	Phos. %
25	Measured	11.1	42.5	3.8	41.9	36.8	21.9	0.63	0.37	0.51
	Indicated	1.4	5.3	3.7	38.2	30.5	23.2	0.57	0.43	0.5
	Measured + Indicated	**12.5**	**47.8**	**3.8**	**41.5**	**36.1**	**22.0**	**0.62**	**0.38**	**0.51**
	Inferred	1.5	5.4	3.5	33.5	23.5	23.5	0.50	0.50	0.52

Notes:

1. *JORC 2012 definitions were followed for estimating Mineral Resources;*
2. *Mineral Resources are estimated at a cut-off grade of 25 % Fe;*
3. *Mineral Resources are estimated using a five year historical average price of US$ 100 per tonne (Source: IndexMundi); and*
4. *Figures may not total due to rounding errors.*

1.6 Geotechnical and Hydrogeological Assessment

1.6.1 Geotechnical Assessment

A preliminary geotechnical assessment has been undertaken by Ramböll. All of the work is summarised in Section 5 and the relevant Ramböll reports are presented in Appendix F.

The work is based mainly on the Rock Quality Designation (RQD) data from the drill cores logging performed in 2012; a review of the structural mapping from old mine maps and, on previously conducted geological studies.

Although some of the early data acquired from the 2014 drilling campaign has been used for this assessment, not all of the new data was available at the time of the assessment. This should be incorporated into the database for any future geotechnical work.

The compilation of RQD is based on the average value of approximately 50 m of the footwall and 50 m of the hanging wall. The rock is generally of a high quality producing good core recovery with high RQD values being measured In general, the cores show a relatively low to moderate joint frequency. There appears to be very little variation between the rock mass quality in different parts of the country rock and the mineralised zones.

Mapping of the drill cores from the 2014 drill programme indicates similar rock qualities measurements to those from previous drill core studies. From the old mine maps, the dominant strike of the fractures is almost parallel to the mineralization, east-northeast direction (N070E), and the dominant strike of the faults is in a southeast direction (N110-160E). This dip and strike of the structures are consistent with the results from core mapping in 2012 and 2014.

The Mathews stability method has been used to evaluate the proposed mining method. The analysis is made with consideration of structural data, possible mining stopes and size of sublevel slices.

The Mathews stability method is an empirical tool commonly used for predicting open-stope stability.

The stability assessment was based on a 10m length and a span of the full width of the orebody span, representing one sublevel blast slice, indicates that openings with a span of up to 10m in most parts of the orebodies will be stable with rock failure unlikely.

Although sublevel caving (SLC) has been historically used to extract ore from the Blötberget orebodies, given the apparent stability and strength of the hanging wall rock mass there is concern over the likelihood of initiating and maintain caving of the hanging wall, which is an essential prerequisite for the success of the method.

In order to assess the stability of SLC stopes the geometry of the joint sets has been considered. The analysis consists of determining what wedges will form in one stope. For this analysis, only structure data from old mine maps has been used, and since this dip and strike of the structures are consistent with the results from core mapping in 2012 and 2014, no update of the figures and analyses has been made.

The stope was given a width of 10 meters and height of 20 meters. This is the geometry of one sublevel. The inclination of the ore is approximately 43° on the 420 m level. Two analyses using ‘Unwedge’ software, were conducted, for each ore direction as well as comparison of wedges on the hanging wall/foot wall and the host rock at the extremities of the orebodies. The analysis indicates that fairly small wedges are formed on the hanging wall and footwall sides with some larger wedges are formed in the host rock at the extremities of the orebodies.

The Mathews stability analysis shows that the rock is mainly in the stable zone of the graph. However, since the boreholes were drilled in the same direction, some of the joint sets may be undetected and when modelled using an assumption that the rock has more fractures, this resulted in more unstable rock. The analysis carried out confirmed that SLC is an option for Blötberget although there is some doubt and risk that SLC can be successfully applied to the Blötberget orebodies. As a result, NIO has adopted a mine design based entirely on the use of sublevel open stoping (SLOS).

The mining operations at the Blötberget mine will alter the in-situ stress conditions of the rock mass surrounding the mineralised bodies. A small lake about 30m in depth, Lake Glaningen, is located approximately 500m to the west of the Blötberget orebodies and the stress re-distributions brought about by the mining activity may influence the water flow paths between Lake Glaningen and the mine. There is a small risk that if the existing flow paths widen or new ones are formed then water from the lake might drain into the mine, increasing the pumping requirements of the mine and creating a negative environmental impact in the area.

To consider this possibility an evaluation of the stress re-distribution and rock deformation, the influence area of this stress re-distribution, and the possible effects on the rock mass and the flow paths between lake Glaningen and the mine using the three dimensional numerical modelling software (FLAC3D) was carried out.

The numerical analysis indicates that stress and displacement changes due to mining operations will affect the rock mass close to the lake when the mining depth reaches 450 m. When the whole resource is mined down to 900 m depth displacements will have extended beneath the lake. The risk of the mining activities impacting the lake increases as the mining depth extends below 450 m.

The fractures in the rock mass have orientations that they will likely be affected by the changes in stress and deformation and thereby may increase the conductivity of the rock mass between the lake and the mine.

DMT recommends that all of the available data now available is compiled and used in the next stage of study for the feasibility study in order to confirm initial findings summarised in this report.

1.6.2 Hydrogeological tests

A number of water pressure tests have been performed in some of the 2014 programme drillholes. The water consumption has been measured between a packer and the bottom of the hole at several depths. The length of the measured sections varies between approximately 20 and 100 m.

Generally the water losses were relatively small in sections that compile the mineralised sections and the adjacent country rock. The larger water losses are concentrated in the hanging wall above the mineralised areas, between 5 and 300 m depth.

Based on the limited available data the conductivity of the rock mass is generally very low. In part, this may be due to the long length of open borehole tested in these packer tests. The calculated conductivity values would probably be locally higher if the measurements had been carried over shorter intervals using double packers to separate the identified conductive zones such as fracture zones. DMT recommends that this should be carried as part of the next stage of hydrogeological investigation.

However, even in sections of fractured and crushed rock where higher water losses occurred, the permeability of the rock mass is low to medium. The “normal” rock mass, close to the orebody, shows an even lower hydraulic conductivity which is in the range supported by the fact that this rock mass has a lower fracture frequency, often with a RQD >80 %.

DMT recommends that further hydrogeological work needs to be carried out to confirm the initial findings summarised above.

1.7 Mining

Mining in the Blötberget area has been carried out since the early 1900’s by a variety of methods, originally by open pit extraction followed by underground exploitation using shrinkage stoping. However, since the 1940’s, the preferred method changed to Sublevel Caving (SLC) which prevailed until the closure of the mines in 1979.

SLC being best suited to the upper levels of the Sandell and Hugget deposits and the introduction of modern diesel powered trackless equipment that replaced the old track bound equipment in the early 70’s.

NIO has prepared a mine plan that utilises the existing underground mining infrastructure as far as possible to minimise the pre-production capital investment.

As a result both the traditional SLC method and Sublevel Open Stoping (SLOS) were considered in detail.

Both mining methods have their merits for adoption at Blötberget however, SLOS has been chosen as the preferred mining method for the whole of the Blötberget deposit for a number of reasons; SLC is only suitable above the -500 m level where the orebody is sufficiently wide enough and dips are greater than 40^o to allow successful caving to occur.

The economic assessment of both methods offers similar mining costs per ton of product. Although ore recovery is higher for SLC than for SLOS the dilution is less for SLOS resulting in a higher headgrade for the feed to the feed, which is likely to be a major driver in processing of the Blötberget ores. In addition, the higher head grade in the early years of production when applying SLOS has an advantageous effect on early cash flow.

The future target production for the Blötberget mine is 3 Mtpa ROM ore and 0.3 Mtpa of waste rock over a LOM of 15 years.

The LOMP consists of three stages:

- Pre-production (initial development and preparation for ore production);
- Production ramp up and;
- Steady state operation at nameplate capacity of 3.0 Mtpa.

Since the mine is currently flooded the first stage of pre-production will be the installation of a dewatering system in the BS Shaft and dewatering of the mine. The dewatering system will consist of submersible pumps suspended from steel pumping ranges, which will be extended as the water level is lowered. NIO already has a permit to discharge mine water at an average annual rate of 150l/s (short-term peak flow of 300 l/s) into the local River Gonäsån, once certain preparatory works have been completed.

Initial mine development from the surface will commence when the water level has been reduced sufficiently to safely allow underground development to be commenced and will continue in parallel with the dewatering operation.

NIO intends to use a contractor to carry out the initial main re-development of the mine and also for the on-going capital development. Ore production and the necessary associated stope development activities will be carried out by the Owner’s workforce.

The Pre-production stage will last 27 months with first access to ore occurring in month 15. An 18 month ramp up phase has been assumed in order to reach 3.0 Mtpa. At peak capacity the mine will be producing around 10,000 tpd.

For the purposes of the Base Case in this Study, the LOMP is based on the following assumptions:

- SLOS mining method throughout
- In-situ tonnes 48.1Mt at 41.6%Fe
- ROM headgrade is 36.1%Fe
- Ore ratio - 76%Magnetite/24% Hematite
- Process weight recovery (yield) - 45%.

A Primary crusher will be located initially at the 420 m level with a conveyor system transporting ore to the surface via a combined conveyor and vehicular access decline.

Main haulage levels will be developed at 80 m intervals at 420, 500, 580 and 660 levels at the mines deepest point, with a twin separate ramp and conveyor decline system developed below 420 m.

The crusher installation will sequentially move down through the mine as extraction advances through the orebodies.

Ore will be transported via ore passes to the Primary Crusher stations using a combination of 50 tonne trucks and 7.5 m^3 capacity LHD's.

Mine ventilation will be provided utilising the existing BS Shaft, Vulcanus Shaft and Central Shafts and internal ventilation raises between the main levels.

The BS Shaft will provide the main intake to the mine with the other two shafts acting as exhaust airways. The main fans will be installed underground at the 370 m level in order to minimise surface noise pollution. A gas powered heating system for the intake air will be provided at the BS Shaft.

1.8 Iron Ore Concentrate processing

The Blötberget ore is very similar to that of the nearby Grängesberg mine which has assisted with the development of a suitable flow sheet for the processing of the orebody, along with testwork carried out via a pilot plant.

The process flowsheet has been developed with the objective of achieving a high recovery of iron into concentrates from ore of variable iron content (33.6 % to 36.1 % Fe) and varying ratios of iron present as magnetite and hematite. The range of the magnetite/hematite ratios used for development of the process flowsheet was approximately between 100:0 and 60:40.

It is notable that negligible quantities of other iron-bearing minerals, especially iron silicates, are present. Thus, the total iron content of the ore is potentially recoverable as iron oxides.

It is also notable that the ore contains very little sulphur and negligible quantities of iron sulphides such as pyrrhotite which can complicate the mineral processing flowsheet because it can be significantly magnetic.

The basic mineral separation process, as developed by comparison with modern practice and supported by metallurgical testwork, comprises the following stages:

1. Liberation of the minerals by comminution (size reduction through a combination of crushing and grinding to a particle size less than 0.10 mm (100 µm)
2. Separation of the magnetite concentrate by LIMS
3. Separation of the hematite concentrate by gravity concentration
4. Removal of phosphorus from the iron ore concentrates by froth flotation

Further development of this basic process has been developed to eliminate gangue at a coarser size thereby saving energy and costs in subsequent fine grinding. The basic process flowsheet is:

1. Partial liberation of the gangue by size reduction to a particle size of less than 1 mm.
2. Separation of a magnetite pre-concentrate by LIMS
3. Separation of the hematite pre-concentrate by WHIMS or gravity concentration

4. Regrinding of the magnetite pre-concentrate to the liberation size of 0.10 mm
5. Separation of a magnetite concentrate by LIMS
6. Regrinding of a hematite pre-concentrate to the liberation size of 0.10 mm
7. Separation of a hematite concentrate by gravity concentration
8. Removal of phosphorus from the iron ore concentrates by froth flotation

Based on the pilot scale testwork, the final hematite concentrate contains 66% Fe and the magnetite 70.5% Fe. The concentrates are dewatered by pressure filtration to produce a final product as filter cake.

Overall the process produces 58% tailings from the ROM feed giving a weight recovery to concentrate of 42%.

1.9 On-Site and Off-Site Infrastructure and Utilities

For Phase 1 of the Project, NIO intends to develop an Industrial complex at Skeppmora, adjacent to the main railway line between Ludvika and Göteborg.

Two land parcels have been permitted for this development namely; Industrial Area 1 and Industrial Area 2.

Industrial Area 1 will be placed close to the railway. The Industrial Area 1 will contain the process plant and associated facilities, vehicle work shop, filling station, office/locker room, stores, incoming transformer, switchgear, drill core archive, emergency stockpile and general maintenance facilities.

Adjacent to Industrial Area 2 will be a gravel area that will be used principally by contractors for portable cabins etc. There will be also a temporary stocking area for waste rock.

The structures and facilities planned at the Skeppmora industrial complex include:

- roads within and between the industrial areas
- service buildings for the mining operations
- the surface portal of the main access decline
- Various stockpiles and storage facilities for ore, waste and iron ore concentrate
- transfer conveyors for ore, waste and concentrate product
- buildings housing the concentration plant, offices etc.,
- fire and fresh water facilities
- electric power installations and distribution network to surface and underground facilities
- communication network
- Railway terminal and concentrate load out facility.

The Mine power supply transmission and distribution network will be derived from a 50kV overhead power line that is currently located on the project site; the mine site will be powered via a single 50kV/12kV transformer rated at 40MW feeding a main substation (STV1) located at Industrial Area No.1.

The mine site water supply will come from a closed water management system that utilises the dewatered mine water, surface water run-off and recycled process water.

Potable water will be provided through connection to the municipality water supply system.

All mine site support facilities will be located within industrial area 1 at the location of the portal. This will include offices workshops and processing plant.

Prior to the dewatering of Blötberget mine and mine water being discharged into the local water courses, it will be necessary to re-route the River Gonäsån.

Two water courses where built in the 1950s to divert the river Gonäsån from the Blötberget mining area. This included two tunnels, one under Blötberget and the other under Främundsberget together with four new water diversion channels.

It is intended that most of the existing channels and tunnels will be used in the new diversion of the River Gonäsån. A new emergency outlet from Lake Glaningen will be excavated towards the duct that leads to the southern channel. The existing outlet will be sealed.

This re-routing work will need to be completed to comply with all the Environmental requirements in advance of the commencement of any dewatering and construction of the mine.

1.10 Product Transportation and Logistics

Ore concentrate will be transported from the processing plant via overland conveyor to a rail load out facility at a rate of 180 tph (nominal). There will be two parallel terminal tracks that can accommodate two separate train sets. Each train set comprising 40 wagons given a total capacity of around 2,500 tonnes.

From the rail load out facility the concentrate will be transported to the port of Oxelösund.

NIO commissioned the Swedish Transport Administration (STA) to design, procure and construct a new terminal at Skeppmora so that the rail switching yard at the mine site can be connected to the main line. The switching yard will consist of 2 terminal tracks approximately 630m long and a loading track approximately 400 m long and can potentially be extended.

This proposed operating system for the terminal will have a design throughput capacity to transport Blötberget iron ore product volumes up to 1.4 Mtpa.

The Swedish Transport Administration has allocated 450M SEK for the upgrading of the main line between Skeppmora-Oxelösund. The distance to the Port of Oxelösund is approximately 270 km and the trains will operate from (Ludvika) Skeppmora.

The Port of Oxelösund has been designed as a dry commodity bulk handling port, with iron ore and coal as its main commodities. It is one of Sweden's largest ports and typically handles 7.6 Mt of cargo.

The port will provide storage capacity of 10,000 tonnes of concentrate local to the ship loading crane and a 5,000 tonne loading silo. Total storage at the port will be in the order of 100,000 – 200,000 tonnes, which equates to 1-2 large vessel ships.

1.11 Environment and Socio-Economics

NIO has already obtained a permit under the current Swedish environmental legislation for mining at both Blötberget and Håksberg in Ludvika.

As Phase 1 of the overall project, NIO will only claim the Blötberget part of the permit.

The Land and Environmental Court has granted NIO a permit, according to Chapter 9 of the Environmental Code as follows:

- to construct and operate Blötberget and Håksberg mines for mining a maximum of 3 million tonnes of ore per annum and mine,
- to deposit waste sand (tailings) from the concentrating plant, and
- to erect, construct and operate the installations required for the work, including, among things, shafts, drifts, ramps and other rock work required for the mining, installations for the transport and storage of mined ore and waste rock above and below ground (tunnels, roads, shafts, conveyor belts, hauling devices, mucking arrangements, intermediate stores etc., plant for crushing above and below ground, a concentrating plant, plant for transporting and loading raw ore and finished products (railway terminals etc.), an installation for transporting waste out of the concentrating plant and a tailings storage facility including clarification dam.

The Land and Environmental Court has granted NIO a permit, according to Chapter 11 of the Environmental Code the key elements of which are as follows:

- To dewater the mine
- to divert the River Gonäsån from its natural bed to excavated channels and rock tunnels under the communities of Blötberget and Främundsberget respectively
- to construct various culverts and construct and demolish settling dams respectively during the construction period,
- to drain land in an area south east of the Blötberget mine and to erect a pumping station
- to construct and operate dams and associated devices for the tailings disposal and settling plants,
- to abstract up to a maximum of 100 l/s of water from Lake Väsman and erect the installations required for this

1.12 Capital Costs

The overall capital cost estimate for the option of producing pellet feed concentrate has been developed by DMT. The cost input parameters and their respective values rely to a significant extent on contributions from the other organisations responsible for the respective sections of this report.

All costs are expressed in US dollars, with exchange rates based on rates quoted by Sweden's Central Bank (Riksbanken) at 31st March 2015, with no allowance made for interest or financing during construction.

The base date of the CAPEX estimate is 31st March 2015. The exchange rates used to convert other currencies into US Dollars are:

- SEK / US$ = 8.6232
- SEK / EUR = 9.2869
- SEK / GBP = 12.7441
- EUR / US$ = 1.0770

The estimated cost to bring the mine into production including design, construct, install and commission of the Project operation and facilities described in this report is US$181.2 million. This amount includes the direct field costs of executing the Project, plus shipping and logistics in Sweden.

Sustaining capital expenditure of the life-of-mine is estimated to be US$70.8 million.

Contingency included amounts to US$23.6 M, equivalent to 9.4% of the total capital expenditure.

Cost estimates are based on the level of engineering presented in this report and are considered to have an overall accuracy of +/-25%.

The capital costs are summarised in Table 1-3.

Table 1-3: Summary of Initial and Sustaining Capital Expenditure

Item Description	Initial Capex US$'000s	Sustaining Capex US$'000s	Total US$'000s
Mining - Underground	49,643	41,877	91,511
Surface Handling	11,444	-	11,444
Rail Terminal	12,055	-	12,055
Port	Incld in port charge	0	Incld in port charge
Site Civils	13,081	-	13,081
Surface Buildings	12,523	-	12,523
Surface Electrical Power	5,371	-	5,371
TMF & pipeline	6,926	14,095	21,021
Process Plant	61,503	3,028	62,715
Environmental & land acquisition	5,143	5,984	11,127

Item Description	Initial Capex US$'000s	Sustaining Capex US$'000s	Total US$'000s
Mine Closure		5,798	5,798
Shipping & logistics in Sweden	3,479		3,479
TOTAL	**181,167**	**70,773**	**251,940**

1.13 Operating Costs

Average life of mine Total Operating costs are estimated to be US$19.64/tonne of ore, equivalent to US$43.96/dry metric tonne of concentrate, based on an average ROM production of 3.0 Mtpa and a process weight recovery of 45%.

Section 13 provides the details of the estimated operating costs over the mine life. These are summarised in Table 1-4.

Table 1-4: Average Opex Costs

Item/Description	Operating Expenditure US$/$t_{ore}$	Operating Expenditure US$/$t_{conc}$
Mining Opex	9.54	21.19
Processing Opex	2.05	4.88
Underground Infrastructure Opex	0.19	0.43
Surface Infrastructure Opex	0.10	0.23
Power (excluding Process)	1.10	2.44
TMF Opex	0.40	0.88
TMF Pipeline & pumps	0.06	0.13
Railway Opex	2.86	6.35
Port handling	1.78	3.95
G&A Costs	1.56	3.48
TOTAL	**19.64**	**43.96**

1.14 Product Marketing and Distribution

Metallurgical testwork programmes on samples of the Blötberget ore carried out by NIO in Sweden and Finland, plus historical records demonstrate that a very high quality iron ore concentrate can be produced from the iron ores of Blötberget. The testwork demonstrates that iron ore concentrates with around 70% Fe (t) content can be produced with low levels of contaminants.

NIO considers that the development of the Ludvika mines is low risk (compared with many developments around the globe) and will provide the market place with low volumes of a niche product to those steel companies wishing to increase their product quality and lower their operating costs.

There has been a severe decline in the prices of the iron ore in the last two years. This has had a major impact on many of the iron ore development projects and an even bigger impact on many operating mines which were developed when prices were high, and when capital spending was less well controlled and operational efficiency less important. The influences that have driven these prices lower are many and are discussed in Section 14 of this report.

NIO has collated significant data from banks and other financial institutions regarding the trends in the iron ore market and has presented its own view of the market for iron ore in Section 14.

1.15 Project Execution Plan

The project execution plan is presented in summary in Section 15.

Appendix H provides the outline development and operational schedule for the Blötberget mine project covering the mine establishment period, production, operation and final mine closure and reclamation.

The schedule covers the two years of pre-production (Year-2, and Year-1,) up until the anticipated start of commercial production and then 15 years of ore production, with final mine closure in Year 16.

It is based upon and related to the activities set out in the pre-development and production schedule covered in the Section 6.7.

1.16 Project Risks and Opportunities

1.16.1 Project Risks

The register of risks and opportunities potentially affecting the success and economics of the project has been derived from a review of the uncertainties referred to in the respective main chapters of this report. Qualitative judgments have been made to categorise the likelihood of the risk arising, and the level of potential impact upon the success and economics of the mine. This is presented in Section 16.

On the whole the levels of risk in terms of the combinations of likelihood and impact are adjudged to lie within a low to medium level of risk. This is largely thanks to the fact that the mine has previously been operated meaning that there are large amounts of data on geology, mining, dewatering and other aspects against which judgments on future activities can be calibrated. This allows the level of uncertainty to be relatively well constrained given that this report is part of the development of a feasibility study.

1.16.2 Project Opportunities

The Blötberget project is Phase 1 of NIO's overall strategy for the development of the former Ludvika mines. Phase 2 of the Project is intended to lead to the development of a much larger operation that incorporates the existing Håksberg mine and the undeveloped and relatively recently investigated Väsman deposit. The development of these later projects will benefit from the establishment of infrastructure provided through Phase 1, the Blötberget Project.

The location of the Project in a major former mining region area of Sweden, within easy access of logistical infrastructure is a major advantage for the project. Support services such as established equipment suppliers and mining service providers are readily accessible in the local region and in Sweden generally. Costs of product logistics are well defined with early agreements at an advanced stage with some of the key parties to this aspect of the Project. Both of these serve to significantly de-risk the project overall and offer the potential for project upside.

With the exception of logistics solution, the level engineering definition of the project described in this Report is at relatively early stage. This suggests that there should opportunities to optimise the engineering and reduce some of the capital costs. This particularly applies to the process plant and industrial area buildings and structures.

Clearly at the time of undertaking this Study, a major risk to the project economics (which have not been assessed in this Study) is the future price for iron ore concentrates, which is hugely uncertain at the current time. In te medium to long-term, it seems reasonable to assume that there will be a recovery of prices from the currently depressed price level back towards US$100/tonne or above. Given the likely timeframe for the commencement of commercial production from the Blötberget Project, this offers an opportunity to enhance the project economics quite significantly.

Whilst recently the project economics have been enhanced by shifts upwards in the value of the US$, it is quite plausible that future currency shifts could go the other way. However it should be noted that under these circumstances commodity prices may well also rise.

1.17 Interpretation & conclusions

1.17.1 Geology

The Hugget and Kalvgruvan/ Flygruvan zones had previously been mined down from near-surface to the 200 m and 240 m levels respectively.

Since acquisition of the property, NIO has undertaken two drilling programmes.

A drilling programme was undertaken by NIO during the summer and winter of 2012 and was completed in November 2012. This 16 hole programme included drilling to confirm the quality of historical drilling data, as well as infill and step-out drilling. NIO completed 16 drillholes totalling 7,430 m of drilling.

The 2014 drilling programme was designed to investigate the area between Flygruvan/Kalvgruvan and Hugget (formally known as “the Wedge” or Betsta area) and to infill the intermediate depth extension of Hugget, in order to improve the confidence of the geological model. 13 drillholes, totalling 7,093 m, were drilled.

'The Wedge' was successfully explored during the 2014 drilling programme and, as a result, Kalvgruvan and Hugget/Flygruvan have now been shown to be continuous zones of mineralisation

Mine maps and historical drilling data have been collected from various sources and digitised, where possible. Drill core from historical exploration drilling in the Blötberget project area has been recovered, re-logged and re-analysed.

In the resource development programme of 2012 and 2014 NIO completed industry standard QA/QC programs to ensure the data is reliable and suitable for resource estimation. The drill density of the resource is adequate for the purpose and is reflected in the JORC compliant resource category classifications of Measured, Indicated and Inferred Mineral Resource.

DMT has relied heavily upon the information provided by NIO, however DMT has, wherever possible, independently verified data provided during the site visits.

DMT was able to overlay licence information on the Mineral Resource estimate area to confirm that the deposit is within NIO’s license. DMT has not undertaken a legal review of the licences and assumes that all the required licences are in place.

The geology of the deposit is fairly well understood and DMT has constructed a wireframe geological model for the Blötberget deposit based upon a combination of logged lithologies and analytical and SATMAGAN magnetite results. This has allowed the splitting of the deposit into geological domains comprising, magnetite-rich material of KALV and hematite-rich material of HUGFLY and SAND.

DMT has undertaken a statistical study of the data, which demonstrates adequate splitting of the data into single iron population domains, and undertaken a geostatistical study to investigate the grade continuity and to provide grade estimation parameters for Ordinary Kriging.

A Surpac block model using all the available geological and sample analytical test data has defined an iron ore resource. At this stage of the investigation most of the mineral resources of Blötberget have been classified into the Measured and Indicated categories.

As a result of the site visits, data base verification and validation and the geological and model generated, DMT has estimated the total Measured and Indicated Resources for the Blötberget Project as 47.8 Mt at a grade of 41.5 % Fe (Total) and 0.5 % P at preliminary COG of 25 % Fe. Of the total estimated contained Fe, the magnetite-hematite ratio is estimated at 62:38.

1.17.2 Mining

The orebodies included in the Blötberget project have been extensively worked in the past down to level 280. A variety of mining methods were used historically including shrinkage stoping, open stoping and sublevel caving.

Although SLC appears to have been successfully applied in the past, NIO has chosen to adopt SLOS as the mining method for this Study. DMT agrees with this approach since it offers more certainty of success at this stage, given the strong hanging wall and the risk that it may not cave successfully. In addition, the use of SLOS allows a lower dilution factor to be adopted, an important element since headgrade to the mill significantly affects iron recovery from the Blötberget ores.

At the time of preparing the LOMP development and production schedule, the new blockmodel was not available. Therefore, this Study is based on assumed in-situ grades although the tonnages are based upon volumes calculated from the latest wireframe.

The mine design is based on utilising as much of the existing underground infrastructure as possible. So for example, the existing shafts (after dewatering) will be used for main intake and exhaust airways. However, main access to the mine and mineral handling of ROM ore and waste rock will be achieved via a new surface access decline driven adjacent to Industrial Area #1 at Skeppmora. The existing BS shaft will not be used for hoisting under the current scheme. All rock from underground will be conveyed to surface via a conveyor system connected to the main access decline, after having been crushed.

DMT considers that sufficient optioneering studies have been undertaken to allow the mine design to be advanced to feasibility study level, without further option studies being needed. Nevertheless, further work is needed to raise the confidence levels and engineering definition to support a full feasibility study.

Costs estimates have been developed to a level of accuracy appropriate to a scoping study. Some costs (notably development and mobile equipment) are based on budget quotations received by NIO from contractors and OEMs, thereby providing some increased confidence in these figures.

1.17.3 Process

The proposed mineral processing plant described in Section 8 has been designed primarily on the results from the small-scale (1 t/h) pilot plant at GTK which operated for 24 hours over a period of several days processing approximately 22 tonnes of a single, composite bulk sample extracted from surface outcrops of the Blötberget orebodies.

The design is further supported by the results of bench-scale laboratory testwork conducted on drill cores recovered from deeper parts of the orebodies in a programme of metallurgical variability testwork although at the time the process flowsheet and the standard metallurgical test for variability included wet high intensity magnetic separation to recover hematite rather than gravity concentration.

As a result, the preliminary flowsheet used for this study is considered to incorporate sufficient flexibility to ensure that the processing plant would be able to treat ore of wide varying quality.

The flowsheet and equipment has been selected in order that the process plant can successfully recover concentrates from ore of variable iron content and varying ratios of iron present as magnetite and hematite.

Equipment has been reviewed and selected on the basis of utilising current "state of the art" items without taking unnecessary risks with unproven plant or process technology.

1.17.4 Infrastructure

The main infrastructure needed to support a mining and processing operation will be located at the two Skeppmora industrial sites, which have already been permitted.

The basis of this study is to place the majority of the infrastructure on Industrial Site #1, where the process plant and railway terminal and loadout facility will be located. The portal for the main access decline will also be located on this site area. This avoids transporting of raw ore or concentrate between site areas, reducing the surface environmental impact.

All of the key infrastructure elements necessary for the operation of the Blötberget mine have been included in the Study. With the exception of the railway terminal, the engineering development of these facilities is at a conceptual/preliminary level of definition, as are the associated cost estimates, and will need to be advanced considerably at the next stage of study in order to support a full feasibility study.

The detailing of the tailings storage facility has been advanced to a preliminary level of engineering appropriate to this Study. Reasonable assumptions have been made as regards the available of suitable construction material for the dams and the ground conditions that exist at the dam foundations. However, ground investigation is necessary to support a feasibility level design.

1.17.5 Environment and Permitting

The Study has been carried paying due consideration to the conditions included in the Environmental Permit already obtained by NIO.

1.17.6 Capital and Operating Costs

The overall capital cost estimate for the option of producing pellet feed concentrate has been compiled by DMT, with significant contributions from the other organisations responsible for the respective sections of this report.

The estimate has been the subject of iterative discussions between NIO and other contributors, and between NIO and DMT.

DMT considers that the overall accuracy level of the cost estimation presented in this Report is approximately +/-25%. However it is noted that some costs such as the capital and operating costs for the logistics system (railway and port) have been estimated to a higher level of accuracy than this.

The estimated cost to bring the mine into production including design, construct, install and commission of the Project operation and facilities described in this report is US$181.2 million. This amount includes the direct field costs of executing the Project, plus shipping and logistics in Sweden.

Sustaining capital expenditure of the life-of-mine is estimated to be US$70.8 million.

Contingency included amounts to US$23.6 M.

Average life-of-mine Total Operating costs are estimated to be US$19.64/tonne of ore, equivalent to US$43.96/dry metric tonne of concentrate at a yield of 45%.

1.17.7 Project Risks and Opportunities

1.17.7.1 Project Risks

The register of risks and opportunities potentially affecting the success and economics of the project has been derived from a review of the uncertainties referred to in the respective main chapters of this report. Qualitative judgments have been made to categorise the likelihood of the risk arising, and the level of potential impact upon the success and economics of the mine. This is presented in Section 16.

On the whole the levels of risk in terms of the combinations of likelihood and impact are adjudged to lie within a low to medium level of risk. This is largely thanks to the fact that the mine has previously been operated meaning that there are large amounts of data on geology, mining, dewatering and other aspects against which judgments on future activities can be calibrated. This allows the level of uncertainty to be relatively well constrained given that this report is part of the development of a feasibility study being undertaken by NIO.

1.17.7.2 Project Opportunities

The Blötberget project is Phase 1 of NIO's overall strategy for the development of the former Ludvika mines. Phase 2 of the Project is intended to lead to the development of a much larger operation that incorporates the existing Håksberg mine and the undeveloped and relatively recently investigated Väsman deposit. The development of these later projects will benefit from the establishment of infrastructure provided through Phase 1, the Blötberget Project.

The location of the Project in a major former mining region area of Sweden, within easy access of logistical infrastructure is a major advantage for the project. Support services such as established equipment suppliers and mining service providers are readily accessible in the local region and in Sweden generally. Costs of product logistics are well defined with early agreements at an advanced stage with some of the key parties to this aspect of the Project. Both of these serve to significantly de-risk the project overall and offer the potential for project upside.

With the exception of logistics solution, the level engineering definition of the project described in this Report is at relatively early stage. This suggests that there should opportunities to optimise the engineering and reduce some of the capital costs. This particularly applies to the process plant and industrial area buildings and structures.

Clearly at the time of undertaking this Study, a major risk to the project economics (which have not been assessed in this Study) is the future price for iron ore concentrates, which is hugely uncertain at the current time. In the medium to long-term, it seems reasonable to assume that there will be a recovery of prices from the currently depressed price level back towards US$100/tonne or above. Given the likely timeframe for the commencement of commercial production from the Blötberget Project, this offers an opportunity to enhance the project economics quite significantly.

1.18 Recommendations

1.18.1 Geology

It is considered that there is only limited additional geological information that can be gained from further, expensive, surface drilling programmes. The bulk of the upper levels of the Blötberget deposit that provide part of the proposed mine plan are within the Measured Resource category.

However, surface drilling for rock mechanical/structural and or metallurgical information for detailed mine planning a part of the feasibility should be considered.

Definition and grade control drilling should commence as soon as there is access to the underground areas after dewatering. This close spaced drilling is required to support the transfer of Measured Resources into (Proven and Probable) Reserves. The underground drilling should follow a similar drill approach to that used historically, with fan pattern of close spaced drilling into the mine blocks , typically at 35-45 m centres, with wider spaced (100 m) deeper down dip drilling to provide increased confidence in the Indicated area of the resources.

Additional hydro-geological investigations on existing drillholes should be undertaken, as DMT considers that insufficient data exists on the hydrological and hydrogeological conditions for underground mining.

Targeted mineralogical and metallurgical testing is suggested to better understand the ore origins and process response during mining operations.

The properties to the north east of the current project, such as Guldkannan and Sandell offer upside potential to the project. At this stage it may prove worthwhile to carry out an evaluation of the properties in readiness for an application for an extension of the mining concession/environmental permit area that would lead to exploitation of the minerals during the development phase.

There was no use of check samples in the historic core re-assay (BGU), this should be addressed as a partial re-run with standards inserted.

The blank samples assayed to date have indicated between 1 % and 2 % Fe. Prior to further analysis being undertaken, the preparation of suitable blanks for insertion into future sample streams should be addressed by NIO.

Check standards have slightly (but consistently) undervalued the results, this should also be corrected ahead of the next phase of core sampling, which will likely be from underground drill locations.

NIO should continue to source historical data and drill core for the purposes of re-assaying, re-logging and integration into the current database.

1.18.2 Mining

The mine plan and design needs to be further refined based on the new blockmodel to allow definition of stope outlines per level in order to optimise the projected ore recovery and to allow Ore Reserves for the Blötberget deposit to be estimated.

In particular the mine schedule should look at the occurrence of the Magnetite: hematite, levels of P, Ti, V and possible other trace elements throughout the mine programme; in order to better define the ROM characteristics during the mine schedule.

A survey of the existing shafts that will be used to dewater the mine and provide the long-term ventilation intake and exhaust airways must be carried out to remove this risk to the currently adopted mine and ventilation design. It is essential that this is done to support the feasibility study.

1.18.3 Process

The pilot plant testwork carried out to date has been on bulk samples taken at surface. For the future refinement of the flowsheet and plant design it would be advantageous to recover samples from deeper parts of the orebodies for the purpose of metallurgical testing.

1.18.3.1 Confirmation of the process flowsheet

In order to refine and optimise the flowsheet consideration should be given to the following issues:

- Reason for the failure of wet high intensity magnetic separation to produce a high grade hematite concentrate in order that this process may be included /excluded.
- Recovery of fine hematite by spiral concentrators or wet high intensity magnetic separation because the recovery (yield) of hematite is relatively low compared to magnetite and crucially important to the financial viability of the project.
- Locked-cycle tests of HPGR to establish probable superiority over rod mills and SAG milling.
- Further SAG milling tests depending upon outcome of HPGR testwork.
- Pot-milling tests with respect to regrinding of rougher LIMS concentrates and rougher spiral concentrates by vertical stirred media mills to determine the improved grindability after HPGR, if applicable.
- Selection of the most suitable cleaner spiral concentrator or wet high intensity magnetic separator for recovery of fine hematite concentrate.

1.18.4 Infrastructure

Further engineering and quotations from vendors is required to refine the engineering and the costs estimates of the underground infrastructure

Basic engineering of buildings and structures will be required to refine the cost estimates for the surface infrastructure.

Detailed ground and topographical surveys are required to develop the cut and fill volumes of the respective industrial sites.

Specific site investigation is required at the Skeppmora industrial sites and at the locations of the North and South tailings storage facilities to confirm foundation requirements.

1.18.5 Environment and Permitting

It should be confirmed that all details of the currently envisaged Phase 1 project as presented in this study are within the current planning and environmental permits.

Any variances of the project as presently envisaged to that approved will need to be amended or discussed with the appropriate authorities during the next stage of the Project.

As mentioned above, there are potential advantages to be gained for the overall economics of the project by the extension of the mining concession and environmental permit to areas that include Guldkannan and Fremundberget orebodies. DMT considers that these should be reviewed.

1.18.6 Capital and Operating Costs

Outline specifications should be developed for the major process plant and equipment items to allow quotations to be sought from vendors and OEMs.

Project engineering should be advanced to a sufficient level to allow more refined cost estimates to be developed. Typically for a Detailed Feasibility Study (DFS), engineering completion would be about 10 to 15%complete in order to allow material detailed take-offs and equipment lists and specifications to be developed for detailed cost estimation.

1.18.7 Overall

In the current difficult economic climate where project financing of mining projects is challenging, NIO's strategy for the development of the Blötberget project as Phase 1 in the re-opening of the former Ludvika mines seems to be appropriate. This will result in a much less capital intensive and potentially financeable Project.

In addition, the reduced activity in the global mining industry offers the possibility to reduce the capital investment needed since suppliers are now prepared to accept orders and supply equipment in a much more timely manner and to do so at reduced costs. The increase in the value of the US$ has the further potential to improve capital costs in non-dollar sourced plant and equipment and increase the value of the revenue in SEK.

The current climate could also be considered an opportune moment to oversee a "lower" cost development with the potential to start production in a rising iron ore market.

The location of the Project in a major former mining region area of Sweden, within easy access of logistical infrastructure is a major advantage for the project. Costs of product logistics are already well defined with agreements at an advanced stage of discussion. Both of these serve to significantly de-risk the project overall and offer the potential for project upside.

Despite the current downturn in iron ore prices at the moment, NIO has an opportunity to advance studies for building a low cost operating mine capable of competing with the other global suppliers.

DMT considers that the Project as described and configured in this Report should be advanced to the next stage of Project development i.e. completion of a DFS.

In advancing the Project to full feasibility a number of steps should be taken towards enhancing this Interim Report. The Life of Mine Plan should be updated to fully reflect the outcome of the April 2015 MRE and the finalised blockmodel with resulting updating of the plant recoveries to reflect the revised headgrades.

The further steps towards feasibility should consist of continued improvement to the engineering design of the mine, the mining schedule, mining infrastructure requirements, process plant and buildings design. Improved specification of the plant and equipment will allow NIO to specify the capital items in much more detail and obtain competitive quotations that provide greater confidence in the overall economics of the project.

Improved engineering details will reduce the overall projects costs as greater confidence allows costs to be defined nearer +/-15%.

2 INTRODUCTION

2.1 General

Nordic Iron Ore AB ("NIO" or the "Company") is a mining and exploration company formed in 2008 with the main aim of resuming mining operations at the currently closed Ludvika mines of Blötberget and Håksberg.

NIO owns 19 exploration permits and two mining concessions covering 9,423 hectares of historical mining land in the Västerbergslagen region, where iron ore mining dates back to the 1600s. Since 2010, NIO has been working on development of a number of brownfield and green-field iron ore sites including a preliminary economic assessment ("PEA") and several internal scoping and trade-off studies.

The company was formed through the contribution in kind of 12 exploration assets by Kopperberg Minerals AB, Archelon Minerals AB and IGE Nordic AB, all whom are current owners of NIO. NIO received funding during 2014 that the Company intended to use for a Feasibility Study (FS) for the development of its Blötberget and Håksberg mines located near Ludvika in the Västerbergslagen region of Sweden.

Following internal reviews, NIO has changed the company strategy in order to focus initially on restarting operations at Blötberget as a standalone project utilising, as far as possible, the existing mine infrastructure from the previous mining operations. As a result, NIO now intends to limit the FS to the Blötberget brownfield mine as a stand-alone project.

NIO commissioned DMT Consulting Limited (formerly IMC Group Consulting Limited or "DMT") to undertake the role of Lead Consultant for the FS of the Blötberget Iron Ore Project in Ludvika, Sweden.

As a result of the current economic climate, and in particular the present depressed state of the global iron ore sector, NIO subsequently has decided to delay the completion of the full FS until later in 2015. Instead of completing the originally intended FS, DMT has been asked to compile a more limited technical report, the Draft Interim Technical Report (the "Report" or "Study"), that updates the 2010 PEA and provides the technical details and cost estimates of Phase 1 of NIO's current development strategy for the Ludvika deposits. This report is an interim report as part of the ongoing FS process and provides the basis of the Project to be taken forward into the FS.

Phase 1 of NIO's development strategy is for the development and re-opening of the Blötberget brownfield iron ore mine (the "Project"). It is the Blötberget project that is the subject of this Study.

2.2 Project Background

In August 2011, the Mining Inspectorate of Sweden granted NIO the mining concessions for the brownfield mines of Blötberget and Håksberg. The environmental permit for the Project, having undergone a final environmental court hearing in early 2014, was granted in June 2014.

The Blötberget mine closed in 1979 but was in operation from the early part of the 1900s producing lump, sinter fines and concentrates.

At the location of the Blötberget mine area historical data exists involving magnetic anomaly surveys carried out during the mining operations, SGU Survey in the 1960's and, more recently, surveys conducted by NIO in 2010/11. These and other surveys, as well as historical mine data were used to focus the efforts on several drilling campaigns in 2012 and 2014.

During 2011/12 Berg och Gruvundersökningar AB ("BGU") was engaged by NIO to log and sample historical archive cores that were stored at the SGU repository in Malå. Cores from 13 holes, totalling 5,077 m were re-logged for geological and geotechnical data (for RQD).

In 2012, a 16 hole drill programme which included twinned drilling to confirm the quality of historical drilling data, as well as infill and step-out drilling was completed by NIO. Drilling for this programme totalled 7,430 m

The 2014 NIO drilling programme was designed to investigate the area between Flygruvan/Kalvgruvan and Hugget and to infill the intermediate depth extension of Hugget in order to improve the confidence of the geological model.13 drillholes, totalling 7,093 m, were drilled

2.2.1 2011 Preliminary Economic Assessment

A Preliminary Economic Assessment ("PEA") was prepared by Ramböll, on behalf of NIO, in November 2011. This was based on a project that combined the two existing Ludvika mines, Blötberget and Håksberg.

This study resulted in positive economics (with the iron ore prices current at that time) and involved the upgrade of the existing declines to the old mines to gain access from surface. A new surface decline was planned for Blötberget, while at Håksberg the existing decline was to be upgraded and extended to the 300 m level. Other existing mine infrastructure was planned to be upgraded to allow the use of modern equipment and higher production rates than was the case historically. The mining method is planned to be a version of the previously used sublevel caving and open stoping methods.

The mined ore was to be crushed and transported to surface via vertical and inclined shafts at both mines. At Blötberget mine the ore would be fed directly to the new concentration plant in Skeppmora, while from Håksberg the ore would be transferred by rail to the concentration plant in Skeppmora.

At a combined production rate of 5.5 Mtpa (2.5 Mtpa at Blötberget and 3 Mtpa at Håksberg) the life of mine was estimated to be approximately 12 years, based on the Indicated + Inferred mineral resource, classified at the time.

2.2.2 Phase 1 – Blötberget Project

The Blötberget project is the subject of this Study.

The Blötberget mine field comprises several independent mineralised units named (from southwest to northeast) Kalvgruvan, Flygruvan, Hugget, Carlsvärdsgruvan, Sandell, Guldkannan and Fremundsberget. The Kalvgurvan, Flygruvan and Hugget zones are mined down to the 240/350 m level. The units dip towards the southeast at between 60^0 - 55^0 in the mined-out areas, near-surface and flatten at depth to ~25^0.

The Blötberget Project is part of the Ludvika mining area that includes Grängesberg and Håksberg. The Blötberget area extends 1.2 km, striking east-northeast at approximately 060^0 and includes several independent orebody units named from west to east:

- Kalvgruvan (apatite-rich magnetite mineralisation);
- Flygruvan (apatite-rich hematite dominated mineralisation with minor magnetite mineralisation);
- Hugget (apatite-rich magnetite-hematite mineralisation);
- Sandell (apatite-rich magnetite-hematite mineralisation).

During mining operations, the orebodies were continually investigated by underground diamond core drilling, where it was often completed at around 25m spacing, particularly nearer the surface and close to the mine face operations with drilling tending to be less dense in the deeper parts of the mine.

The Ludvika orebodies contain both magnetite and hematite (martite) ores which may require differential processing.

Since 2010, when NIO acquired the assets, several independent Mineral Resource Estimates (MREs) have been carried out the latest of which, MRE IV, was dated January 2014 and prepared for NIO by Geovista AB.

Table 2-1 –Mineral Resource Estimate as of January 2014 (Geovista 2014)

	Hugget			Sandell			Flygruvan			Kalvgruvan			Global		
	Mton	Fe [%]	P [%]	Mton	Fe [%]	P [%]	Mton	Fe [%]	P [%]	Mton	Fe [%]	P [%]	Mton	Fe [%]	P [%]
Measured	10.7	34.3	0.3										10.7	34.3	0.3
Indicated							8.2	36.5	0.4	19.2	48.3	0.5	27.4	44.8	0.5
Meas+Ind													38.1	41.9	0.4
Inferred	20.8	32.55	0.38	0.9	43.14	0.81							21.7	33.0	0.4

Table 5: Mineral resources for Blötberget, effective on January 25, 2014.

During 2014, NIO undertook a further exploration and testwork programme and this has been incorporated into the geological database and will be used to produce an updated Mineral Resource Estimate, which provides the basis for this Study.

The run of the mine (ROM) ore from Blötberget is planned to be 3.0 Mtpa (million tonnes per annum) when fully operational with an expected production of about 1.4 Mtpa of high quality finished iron ore products with an average iron content of approximately 69%.

2.3 DMT Draft Interim Technical Report

In 2013, NIO changed the company strategy in order to focus initially on restarting operations at Blötberget as a standalone project, utilising as far as possible, the existing mine infrastructure from the previous mining operations.

The primary purpose of this Study and the Draft Interim Report (the Report) is to present the technical and economic aspects of the standalone Blötberget Project at a preliminary level of engineering and cost estimation.

DMT notes that certain elements of the Project have been developed to an advanced level of engineering definition, whilst others are at a conceptual level only. The former includes the rail terminal, for which the engineering has been carried out for NIO by STA, who contracted WSP.

Some of the engineering and cost estimation has been carried out by other consultants directly appointed by NIO. This Report has been compiled and reviewed by DMT from the contributions provided by the other Project consultants including NIO and these are listed below. Although DMT takes responsibility for the overall report, reliance has been placed by DMT on the other consultants having applied due professional skill and care in the carrying out of their assigned work.

- DMT
- Ramböll
- Tata Steel Consulting (TSC)
- WSP
- Nordic Iron Ore (NIO)
- Oxport

2.4 Units & List of Abbreviations

Units of measurement used in this report conform to the metric system.

All currency in this report is stated in US Dollars (“US$”) unless otherwise noted.

Table 2-2 List of abbreviations

Abbrv.	Description	Abbrv.	Description
°	degrees	m^3/hr	cubic metres per hour
°c	degrees Celsius	m^3/t	cubic metres per tonne
%	percent	Ma	million years
<	less than	Magsus	magnetic susceptibility
>	greater than	masl	metres above sea level
BCM	bank cubic metres	mm	millimetre
BGU	Berg och Gruvundersökningar AB	MOP	mine operation period
CAPEX	capital expenditure	MRE	mineral resource estimate
cm	centimetre	Mt	million metric tonnes

Abbrv.	Description	Abbrv.	Description
DGRF	definitive geomagnetic reference field	Mtpa	million metric tonnes per annum
DMT	DMT Consulting Limited	m/min	metres per minute
DSCO	drill core structure orientation	m/s	metres per second
E	east	N	north
EIA	environmental impact assessment	NIO	Nordic Iron Ore
ESIA	environmental & social impact assessment	OPEX	operating expenditure
EMP	environmental management plan	OREAS	Ore Research and Analysis Australia
Fe	iron	P	phosphorous
g	gram	QA/QC	quality assurance / quality control
Ga	billion years	REE	rare earth element
ha	hectare	ROM	run of mine
HCl	hydrochloric acid	RQD	rock quality designation
ICP-AES	inductively coupled plasma–atomic emission spectroscopy	S	south
IEC	International Electrotechnical Commission	S	sulphur
IOCG	iron oxide copper gold (deposit)	SATMAGAN	saturation magnetisation analyser
ISO	International Organisation for Standardisation	SEK	Swedish Krona
JORC	Joint Ore Reserves Committee	SG	specific gravity
kg	kilogram	SGU	Swedish Geological Survey
km	kilometre	SiO_2	silica dioxide
km^2	square kilometres	TGA	thermo gravimetric analyser
ktpa	kilo (1,000) metric tonnes per annum	t/m^3	tonnes per cubic metre
kV	kilovolt	tonnes	metric tonnes
LIMS	low intensity magnetic separation	ToR	terms of reference
LOI	loss on ignition	US$	United States Dollar
LOM	life of mine	UV	ultra violet
m	metre	W	west
m^2	square metre	XRF	x-ray fluorescence
m^3	cubic metre		

3 PROJECT DESCRIPTION

3.1 Location

The Blötberget Project is situated in Dalarnas County in central Sweden, approximately 500 m south east of the village of Blötberget, and near to the town of Ludvika. Sweden's largest city, Stockholm, is located to the South East, within driving distance along Route 66 and E18 (228 km). The country's second largest city, Göteborg, to the South West is within driving distance along Route 50 and E20 (400 km).

The Project region is known as the Bergslagen District, famous for its very long mining and steelmaking history, with notable former and current production areas within this region including the Grängesberg iron ore mine, Zinkgruvan sulphide mine, Garpenberg sulphide mine and Falun sulphide mine. Ludvika is located at the southern shore of Väsman lake at an elevation of around 157 m above sea level ("**masl**").

Figure 3-1 Location Map (Source: **Google Maps)**

3.2 Accessibility

Blötberget is located 2.5 km west/north-west of Route 50 and is directly accessible along well maintained asphalt roads.

3.3 Climate

The climate at the Project is classified as cold and temperate (sub-arctic or boreal), characterised by short, cool summers and long cold winters.

There is significant precipitation throughout the year, with an average annual precipitation of 713 mm. The driest month is March with an average of 35 mm of precipitation. Most precipitation falls in August, with an average of 87 mm for the month.

The average annual temperature is around 4.6 °C. The warmest month of the year is July with an average temperature of 16.1 °C. The coldest month is February, when the average temperature is -5.6 °C. The average temperature fluctuation throughout the year is 21.7 °C.

Although unlikely, the cold climate at the Project site has the potential to affect surface operations during the winter months, should the mine become operational.

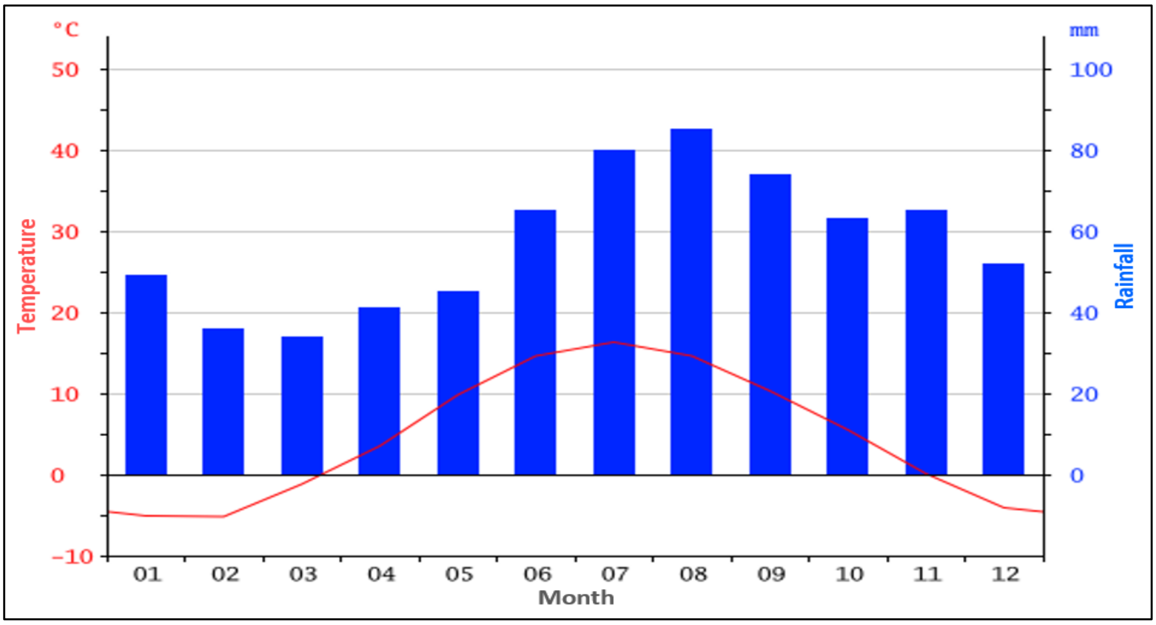


Figure 3-2 Climate data (Ludvika) (Source: **http://en.climate-data.org)**

3.4 Local Resources and Infrastructure

3.4.1 Communities

The closest large town to Blötberget is Ludvika, which is located 6 km east along Route 50. Ludvika has a population of approximately 14,500 and population density of 15.5/km^2 as of 2012 (*Urbistat.it, 2015*). Ludvika offers general services including medical care, telecommunications, banking, housing, hotels, vehicle repair and schooling.

The local community has a 7.8 % unemployment rate, lower than the Swedish average of 8.0 % (*Ekonomifakta, 2014*). The multinational engineering company ABB has manufacturing facilities in Ludvika for power transformers, capacitors and equipment for power transmission. ABB, as well as Sandvik and Atlas Copco are major employers in the area.

3.4.2 Roads

The national highway, Route 50, which runs north-south, passes close to the Project.

3.4.3 Rail

Blötberget does not have its own railway station, however the main line lies adjacent to the Project. This line is the main Northern Swedish railway from Göteborg to Gävle which primarily runs parallel to route 50. There is also a railway station in Grängesberg, 9.7 km to the South West of Blötberget.

The railway that passes through Ludvika and close by the Project, offers connections to three port towns/cities namely; Gävle (180 km) and Oxelösund (270 km) at the Baltic Sea and Lysekil (410 km) on the Swedish west coast.

3.4.4 Air

The nearest airport with domestic flights to and from Arlanda International Airport is Dala Airport, located in the neighbouring town Borlänge, approximately 55 km northeast of Ludvika.

3.4.5 Power

The electrical power required for mining and milling operations will be sourced from the main power line (50 kV), operated by VB-Kraft, which is approximately 1 km to the north east of Blötberget.

3.4.6 Water

Water for the industrial areas and process plant can be sourced from nearby lakes.

3.5 Physiography

The Project is located in an area dominated by arboreal forest.

In general, the local terrain consists of gently undulating hills, except for the area around Blötberget, which is predominantly flat and marshy. The elevation in the Project area varies between 150 and 250 masl.

3.6 Surface Rights

The agreements fully in place are those with the landowner of the water rights for the mine (for dewatering), necessary before the mining concession rights are granted.

Access has been agreed with the landowner to allow the building of the sedimentation ponds, required for dewatering.

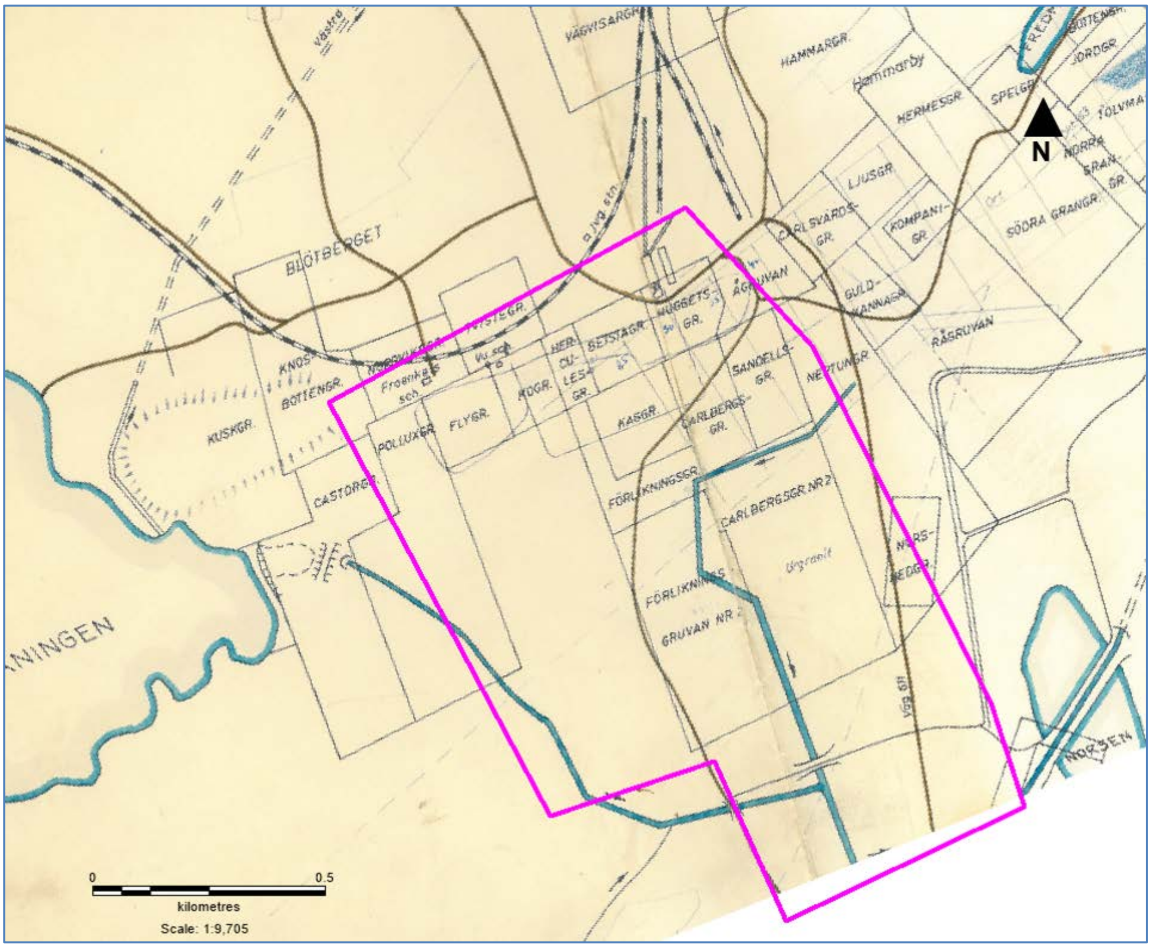
Currently, NIO does not have final agreement for all the industrial areas or the tailings dams. However, discussions and agreements are pending commitments for further investment and advancement of the Project.

Importantly, all the arrangements for the development of the project and the surface rights are now covered by the Mining Laws. In the event of disputes, the Government representatives of the mining law can either arbitrate an agreement or, in extreme cases, expropriation can be enforced if agreement cannot be achieved through negotiations.

3.7 Land Tenure

The Blötberget area was historically subdivided into six fields based on the old mining concessions, namely:

- Kalvgruvan;
- Flygruvan;
- Betsta;
- Hugget;
- Sandell; and
- Guldkannan.



Source: NIO

Figure 3-3 Historical mining and current concessions (pink border) - Blötberget

NIO currently holds 12 exploration permits, which together cover an area of 3,044.36 hectares ("**ha**"). NIO also holds two mining concessions - Blötbergsgruva K nr 1 and Håksbergsgruva K nr 1, covering an area of 262.7 ha. All areas, besides those covered by the Väsman concession and parts of the Håksberg concessions are so called "brownfield" sites and have previously been worked and contain abandoned mines.

NIO applied for a mining license within the Blötberget area in October 2010 and it was granted by the Mining Inspectorate of Sweden in August 2011. The mining concession, which runs for 25 years with possibility of extension, implies the right of exploitation and utilisation of iron, rare earths, and apatite. The environmental permit for this concession was granted in late March 2014. The licence locations and descriptions are shown in Figure 3-4.

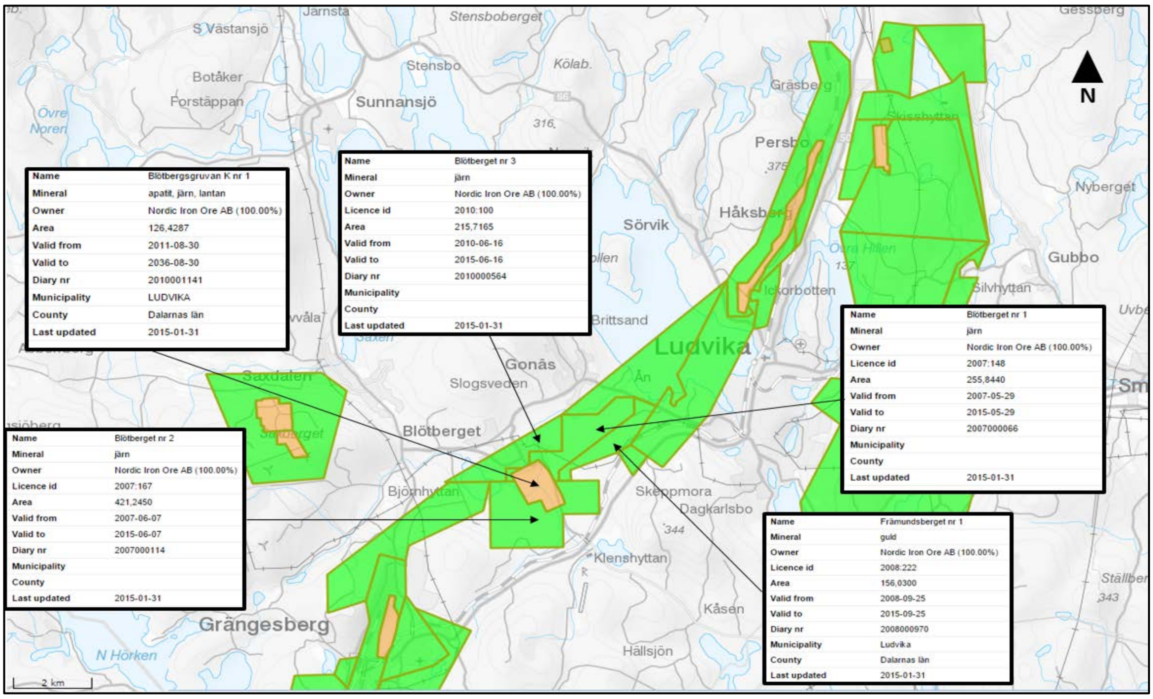


Figure 3-4 Licences Map

4 GEOLOGY AND MINERAL RESOURCE

4.1 2015 Mineral Resource Estimate

The contents of Section 4 "Geology and Mineral Resource Estimate" of this report are summarised from DMT's Mineral Resource Estimate Report ("**MRE**") dated April 2015. A copy of the complete MRE is attached as Appendix E.

4.2 Regional Geology

Regional geological maps over the area have been published by the SGU. Publications include a colour version of map sheet Ludvika AF158, 1:50 000 (1986), a more detailed map in scale 1:50 000 (2005).

The Blötberget apatite-iron oxide deposit is located in the western part of the intensely mineralised Paleoproterozoic Bergslagen Province ("**the Province**") in south central Sweden.

The Province is volumetrically dominated by several generations of intrusive rocks, which enclose inliers of metasedimentary and metavolcanic rocks. The Province has been described as being an extensional, continental back-arc, magmatic example. The metasedimentary and metavolcanic inliers are of great importance as they host an overwhelming majority of the more than 6,000 known metallic mineral deposits and prospects in the Province.

These rocks have been subjected to multiple-phase deformation and metamorphism under mainly greenschist to amphibolites facies conditions. Pre-Svecofennian rocks are not exposed but various isotopic, petro genetic, and trace element studies of supracrustal rocks imply that an older Proterozoic, possibly in part Archaean, felsic basement underlies the western part of the Province.

The deposits in the neighbouring area occur along a ~40 km long, broad zone This zone of mineralisation is the third largest iron ore deposit in Sweden by production, only outnumbered by the giant Kirunavaara and Malmberget iron ores in Norrbotten, northern Sweden.

4.3 Property (Local) Geology

The mineralised zone at Blötberget appears as a set of vertically narrow, elongated lenses dipping 50°–70° to the SE. Airborne geophysical surveys and historical drillholes indicate that mineralisation extends to a depth of at least 900 m below surface.

The host rocks to the Blötberget iron mineralisation have traditionally been classified as belonging to the "leptite formation", i.e. mainly felsic to, more rarely, intermediate, regionally metamorphosed (c. 1.90–1.87 Ga) volcanic rocks. In most parts of the Bergslagen ore province these leptites are predominantly SiO_2-rich and have mainly rhyolitic to dacitic compositions, yet, the immediate host rocks to Blötberget ores exhibit significantly more of intermediate to basic compositions. The metavolcanic rocks are locally feldspar-porphyritic, fine-grained and generally range between rhyolitic-dacitic to basaltic/andesitic in composition. A number of the observed leptites within Blötberget area, particularly in the mining concession, also exhibit crosscutting relations to various rock units and have been interpreted as sub volcanic in origin.

Alteration is evident in these host rocks, both in the form of regional-style sodic or potassic alteration and locally, as disseminated as well as discrete phyllosilicate (mainly biotite + chlorite) and amphibole-rich zones. These alteration assemblages systematically occur in and around the main ore zone.

4.4 Mineralisation & Geochemistry

The mineralisation at Blötberget is a so-called "apatite lake ore" which, besides the iron mineral magnetite and hematite, also contains the phosphorus mineral apatite, which previously caused problems in the production of iron. With the technological developments that occurred when the so-called Thomas process was invented in 1879, it became possible to also take advantage of ore that was rich in phosphorus.

The Blötberget field consists of five mineralised bodies, from west to east, these are:

Table 4-1: Mineralisation zones

Mineralised Body	Short Form Name	Mineralisation
Kalvgruvan	KAL-	apatite-rich magnetite
Flygruvan	-FLY	apatite-rich, hematite-dominated, minor magnetite
Hugget & Betstamalmen	'Betsta' or 'the Wedge'	Apatite-rich magnetite-hematite
Sandellmalmen	'Sandell'- SAND	Apatite-rich magnetite

The Hugget/Flygruvan and Kalvgruvan/Flygruvan zones had previously been mined down from near-surface to the 200 m and 240 m levels respectively. The units dip towards the southeast at between 50° and 55° in the near-surface mined-out areas, and flatten at depth to ~25°.

The area, previously known as 'the Wedge' or Betsta, was an unknown area between the two former mining concessions, Vulcanus and Blötberget. The Wedge was successfully explored during the 2014 drilling programme.

Kalvgruvan and Hugget/Flygruvan have now been shown to be continuous zones of mineralisation (Figure 4-1).

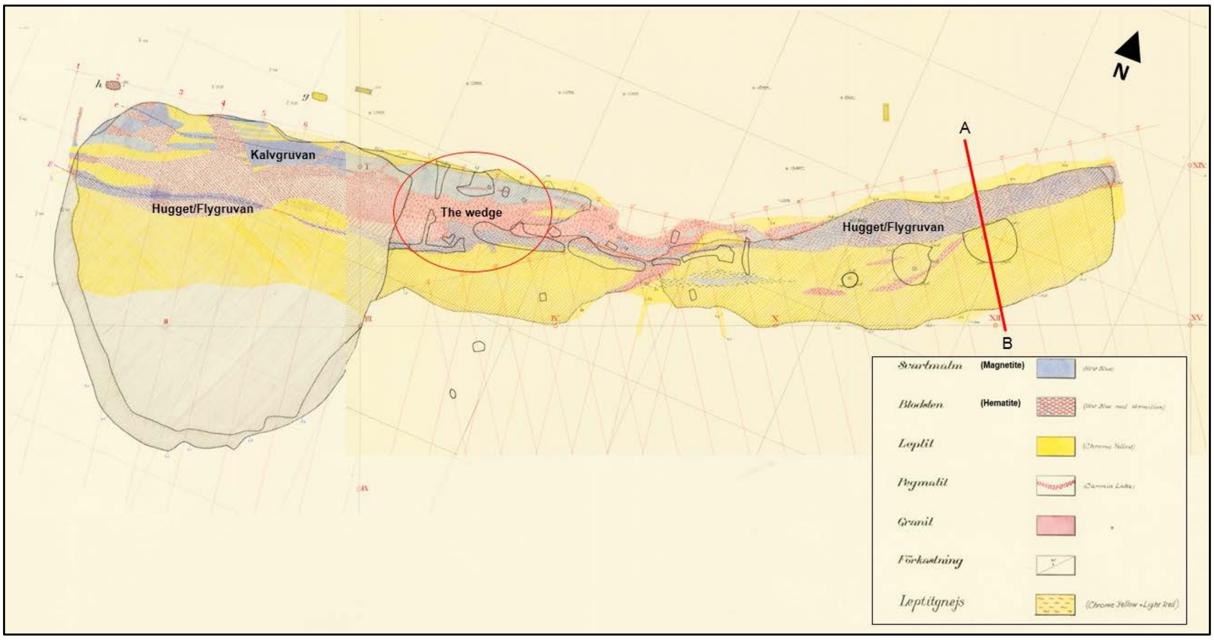


Figure 4-1 Location of mineralised zones

4.5 Deposit Type

The Blötberget deposit and its NNE continuation to Idkerberget constitute an anomaly in the Bergslagen Province. Of the >6000 deposits in the Province, registered in the SGU mineral deposits database, 5500 are iron oxide deposits. Of these, most are either banded iron formations or skarn-type deposits, except those in the spatially restricted Grängesberg-Blötberget-Idkerberget area.

The Blötberget deposit and its northern extension to Idkerberget thus represent a significant ore genesis and geological anomaly in the Province. Based on the mineralogy, deposit geometry, host rock relations and geochemical character, it is evident that the Blötberget apatite-iron oxide deposits represent Paleoproterozoic Kiruna-type deposits that have been deformed and metamorphosed to amphibolite-facies grade.

The similarities with the Kiruna deposits were acknowledged early, but hypotheses concerning the origin of these ores have varied over time, from direct magmatic, exhalative sedimentary to hydrothermal and metasomatic. Two main hypotheses on the origin of apatite-iron oxide ores have dominated the discussions during recent years, namely:

- Hydrothermal or orthomagmatic origin i.e. formed directly from a melt.
- Direct-magmatic origin, noted through several textural similarities between the Kiruna deposit and the much younger apatite-iron oxide ore at El Laco in Chile.

The magmatic model was challenged by Sillitoe & Burrows (2002), and a theory for a hydrothermal replacement process was proposed in its place. This was subsequently rejected based on the magmatic textures and relationships between apatite iron boulders and the host-rock. However, a hydrothermal origin for other apatite-iron oxide ore deposits was proposed.

A comparative study of different apatite-iron oxide ore deposits in North America was completed and suggested that these deposits, characterised with respect to age, tectonic setting, mineralogy and alteration, ought to be referred to as Iron oxide-copper-uranium-gold-REE deposits. These were later to be called Iron Oxide Copper Gold ("**IOCG**") deposits. The deposits of Kiruna-type were considered a sub-set within this IOCG concept, and a primary, shallow-level, hydrothermal origin has been suggested.

The IOCG concept marked the onset of an exploration frenzy for these deposits and subsequently more research has been conducted.

It has been shown that there is great variation between different possible IOCG occurrences, tentatively related to different ore forming processes. The trace element composition in apatite from the Tjårrojåkka deposit in Norrbotten is, for instance, very different compared to other IOCG deposits and the question remains whether Kirunavaara should be considered an IOCG type deposit at all. This statement is also valid for the Blötberget deposit, as it does not contain any significant concentrations of either gold or copper, which perhaps emphasizes its similarity with the Kirunavaara ore. Indeed, the concept of an orthomagmatic origin for the Kiruna-type deposits suggest they are "non-IOCG" (*Nilsson et al. 2013*)

Besides iron and apatite being present in these rocks, there are significant accumulations of rare earth elements ("**REEs**") and phosphorous (*Nilsson et al. 2013*).

4.6 Exploration

During the 1950s and 1960s, ground based and airborne geophysical surveys respectively were carried out over the Project area.

Since the formation of NIO, several surface sampling campaigns have taken place. The majority of these have been within the mining concession areas but some have extended to include the surrounding exploration concessions in order to allow a better understanding of the geochemical relationship between the satellite deposits and the main Blötberget mineralised zone. This work assisted with the realisation of potential sites for a bulk sample/test mining site. A handheld magnetic susceptibility (KT-10) device and a Thermo Niton x-ray fluorescence ("**XRF**") XL3 were used to ascertain iron and magnetite percentages of outcrop samples. Rock samples were then sent to ALS for chemical assay.

During 2009, Kopparberg Mineral AB carried out a more detailed magnetometry survey over a limited part of the Blötberget area on behalf of NIO (Figure 4-2).

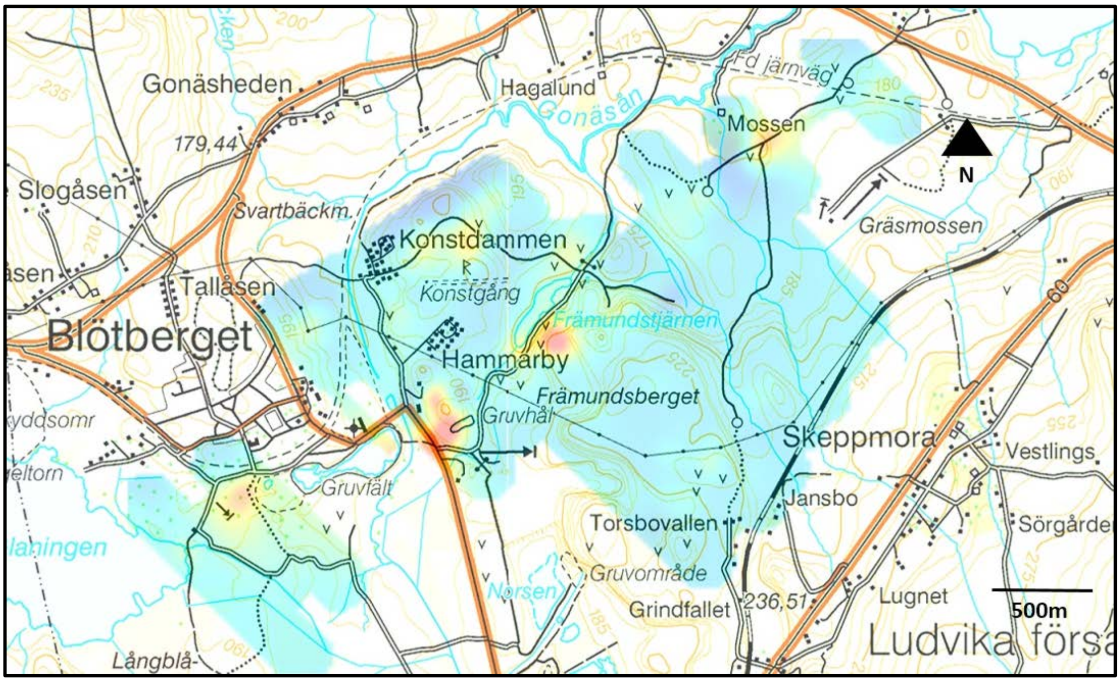


Figure 4-2 Ground magnetic anomaly map (November 2009)

4.7 2012 Drill Programme

A drilling programme was undertaken during the summer and winter of 2012 and was completed in November 2012. This 16 hole programme included twinned drilling to confirm the quality of historical drilling data, as well as infill and step-out drilling.

NIO completed 16 drillholes totalling 7,430 m of drilling. The NIO drilling in 2012 was carried out by the Swedish contractor, Drillcon Core AB, or by their Finnish subsidiary Suomen Malmi Oy ("**SMOY**") using Onram 1000 and Onram 1500 drill rigs and wireline 56 methodology, the programme recovered 39 mm diameter drill core.

One hole, BB12015-MET, was drilled for the purposes of generating material for metallurgical sampling, using HQ-size equipment to recover 63.5 mm diameter core. To date, this has been the only hole drilled using oriented core.

No drillholes were water pressure tested during this drilling campaign.

Three holes were left uncompleted as they hit highly fractured and clay altered rock which collapsed the drillhole. These holes were BB12003 (@ 400 m), BB12012B (@ 810 m) - BB12012 was a re-drill of this hole - and BB12014-MET (@ 30 m). No mineralisation was encountered in these holes.

Several deviations from the planned targets occurred during this drilling campaign, due largely to the small drill equipment and small drill diameter used.

4.8 2014 Drill Programme

The 2014 drilling programme was designed to investigate the area between Flygruvan/Kalvgruvan and Hugget (formally known as "**the Wedge**" or Betsta area) and to infill the intermediate depth extension of Hugget, (-320 m to -660 m; measured from surface depth and relating to mining blocks) in order to improve the confidence of the geological model.

At the onset of planning for this drill programme, it was deemed necessary that a larger diameter drilling method, NQ2 (50.6 mm core diameter), and larger, more powerful drilling rigs were to be used to alleviate deviation; hole losses due to fractured/clay strata and to improve core recovery.

13 drillholes, totalling 7,093 m, were drilled by the Finnish contractor Kati using a Sandvik DE 140 and Onram 1000 and NQ2 drilling methodology, recovering 50.6 mm diameter drill core and producing 75.5 mm diameter drillholes. Kati used a hexagonal reamer which helped ensure that the drillholes had minimum deviation.

One of the drillholes (BB_14-011) was drilled down-dip for geotechnical purposes.

All holes were drilled with orientation information, using either a Devico Devicore or ACT II Reflex tool to provide accurate structural information. Eight holes were subject to pump testing to provide information on the potential for water bearing fracture zones. Each hole was measured for deviation using a Devicore Deviflex gyroscope.

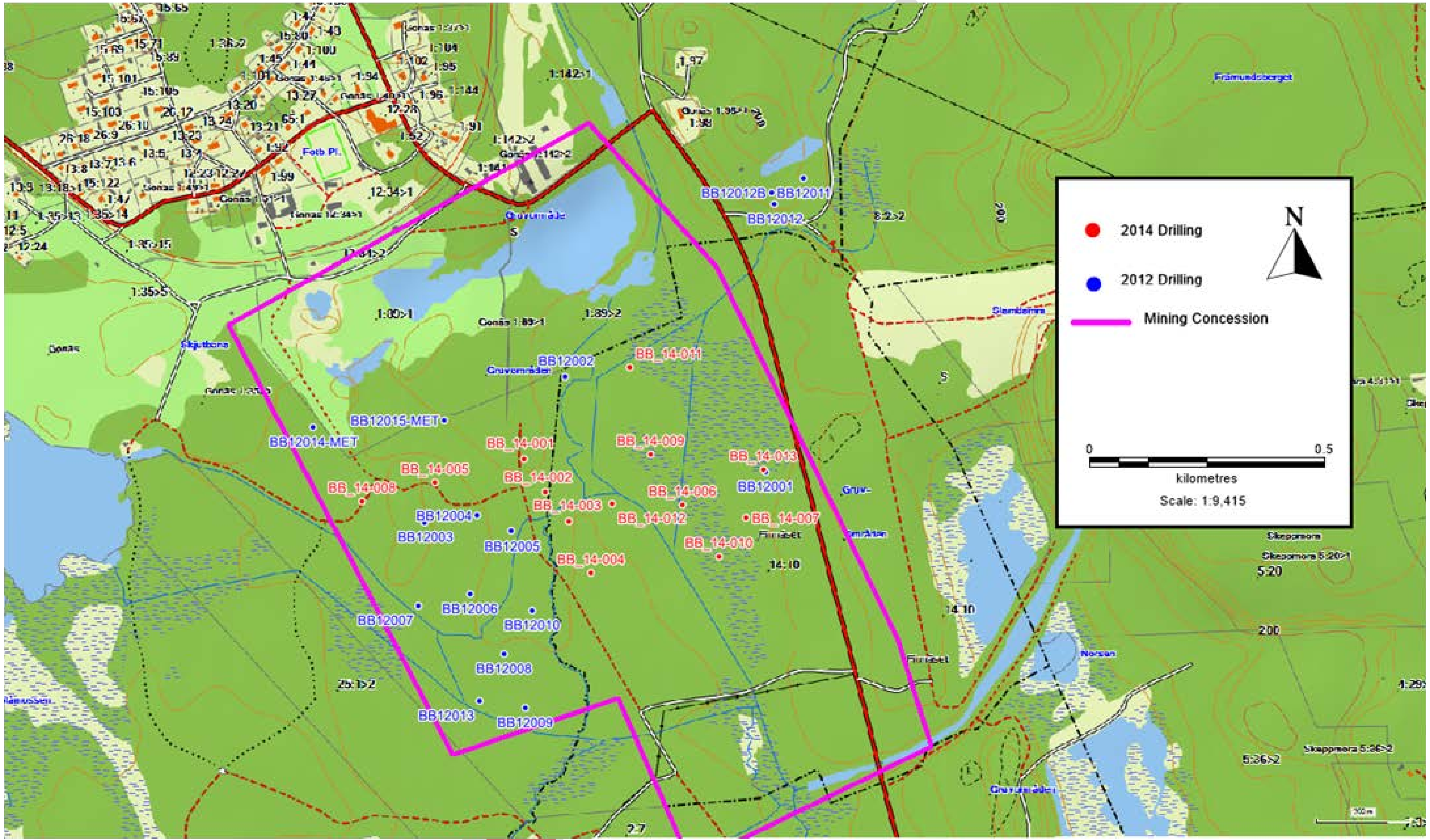


Figure 4-3 **2012 and 2014 drillhole locations**

4.9 Data Verification

All of the data from 2012 and 2014 have been acquired based on standard operating procedures ("**SOPs**"). Surface historical drillholes have been re-surveyed and a number of historical drillholes have been re-logged and re-assayed.

The locations of all historic drillholes and geological underground maps have been converted from a local mining grid to map datum SWEREF99-TM. A local height reference system has been established which is 227.95 m below the RH2000 height system. The geo-referencing and coordinate conversion work has been carried out by Tyrens, Sweden.

Borehole collar surveying was completed by Ludvika Kommun ("**LK**") after the completion of the 2012 and 2014 drilling programmes. LK surveyed drillhole collar locations, X, Y, Z, dip and azimuth using high resolution Real Time Kinetic ("**RTK**") GPS. During the 2014 programme LK also re-surveyed historic drillhole locations to confirm the translation of historic coordinates

A digital terrain model ("**DTM**") has been prepared by Tyrens on behalf of NIO. This DTM is a triangulation based on contour lines with 5 m spacing. The morphology in the license area has a relatively low topographic range from -46 to -36 m; predominantly on the -46 m (Figure 4-4).

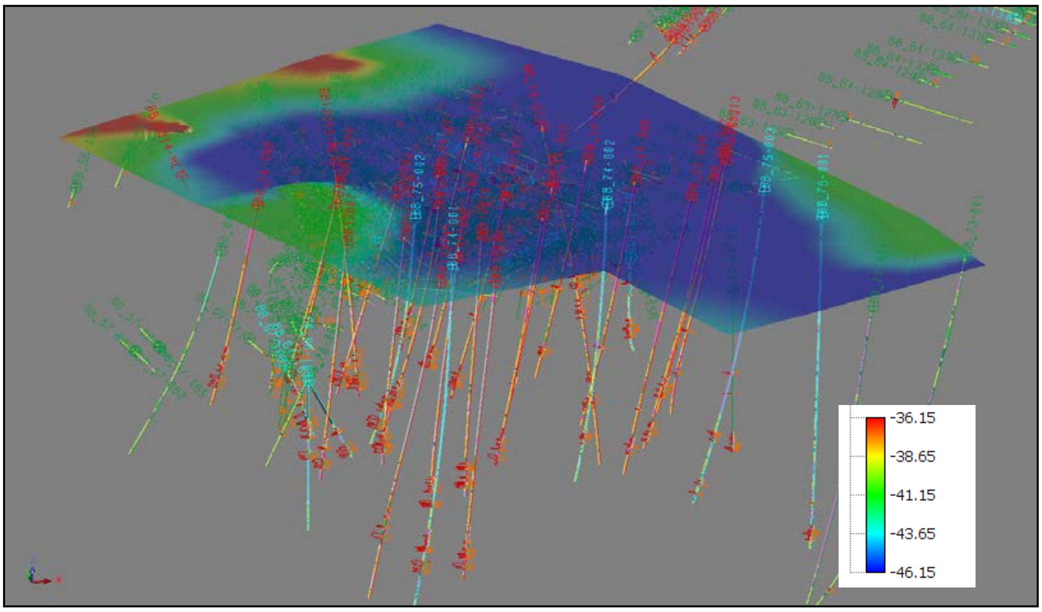


Figure 4-4 Digital terrain model (DGM) in license area

As part of the verification program, NIO has re-logged and re-assayed many of the located historical cores. In total, 45 drillholes from Blötberget were found in Malå and 15 at the former mine storage facility in Håksberg.

There has been re-logging of 31 of these cores (6036 m), 950 m of mineralisation has been re-sampled and re-assayed according to current industry practice and standards. This included mineralised core that had not been sampled historically as it fell below the (visual) historic cut-off grade of 35 %. Approximately 5-10 m of mineralised core was sampled beyond the boundaries of the historical sampled sections.

Geological maps and underground level mine plans have been used to estimate the maximum depth level of historical mining activities and the volume of mined out material. This process was undertaken by NIO technical personnel and a surface was supplied to DMT showing the depth limit of mining activities (Figure 4-5).

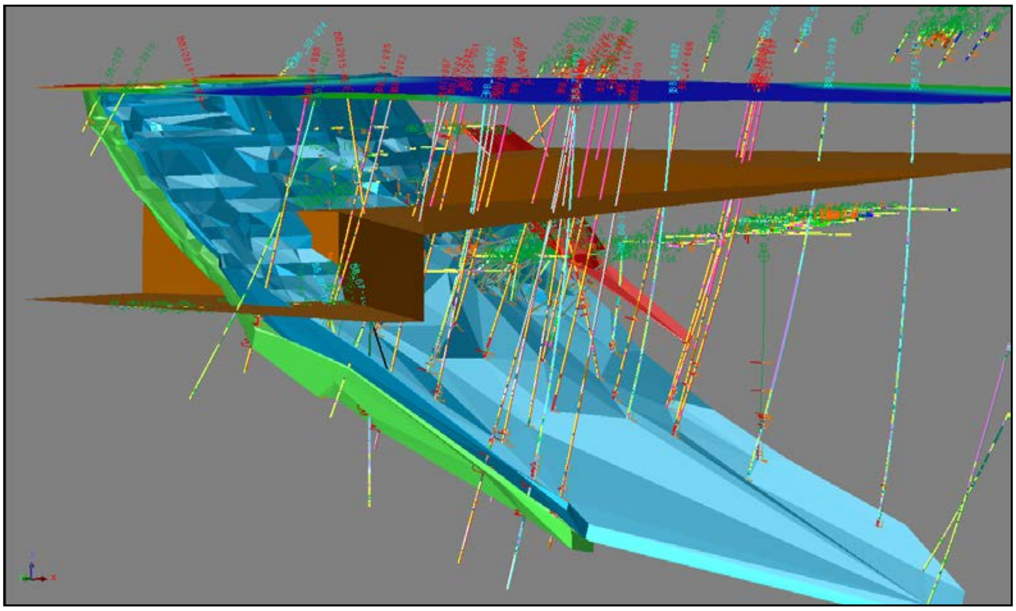


Figure 4-5: Topographical Surface (Dark Blue) and Lowest Mined Level (Brown plane)

The collar locations, dips and azimuths of the historical holes, in all but a few instances, come very close to the information given in the historical protocols. The error in location is typically less than 0.5 m. For a small number of drillholes, no collar information has been found in the records. As a result, their locations have been taken from the plans.

4.10 Database

All data from historical and recent drilling programmes have been stored in an industry standard database software to include the following information;

- Collar survey
- Down hole survey
- Geology (and abbreviation codes)
- Sampling
- Laboratory Assay data
- Digitized chemical data
- Magnetite data of SATMAGAN

- Davis tube recovery
- QA/QC sample set
- Bulk density

The database has been rigorously checked by DMT for completeness and error for each drillhole and cross checked with the core photographs. All data has been exported and implemented into an industry standard (Geovia Surpac) modelling software package.

The accuracy and precision of applied sample preparation and assaying methods has been verified by DMT and the resulting data of Fe, P and magnetite values are deemed reliable and representative to be used in resource modelling.

While the historic data of Fe are assessed as acceptable to be used in the resource model, the historic data for phosphorous could not be considered acceptable and was not used in the resource model and resource estimate.

The drilling recovery of the 2012 and 2014 drilling programmes is close to 100 %, hence, sample or assay bias caused by poor core recovery is negligible.

A slight offset in sample location or orientation will not have any influence on the resource estimate. Significant risks with the underlying data used for mineral resource estimation were not identified by DMT.

Table 4-2 Summary drillholes and data used in the Resource Estimate

Wire Frame domain	Number of Drillhole Intersections	Metres of Drillhole Intersections	Sampled metres assayed by ALS	Metres with digitised Fe results	Total metres with Fe results	Sampled metres with SATMAGAN (Mg) analysis	Metres of density data
Hugget/ Flygruvan	273	4806	769	2312	3081	769	714
Kalvgruvan	103	1579	447	571	1018	447	421
Sandell	18	121	18	73	91	18	18
Total	**394**	**6506**	**1234**	**2955**	**4190**	**1234**	**1153**

4.11 Wireframe & Block Modelling

The interpretation followed the geological concept of a laterally continuous seam-like geometry, which is flexured along the dip direction of 145^o with dips ranging from 50^o at the surface to 35^o at a depth of 800 m below surface.

Three main iron rich zones lie as narrow mineralised envelopes , from the upper (hanging wall) zone to lower (footwall) zone these three zones are referred to as Sandell (SAND), Hugget-Flygruvan (HUGFLY) and Kalvgruvan (KALV). Each of these zones required wireframe modelling for grade estimation purposes.

Individual hanging and footwall triangulated surfaces were produced based on drillholes intersecting the mineralised zones and a set of underground maps of historic mining areas. The surfaces were extended with half the distance to nearest drillhole to define the lateral limit of mineralisation. A fully enclosed 3D triangulated solid of each zone was achieved by cross-linking the boundary strings. A 15 % Fe cut-off grade has been applied to model the contacts of the mineralised zones. Some intersections did not show a composite grade above 15 % Fe. Consequently, these low grade intersections were also included in the mineralised zone in order to best represent the lateral continuity of the 'seam-like lava flow' model.

The three solid models are representing the most optimistic envelopes which also consider waste material and low grade ore.

The shape and orientation of the mineralisation and the geological and mineralogical data (including dip and dip direction) suggest that there is no additional tectonic influence on the distribution of mineralisation..

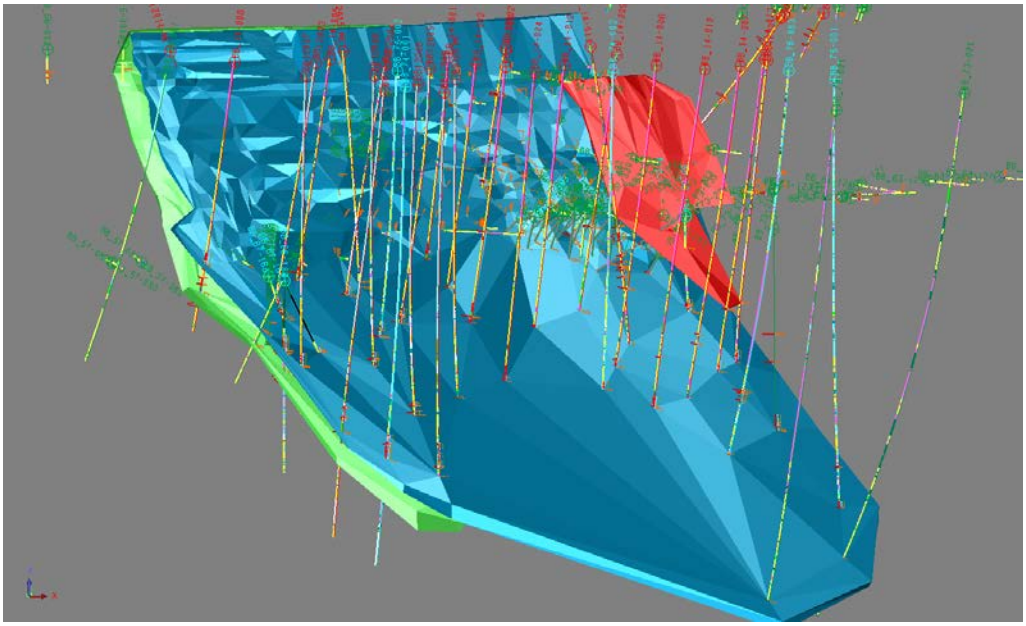


Figure 4-6 Wireframe 3D view North :SAND (red), HUGFLY (blue) KALV (green

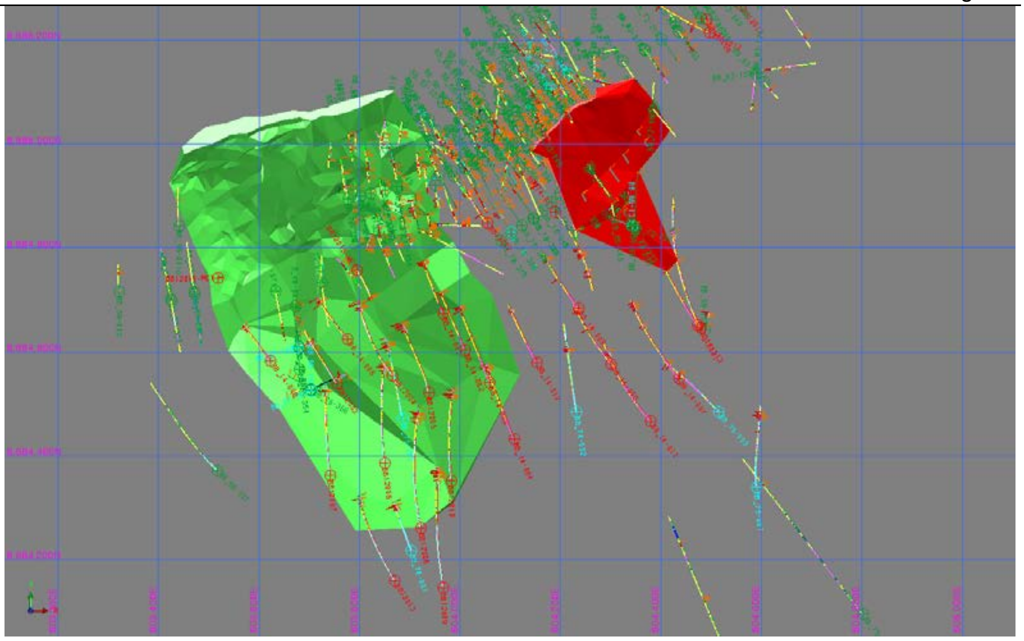


Figure 4-7 Wireframe 3D of the mineralised envelopes: (HUGFLY removed).

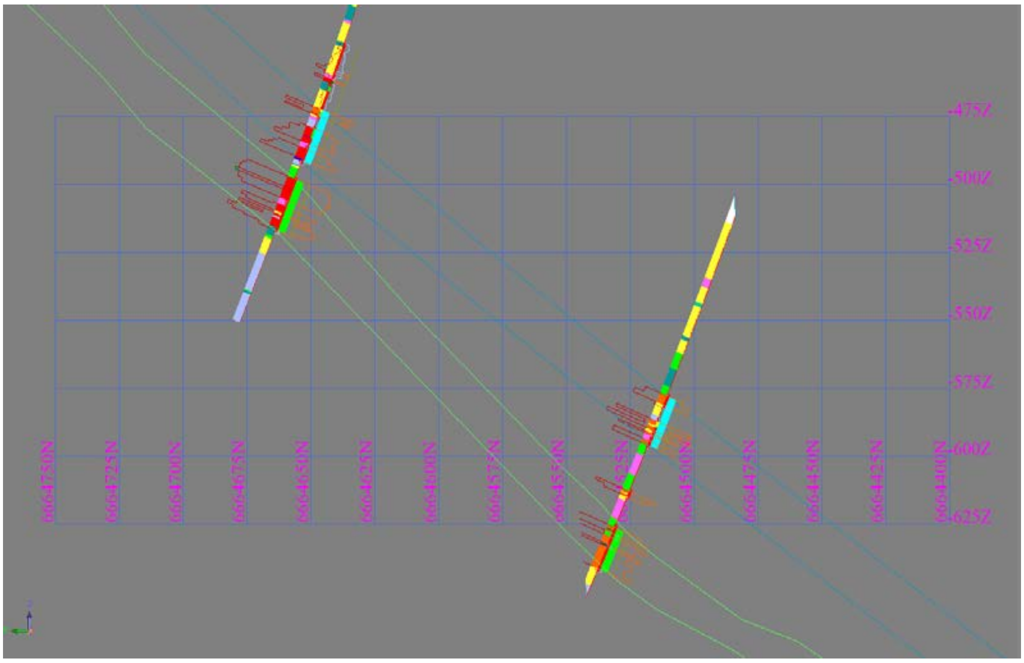


Figure 4-8 HUGFLY example cross section looking north east

The block model uses regular block size of 10 m length, 5 m width and 10 m height. These block dimensions are considered to be the most appropriate, considering the morphology of the mineralisation and the proposed mining method. The block model is rotated to the same strike as the mineralisation, N55°E.

The dimension of the block model is maximum 1540 m along strike direction and 1940 m perpendicular to strike (down- dip); adapted to the drilled area and license area. The total height is ranging from 200 m to -1200 m. The total number of blocks is 427,525. No sub-blocking is applied.

Attributes have been added to the block model and filled with content. Detailed explanations about the attributes are given in the MRE in Appendix E

For all blocks lying within or intersected by wireframe HUGFLY, SAND and KALV a partial percentage attribute has been calculated. This attribute adds a volume portion ranging from 0.000 to 1.000 to each block lying within or intersected by these wireframes, which has been used for volume correction in the resource estimate.

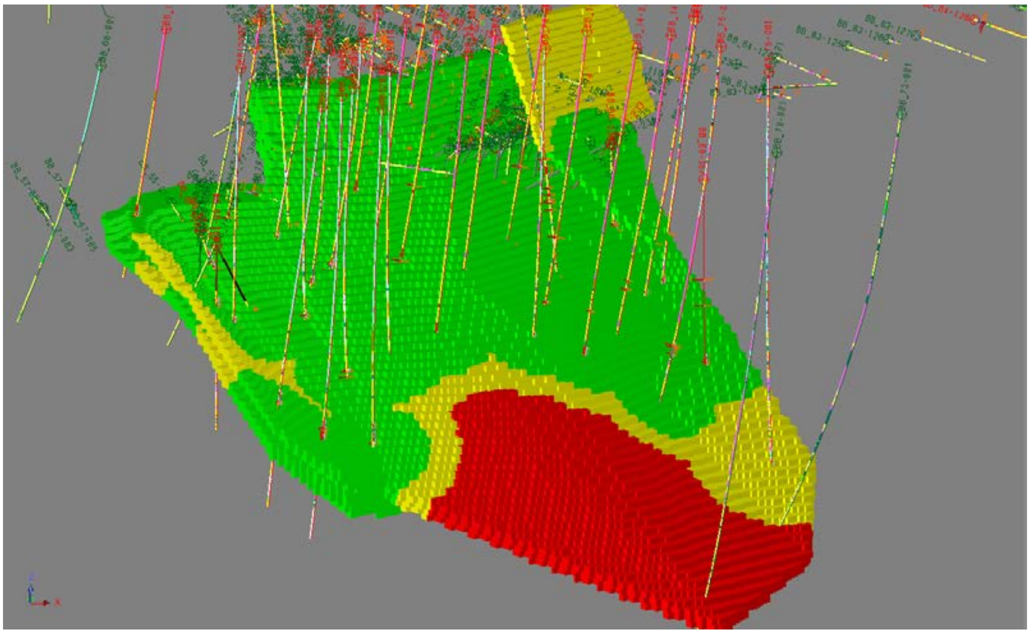


Figure 4-9: Blötberget 2015 Resource Block Model

4.11.1 Cut Off Grade Assumptions

4.11.2 Preliminary Economic & Mining Assumptions

Initially, DMT did not apply any economic cut-off grades or mining criteria to the global resource estimate which was generated within the confines of the wireframes.

DMT used the wireframes and a set of technical and economic input assumptions, summarised in Table 4-3, to create a preliminary block model, using Geovia Surpac software, in order to constrain the estimated Mineral Resources and to demonstrate reasonable prospects for eventual economic extraction.

Commodity price assumptions are based on typical China import sales, over the past five years, of iron ore fines (62 % Fe) (Figure 4-10).

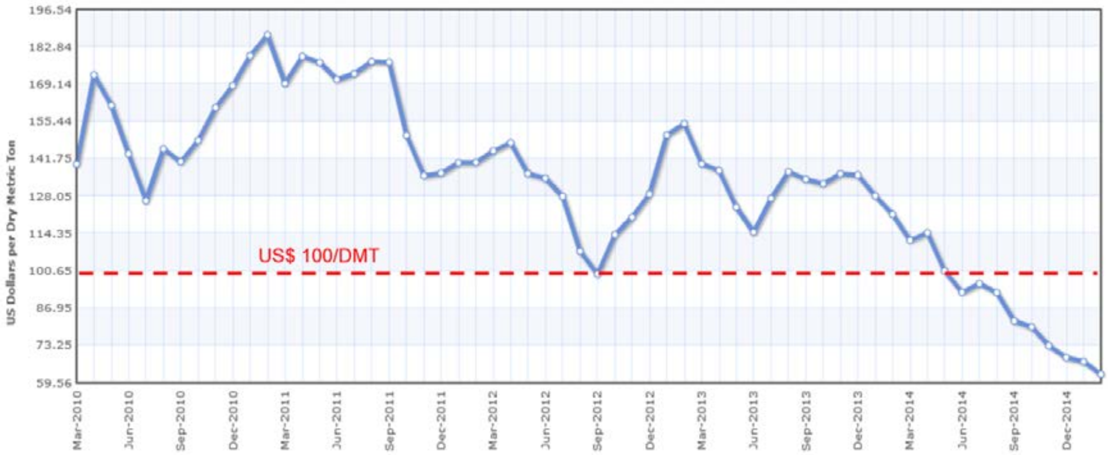


Figure 4-10 Five year historic China import spot price (FR Tianjin port)

Source: Index Mundi

Using preliminary economic input parameters for the proposed mining method, processing and selling-related costs, the cut-off grade has been estimated by applying the below formula:

$$\frac{\text{Costs for mining plus processing [US\$/t ore]}}{\left(\frac{\text{Price of concentrate [US\$/t conc]}}{\text{Fe grade of concentrate [Fe[\%]/t conc]}}\right)} * \frac{1 + \text{Dilution [fraction]}}{\text{Processing recovery [fraction]}} = Fe\ Cutoff\ grade$$

Table 4-3 Block Model preliminary economic input parameters

Parameter	Cost/Value t/Revenue
Costs for mining plus processing [US$/t ore]	20
Price of concentrate [US$/t conc.]	100
Fe grade of concentrate [Fe[%]/t conc.]	63
Dilution [fraction]	0.1
Processing recovery [fraction]	0.9

The wireframe shells provide a constraint for the reported block model resources based on the JORC definition of Mineral Resources having “reasonable prospects for economic extraction”.

When the basic economic input parameters are applied to the wireframes, an indicative COG of 23 % Fe is arrived at.

A tonnage / grade sensitivity study has been carried out by DMT at COGs ranging from 0 % to 60 % Total Fe. A COG of 25 % has been highlighted as this is the nearest rounded up percentage COG.

Table 4-4 Resource Sensitivity Grade / Tonnage Cut Off Grade

Fe Cut-off [%]	Volume [Mm³]	Tonnage [Mt]	Density [t/m³]	Fe [%]	Magnetite [%]	Hematite [%]	magnetite prop [%]	hem prop [%]	Phosphorous [%]
0	17.2	62.2	3.6	36.2	31.6	19.0	0.62	0.38	0.46
5	17.2	62.2	3.6	36.2	31.6	19.0	0.62	0.38	0.46
10	16.9	61.3	3.6	36.6	31.9	19.3	0.62	0.38	0.46
15	16.2	59.1	3.7	37.5	32.7	19.8	0.62	0.38	0.47
20	14.6	54.3	3.7	39.2	34.2	20.7	0.62	0.38	0.48
25	12.5	47.8	3.8	41.5	36.1	22.0	0.62	0.38	0.51
30	10.5	41.1	3.9	43.8	38.4	22.9	0.63	0.37	0.54
35	8.4	33.8	4.0	46.2	41.3	23.3	0.64	0.36	0.57
40	6.4	26.2	4.1	48.7	44.9	23.1	0.66	0.34	0.60
45	4.1	17.7	4.3	51.6	48.6	23.5	0.67	0.33	0.60
50	2.1	9.2	4.5	55.5	51.8	25.7	0.67	0.33	0.64
55	1.0	4.6	4.6	58.7	54.5	27.6	0.66	0.34	0.70
60	0.3	1.6	4.8	61.5	58.6	27.3	0.68	0.32	0.66

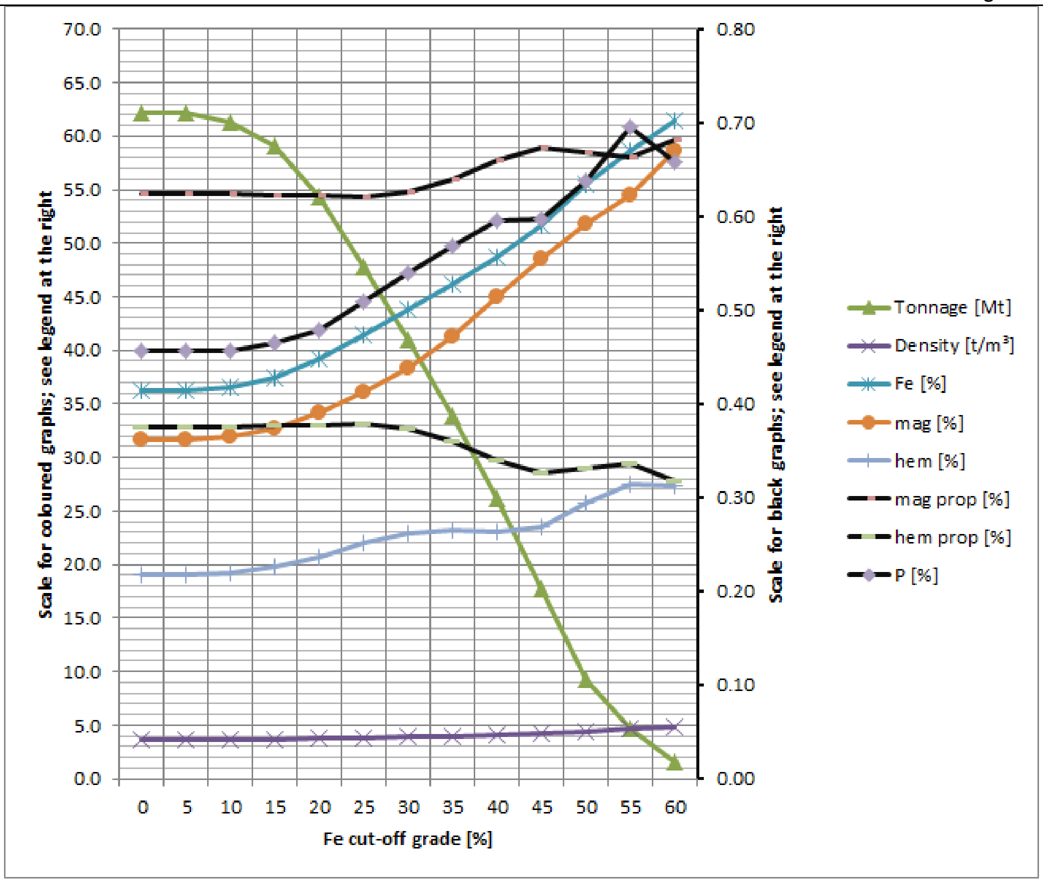


Figure 4-11: Resource Grade /Density /Tonnage Curve

4.12 Mineral Resource Classification

The definitions for resource categories used in this report are consistent with the JORC Code 2012.

Under the JORC classification system, a Mineral Resource is defined as:

…"a concentration or occurrence of natural, solid, inorganic or fossilised organic material in or on the Earth's crust in such form and quantity and of such grade or quality that it has reasonable prospects for economic extraction.

"The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."

Resources are classified into Measured, Indicated and Inferred categories based upon geological knowledge and confidence (Figure 4-12).

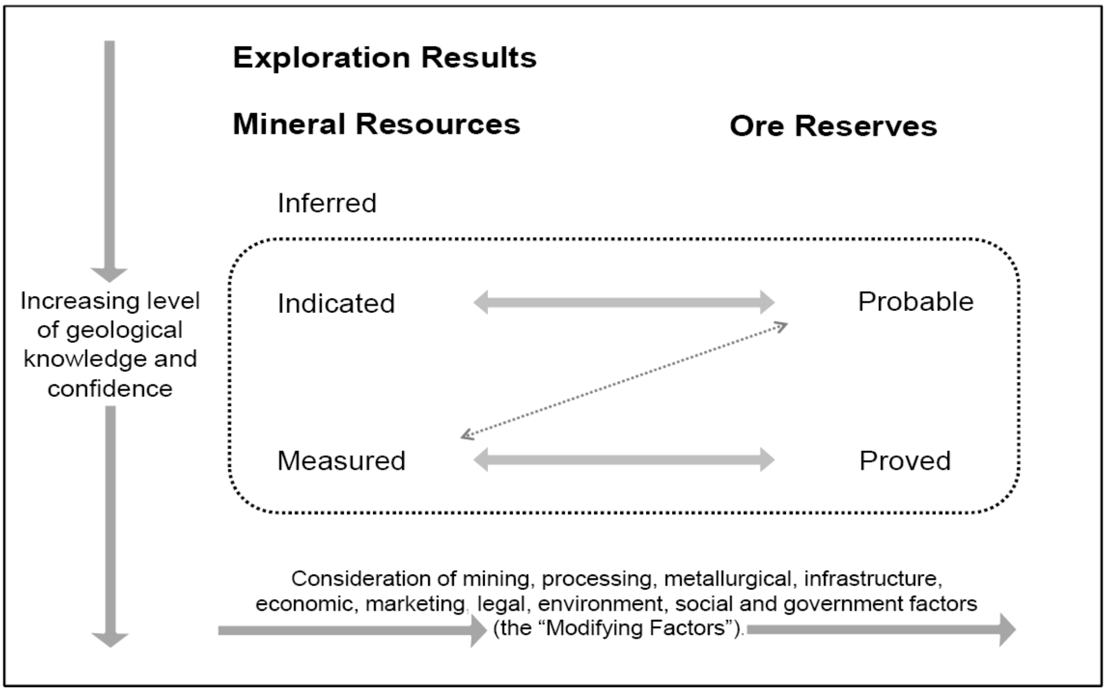


Figure 4-12 Relationship between Exploration Results, Mineral Resources & Ore Reserves

Resource classification within mineralisation envelopes is generally based on drillhole spacing, grade continuity, and overall geological continuity. The distance to the nearest composite and the number of drillholes are also considered in the classification.

In classifying the resource estimate, the following key factors have been considered:

- Confidence in data quantity and specifically sample spacing of Fe and magnetite data;
- Confidence in the geological interpretation and continuity (geological complexity); and
- Confidence in mineralisation / grade continuity (complexity of spatial grade distribution).

Considering the above, the following criteria have been applied for classification into the various mineral resource categories for this estimate:

4.12.1 Measured Resources

- All blocks whose distance to the nearest magnetite sample is less than 2/3 of the variogram range (i.e. <100 m) - excluding distally located drillhole BB_75-001.
- All blocks which are surrounded by measured blocks; and
- All blocks near the historic underground workings.

4.12.2 Indicated Resources

- All blocks whose distance to the nearest magnetite sample is equal or above 90 m and less than the full variogram range of 140 m - including distally located drillhole BB_75-001.

4.12.3 Inferred Resources

- All blocks which are not defined as Measured or Indicated but are included in the interpreted wireframes.

4.13 Mineral Resource Estimate

DMT has prepared a Mineral Resource estimate for the Blötberget Project with a drillhole database cut-off date of 1st January, 2015.

The Mineral Resource estimate has an effective date of 30th January, 2015 and has an issue date of 10th April 2015.

DMT applied basic mining and economic parameters including commodity price and wireframe assumptions, to estimate a cut-off grade for resource estimation of 25 % Fe (Total).

The total Measured and Indicated Resource estimated for the Blötberget Project, at a preliminary economic cut-off Grade of 25 % Fe, is 47.8 Mt at a grade of 41.5 % Fe (Total) and 0.5 % P.

1) HUGFLY contains an estimated 26.5 Mt of Measured and Indicated Resources at a grade of 38.5% Fe (Total) and 0.5% P.
2) KALV contains an estimated 19.8 Mt of Measured and Indicated Resources at a grade of 45.6 % Fe (Total) and 0.5 % P.
3) SAND contains an estimated 1.4 Mt of Measured and Indicated Resources at a grade of 38.5% Fe (Total) and 0.5% P.

Of the total estimated contained Fe, the magnetite proportion is estimated at 62% and the hematite at 38%.

DMT has reported all the material of magnetite-rich ore of KALV and hem-rich ore of HUGFLY and SAND contained within the resource block model limited by the licence area and excluding the material mined out by historical mining activities.

DMT considers all of the material reported as Measured and Indicated Resources to have ‘reasonable prospect of economic extraction’ given appropriate economic and technical considerations.

Table 4-5 and Table 4-6 summarise the Mineral Resource estimate for the Blötberget Project as of 30th January, 2015. . The Block Model has been constrained using basic economic and mining parameters and the Mineral Resources are estimated at a COG of 25%.

Table 4-5 : Measured and Indicated Resources for the Blötberget Iron Project - January 2015

Fe Cut-off % Fe	Resource Category	Volume Mm^3	Tonnage Mt	Density t/m^3	Fe %	Magnetite %	Hematite %	Magnetite proportion %	Hematite proportion %	Phos. %
25	Measured	11.1	42.5	3.8	41.9	36.8	21.9	0.63	0.37	0.51
	Indicated	1.4	5.3	3.7	38.2	30.5	23.2	0.57	0.43	0.5
	Measured + Indicated	**12.5**	**47.8**	**3.8**	**41.5**	**36.1**	**22.0**	**0.62**	**0.38**	**0.51**
	Inferred	1.5	5.4	3.5	33.5	23.5	23.5	0.50	0.50	0.52

Notes:

1. *JORC 2012 definitions were followed for estimating Mineral Resources;*
2. *Mineral Resources are estimated at a cut-off grade of 25 % Fe;*
3. *Mineral Resources are estimated using a five year historical average price of US$ 100 per tonne (Source: IndexMundi); and*
4. *Figures may not total due to rounding errors.*

Table 4-6 : Deposit specific Resources for the Blötberget Iron Project - January 2015

Fe Cut-off % Fe	Deposit	Volume Mm^3	Tonnage Mt	Density t/m^3	Fe %	Magnetite %	Hematite %	Magnetite proportion %	Hematite proportion %	Phos. %
25	HUGFLY	7.2	26.5	3.7	38.5	20.4	34.0	0.37	0.63	0.5
	KALV	5.0	19.8	4.0	45.6	58.0	5.2	0.92	0.08	0.54
	SAND	0.4	1.4	3.8	40.6	25.4	31.8	0.44	0.56	0.25
	TOTAL	**12.5**	**47.8**	**3.8**	**41.5**	**36.1**	**22.0**	**0.62**	**0.38**	**0.51**

Notes:

1. *JORC 2012 definitions were followed for estimating Mineral Resources;*
2. *Mineral Resources are estimated at a cut-off grade of 25 % Fe;*
3. *Mineral Resources are estimated using a five year historical average price of US$ 100 per tonne (Source: IndexMundi); and*

4. *Figures may not total due to rounding errors.*

4.14 Comparison with Historical Resource Estimates

Previous JORC compliant MREs have been undertaken by GeoVista, Sweden. The initial GeoVista MRE estimate was carried out in 2011, with subsequent updates in 2012, and 2014.

The GeoVista January 2014 MRE established a COG based on similar preliminary economic assumptions to those made in this DMT resource estimate. However, the parameters were not applied to the 2014 resource statement.

Table 4-7 therefore compares the resource estimates of 2014 and 2015 without a COG applied (i.e. 0 % Fe Total).

The January 2014 MRE did not allow for the loss of volume and tonnage created by the former mined out areas in the upper levels of the mine. Subsequently, and due to improved information, the 2015 estimate has excluded some areas that have been mined out or are believed to be "un-mineable" areas.

Since the 2014 MRE, additional resources have been added in the ‘Wedge - Betsta’ areas a result of the 2014 drilling programme. In 2014, no Fe grade (i.e. 0 %) was applied to the internal waste or country rock material, whereas the 2015 estimate has applied an 8 % average Fe grade to these parts of the block model.

The additional material recently explored from the Wedge area connects the two bodies HUG and FLY, to create HUGFLY. The upgrading of the January 2014 Inferred Resource for HUG and FLY (low Fe grade) to Measured and Indicated Resources leads to slightly lower Fe grades for Measured and Indicated Resources of HUGFLY but the overall tonnage has approximately doubled.

The DMT 2015 MRE shows a slightly higher tonnage for SAND but a lower Fe grade.

Estimates for density, magnetite, hematite and Mag-hem ratios were not available in the 2014 MRE.

Table 4-7 Comparison of GeoVista 2014 estimate and DMT estimate March 2015

Resource Category	Resource Estimate 2014			Resource Estimate 2015		
	Tonnage Mt	Fe %	P %	Tonnage Mt	Fe %	P %
Measured	10.7	34.3	0.3	53.7	37.0	0.46
Indicated	27.4	44.8	0.5	8.5	31.1	0.43
Measured + Indicated	**38.1**	**41.9**	**0.4**	**62.2**	**36.2**	**0.46**
Inferred	21.7	33	0.4	10.5	27.3	0.48

Resource Category	Resource Estimate 2014			Resource Estimate 2015		
	Tonnage Mt	Fe %	P %	Tonnage Mt	Fe %	P %
Total	**59.8**	**38.6**	**0.4**	**72.7**	**34.9**	**0.46**

Note: These resources are **global** estimates with no cut-off parameters applied and are for comparison purposes only

5 MINE AND SURFACE GEOTECHNICS

5.1 Introduction

In the Blötberget mines, sublevel caving was the chosen main mining method used up to the closure of mining in the mid 1960's for the Kalgruvan /Flygruvan (KAL/FLY) orebodies, and in 1979 for the Hugget/Betsta (HUG/ BET) section of the mine. Open stoping method was also used in the Sandell orebody.

The historically chosen mining method may influence the planning of new mining operations at levels below those already extracted.

This section is a compilation of geotechnical work based mainly on the Rock Quality Designation (RQD) method of logging drill cores performed in 2012; on a review of old mine maps and, on previously conducted geological studies. Although some of the data acquired from the 2014 drilling campaign has been used for this assessment, not all of the new data was available at the time of the assessment. This should be incorporated into the database for any future geotechnical work.

5.1.1 Geotechnical compilation

The geotechnical work is a compilation of base data such as

- RQD
- Point load tests
- Water pressure tests
- Structural mapping from old mine maps. Fracture orientations are according to the right hand rule if not stated otherwise.

5.1.1.1 RQD

A compilation of RQD values were made in a number of boreholes (Figure 5-1), to determine variation in fracturing within and around the mineralised zones . Drillholes are drilled from surface through the hanging wall (HW) and through the mineralised zone to footwall rocks (FW) (HW→ore→FW). (Figure 5-2).

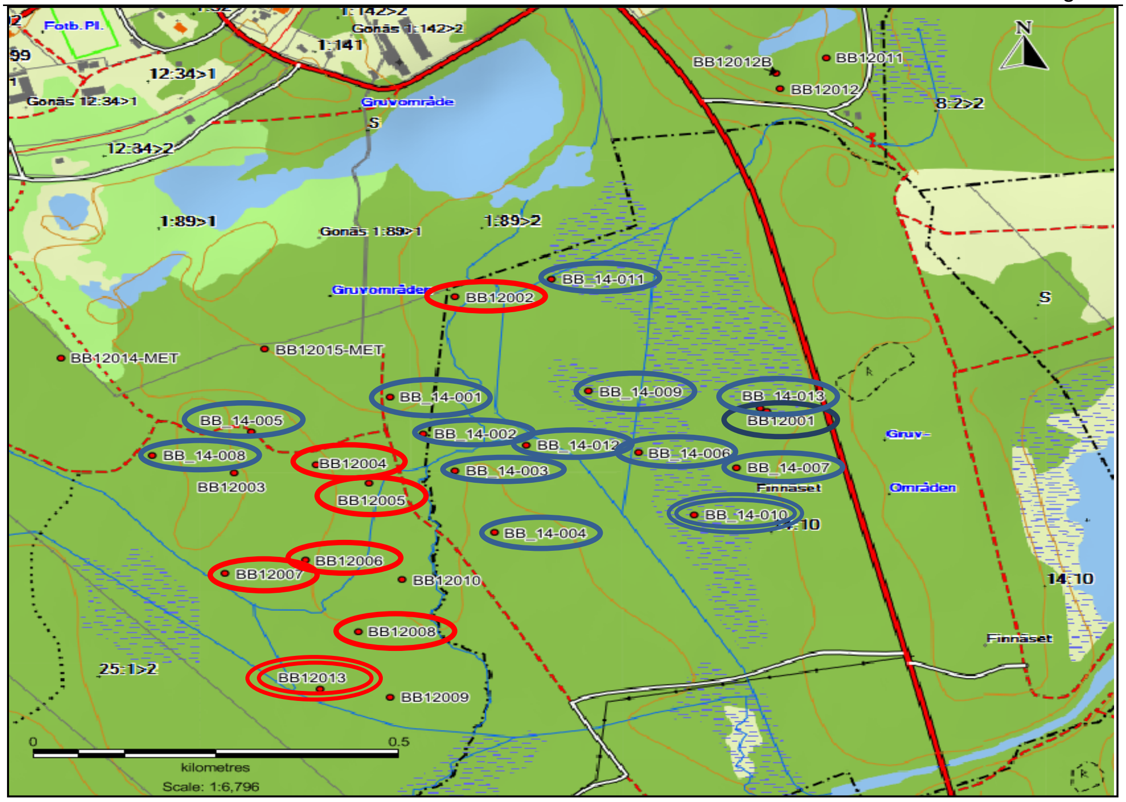


Figure 5-1 Locations of main boreholes used in analyses.

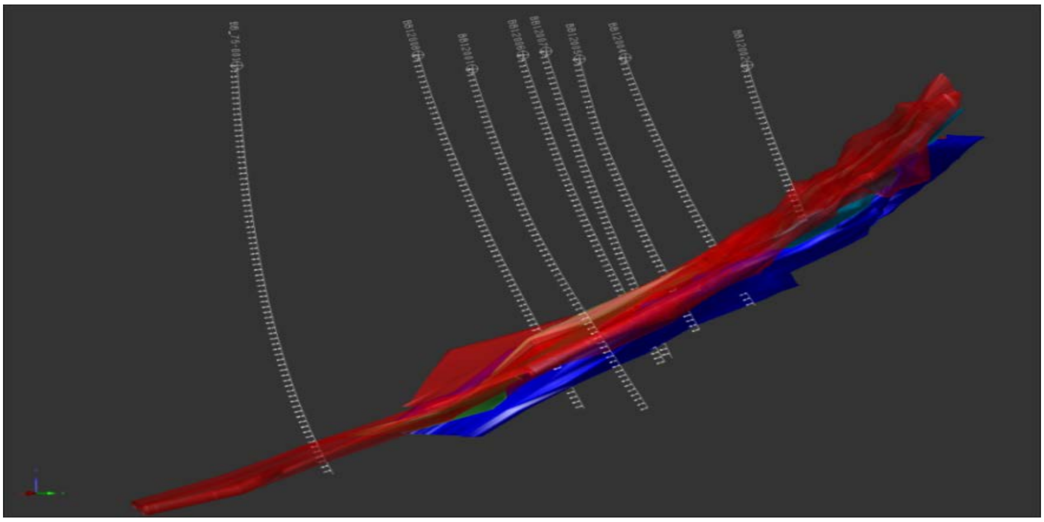


Figure 5-2 Cross section of orebody and location of drill cores (2012)

The compilation of RQD is based on the average value of approximately 50 m of the footwall and 50 m of the hanging wall. The average value for the mineralised seams is made for rock containing more than 20% of iron (>20% Fe Total). No significant variations are noted in the between the rock mass quality in different parts of the country rock or mineralised sections.

Mapping of the drill cores from the 2014 drill programme indicates similar rock qualities measurements as previous drill core studies. The rock is generally of a high quality producing good core recovery with high RQD values being measured In general, the cores show a relatively low to moderate joint frequency. Refer to Appendix F.

5.1.1.2 Rock material strength testing

The rock material strength is analysed through point load tests on drill cores BB12011, BB12012 and BB12013. The tests are documented as Point Load Massive, Point Load Parallel, or Point Load Perpendicular. All values are adjusted to the 50 mm diameter equivalent standard size (Is_{50}).

Testing indicates point load test results for different rock types which show a variation in strength between 2 and 14 MPa, which corresponds to 40–280 MPa as Uniaxial Compressive Strength (UCS) (Figure 5-3).

No separate UCS tests were carried out on the drill core material coming from the 2014 campaign. However, UCS tests were performed on the core acquired during the 2012 campaign by Petroteam.

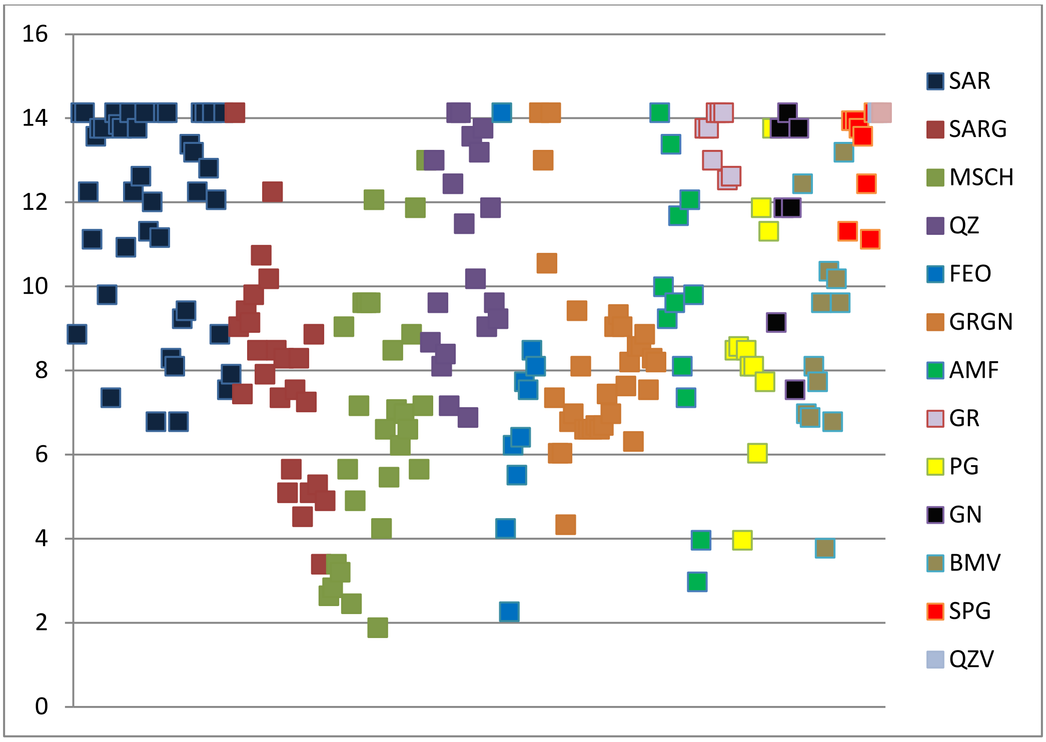


Figure 5-3 Point load test results for the different rock types

The point load tests show no significant variation between different rock types in the host rock. However, the rock types SARG and MSCH (refer to Appendix F) seem to show a greater amount of test results with lower strength of 2-6 MPa.

The tested iron mineralization showed a significantly lower strength than the host rock with an average of about 6 MPa (USC=120 MPa). Please note that only 8 tests were carried out on mineralised specimens.

5.1.1.3 Hydrogeological tests

A number of water pressure tests have been performed in some of the 2014 programme drillholes. The water consumption has been measured between a packer and the bottom of the hole at several depths. The length of the measured sections varies between approximately 20 and 100 m. The measurements are usually performed with a series of pressures 0.5, 1 and 0.5 MPa, each measurement during 5 or 10 minutes.

5.1.1.4 Lugeon values and hydraulic conductivity

Generally the water losses are relatively small in sections that compile the mineralised sections and the adjacent country rock. The larger water losses are concentrated in the hanging wall above the mineralised areas, between 5 and 300 m depth. However, the calculated water conductivity (Lugeon) values, the water loss is still comparatively very low, even in sections with higher water losses due to increased fracturing and related core losses.

The calculated Lugeon values are generally very low, which may, in part, be due to the long length of open borehole tested in these packers tests. The Lugeon values would probably be locally higher if the measurements had been carried over shorter intervals using double packers to separate the identified conductive zones. DMT recommends that this should be carried as part of the next stage of hydrogeological investigation.

In sections with the higher water losses due to fractured and crushed rock, the hydraulic conductivity is in the range of $1x10^{-7}$ to $3x10^{-7}$ m/s. This can be considered as a low to medium permeability rock mass even in the fracture zones. The “normal” rock mass, close to the orebody, shows an even lower hydraulic conductivity which is in the range of $1*10^{-9}$ to $2*10^{-8}$ m/s. This rock mass has a lower fracture frequency, often with RQD >80 %.

5.2 Information on Historic Mining

5.2.1 Structures from old mine maps

On historical mine maps (dating back to the 1800’s), major structures and faults were marked. A compilation of the structures on these maps have been done. The strikes of the structures are measured directly on the mine maps, based on the coordinate systems given on the maps. In total 213 fractures and 103 faults were found on the maps.

These observations are compiled from the uppermost levels down to elevation 540 m. An example of an area with structures marked on the map is given in Figure 5-4.

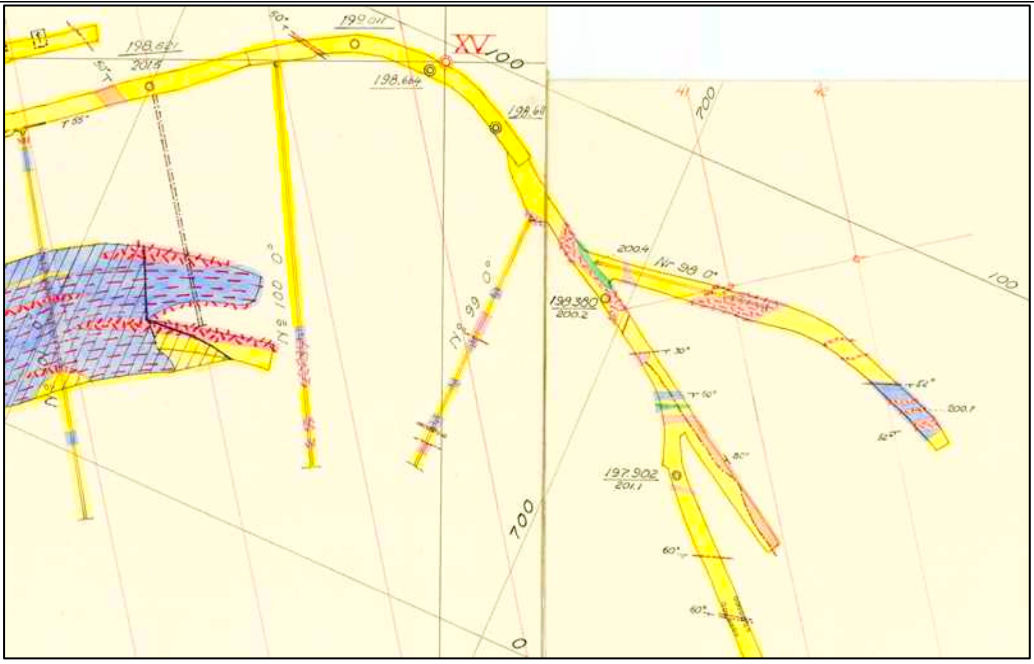


Figure 5-4 Part of mine map showing structures in drifts and boreholes

On the maps the structures are marked separately as joints or faults. Most of the structures are marked on drifts or on the side wall, and only a few are found in the mineralised zones. Figure 5-5 shows one fault, supported with timber, marked in the drift (red circle). The support indicates that this fault is possibly a wider zone than other faults without support (blue circle) in the same area.

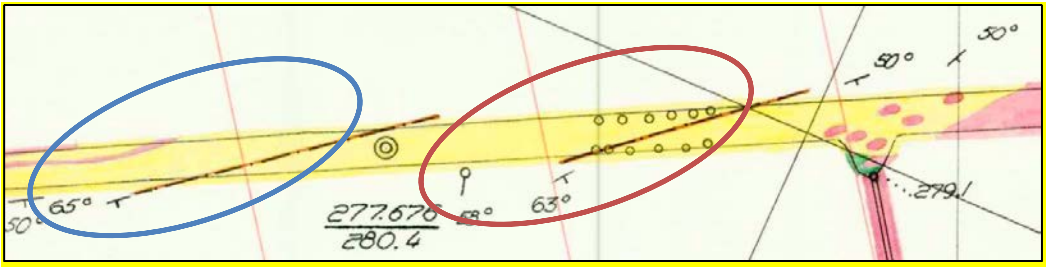


Figure 5-5 Example of faults, tunnel support and general markings

The structures were compiled and divided into joints and faults independent of elevation, The structures show a uniform distribution throughout the mine independent of where they are located, both in horizontal and longitudinal level. The joints are mainly occurring in direction N050-080E. The main directions of the faults are N070E and N110-N160E. No faults can be traced on several sublevels. Refer to Appendix F.

The dip and strike of the structures are consistent with the results from core mapping conducted in 2012.

The drill cores made in 2014 show similar fracture orientations as the old mine maps and drill cores from 2012.

5.3 Subsidence Implications

5.3.1 Mathews stability graph method

The Mathews stability method is an empirical tool for predicting open-stope stability. The method is based on a stability graph relating two main factors; the shape factor (S) and the stability number (N), see Figure 5-6. The principal concept of the stability graph is that the size of an excavation surface can be related to the competence of the rock mass to indicate stability or instability of the open stope. The stability graph can present numerous excavation surfaces with a specified range of stabilities.

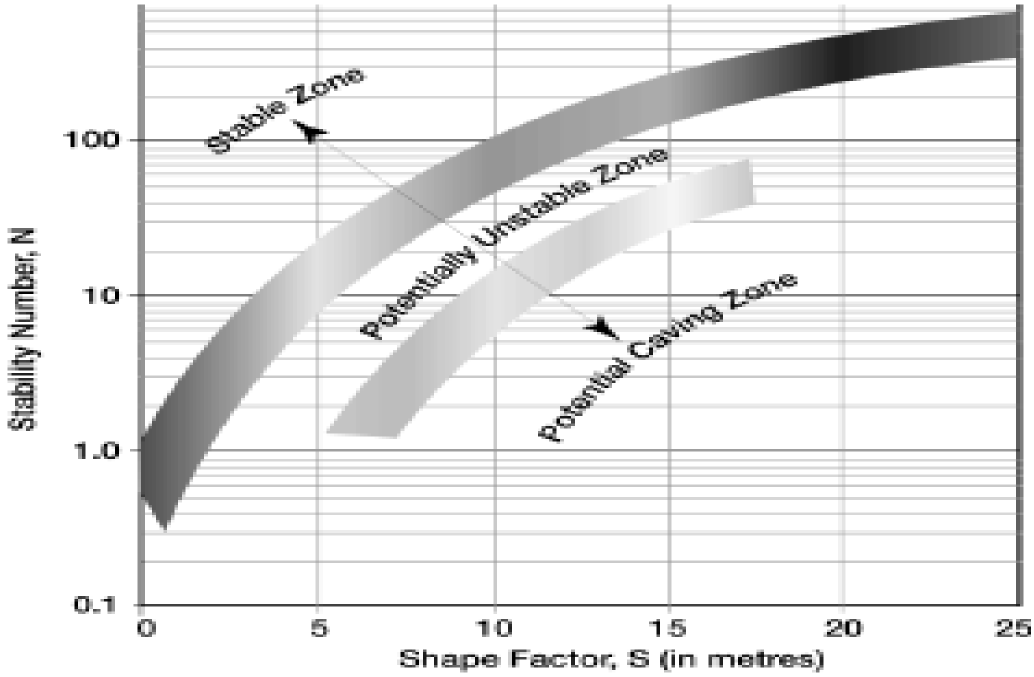


Figure 5-6 Three stability zones of the original Mathews stability graph

5.3.2 Applied stability analysis for the Blötberget mine

The Mathews stability chart has been used to evaluate the proposed mining method. The analysis is made with consideration of structural data, possible mining stopes and size of sublevel slices.

In 2011, Vattenfall published an investigation of the stability situation in the Blötberget mines based on the Mathews stability method. For the most recent stability estimation some parameters were given the same values as in 2011.

Factors J_n (joint number), J_r (joint roughness) and J_a (joint alteration) are the same in this evaluation as in 2011. Factor A, regarding rock stresses, has been estimated from measurements done in the 1950's. These measurements show high horizontal stresses. Factor B, regarding joint orientation, is based on estimations made in 2011. For this study, factor B is assumed to be the same for all walls of the stope (hanging wall, footwall and side walls).

The remaining parameters have been estimated in this study. RQD was set to an average value at level 420 meters. Factor C is based on the inclination of the orebody in relation to the ground surface for the hanging wall and footwall, which is 42–44°. For the side walls, the inclination is set to 90°.

In order to analyse the stability of Blötberget, several scenarios were studied. For each scenario, the side walls are assumed to be identical. The entire ore area was divided into several stopes, or spaces, of different sizes since the ore width varies, (Figure 5-7). Hence, the ore width is the width of the mining space.

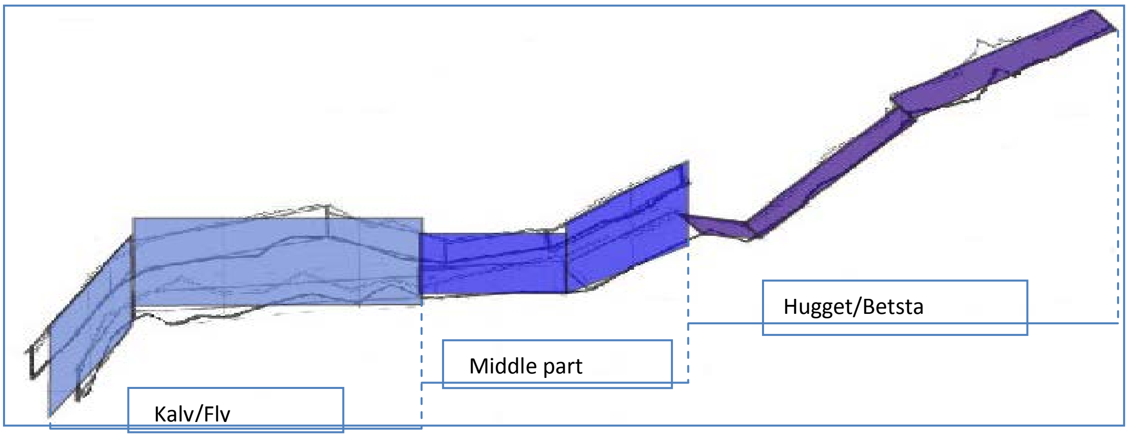


Figure 5-7 The different mining spaces (each rectangle is one space)

In the first scenario, each space in the figure is analysed individually. The height varies between 20 and 26 meters, which is the height of the mining fan.

In the second scenario, the stope is one sublevel fan of length 10 meters, which is a reasonable length to be able to take out rock. The height is 20-26 meters, which is equivalent to one fan height when mining. This means that the rock above the fan is intact.

In the third scenario, the stope is one sublevel fan of length 10 meters and height 420 meters. This scenario demonstrates the stability when the rock above is not intact and possible fallout of rock can go all the way to the ground surface, which is 420 meters above. These parameters are shown in the report included in Appendix F.

5.3.3 Results of Mathews stability method

- Scenario 1 - regarding the larger mining spaces, indicate that all spaces are stable and rock failure is unlikely.
- Scenario 2 - with an opening length of 10 m, show that all stopes are stable and rock failure is unlikely.
- Scenario 3 - with an opening length of 10 m and height 420 m are shown in Figure 5-8. Most values are stable. The exceptions are the host rock at the extremities of the KAL/FLY orebodies and the middle part (K/F P1 Side, K/F P2 Side, Mid P1 Side, Mid P2 Side).

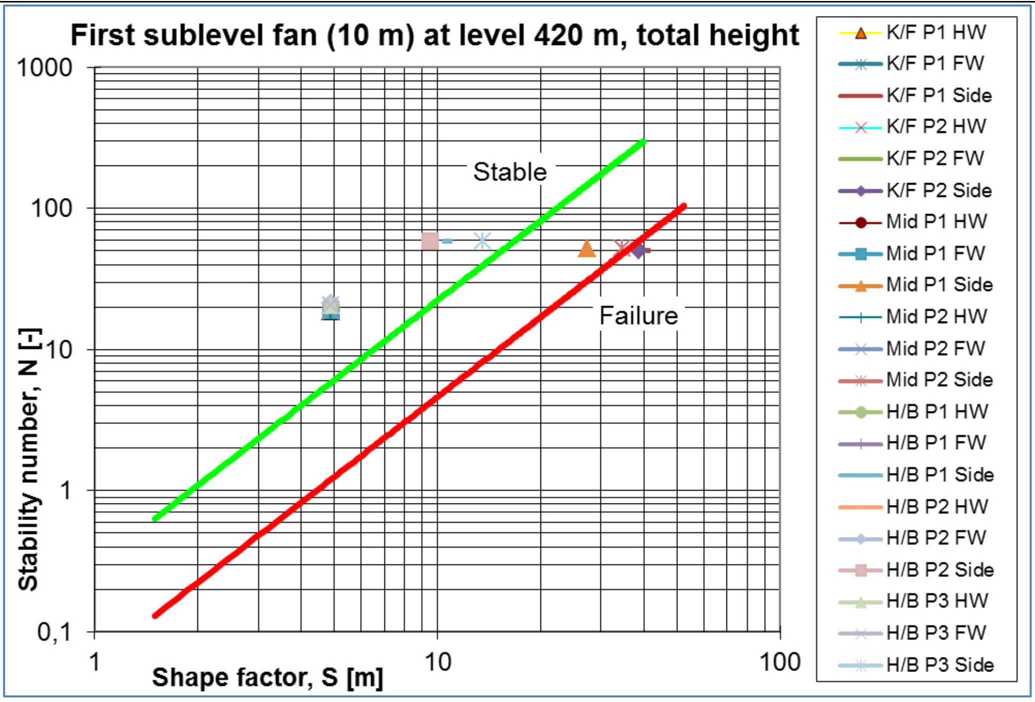


Figure 5-8 Relationship between stability and shape

For the rock cores examined in 2012 and 2014, the boreholes were drilled in the same direction (hanging wall → ore → footwall), leaving some of the joint sets undetected. However, the old mine maps indicate that three or more joint sets are probable, resulting in a higher joint set number, J_n, of 12. Using this value, the rock is more jointed. Thus, the stability graph is changed and more of the rock is within the unstable part of the graph, (Figure 5-9).

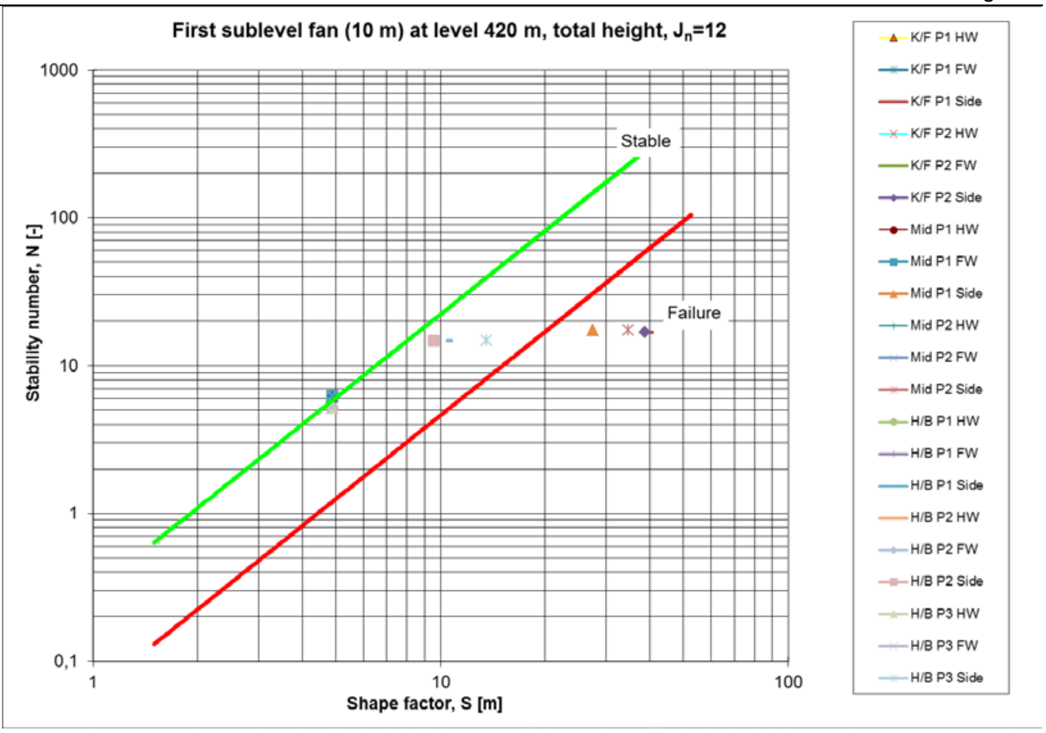


Figure 5-9 Mathews stability graph for a sublevel fan of 10m length, 420 m height and a joint set number of 12

5.3.4 Sublevel caving stability

In sublevel caving (SLC), the side rock breaks and fills the voids created by mining of the ore in a controlled manner. To create a successful SLC, a large enough open space must be created to enable the rock to fall out. Depending on the rock quality, the caving may be induced on the longitudinal side rock contact after just one fan is blasted, or after some 10–20 m of mining operation, (Figure 5-10). As the opening increases in size, instability is more likely. If the excavation interferes with old caving areas, the probability of a failure is likely to be high.

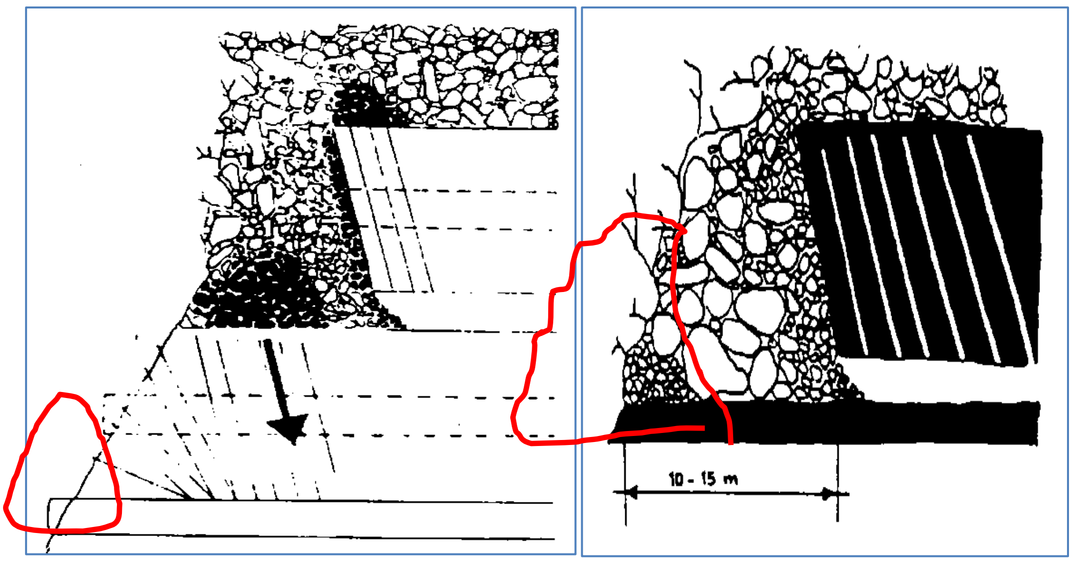


Figure 5-10 Longitudinal figure of the caving

The orebodies in Blötberget are showing variations in geometry. For areas with more narrow ore width, mining will be conducted through one drift; meanwhile in areas with a wider orebody, several drifts are required, (Figure 5-11).

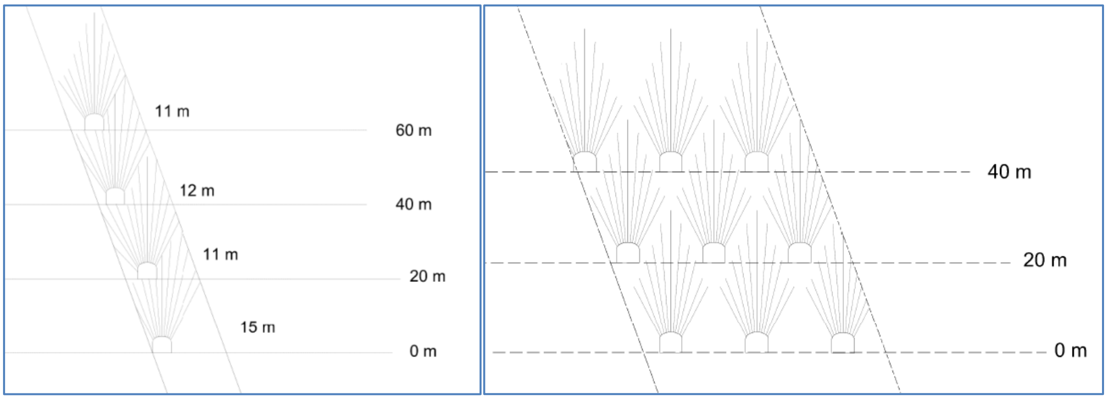


Figure 5-11 Cross section of one drift in a more narrow orebody (left) and several drifts in a wider orebody (right)

For this report, a general picture of the stability is given. A stability assessment of the adjacent sublevel is not studied, but this may have to be assessed to determine the mining sequence. For such an analysis, more data of the ore quality are required.

5.3.5 SLC stopes

In order to show the influence on the mined SLC stopes from the fracturing, the geometry of the joints in relation to the geometry of the stope has been analysed. The analysis consists of determining what wedges will form in one stope. In this analysis, only structure data from old mine maps is used, and since this dip and strike of the structures are consistent with the results from core mapping in 2012 and 2014, no update of the figures and analyses has been made. However, this should be updated when these drill cores are entirely mapped in order to get a more detailed overview of the area.

The structures were divided into three sets, are shown in Table 5-1.

Table 5-1 Average values for each joint set

Set #	Dip	Strike
1	48	66
2	43	134
3	87	134

The stope was given a width of 10 meters and height 20 meters. This is the geometry of one sublevel. The inclination of the ore is approximately 43° on level 420 meters.

The ore in Blötberget has two main directions; N090E and N035E, (Figure 5-12). Therefore, two analyses have been conducted in 'Unwedge' (computer software that calculates the wedges that will be formed), one for each ore direction as well as comparison of wedges on the hanging wall/foot wall and the host rock at the extremities of the orebodies.

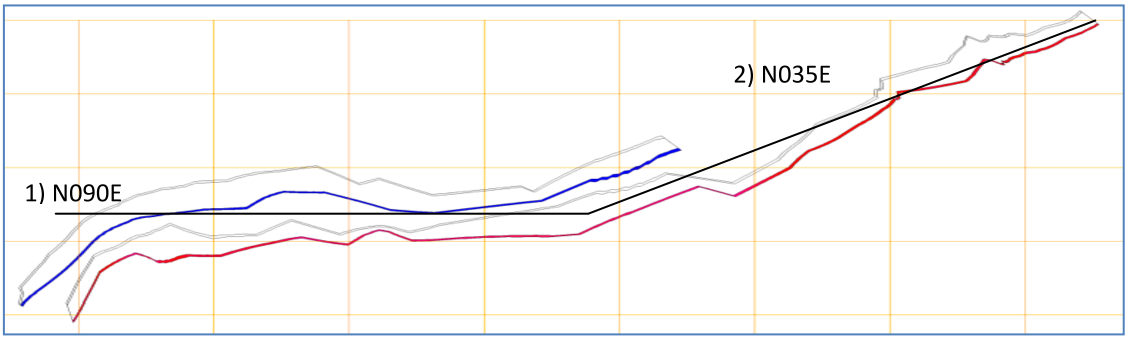


Figure 5-12 The two ore directions

The wedges that are formed on the hanging wall and footwall are rather small. However, larger wedges are formed on and the host rock at the extremities of the orebodies, which may cause problems in the mined area with large blocks.

5.3.6 Sublevel geometry

The size of the wedges that can form depend on the geometry of the stope. Analyses are done in Unwedge.

5.3.6.1 Comparing heights

On the old mine maps, each sublevel was around 5 meters apart. In order to see what wedges will form depending on the height of the stope, an analysis has been done of three alternative heights; 5, 10 and 20 meters.

The resulting wedges for the two mineralised directions are compared .(Figure 5-13).

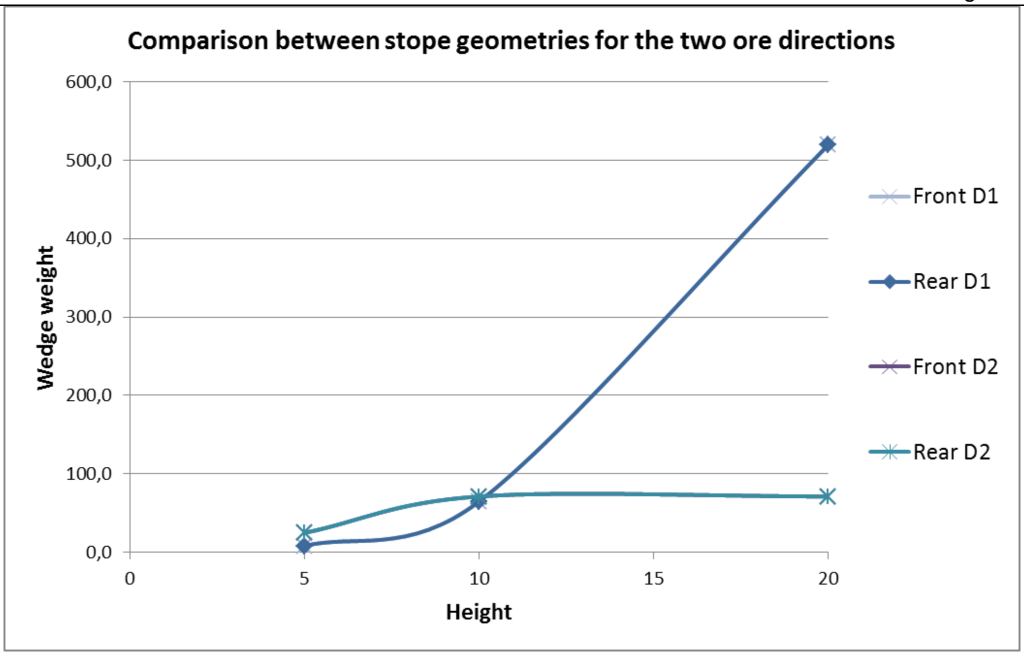


Figure 5-13 Wedges formed for the three alternative sublevel heights

5.3.6.2 Comparing widths

Four different widths are compared (Figure 5-13). The height is 20 meters for all cases.

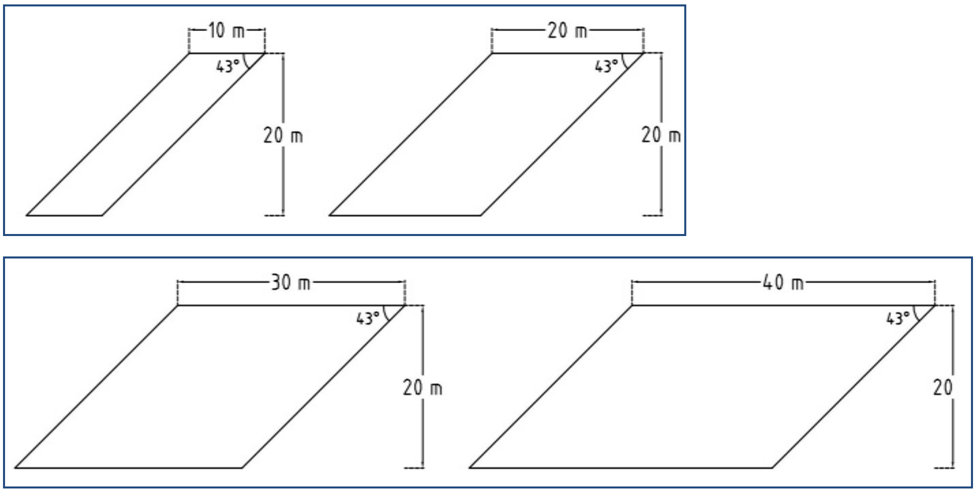


Figure 5-14 The different widths that are compared

A comparison between the four cases for the two ore directions is shown in Figure 5-15.

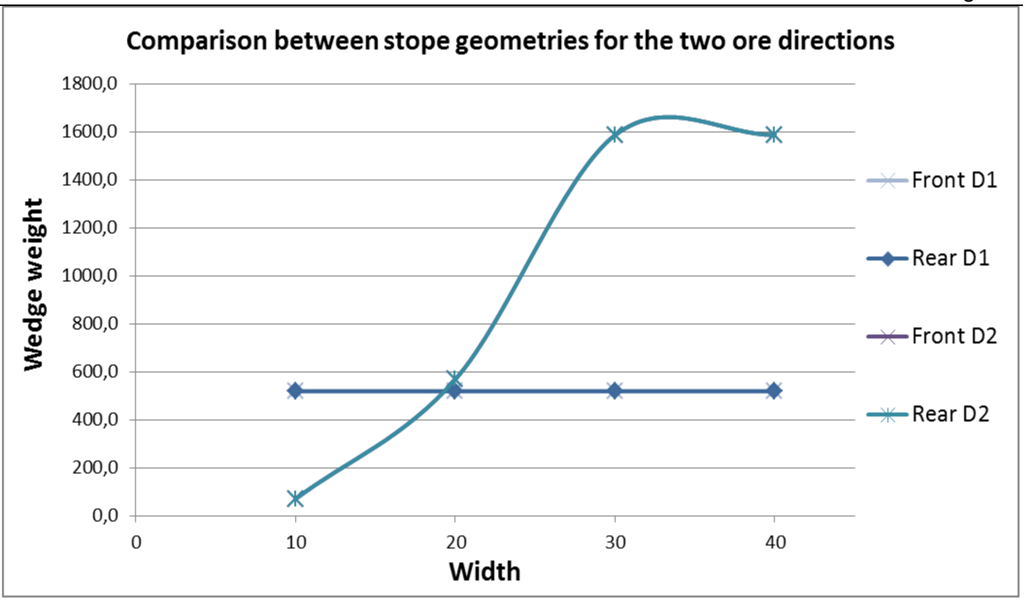


Figure 5-15: Comparison between widths for the two ore directions

5.3.6.3 Ramp design

The ramp should be designed to follow the main ore direction, 45° off the mineralisation direction or vertical to the ore direction. Therefore, these three options are analysed with regards to joints and the blocks that can be formed. Direction 1 (N090E) has been set as the main direction of the mineralisation. Thus, the ramp system will be designed with a main direction parallel to the main mineralisation (N090E), 45° off the strike (N135E) or perpendicular to the strike direction (N180E).

In order to decide which direction is most optimal, the three options are analysed in Unwedge. Figure 5-16 shows a comparison between the different wedges that are formed. The locations of the possible wedges are displayed to the right (block 5 is not formed in any of these cases and is thus not included in the Figure).

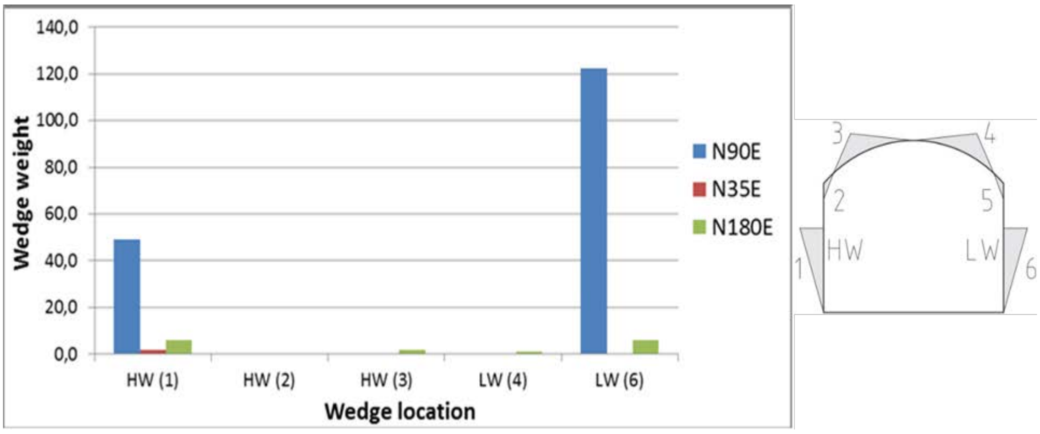


Figure 5-16: Diagram of the different wedges. Wedge locations are shown to the right

Designing the ramp in the same direction as the ore will create the largest blocks. Choosing one of the other options would be more optimal regarding block size.

5.3.7 Blend of waste rock and ore

When mining, some of the sidewall rock will break and fall out together with the mineralised material,. this will result in dilution.. This dilution blend was analysed in Unwedge for a stope of size H20 x W10 x L4 meters. The result is shown in Table 5-2.

Table 5-2 Blend of waste rock and ore when mining

	Volume [m^3]	Unit weight [t/m^3]	Weight [t]
Ore	800	5.2	4160
Waste rock			
Dir N090E	2.7	2.6	7
Dir N135E	19.6	2.6	51
Blend		Percentage	
Dir N090E		99.7%	
Dir N135E		97.6%	

The amount of waste rock dilution is considered small for a stope of this size.

5.4 Stress Redistribution due to Mining

5.4.1 Introduction

Mining operations at Blötberget mine will alter the stress conditions of the rock mass surrounding the mineralised bodies. The stress re-distributions might influence the flow paths between lake Glaningen, located 500m to the west of the orebodies and the mine. If the flow paths widen or new ones formed, water from the lake might drain into the mine, increasing the pumping requirements of the mine and have a negative environmental impact in the area.

An evaluation of the stress re-distribution and rock deformation, the influence area of this stress re-distribution, and the possible effects on the rock mass and the flow paths between lake Glaningen and the mine was carried out using the three dimensional software FLAC3D (ITASCA).

Ore and lake geometries were obtained from NIO. The depth of the lake was estimated to be approximately 30 m.

5.4.2 Geotechnical setting and input data

5.4.2.1 Discontinuity sets

The discontinuity characteristics of the rock mass in Blötberget are presented in (Anläggningsprojektet Ludvika gruvor, Blötberget, Per-Erik Söder, Ramböll, 2014) Appendix F. Three sets were found with the orientations presented in Table 5-3.

Table 5-3 Joint sets derived from old mine maps (Söder, 2014)

Set #	Dip	Strike

1	48	66
2	43	134
3	87	134

5.4.3 In Situ stress

The in-situ stress in the area is not well known. The values used in this study are in the report “Bilaga 4-3 Rasvinkel Blötberget, Vattenfall 2011.

- v= 0.027z MPa
- h= 2 + 0.025z MPa
- H= 5 + 0.040z MPa

The orientation of the major principal stress is set perpendicular to the orebody.

5.4.4 Numerical model

In order to minimise boundary effects, the numerical model was created as a block with sides 10 km long and 3 km deep. The ground surface is modelled according to the 3D map of the area. The size of the block elements close to the orebody is approximately 10 m and at the model boundaries 100 m.

An elastic analysis has been performed for this work and the Young’s modulus in the model is set to 40 GPa as an average for the rock mass.

The orebody was simulated as a void. Three different mining depths have been modelled 250m, 350m and 450m, as well as excavation of the whole resource to approximately 900 m depth (Figure 5-17).

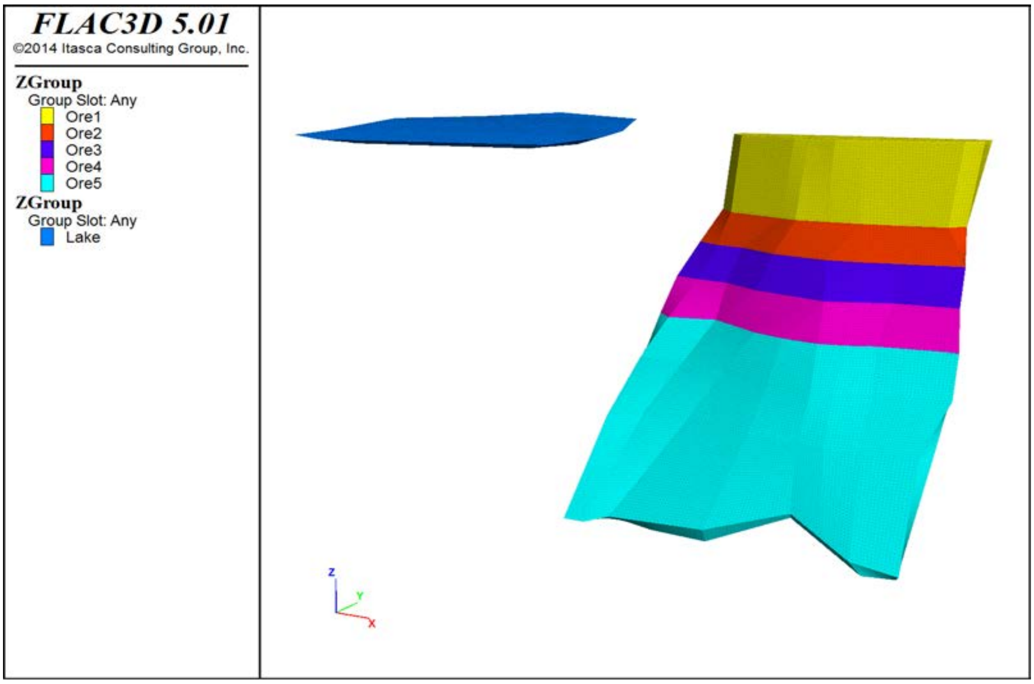


Figure 5-17 Overview of the numerical model with the lake and the orebody

5.4.5 Interpretation of numerical results

The focus of the results is on the minor principal stresses and displacements. The major principal stress is always compressive as a consequence tightens fractures perpendicular to its orientation. The minor principal stress induce tensile stresses in the rock mass dilating fractures and creating new flow paths in the rock mass between the lake and the mine.

The results indicate that the rock mass is subjected to tensile stresses closer and closer to the lake as the mining progresses. From the depth of 450m and below tensile stresses will develop near the lake. When the whole deposit has been mined tensile stresses will develop beneath the entire lake (Figure 5-18 and Figure 5-19).

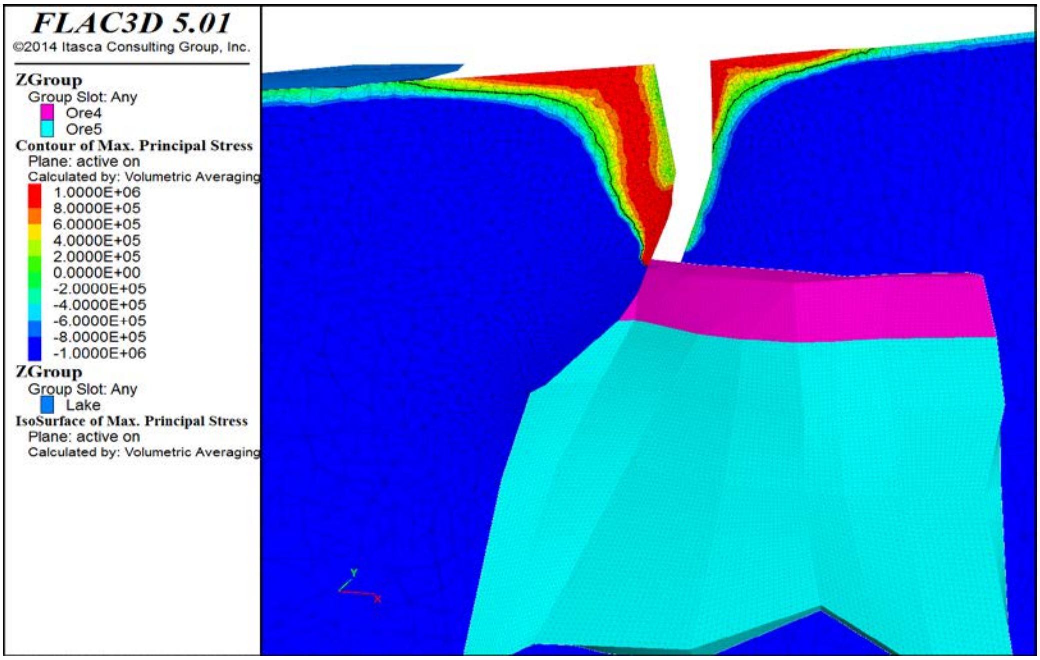


Figure 5-18 Minor principal stress for a mining depth of 450m

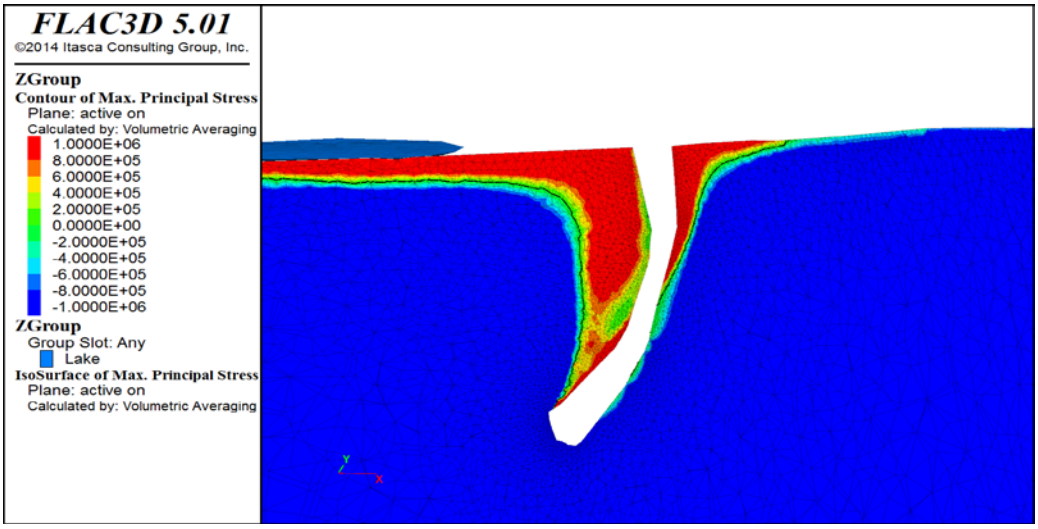


Figure 5-19 Minor principal stress after mining the known resource

As the orebody is mined out, the rock mass will displace towards the mined void. At ground surface the displacement extends almost to the lake as illustrated in Figure 5-20. When the entire resource is mined displacements will likely occur beneath the lake as well. The total area influenced by displacements will be approximately 3 km South and 2 km North of the mine (Figure 5-21).

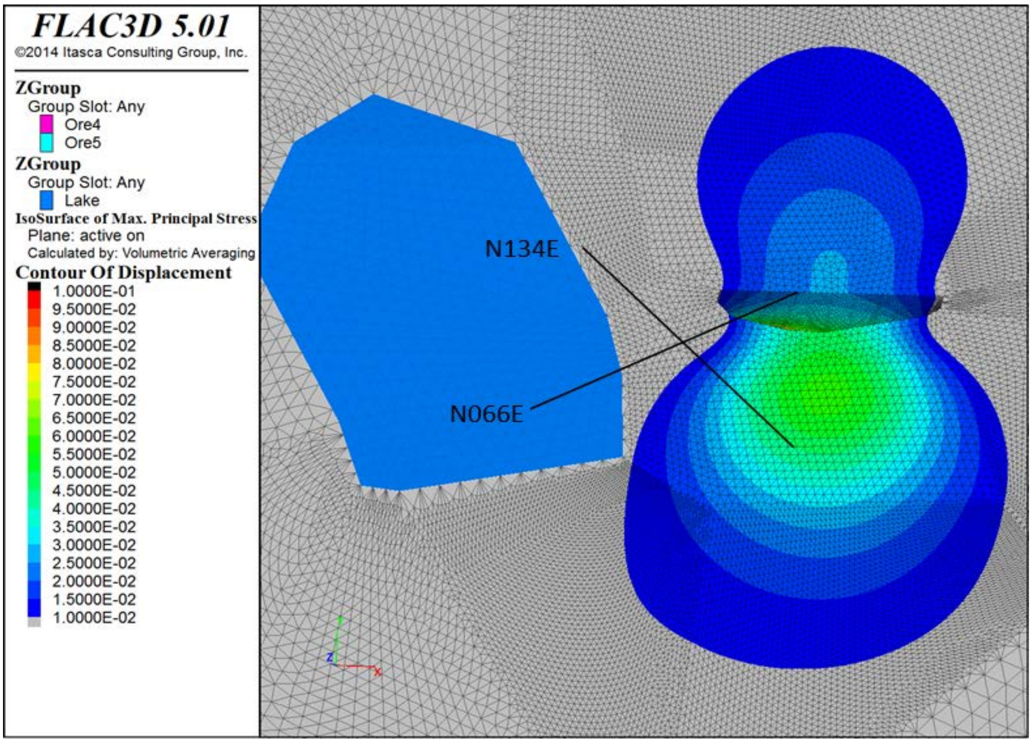


Figure 5-20 Illustration of total displacement at ground surface

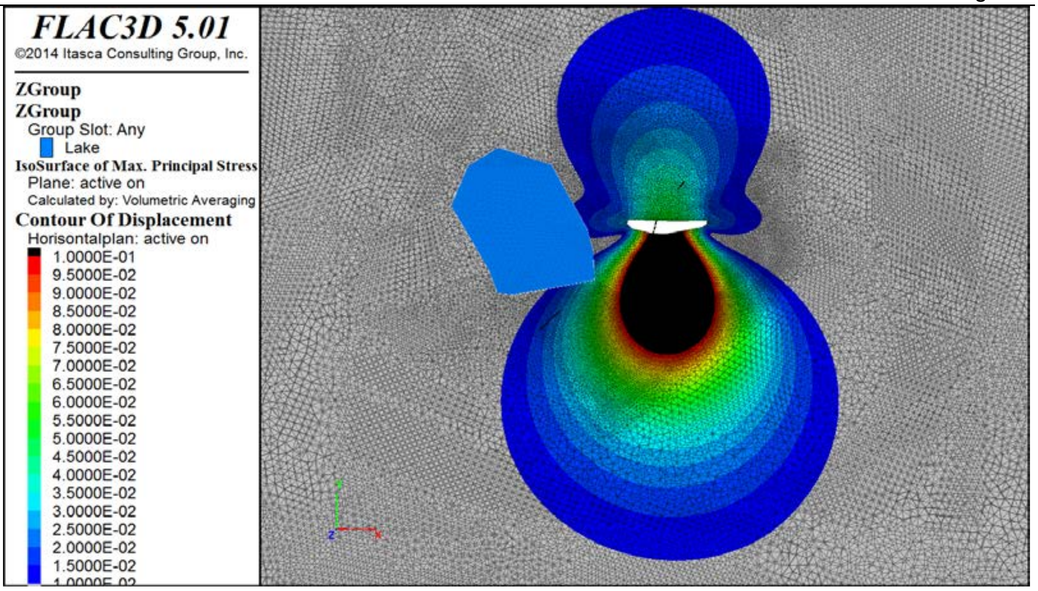


Figure 5-21 Illustration of total displacement at ground surface. Mining depth 900 m

5.5 Conclusion

No major differences regarding RQD are found between the different boreholes or between the different parts of the ore. There are differences in RQD within each borehole, but in general the boreholes show high values of RQD values, mainly 80–100%.

Based on the limited data currently available, the estimated water losses are relatively small in sections that comprise the orebody and the adjacent rock in the hanging wall and the footwall. However, as identified elsewhere further data acquisition and analysis needs to be carried out to support this interpretation.

From the old mine maps, the dominant strike of the fractures is almost parallel to the mineralization, east-northeast direction (N070E), and the dominant strike of the faults is in a southeast direction (N110-160E). This dip and strike of the structures are consistent with the results from core mapping in 2012 and 2014.

The Mathews stability analysis shows that the rock is mainly in the stable zone of the graph. However, since the boreholes were drilled in the same direction, some of the joint sets may be undetected. Hence, one analysis was made with the assumption that the rock has more fractures, which resulted in more unstable rock. Sublevel caving is an option for Blötberget. Parallel drifts can be mined independently of each other. However, front and rear wedges may cause problems.

The sublevel geometry is almost insignificant for the ore in direction 2 (N135E), since the block size does not change significantly when changing neither the height nor width. For direction 1 (N090E) however, the block size changes exponentially with change of stope high and width hence the geometry of the sublevel can be very important parameter importance.

If the ramp design follows the main ore direction (N090E), the largest wedges will be created, out of the three optional directions. Designing the ramp in one of the other directions (N135E or N180E) generates much smaller wedge sizes.

For a stope of size H20 x W10 x L4 meters, the amount of waste rock blended with the ore will be small (<3%).

The numerical analysis indicates that stress and displacement changes due to mining operations will affect the rock mass close to the lake when the mining depth reaches 450 m. When the whole resource is mined down to 900 m depth displacements will have extended beneath the lake. The risk of the mining activities impacting the lake increases as the mining depth extends below 450 m.

The fractures in the rock mass have such orientations that they will likely be affected by the changes in stress and deformation and thereby may increase the conductivity of the rock mass between the lake and the mine.

5.6 Recommendations

The orientations of the rock mass discontinuities close to the ground surface should be further investigated since the currently available information is based on discontinuity recordings at depth.

Re-visiting of the hydrogeological conductivity tests is recommended using double packer configuration over shorted sections of the boreholes to isolate and identify more accurate conductivity values in the fractured sections of the deposit.

Further analysis is recommended taking into consideration the impact of the ground water changes of the in situ stress in the rock mass due to the mining activities.

The numerical results are dependent on the use of a parameters. The in situ stress field is currently not well known. Rock stress measurements should be performed relatively close to the mine. The results should be used as input data in numerical models to increase the accuracy of the results.

For this report, all fractures have been analysed together and no distinction between hanging wall, ore and footwall has been done. The characteristics for the ore should be further evaluated, so that differences between different ore parts can be studied. That has been considered a question for the detailed mine planning and not been studied in detail at this stage.

Even though the material available so far gives a quite consistent picture, we recommend, in order to create a more holistic picture, that full mapping and orientation of all cores should be conducted. This report should then be updated with the new data. The RQD analysis has been updated with data from all mapping that has been done so far, but the Mathews stability graphs and the Unwedge analyses are only based on average data from boreholes mapped in 2012. Data from mapping of drill cores in 2014 have not been taken into consideration in these analyses, since they show the same values in general.

Both the 2012 and 2014 drill cores are recommended to be entirely mapped and then all analyses should be updated, if a more detailed evaluation should be produced.

Two boreholes in 2012 years program were stopped in fractured rock in the western fringe of Blötberget. These two holes should be studied further in order to show whether these are drilled in true crushed zones or if the more fractured rock has more correlation to the boudinage structure of the mineralization.

6 MINING

6.1 Introduction

Mining and exploration in the Ludvika area has been carried out in different periods since the 1600's. The majority of mining was focused on iron, except for two periods, 1701-1711 and 1885-1889, when copper was recovered at Iviken, in the most southern part of the Håksberg ore field. In the 1800's and early 1900's comparably lower quantities of ore were produced.

In 1937, when the mines were taken over by German owners all of the different mines within the Håksberg ore field were merged into one operating unit and a central hoisting and concentration plant was erected at Håksberg. This allowed more efficient mining with the transportation optimised and the facilities at the central shaft utilised while operating at its full capacity.

After the Second World War, Sweden regained ownership of the mines and continued production with two different companies, until mine closure in 1979.

At Blötberget, two mines with separate shafts were in operation simultaneously between 1950 and 1966; the Vulcanus "original" mine and the Blötberget "new" mine, which started operation in 1944 by sinking the new shaft to 300 m level and building the new central plant.

Since the mines closed in 1979, the deposits have been owned by various companies until taken over by NIO in 2008.

Thanks to the previous mining activities, a considerable amount of underground infrastructure exists at Blötberget.

6.2 Historic Mining Methods

The history of Blötberget is summarised below:

- 1900 - Mining Co Vulcanus started large-scale mining at Kalvgruvan and Flygruvan under German ownership.
- 1944 - Stora Kopparberg Bergslags AB started mining in an adjacent claim and sunk a new shaft (BS-shaft) together with complete new surface structures, head frame, concentrator, storage /loading facilities.
- 1949 - Stora Kopparberg bought Vulcanus.
- 1950 - to 1966 both mining areas were mined simultaneously, using both shafts. The production rate was ca 400 kt / year of ore and 220 kt of product.
- 1968 to 1975 - the BS-shaft was further sunk to 570 m depth. The hoisting facility was modernized and upgraded to 600 kt/year production capacity. The new plant commenced operation in December 1975.
- 1977 - Swedish Steel (SSAB) was founded and the mines (Blötberget and Håksberg) were sold to SSAB the same year.

- When the operation ceased in June 1979 a total of 19 Mt of material, averaging 37% Fe Total, 0.55 – 0.8% P and <0.01% S had reportedly extracted.

Mining of the Blötberget orebodies was started in 1900 by the Vulcanus Mining Co by open pit methods. Some years later the operations changed to underground exploitation using Shrinkage stoping.

In the 1940´s the mining method was changed again to Sublevel Caving (SLC), primarily using longitudinal stoping because of long and narrow ore-lenses: the Ko- and Flygruvan concessions were mainly exploited during this time

After acquisition by Stora Kopparberg Bergslags AB (“Stora”), mining of the Fly- Kalv- and Ko-orebodies continued until 1966, by which time mining had reached the 340m level (equivalent to 380m below surface (mbs)) and mining activities were stopped.

In 1944, Stora also started open pit mining in the eastern orefield; a separate mine comprising the Hugget and Betsta mining concessions.

At the end of the 1940´s Stora built a new mine-complex east of Vulcanus, centralized around the new BS-shaft that had been sunk to the 300 m level and the shaft was later deepened to the 570 m level in the 1970´s. As a result underground mining could now start in the eastern area, below the open pit, starting at 80 m level using Shrinkage stoping and partly open stoping. During the 1960´s the mining method was also gradually changed to SLC in Hugget/Betsta-area and after 1966 all production came from this area.

The upper part of the Sandell orebody was mined using SLC and open stoping during the latter 60´s and 70´s. During the 1970´s the use of track bound compressed air powered equipment was phased out and modern diesel powered LHD loading and hauling to ore passes replaced the old track bound haulage.

To assist this change, an internal decline was developed between the 160 m and 280 m levels to create easy access to the SLC-levels. The cross-section of the drifts in the lower mining levels 240 – 280 was also increased from an area of 6-10m^2 to 15m^2 to accommodate the new trackless equipment.

The last active main haulage level before closure was the 280 m level but preparation of next main level on 330 m had started.

In the final years before closure in 1979, the ore was transported on the main levels in 5m^3 Granby-cars pulled by diesel locos to a central orepass system that dropped ore to the primary crusher station on the 480 m level.

6.3 Current Mining Strategy

NIO intends to utilise the existing Blötberget mine to re-start ore and iron ore concentrate production from the Ludvika mines as Phase 1 of the Project.

The main drivers defining NIO’s strategy and concept for re-establishing the Blötberget mining operations are:

1. Maximize utilisation of the existing underground infrastructure;
2. Minimize pre-production capital investment;
3. Early cashflow i.e. early access to ore;
4. Target annual production of 3 Mt run-of-mine (ROM) ore.

To achieve the strategy, re-commencement of underground production will be achieved in three phases:

1. Pre-production with first access to ore;
2. Ramping up to full production and;
3. Full production at 3.0 Mtpa.

The mining plan is based on one-product-mine throughout the ore treatment and applying sublevel open stoping (SLOS) as the main mining method. This will secure a high-production level, high mining recovery, a reasonably high ROM grade and the possibility to selectively mine some of the orebodies.

Ore and waste will be transported from the mine on a system of conveyor belts, fed from a primary crusher. Ore will be delivered to the primary crusher by an orepass system from the production levels and delivered to the secondary crusher on the Surface by conveyor..

The development plan includes a surface access decline to the mine, which will be used for mineral and waste transport via a conveyor and a single lane vehicle access road. The decline will be developed during Phase 1, when access to the first primary crusher at level 420 will be achieved.

The conveyor system will be extended in a dogleg decline system with haulage levels and crusher installations on three deeper main levels as mining extraction sinks through the orebodies. An internal ramp system will allow vehicle movement between levels.

Other mine infrastructure, e.g. ventilation, pumping, power distribution, service facilities is designed to maximize the re-use of existing underground workings.

The existing BS and Vulcanus shafts will be used for ventilation and mine services (power and pumping) only.

6.4 Trade-off studies

A number of trade-off studies have been carried out by NIO in the PEA and in late 2014 and early 2015 in order to determine the best options for the re-development of Blötberget.

These studies included optimisation of mining capacity, selection of mining method, mine level layout and selection of the mineral transport method.

A summary of these studies follows.

6.4.1 Mine Output capacity

To define the optimum production capacity from the Blötberget deposit, the planned production chain was evaluated in order to define the most critical operation that affects the overall production capacity.

While for all practical purposes, everything in the minerals handling chain can be designed and constructed to accommodate a wide range of production capacities, it is the shape and physical characteristics (width, length and dip) of the main Blötberget orebodies that are the limiting factor as regards the maximum mine output and extraction rate.

To evaluate the maximum achievable capacity a typical mining level was created based on the orebody wireframes. The level comprised the ore outline, preliminary production drive configuration (single or twin-parallel drive along the orebody) and delivery point to ore pass system.

The average number of drawpoints used in this assessment was ten. By applying typical production rates or LHD units and availabilities for each extraction point, it was determined that a maximum of three loaders can operate on any given level at any given time, with one additional unit working on the level below. This configuration results in the potential to produce between 600t and 2000t per drawpoint per day, depending on the ore geometry in the various parts of the orebody. This equates to an overall mine capacity of between 2.6 Mtpa and 3.2 Mtpa.

A LHD simulation undertaken by Atlas Copco further supports this production level.

As a result of the production assessment carried out by NIO, an annual capacity of 3Mt has been selected for this Interim Technical Report.

6.4.2 Proposed Mining methods

Two bulk mining methods have been used in the past at Blötberget; these being sublevel caving and sublevel open stoping.

6.4.2.1 Sublevel Caving (SLC)

Sub-level caving is the dominant mining method in the iron ore mines of LKAB in northern Sweden. It has been used for many decades and increased in scale over the years partly due to the greater degree of remote control and automation. The increase in scale refers to both a greater distance between cross-cuts in horizontal or vertical orientation, larger drifts and larger and more efficient mobile equipment. Large-scale sub-level caving reduces the development portion of the mining cost, which is a large portion of operating costs.

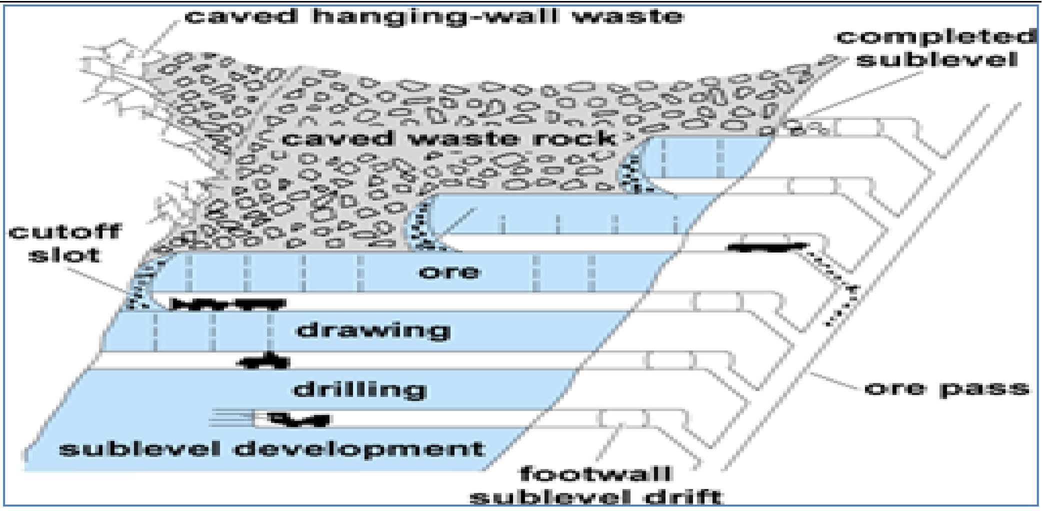


Figure 6-1: Generic schematic of the SLC method

Sub-level caving is not a widely used method for world-wide ore extraction. The method has been developed specifically for the bulk mining of low value ores (on a tonnage basis) such as iron ore, where a certain degree of ore loss is acceptable. This system is one of the cheapest methods of underground production with only block caving having a lower cost per tonne mined.

Historically, parts of Blötberget mine have been mined using the SLC mining method. When sub-level caving was first used in Swedish iron ore deposits, both horizontal distance between cross-cuts and sub-level interval was approximately 10 m. Techniques have gradually improved and more current development in Swedish iron ore deposits shows cross-cut distance of 22 m to 25 m and sub-level interval of 27 m to 30 m.

The method has some obvious advantages over other stoping methods with perhaps the lower operating costs being the most important. However the downside can be loss of control in terms of hanging wall waste rock dilution, which will negatively affect head grade.

Typical prerequisites for applying SLC to an orebody include a steeply dipping orebody, reasonable orebody width (>6 m), a tabular or massive shape, uniformity of grade, moderate to strong ore and weak to moderately strong waste rock that will cave naturally.

It is essential that the hanging wall caves after extraction of the ore and that the ore geometry supports free-flowing rock i.e. the dip is in excess of 40 degrees.

6.4.2.2 Sublevel Open Stoping

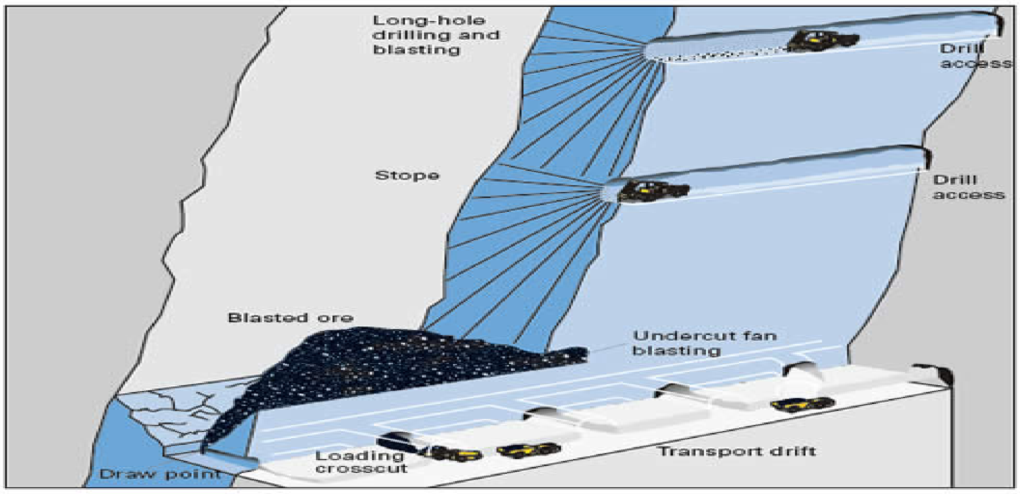
This method is usually considered as an alternate to sublevel caving when a lower dilution method is needed and where the rock is fairly competent. The method is a high production relatively low cost method of mining and is fairly safe as the mining tends to retreat away from previously mined or unsupported areas.

The mining method is similar in principle to SLC except that the stope remains open after the ore has been extracted, without any caving of the hanging wall.

Ore bodies that are ideal for sublevel stoping are typically tall and have a medium to narrow width, but can have an extensive length. The ideal width of the orebody should be greater than 6 meters but less than 30 meters. Most of the time this type of orebody has a dip greater than the angle of repose of the ore, or steep enough so that the ore can fall efficiently into the open stope. Preferably the dip would be between 60 and 90 degrees, however anything above 45 degrees may be adequate.

Sublevel stoping requires a midrange ore grade in order to pay for its development. This mining method can be used at a wide range of depths. The orebodies must be made of competent rock, and be surrounded by competent wall rock. Due to the nature of the drilling and blasting of the rock it is best if the orebody has a regular outline with well defined edges, as this lowers the amount of dilution. Since this mining method requires that the ore be blasted on multiple occasions, the ore should ideally have a high compressive strength and few planes of weakness such as joints, bedding planes or faults.

A typical arrangement is shown in Figure 6-2.



Sublevel open stoping layout

Figure 6-2: Generic SLOS Stope

6.4.2.3 Selection of Mining Method

The Blötberget orebodies generally meet the criteria for the application of SLC above 500 m level; however, below this level the orebodies flatten out and are insufficient in width to utilise this method. Rock mechanics also impact on the selection of mining method; details of their impact are available in section 5.3.4.

DMT considers that the use of SLC mining methods could be appropriate for the future mining of some of the upper levels of Blötberget orebodies. However due to the dip of the orebodies, which gets less with increasing depth, the modern SLC method could only be applied safely above the 500 m level with Sublevel Open Stoping (SLOS) being applied below the 500 m level.

However, to select the most appropriate method to apply to the Blötberget orebodies in this Study, a comparative economical assessment of SLC versus SLOS was conducted.

Parameters used for the comparison were ore recovery (SLC 100%, SLOS 80%), waste dilution (SLC 25%, SLOS 15%) and weight recovery ore-to-product (SLC 41%, SLOS 45%). These parameters were applied to the global resource.

The results of this assessment indicate that both methods offer a similar mining cost per ton of product. The higher headgrade in early years of production when applying SLOS has an advantageous effect on the early cashflow. However, the effect of applying SLOS to the entire resource does result in a lower total ore recovery and the overall return on investment is thus lower.

The key factor affecting the selection is nevertheless the grade output in the early years of production, which is clearly higher when applying SLOS.

Due to the positive impact on early cashflow, SLOS has been used as the mining method for the whole mineable inventory for this Interim Technical Report.

6.4.3 Minerals Handling

Historically the Blötberget mine utilised shaft access and skip hoisting for ore and waste from the mine.

For the future planned operation with a production of 3 Mtpa ROM ore and 0.3 Mtpa waste rock, alternative hoisting methods have been evaluated. The alternatives considered were shaft hoisting, combined shaft/conveyor, trucking and conveying.

The first three options included a permanent primary crushing station at a fixed location, while the conveyor option adopted primary crushing at 80m interval haulage levels (located at 420, 500, 580 and 660 levels).

Using database CAPEX and OPEX as the basis for the costs estimates of all alternatives with some project specific budget quotes, the conveying option with conveyors from each of the haulage levels is the most cost efficient way of transporting ore and waste to surface. Factors contributing to this conclusion are ore geometry (dip flattens out with depth, adding to haulage distance to any one centrally located crusher), horizontal distance on surface from the BS shaft to processing plant (circa 2km) and environmental restrictions on the surface infrastructure between the mine area in Blötberget and the planned processing plant location at Skeppmora.

For the purpose of this Interim Technical Report, a belt conveyor system to surface with primary crusher stations located at each of the main levels at 80m intervals has been adopted.

6.5 Mining Layout

6.5.1 Mine Access

The mine will be accessed via a new surface connection, using a combined conveyor and vehicle ramp developed from the Skeppmora industrial site West of the Blötberget mine area; the decline roadway will connect to the existing workings at the 320 m Level. From this level a new internal ramp system will be developed to provide access to the first main haulage level at 420 m. The layout is shown in Figure 6-3 below.

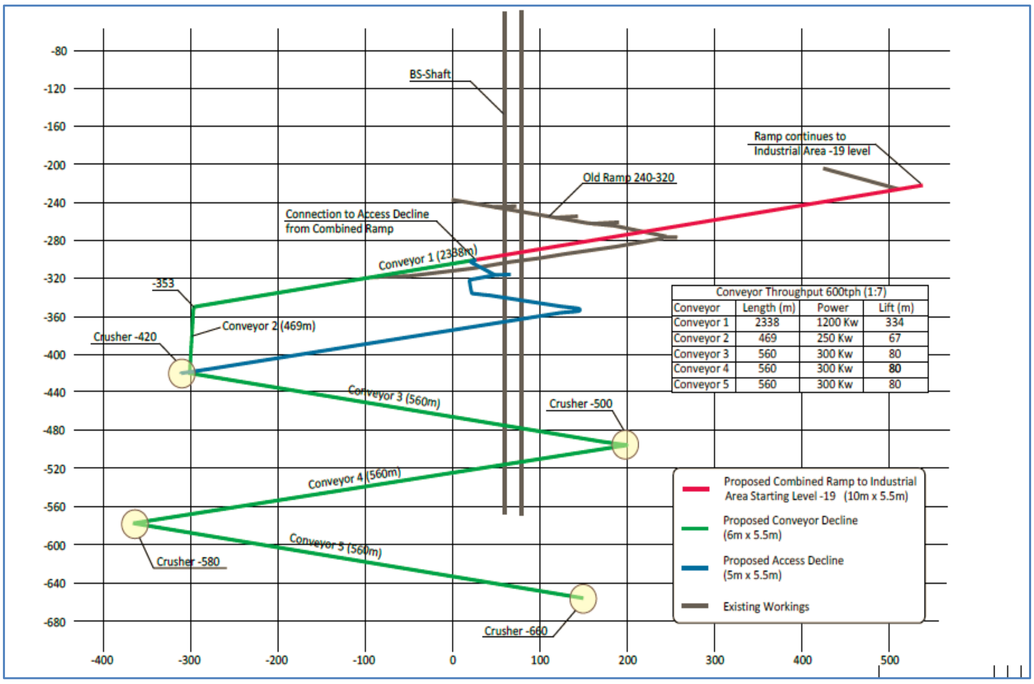


Figure 6-3: Schematic of main access declines and ramps

The ramp will contain an access lane for vehicles and the conveyor for ore and waste handling (see Figure 6-4). The cross sectional area will be 60m^2, driven at a down dip gradient of 14% (1 in 7) and a length of approximately 2340 m from the portal at an approximate elevation of -19m at Industrial Site 1.

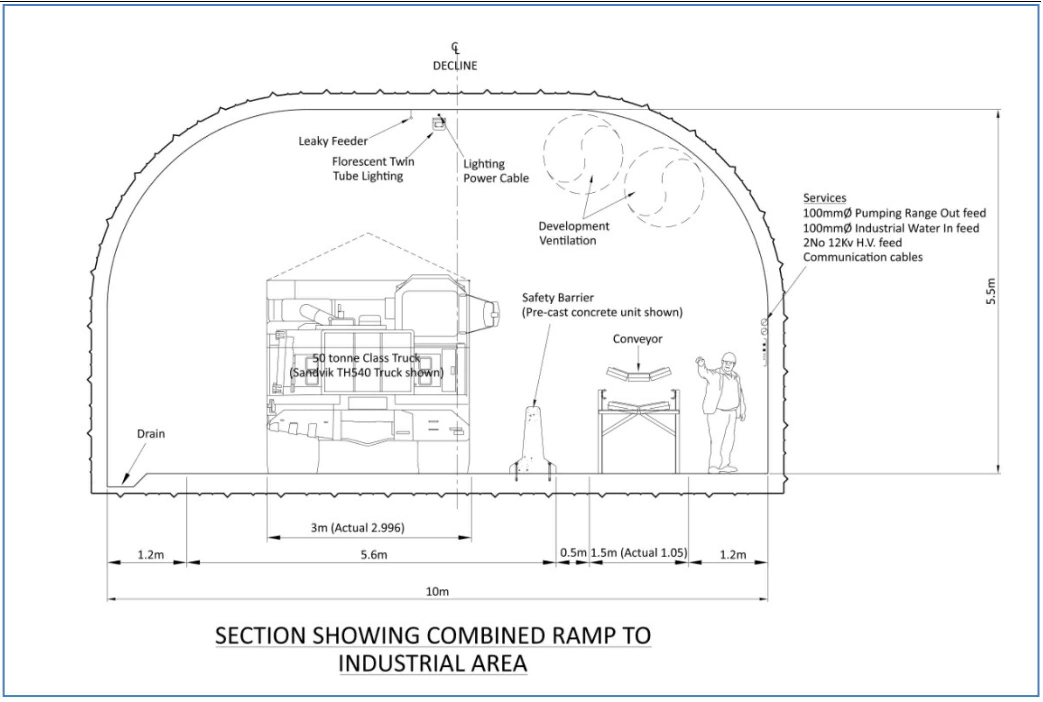


Figure 6-4: Main Surface Access Ramp

The main Access Decline will provide a connection to the BS shaft (which will be utilised as the fresh air intake), the Sandell orebody and to the existing infrastructure on the 320 m level.

Main haulage levels will be developed at 80 m intervals at 420, 500, 580 and 660 levels with a twin separate ramp and conveyor decline system developed below 420 m. The levels will be developed at a size of 6m wide by 5.5m high to accommodate large LHDs and 50 tonne underground haul trucks.

The internal ramp system will continue at a section of 5m wide by 5.5m high to allow mobile equipment movement between the main levels and sublevels (see Figure 6-5).

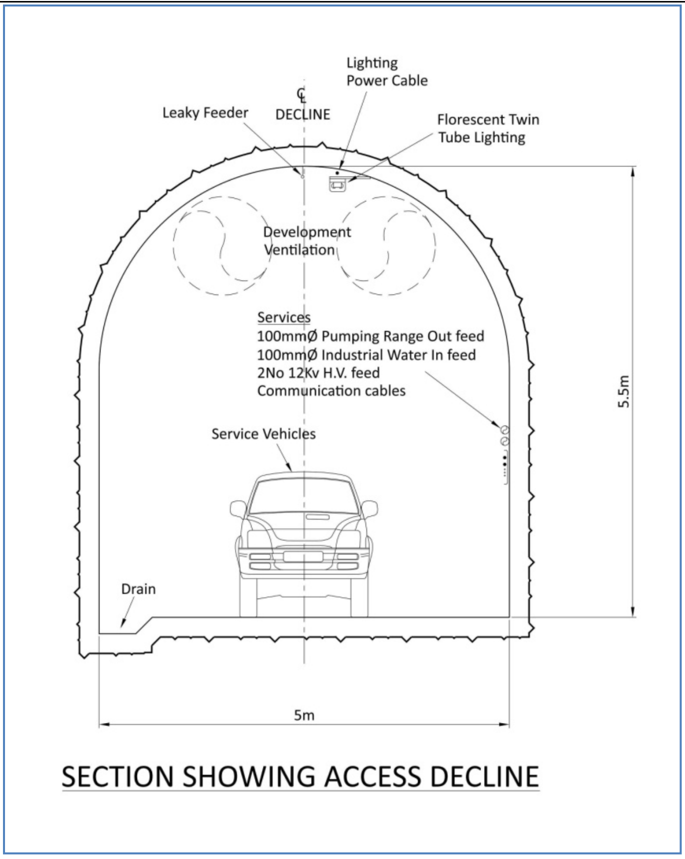


Figure 6-5: Typical Internal Access Ramp

The conveyor declines will be developed between the main levels and connect to the crusher stations at each main level. The section will be sufficient to accommodate a service vehicle at the side of the belt conveyor. See Figure 6-6

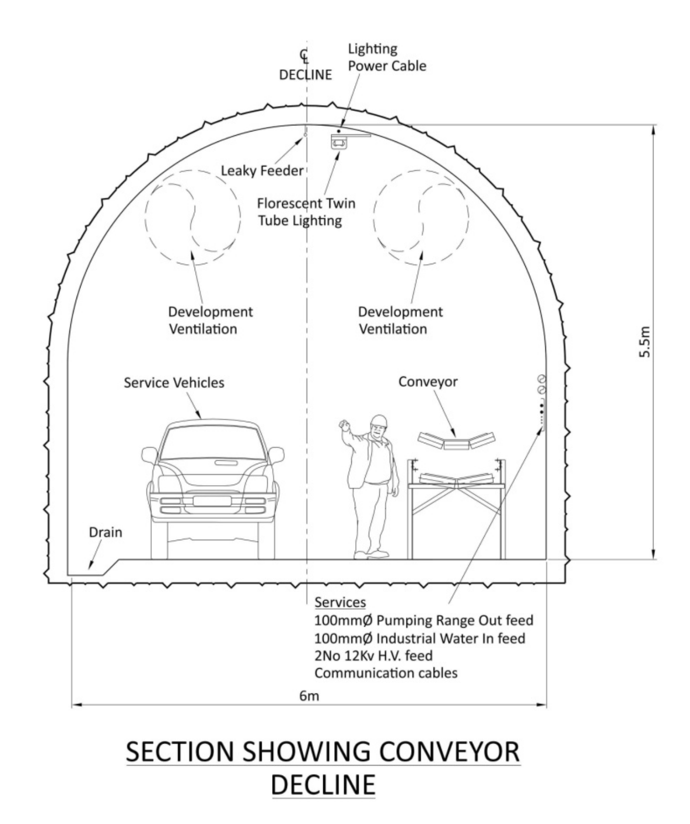


Figure 6-6: Conveyor Declines

6.5.1.1 Sublevel Open Stoping Design for Blötberget

All access ramps, declines, major infrastructure and primary development drives are placed in the footwall of the orebody at a minimum stand-off distance of 30m from the stope edge.

The SLOS layout for Blötberget has been based on the following parameters:

- Sub-levels are spaced 20 m floor to floor vertically;
- Haulage levels are located at 80m intervals floor to floor vertically;
- Due to the narrow width of the orebodies most sublevel drives will be driven in the footwall to ore.
- Production and drill drive cross-cut section will be width 6m by 5.5 m height.
- Radial blast holes will be drilled overhand from each drill drive

6.5.2 Backfilling

Open voids i.e. stopes left behind after ore excavation (applying SLOS) can be backfilled with development waste.

Based on the current stoping layout on each level there will be two to three access points to each open stope remaining after completion of the stope. These accesses can be utilised to place waste rock into the open stope void using a combination of trucks and LHD’s. However, the volume for backfilling purposes is limited by dump location and the stope geometry and it is estimated that an average 40% of stope in-situ volume can be backfilled.

As an example, for the first production stopes, located between levels 320 and 380, the open stope is estimated to be approximately 400 000m^3. At 40% the effective volume that can be backfilled is 160 000m^3. This equates to 250kt of development waste that can be deposited in the stopes between levels 320 and 380 and compares to an annual average waste production of approximately 300kt to 450 ktpa.

At steady state production (below level 390) the approximate total void available for backfilling per 60m lift is 640 000m^3, of which effective 40% 250 000m^3 or 410kt can be backfilled. So overall the open stope volume created is sufficient to accommodate 80-100% of development waste in the worked-out stopes.

On the current production schedule, these stopes are completed by end of Year 3 of production, at which point the development will be at level 440 and below. As a result the waste must be hauled to level 340 rather than to level 380 and from there via the ore passes, crusher and conveyor system to surface.

Placing waste rock as backfill into worked out stopes would result in the following cost savings:

- Waste loading into orepass (5SEK/ton)
- Transport on haulage level (1.5SEK/ton)
- Primary crushing (5SEK/ton)
- Hoisting to surface (5SEK/ton) and
- Material handling on surface from portal to Industrial Area 2 (10SEK/ton)

In total these costs are estimated at 26.5SEK/ton waste.

For an average annual waste handling of 300kt, the cost saving is in the region of 8MSEK (2.65SEK/ton ore). Some additional costs will be incurred in securing and enlarging the tip locations at the top of each open stope.

Whilst using waste rock underground as backfill results in a reduction in operating costs, it will still be necessary to transport some waste rock to the surface for use in the TSF dam wall raises.

6.6 Mineable Inventory

6.6.1 Mineral Resource Base

The mineral resource base used for the mineable inventory that forms the basis for the LOMP is described in Section 4.12 and shown in Table 6-1.

Table 6-1: Mineral Resource Estimate at 25% Fe COG

Fe Cut-off [%]	Rclass	Volume [Mm³]	Tonnage [Mt]	Density [t/m³]	Fe [%]	Mag [%]	hem [%]	Mag proptn	hem proptn	P [%]

Fe Cut-off [%]	Rclass	Volume [Mm³]	Tonnage [Mt]	Density [t/m³]	Fe [%]	Mag [%]	hem [%]	Mag proptn	hem proptn	P [%]
25	M	11.1	42.5	3.8	41.9	36.8	21.9	0.63	0.37	0.51
	IND	1.4	5.3	3.7	38.2	30.5	23.2	0.57	0.43	0.5
	M + IND	12.5	47.8	3.8	41.5	36.1	22.0	0.62	0.38	0.51
	Inferred	1.5	5.4	3.5	33.5	23.5	23.5	0.50	0.50	0.52

The mineral inventory used in the LOMP includes only Measured and Indicated Resources. No Ore Reserves have been estimated because DMT considers the overall level study to be below the requisite level of a pre-feasibility study.

When a more detailed mine design is prepared in the next stage of engineering, the Mineral Reserves estimated will by definition exclude all Inferred Resources.

The Mineable inventory was estimated by using the resource model wireframes to estimate the tonnage for each 20m-mining level to generate the global tonnages available for that level. Mining factors of ore recovery, waste dilution and waste rock Fe grade have then been applied to arrive at the estimated mineable inventory.

For the reasons explained earlier in this section, the mining method selected for this Study is the SLOS method of mining.

For the purposes of the Base Case in this Study, the LOMP is based on the following assumptions:

- In-situ tonnes 48.1Mt at 41.6%Fe
- ROM headgrade is 36.1%Fe
- Ore ratio - 76%Magnetite/24% Hematite
- Process weight recovery - 45.0%.

6.6.2 Internal Dilution

Internal dilution in the blockmodel is the result of barren or low grade material in the boreholes. Internal dilution has been included in the wireframes and blockmodel at a grade of 8%.

Any low grade material that occurs in designed stopes will be excluded in pillars if practical.

6.6.3 External Dilution

External dilution is the waste rock or low grade material that inevitably enters the stope from beyond the stope cut-off outline and is drawn off as part of the ROM ore. The quantity of dilution is a function of the mining process (blasting control) and the geomechanical characteristics of the hanging and footwall rock.

6.7 Life of Mine Plan

6.7.1 Overview

The life-of-mine plan (LOMP) essentially consists of three stages:

- Pre-production (initial development and preparation for ore production);
- Production ramp up (until full capacity is reached) and
- Steady state operation at nameplate capacity of 3.0Mtpa.

The Pre-production period targets are to provide access to the orebodies and the completion of the hoisting, ventilation, power and dewatering systems. Duration of period is estimated at 27 months with milestones defined at month 15, when access to Sandell ore is established, month 18 when access to historical underground infrastructure is established and at month 21 when access to ore on level 340 is established. The period can be divided into individual work packages, which are commonly awarded to external contractors, as the key driver is speed of advancement across the tasks.

6.7.2 Stage 1 - Pre-Production/Development Schedule

Key milestones identified during the pre-production/development schedule are month 15, month 18 and month 21 as detailed above.

The pre-production period, is estimated to take approximately 27 months and will include the following Major works:

- Dewatering the mine via BS shaft (5 Mm^3);
- Establish permanent pumping facility at 370 m level;
- Development of access ramp and conveyor drive to 420 m level;
- Excavation of crusher cavern at 420 m level;
- Crusher installation at 420 m level;
- Installation of No.1 and No.2 Conveyors (surface to 353 m) and (353 m to 420 m) respectively;
- Installation of ventilation equipment at 370 m level;
- Installation of underground power distribution system;
- Construction of ore passes between 320 m and 420 m levels;
- Preparation of mining levels on block 340-380 in Hugget;
- Development and access to Sandell orebody from the main ramp at elevation 230 m;
- Sandell internal access ramp

NIO intends to use external contractors to carry out the pre-production phase of the mine development.

6.7.2.1 Pre-production Development

The total pre-production development required to access first ore is approximately 5,050m (circa 195,000m^3) at an assumed rate of advancement in the range of 150 meters per heading per month.

Development of the main access ramp from surface to 420 m level is on the critical path of the Project, and it is essential that a high rate of advancement is maintained in those particular headings.

The fixed installations required for ore production are installed during the pre-production period; this includes the primary crusher at 420 m level and the conveyor system to surface.

6.7.2.2 Ore access

First access to ore is achieved at Month 15 in the Sandell ore-body.

Any ore extracted during the pre-production period will be trucked to surface, crushed to minus -70mm and stockpiled in the immediate vicinity of the portal. A stockpile of approximately 0.25Mt is anticipated by end of the prep-production, predominantly from Sandell. This stockpile will be fed into the mill feed via the ROM stockpile reclaim conveyor once the processing plant has been built and commissioned and is ready for commercial production.

6.7.3 Stage 2 - Production Ramp-up

An 18 month ramp-up period has been allocated to bring the ore production up to 3.0Mtpa. During this period ore is accessed and produced at Sandell between 180 and 240 levels and in Hugget from the 340 level downwards.

The full width of the orebodies is only accessible from 380 and below, meaning that full production can only be achieved when production is at or below that depth.

Due to the historical mining activities in the Blötberget area, mined out and non-accessible areas have been assumed above the 240 level in Hugget and above the 380 level in Flygruvan and Kalvgruvan deposits.

The ramp includes:

- Sandell ore development and subsequent production from levels 180-240;
- Hugget ore development and subsequent production from levels 340-380.

Development metres during ramp-up will consist of production drives in Sandell and Hugget 340-380 for an estimated total of 4,595m; split between 3,930m in Hugget and 665m in Sandell.

Approximately 3,750m of drivage (94 000m^3) is in waste rock.

The first ore originates from Sandel 180 level. With a limited footprint and single-drive configuration per level, Sandel is estimated to produce on average 1250 tpd.

Hugget level 340 will be available for production operations approximately 3 months after Sandel production operations commence; and the level is accessed via a down dip access decline from level 320. For levels 340-380 an overall production rate of 6000 tpd has been estimated.

6.7.4 Stage 3 - Steady state Production Schedule

As the mine has been partially developed to the 320 level in Hugget, this provides an opportunity for early production once access to the 240-320 block has been established, however, due to current uncertainties in mapping the existing infrastructure it is likely that the production from this block would be at a lower extraction rate.

A wider orebody and new infrastructure on levels 340-360 gives a further opportunity for higher production.

Steady state production at nameplate capacity of 3.0Mtpa is reached when level 380 is in the mature production phase i.e. the full width of orebody is accessible for production.

From this level onwards, an overall production capability of 10 000 tpd is assumed. Due to the nature and geometry of the deposit, Flygruvan must always be mined ahead of Kalvgruvan however in practise one level difference is sufficient. Due to this constraint, below 380 level (the lowest mined out level in the Kalvgruvan and Flygruvan orebodies) two levels must be in production at any given time.

As far as geotechnical constraints allow, SLOS stopes will be maximised in size to maximise the mucking capacity. For purposes of this study, stopes with the following typical dimensions have been assumed; length160m, height 60m and width of orebody (15-30m).

Over the LOMP, the average vertical sinking rate is 20-25m per annum.

The development requirement to maintain extraction sequence and forward planning is approximately 3,000 metres annually of which approximately 2,300m is waste development in the stoping blocks.

Development intensity is highest during the early years, when access to the first haulage level at 420 is required.

Ore between the previously mined out area (in Hugget above level 240) and the top of new mining area (first stope between 320-340 levels) will be mined during years two and four with an estimated extraction rate of 6000tpd. To ensure that the 240-310 block can be mined independent of mining operations below 320 level, a 10m thick crown pillar at 320-310 will be left unmined. For each of the upper levels it is estimated that 900m of development is required.

The ore production and development profile for the LOMP is shown in Figure 6-7.

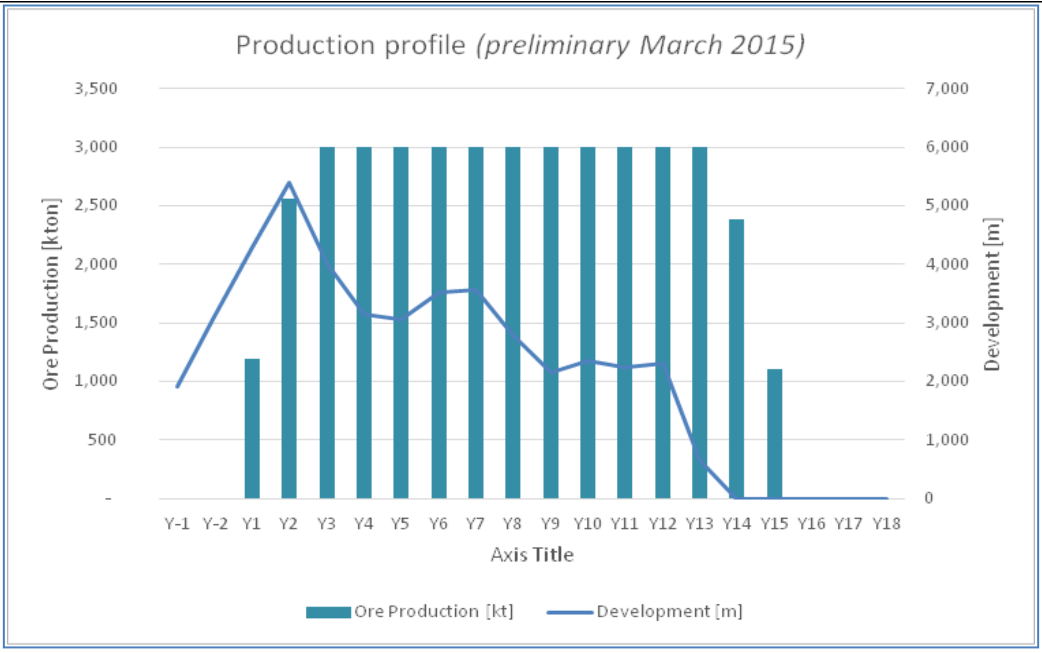


Figure 6-7: Ore Production and Development Profile

6.8 Mining Equipment

Underground capital development and rock excavation works (access and conveyor ramps, haulage levels, crusher stations etc) will be carried out by an external contractor; therefore development equipment for this work is not included in the capital investment estimate for the mine.

Ore production and development (sublevel development and ore loading) will be carried out by NIO using the mobile equipment listed in Table 6-2. Also support functions (main ventilation, high voltage power distribution, permanent dewatering, planning and management) for both capital and operational works are provided by NIO.

Table 6-2: Mobile fleet for Ore Production and Development

OPEX development	active	in reserve	Notes
Jumbos	1		
Shotcrete sprayer			*Contracted*
Concrete trucks			*Contracted*
Bolters	1		
Cable bolting			*contracted when req.*
Scaling	1		
Loader – 10t	1		*10-12t*
Face charging unit			*Contracted*
Lorries (development)			*Contracted*
sum:	4	0	
Production	**active**	**in reserve**	

OPEX development	active	in reserve	Notes
Long-hole rigs 4	4	0	*110mm*
Box-cut rig	1	0	
Longhole charger			*Contracted*
Loaders - 18t	4	1	*incl remote*
50t trucks	2	1	
Secondary breakage	1	1	*hammare alt hydr. Split*
sum:	12	3	
Auxiliary fleet	**active**	**in reserve**	
Long-hole rig - infra	1		
Light vehicles x 10	10	1	
Lifts – long reach 2	2		
Work platforms (vent., services)	2		
Material transport	2		
Work platforms with lifts (vent., services)	2		
Fuel truck - mine internal	1		
Water truck	1		
Workshop-forklift	2		
Powder truck			*Contracted*
Road grader			*Contracted*
sum:	23	1	

6.9 Manpower

The following manpower assessment assumes that all underground capital rock excavation works are executed by a contractor.

The estimated NIO workforce for ore production and mine maintenance and service functions are detailed in Table 6-3.

Table 6-3: Owner Production labour

OPEX development	per unit	units	shifts	dayshift	total	Notes
Production foreman				1	**1**	
Jumbo	1	1	2		**2**	*M-F, 2shifts per day*
Charging unit (crew above)					**0**	*Contracted*
Rock support crew (scaling/bolting/shotcrete)	2	1	2		**4**	*M-F, 2shifts per day*
Concrete trucks (drivers below)					**0**	
Loader					**0**	

OPEX development	per unit	units	shifts	dayshift	total	Notes
Lorries 5 units						*Contracted*
					7	
Production	**per unit**	**units**	**shifts**	**dayshift**	**total**	**Notes**
Production foreman				1	**1**	
Loaders	1	4	5		**20**	*continuous 5-shift*
Longhole drilling	1	4	4		**16**	*continuous 4-shift*
50t trucks	1	2	5		**10**	*continuous 5-shift*
Oversize handling	1	1	2		**2**	*M-F, 2shifts per day*
Long-hole charging					**0**	*Contracted*
					49	
Services	**per shift**		**shifts**	**dayshift**	**total**	**Notes**
Workshop - mobile fleet service					**0**	*Contracted*
Fixed plant, workshop, services foreman				1	**1**	
Fixed plant maintenance (crusher, conveyors)	2		5	4	**14**	
Ventilation, pump crew				4	**4**	
Mine construction crew				4	**4**	
Electricians	1		5	1	**6**	
					29	
Engineering, Management				**dayshift**	**total**	**Notes**
Geology				2	**2**	*jnr, snr*
Planning, Survey				3	**3**	*jnr, snr, survey*
Mine- and Technical Services Mng, Mine Superintendent				3	**3**	
Admin				**0**	**0**	
					8	
				grand total:	**93**	

6.10 Mine Ventilation

6.10.1 Background

The following requirements, as prescribed in the Swedish regulations (Work Environment Authority, AFS 2005:17), have been used to determine the mines ventilation requirements; these are:

- Carbon monoxide (Level Limit Value 35 ppm, if diesel exhaust 20 ppm);
- Carbon dioxide (Level Limit Value 2 ppm, if diesel exhaust 1 ppm);
- Organic dust (5 mg/m^3);
- Silica dust (0,1 mg/m^3).

The ventilation design utilises the existing BS Shaft, Vulcanus Shaft and Central shafts as the main airways, with the BS Shaft used for the intake airway and the Vulcanus Shaft and Central shafts used as the main return airways.

Air is drawn down to level 370 (existing) via the BS Shaft from where it is delivered to the sublevels via series of ventilation raises. As the BS shaft is located at the eastern flank of the Blötberget orebody all intake ventilation raises will be located at the eastern end of each sublevel.

From each of the ventilation raises, the air is directed via ducts and if needed, secondary fans into the working areas (a maximum of three per level). The ventilation design assumes that a maximum of two sub-levels are in ore production at any one time in addition to three/four active development drives. A maximum of ten working faces therefore require ventilation at any given time, each consuming 25m^3/s; 255m^3/s in total.

Exhaust air is directed toward the West end of the mine where the Vulcanus and Central shafts are located in auxiliary exhaust fan is installed at each exhaust ventilation raise.

6.10.2 Pre-production phase

During the pre-production phase, all development operations down to -320 level are force ventilated from surface using two, 90kW fans and two, 1200mm x 2200m ducts, with exhaust routed through the main ramp to surface.

6.10.3 Ramp-up phase – Sandell, Hugget

Mining of Sandell and later Hugget 340-380 levels will take place during the pre-production period when the permanent ventilation solution is unavailable. Fresh air will be taken from surface via the decline roadway at first and later via BS shaft. The exhaust will be routed through the main ramp to surface.

6.10.4 Steady state phase – early years

The BS Shaft will become available for ventilation purposes in the early phase of production operations. Mining blocks between levels 240-320 will be ventilated with intake from the BS shaft via 320 level with exhaust air being directed to Central shaft at 320 level at the western end of the mine.

6.10.5 Steady state ventilation

When in full production i.e. below 380 level, the access to the existing 370 level is established and two 250kW, 1800mm, Main Fans will be installed and utilised to provide intake ventilation for the mine; provision will be made for a third fan should it be required in the later stages of the mining operations Intake air will be heated at the BS Shaft to prevent ice forming in the intake roadways.

It is estimated that a total of 34 auxiliary fans will be required for the mine, with an installed power of 1800kW.

Gas or electric heaters can be utilised to heat the intake air at the surface of BS Shaft. Calculations using base data of 255m^3/s and temperature delta of 25C, predict a power requirement of approximately 7.7MW.

For the purposes of the Interim Technical Report, the gas alternative is selected for heating method.

7 UNDERGROUND MINE INFRASTRUCTURE

7.1 Introduction

The purpose of this section is to provide outline engineering infrastructure requirements for the mine. This consist of, but is not limited to, the underground electrical and mechanical requirements for the Blötberget mine operations, inclusive of mine dewatering, access declines, materials handling and services.

7.2 Mine dewatering

The dewatering plan for the Blötberget mine is to pump the mine clear to an existing inset level of -373m. The majority of historical mining activities have previously taken place above or on the 280 m level; consequently the largest volumes of water are believed to be found in the top 280 meters.

The Environmental permit allows the mine water to be pumped out of the Blötberget shaft at an average rate of 150 l/s (540 m^3/hr) with a maximum of 300 l/s (1,080m^3/hr).

Previous dewatering facilities were located in the BS-shaft and pump lodges were situated on level 80 m and 280 m. On level 80 there were two pumps with a total capacity of 59 l/s (213 m^3/h) and on level 280 m there were three pumps with a total capacity of 82 l/s (294 m^3/h).

7.2.1 Initial dewatering process

Prior to the dewatering operations at Blötberget mine and discharging mine water into the local water course, it will be necessary to reroute the River Gonäsån.

The proposed pumping operation would utilize up to 3 submersible pumps mounted on a steel support frame over the shaft supporting the shaft ranges and pumps which will be installed down to the -373m level. The mine water could be pumped out by the 2 or 3 submersible pumps which have a combined rate of 300l/s (1,080m^3/hr) in total.

The estimated water make in the mine currently is stated at 5Mm^3 of mine water plus an historical estimated inflow of 40l/s (144m^3/hr).

Therefore, based on the permissible environmental pumping rates (150l/s average and 300l/s max) the pumping time required to lower the mine water utilizing two of the three pumps is:

(5,000,000 m^3 / (720m^3/hr – 144m^3/hr) = 8,680 hours (361 days)

If the mine water inflow is greater than the estimated 40l/s or there is a greater make up of water all three pumps could be utilized to pump the mine water clear at the maximum permissible rate. Assuming an increase of 50% water inflow (40 x 1.5 = 60l/s) then the pumping time required to clear the mine water to the lower workings utilizing all three pumps is:

(5,000,000 m^3 / (1,080m^3/hr – 216m^3/hr) = 5,787 hours (241 days)

When the mine water level has been lowered sufficiently past the -373 level, a permanent pumping station can be installed underground close to the shaft inset, which ultimately replaces the submersible pumps to control the mine water.

The permanent pumping station is designed to manage indicated mine water inflow and operational water and would consist of two centrifugal pumps rated at 100l/s each (360m^3/hr, 50% more than estimated inflow) as a duty and standby configuration pumping through permanently installed ranges in the BS shaft. Two of the 250mm diameter ranges used for the submersible operation can be installed permanently on the shaft walls to avoid purchasing additional ranges whilst the third submersible pump manages the mine water make in the shaft during the installation period.

All operational mine water make and mine water inflow at the working areas will be managed by localised sump arrangements and nuisance water pumping equipment which can be installed and relocated relatively easily. The nuisance water pumps (electrical/pneumatic) will pump in series from various operational levels/locations to the pump station where the mine water can be pumped to the surface.

Nuisance water pumps (electrical/pneumatic) should also be utilized to pump water from the main incline during development to a surface settlement pond at the Industrial Site 1 prior to pumping the water to the tailings facility for settlement and recycling of the water.

7.3 Mine Water Pumping

The Blötberget Mine pumping system should comply with best practice and flow velocities are required to ensure that suspended solids remain in suspension i.e. 0.7m/s or above. However flow velocities should be maintained below 3.0 m/s to avoid pipe/joint corrosion/erosion. Table 7-1 indicates the calculated pump flows for the proposed pumping system and indicates the system complies with best practice.

Table 7-1 Shaft Pumping Velocity

Blötberget Mine Drainage Pumps 3 x 100l/s x 250mm diameter ranges						
Flow velocities are required to ensure that suspended solids remain in suspension i.e. 0.7 m/s or above. However flow velocities are required to be maintained below 3.0 m/s to avoid pipe/joint corrosion/erosion.						
	m^3/hr	l/sec	m	m^2	m^3/sec	m/s
	360	100				
Total Capacity	360	100				
Pipe Radius			0.125			

Blötberget Mine Drainage Pumps 3 x 100l/s x 250mm diameter ranges						
Flow velocities are required to ensure that suspended solids remain in suspension i.e. 0.7 m/s or above. However flow velocities are required to be maintained below 3.0 m/s to avoid pipe/joint corrosion/erosion.						
Static Head			354			
Mine Pumps						
Q=V/A, (were V=Velocity (m/s), q=Quantity (m³), A=Area (m²)						
Velocity						2.04
Quantity					0.1	
Area				0.0491		

The pump power requirements are calculated as follows.

7.3.1 Submersible pumping from -373 to surface -19

Figure 7-1 indicates the calculated power for a submersible or centrifugal pump installation to pump from -373level to the surface at 100l/sec.

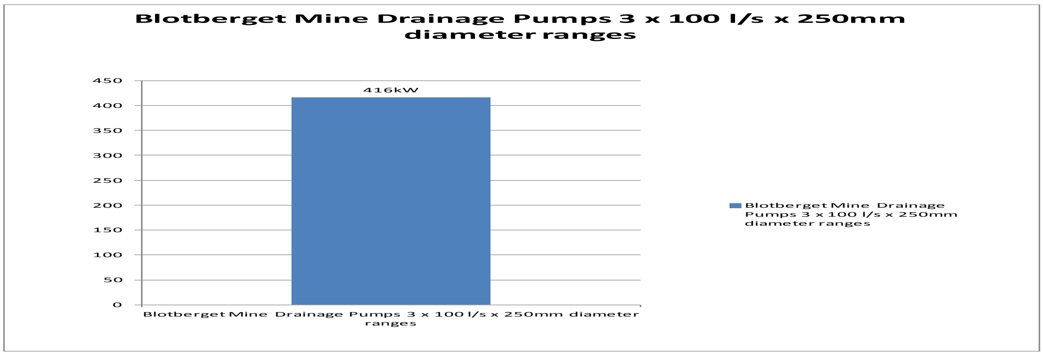


Figure 7-1 Power requirement for Submersible Pump at Blötberget Mine

The proposed pumping system could potentially utilize two of the three submersible pumps and ranges installed to dewater the mine at a combined rate of 200l/s. In the event of excess mine water the third pump could be utilized to overcome the mine water inflow and pump at the maximum permitted rate of 300l/s.

7.3.2 Industrial Mine Water

A permanent surface pond (400m^3) located within the processing plant area will receive recycled water pumped from the tailings clarification pond (required as a surge facility) and should be constructed to provide the requisite make up water for the processing plant estimated at 50l/s (180m^3/hr) approximately and provide a maximum of 25l/s (90m^3/hr) supply for underground operations. The underground supply will also provide the requisite fire-fighting supply along the conveyor installation. Mine water can also be pumped from the BS clarification ponds to the TMF as make up water as indicated in the schematic below.

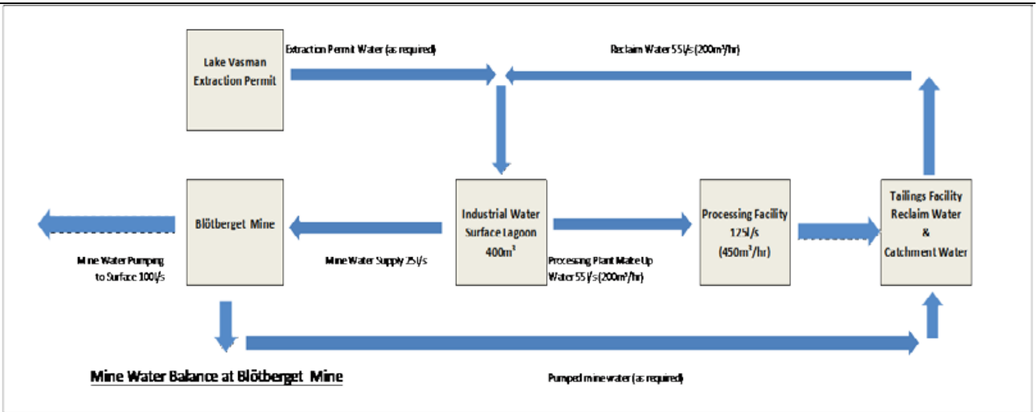


Figure 7-2 Provisional Water Balance at Blötberget Mine

7.3.3 Treatment of mining water

Operational water from dewatering the mine contains suspended particles, residues from explosives and may also contain pollution from machines and vehicles deployed in the mining operation. The clarification ponds will help remove any contaminants from the mine water

The pumped mine water will be discharged into one of two clarification ponds which will be constructed close to the Blötberget shaft on the surface. The primary and secondary sedimentation ponds should be inter-connected by pipe ranges and each measure 75 x 10 x 2 metres (minimum) providing approximately 1.4 hours storage at a maximum pumping rate of 300l/s. Mine water from the sedimentation ponds can be pumped to the clarification pond at the tailings facility to make up the water requirement or can be discharged from the surface sedimentation pond to a part of River Gonäsån that will have a much reduced flow compared to present flow due to the proposed re-routing.

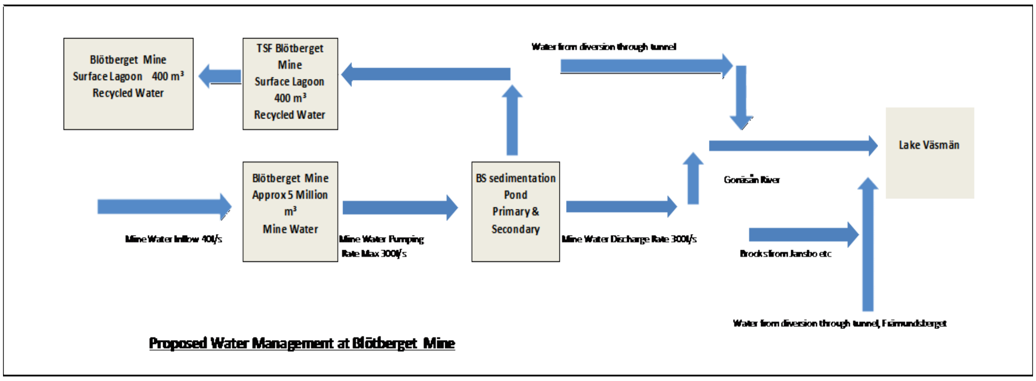


Figure 7-3 Mine Water Management at Blötberget Mine

7.3.4 Initial water treatment

Water from dewatering the mine may contain suspended particles, residues from explosives and may also contain pollution from machines and vehicles deployed in the mining operation.

The pumped mine water will be discharged into one of two clarification ponds which will be constructed close to the Blötberget shaft. The primary and secondary sedimentation ponds should be inter-connected by pipe ranges and each measure 75 x 10 x 2 metres (minimum) providing approximately 1.4 hours storage at a maximum pumping rate of 300l/s. Mine water discharge from the surface clarification pond will be discharged to a part of River Gonäsån that will have a much reduced flow compared to present due to the re-routing.

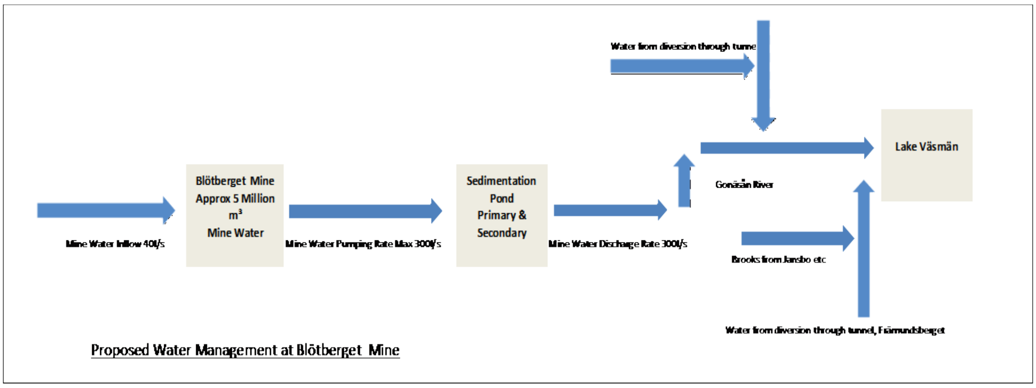


Figure 7-4 Proposed Mine Dewatering Management at Blötberget Mine

7.3.5 Permanent Mine Pumping Scheme

When the mine has been pumped clear of water below the -373 level a permanent pumping station can be constructed underground. The above power table also indicates the power requirement for a duty standby arrangement of centrifugal pumps located at the permanent pumping station underground (-373) to control and pump the operational water make and predicted mine water inflow to the surface at 100l/s (50% greater than estimated).

It may also be necessary to pump mine water from the surface settlement pond facility at the Blötberget shaft to the tailings dam facility as required to supplement the makeup water required for the mining and processing operations.

7.4 Minerals Handling

7.4.1 Introduction

The materials handling system for transporting the ROM production from Blötberget underground mine to the surface will consist of a series of conveyors in association with one or two primary crushers. This is shown schematically in Figure 7-5.

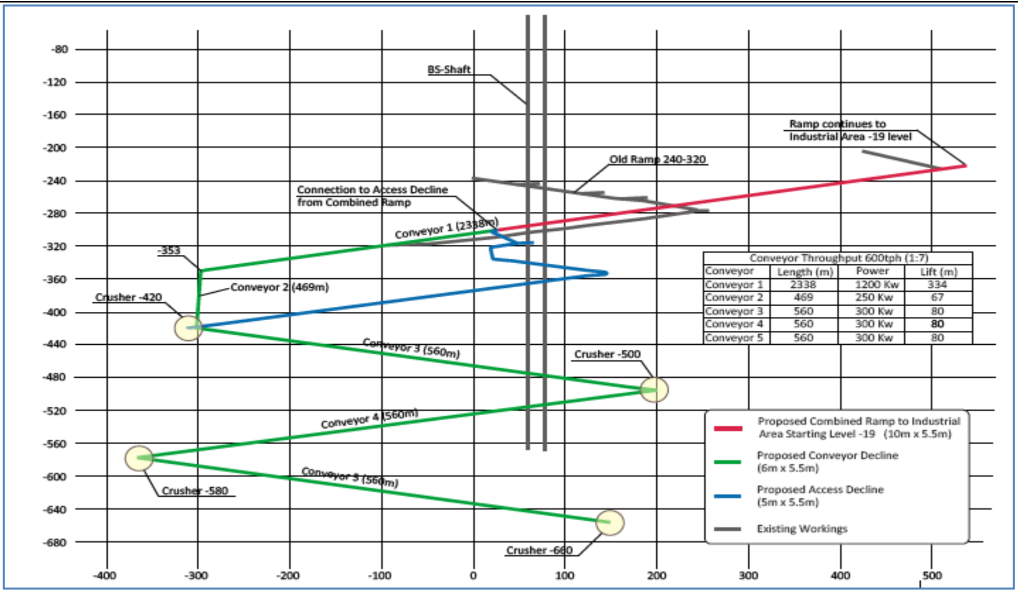


Figure 7-5: Blötberget Conveyor/Access Schematic Plan

The design capacity of the Blötberget main decline conveyor system will be required to convey a combined annual throughput from the underground mine of 3,300,000 tonnes per year (3,000,000 tonnes of Ore + 300,000 tonnes of waste). The proposed conveyor configuration for the underground system will utilise conventional troughed belt installations as straight conveyor flight arrangements for Phase1 of the mining operation and will be installed in the respective declines. The conveying systems are outlined below.

Table 7-2 Materials handling equipment

Underground Infrastructure Requirements			
Description	**Capacity**	**Amount**	**Comments**
Surface Conveyors	690 ton/hr	2	ROM ore and waste rock conveyors
Main Surface Decline Conveyor	690 ton/hr	1	2338m conveyor length at 1:7 inclination
UG Conveyors	690 ton/hr	4	1 conveyor required per crusher level
Crushing & Belt Conveyor Loading	690 ton/hr	45 m3 Hopper	Potentially C140 Jaw Crusher

The conveying system will be extended in-line with production operations and will eventually include four (ramp) conveyors down to -660m level.

7.4.2 Surface Conveyors

The production ROM from the underground operations will be conveyed by the main incline conveyor and delivered onto either,

1. A waste conveyor arrangement which will deliver the waste rock to a spoil heap located close to Industrial site 2 where the waste will be managed and utilized as required, or

2. Directly into a secondary cone crusher which will crush the ROM ore <70mm and will convey the ore to a storage silo for feeding into the processing plant. An additional ore stocking ground will be provided close to the silo as an emergency stockpile facility to ensure continuity of processing. Any ore overspill from the silo will be discharged onto the ground via a bypass chute and transported to the stockpile by front end loader (FEL). A reclaim facility will allow the ROM ore from the emergency stockpile to be loaded onto the processing plant in feed conveyor at the required rate.

7.4.2.1 Waste conveyor

The surface conveyor receiving section will be located directly below the delivery of the main incline conveyor and will convey the underground waste overland approximately 690m to the waste disposal area. The overland conveyor will need to be rated at 690 tph to match the main incline conveyor as there is no inline Bunkerage to facilitate peak loads.

The underground waste will be conveyed separately to the iron ore potentially on one shift of the daily working shifts and hoisting will be in batches between ore and waste with process controls to manage the execution.

When conveying the waste, a two way chute will divert the mineral away from the secondary cone crusher into a discharge chute onto the waste conveyor. At the delivery end of the conveyor, the waste will be discharged onto the ground where front end loaders (FELs) and trucks will load and transport the material to the appropriate area. The waste material may be utilised as infill for the civil works; however this will be undertaken by a contract firm and is not covered in this current scope of work. The figure below shows a schematic proposal for the overland conveyor to convey the waste mineral from the portal delivery to the waste stocking ground.

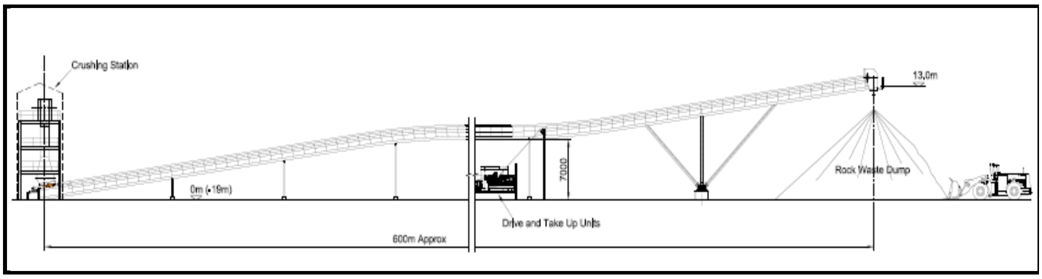


Figure 7-6 Proposed overland waste conveyor

7.4.2.2 5,000 Tonne Silo feed conveyor

The surface silo feed conveyor will be located directly below the delivery the secondary cone crusher. The crusher will deliver the ROM ore <70mm onto the conveyor and then into the surface silo of 5,000 tonne capacity. A plate feeder located beneath the silo will control the ROM feed onto the processing plant feed conveyor at a design feed rate of 416 tonnes/hr. The silo feed conveyor will need to be rated at 690 tph to match the main incline conveyor delivery.

When the silo has reached its capacity of 5,000 tonnes ROM, a traverse chute will operate and divert the ROM product down a chute to the ground overspill area. Front end loaders (FELs) will transfer the overspill onto the emergency stockpile pad (10,000 tonnes) where it can be stockpiled until it is required to be reclaimed for feeding into the processing plant. FELs will reclaim the product and deliver it onto the process plant feed belt via a hopper and feeder arrangement delivering the ROM ore at the required rate. The figure below shows a schematic proposal for the silo feed conveyor to convey the ROM mineral from the portal delivery into the silo.

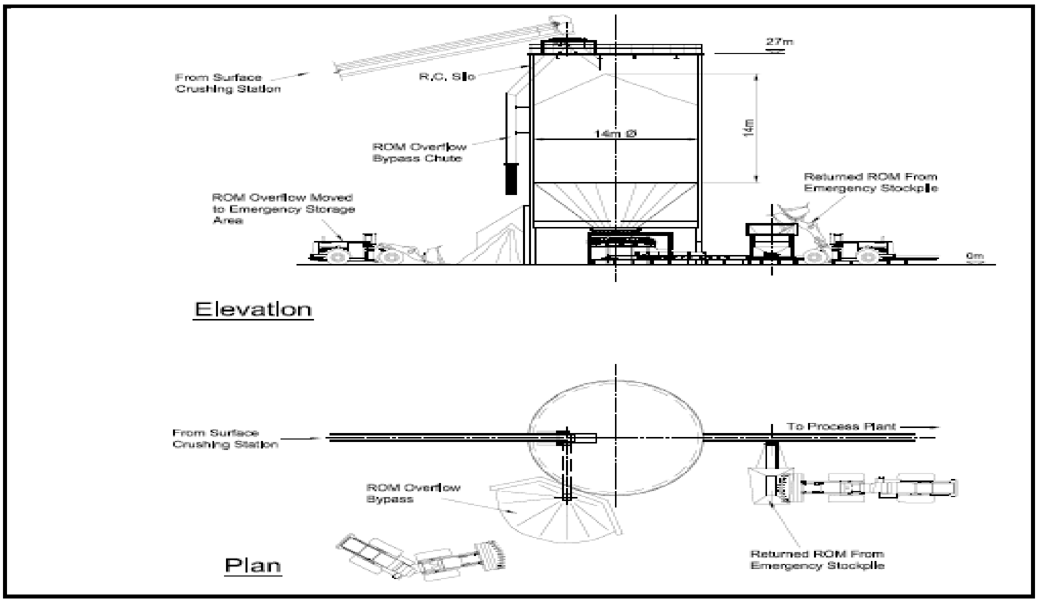


Figure 7-7 Proposed 5,000t ROM Silo feed conveyor

7.4.3 Main Incline Conveyor

The main surface decline conveyor will be operating from the surface -19 level down to the -353 level and will be installed after the decline has been developed and completed at a grade of 1:7 using drill and blast methods.

The conveyor will be approximately 2,338m long with a design throughput capacity of 690 tph and an installed power of 2 x 600kW discharging the <250mm ROM ore into the 2nd cone crusher. The crusher capacity will be sufficient to cope with the feed rate from the underground ROM conveyor with some of the ROM screened off before passing through the crusher.

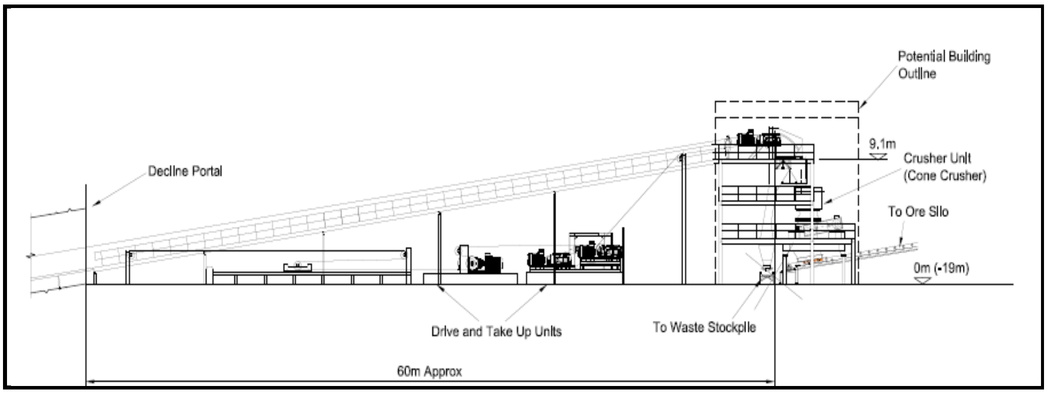


Figure 7-8 Main Incline Conveyor Surface Discharge

7.4.4 Underground Crushing Facilities

It is planned to locate a primary crusher (typically Nordberg C140) facility underground at the 420 level to receive the ore and waste flow from drawpoints into ore passes and onto trucks and crush to <250mm prior to conveying to the surface. The crusher facilities will be re-installed at lower levels as the mine operations progress. The current mine plan identifies the future lower levels for the crushing operations as -500, -580 & -660 Figure 7-9 Underground Crusher and Conveyor Discharge

The initial primary crusher facility will be installed at the 420 m level to an excavated volume of 3,500m ^{3} to allow the installation of the equipment and the conveyor arrangement to operate. The ROM material will be delivered into a hopper arrangement by conventional trucks, front end loaders (FELs) or LHD from the ore piles at the base of the ore passes or the draw points. When this level of the mine has been mined out the crusher facility will be relocated to the next level -500 m and then at -580 m and -660 m respectively. Similar loading arrangements and conveyors will be utilised at all the installations (Drawing C22-002 in Appendix C).

The in-feed hopper for the crusher is designed to provide adequate capacity to allow rapid unloading of vehicles as a continuous process. The feed rate from the apron feeders installed below each bunker will be regulated to prevent conveyor overloads.

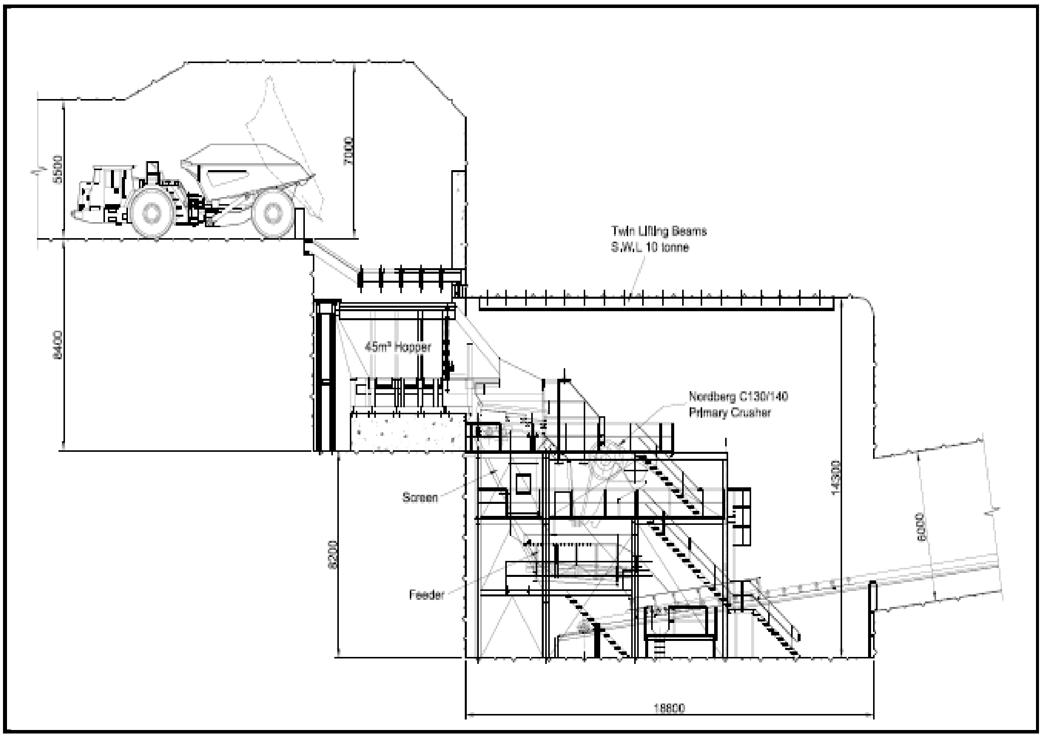


Figure 7-9 Underground Crusher and Conveyor Discharge

7.4.5 Conveyors Installations

As shown schematically in Figure 7-5, a series of belt conveyors will be installed throughout the life of mine to convey the crushed ROM from the crushing facility to the main incline conveyor.

The conveyors are all rated at 690 tph and installed in series and synchronized to ensure that continuous mineral conveying is maintained. Ore and waste will be conveyed campaign wise and separated at the surface before the 2nd crusher. Crushing and conveying of ore and waste will be controlled by process flow (material flow) management systems from draw point to ore pass to crusher.

7.5 Services

All necessary services for the underground operations such as water, power and compressed air will be clearly shown on a schematic mine plan in Appendix C.

The mine plan also identifies all services required for the mining operations such as ventilation.

Emergency areas and services such as meeting areas, fire fighting and evacuation points shall also be clearly identified on the mine plan.

7.5.1 Mine water supply

Industrial water for mining operations, dust suppression and fire fighting will be supplied by steel ranges located in the designated roadways and shown on the roadway cross section. The requirements for the mine industrial water supply is based on the mining plan which will produce 3,300,000 tonnes of ROM per annum with the planned number of working locations and respective requirement for process water. Table 7-3 below outlines the main operations that will require an industrial water supply;

1. Underground maintenance areas;
2. Mobile mining fleet using water;

Based on the above operations the process water requirements have been determined at 22l/s maximum and this will be supplied via a 100mm diameter pipe range in the main incline. This underground water feed will also be utilised to provide a fire fighting supply for the planned conveyor installations.

Table 7-3 Underground Industrial Water

Underground Industrial Water Requirements				
Equipment/Utility	Number	l/s	Total Quantity l/s	Comments
UG Conveyors Dust Suppression	5	2.5	12.5	Maximum of 5 conveyors required for LOM
Drill Rigs	8	1.1	9	Only during

Underground Industrial Water Requirements				
Equipment/Utility	Number	l/s	Total Quantity l/s	Comments
				drilling operations
Longhole drill rig	4	3	12	Only during drilling operations
Shotcrete Machine	2	0.5	1	As required
Maintenance Area & Miscellaneous	2	3	6	Workshops various activities vehicle wash down etc
Total			**41**	

No supply has been considered for potable (drinking) water in the above estimate as it is assumed this will be supplied from connection to the municipality water supply.

7.5.2 Underground Compressed Air Supply System

Some of the machines used in the mining operations necessitate compressed air to operate and others such as tyred truck vehicles will require compressed air for servicing in underground workshops. All compressed air requirements will be supplied by a number of mobile air compressors which will be located and installed and commissioned where necessary.

7.5.3 Fuel Storage and Distribution System

There will be a large fleet of diesel vehicles operating in the mine . It is planned to store and distribute the diesel fuel underground during mining operations. The diesel fuel will be transported underground in fuel trucks and transferred to refuelling stations installed at specific locations close to the maintenance area and mining activities

The locations of the diesel fuel filling stations will fulfil all necessary regulations for health and safety aspects.

It is recommended that initially two diesel storage tanks are employed on the surface at a permanent location but are portable and can be relocated if required. The surface tanks will be used mainly to provide fuel for vehicles operated or serviced on the surface and vehicles that regularly come out of the mine. The tanks on the surface will be refilled by fuel trucks as required. A specific road tanker operated by the mine should be utilised to supply and replenish the underground diesel filling stations as required.

The majority of underground operational vehicles are intended to be maintained and serviced underground in the maintenance workshop. The vehicles will be fuelled from the underground filling stations . Major repairs and maintenance will be carried out on the surface in the vehicle workshop.

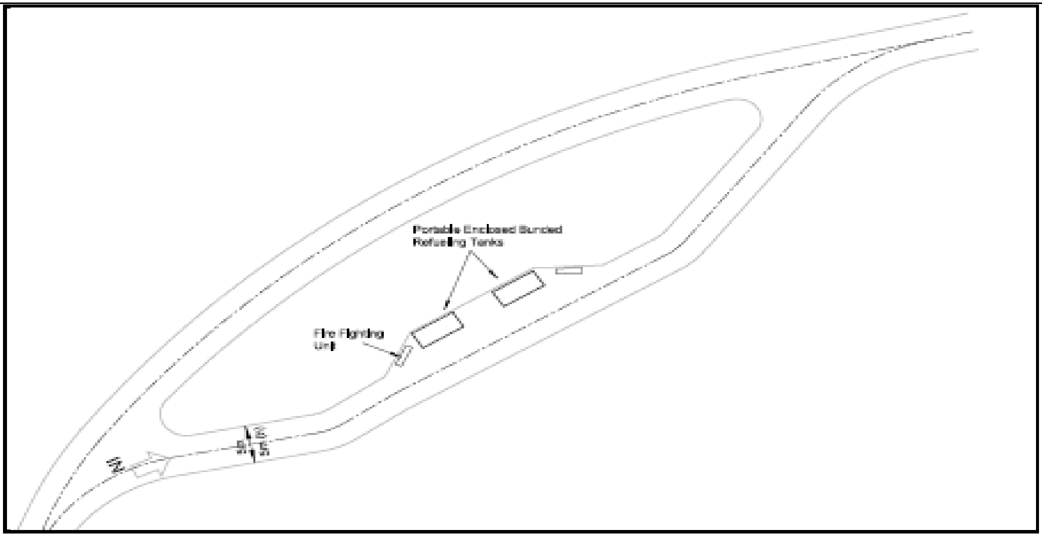


Figure 7-10 Underground Fuel Filling Station

7.5.4 Underground Workshops, Stores and Meeting Places

The underground operations will necessitate the construction of workshops, stores and meeting places to allow the mining operations to be carried out efficiently and safely. It is proposed that a maintenance area will be situated at the 320 level, however, the location of the proposed maintenance area location will be confirmed when the mine plan is confirmed.

The facility will be the main maintenance workshop for the mining fleet with smaller satellite facilities providing localised maintenance. The final location and layout for all the maintenance area's will be confirmed by the mine management team based on the operational requirements that will best facilitate the mining operations.

The workshop area has been designed to facilitate the maintenance of the mining fleet underground as necessary and includes designated areas with sufficient clearance to allow safe working on the vehicles as well as safe passage for pedestrians in the immediate vicinity. The main workshop facility is designed to facilitate two trucks and two auxiliary vehicles at any one time. See Figure 7-11.

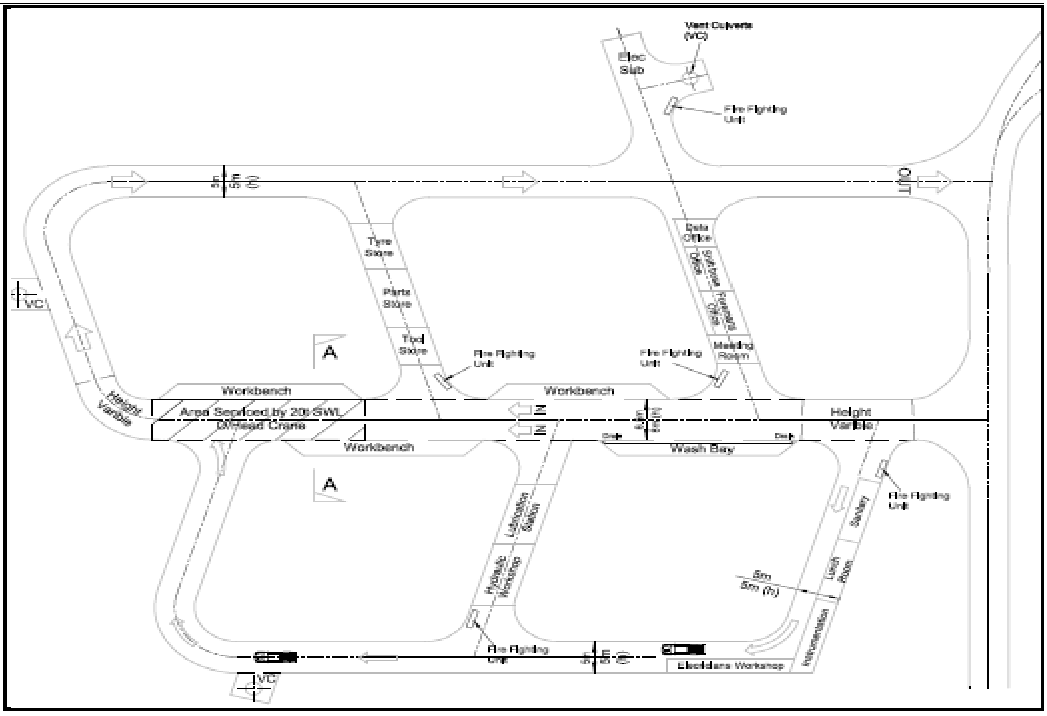


Figure 7-11 Conceptual Underground Maintenance Complex Layout

The maintenance area will have designated sections for the logistical requirements such as offices, workshops, stores facilities and also welfare amenities for the workforce. It is also planned to locate the fuel filling depots for the mine fleet close to the maintenance facilities. Other infrastructure logistics such as compressed air, water supply for wash down bays and pumping facilities will also be incorporated into the final layout design. A high priority for the designated facility will be adequate lighting throughout to ensure that persons working, passing or resting in the location are able to see all activities that are being carried out at any time and remain safe.

The maintenance layout has also taken into consideration additional services that would be required such as vehicle wash down facilities, overhead crane installation, tyre changing bays, lubrication services, workshops and hot work areas. These are identified as specific locations to ensure that when in use they do not conflict with other operational activities. Ramp access and vehicle testing areas are also designed in the maintenance area.

There will be potentially a high volume of traffic utilising the facilities, therefore it is proposed that the route for vehicles is operated in a one way direction to alleviate any potential for vehicles to collide by travelling in the opposite direction.

Oil bays will be located near production levels as the mine is deepened and the distance to the workshop increases.

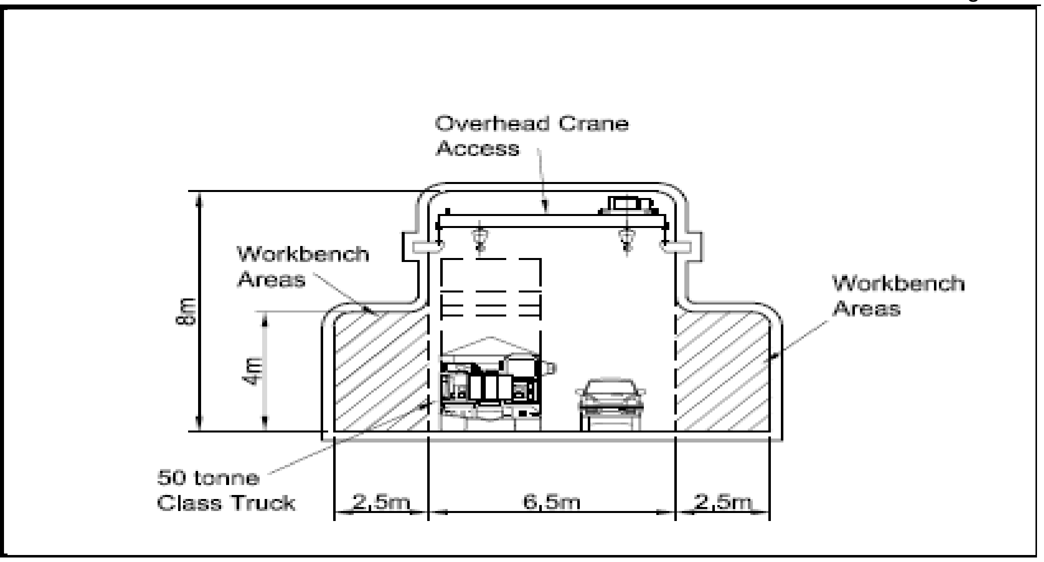


Figure 7-12 Proposed Lifting Facilities for Maintenance Complex

A typical equipment list to populate the workshops, work areas and offices is shown in Table 7-4. Although comprehensive it is not exhaustive and is intended for guidance whilst representing the majority of equipment that would be necessary to carry out the required maintenance tasks.

Table 7-4 Typical Workshop Equipment requirement

Truck Workshop	
10 tonne Overhead Crane	1
Workshop Compressor	1
Pillar Drill	1
Pedestal Grinding Machine	1
Power Saw	1
Battery Charger/Starter	1
Bench Grinder	1
Injector Test Rig	1
Tyre Changing - main workshop	
Tyre levers	2
Flogging spanners	2
Tyre inflation gear	2
Tool Kit - Snap on Type	1
Messing and Safety	
Office Furniture & equipment	1
Safety Equipment	1
First aid station, fully fitted	1
Clean Room for Rebuilding components	
Air con	1
Benches, vices, etc	1

Truck Workshop	
20' Container	1
Plastic curtain	1
Tool Kit Snap On Type	1

7.6 Underground Electrical, Communications and Control

7.6.1 Introduction

The Mine transmission and distribution network will be derived from a 50kV overhead power line that is currently located on the project site; the mine site will be powered via a single 50kV/12kV transformer rated at 40MW feeding a main substation (STV1) at Industrial Area No.1.

The underground power network will utilise a main power grid rated at 12kV, 50Hz with the main supply transformed down to a variety of supply voltages for use at the working and remote areas of the mine, using approved mine-type transformers; identified voltages include 3.3kV, 1000V, 415V and 240V. The final underground system comprises of three main 12kV substations and a series of auxiliary substations, operating on a 3phase 50Hz system.

The underground complex will be supplied at 12kV using two separate feed cables from Surface Incoming Substation STV1, both routed via the Conveyor roadway portal at No.1 Industrial Area. The cables will be of single wire armoured (XLPE) construction, rated at 12kV each with sufficient capacity to meet the full load requirements of the mine; based on current plans, expected to be approximately 325A, this being sufficient for the planned development, ventilation, mine waste water pumping requirements, ROM conveying and planned production operations in concurrent working areas.

The supply cables from the Main Surface Substation (STV1) will be terminated onto a suitable 12kV switch-board in the Underground Substation (SS1), located adjacent to the surface portal of the Conveyor decline; the switchboard will consist of two incoming switches and five outgoing switches (drawing C22-023). The use of two supply cables, with each cable able to cater for the full expected underground operational load, provides security of supply to the underground mine.

A third 12kV feed, supplied from BS Shaft Substation STV3 will supply the temporary shaft de-watering system at Blötberget Shaft (Surface);; the feed will supply a 12kV/3.3kV transformer rated at 1500kVA. The supply will be removed once the water level has been pumped below -373m level and a permanent underground pumping system has been installed and commissioned at this level.

The 12 kV supply is utilised to feed the underground H.V. switchgear; system voltage is then transformed down to 3300V/1000V/660/415V to be utilised in and around the mine complex as distribution voltages.

Substation switchgear details, i.e. type, switch ratings, control & monitoring details, earthing arrangements, etc and outgoing cable details, i.e. size, type, lengths, gland arrangements, etc, will be specified in detail during the detailed engineering studies.

The mines underground operating load includes large capacity conveyor drives, which will operate on a 3.3kV system and smaller capacity equipment operating on 1000V/660/415V systems, as required.

7.6.2 Power Requirements during the Development of the mine

An initial, temporary power network will be utilised during development activities with the operational power network being installed when the development reaches -310m area.

The operational power network will be supplied using a twin-cable system to feed all main substations. The use of two supply cables, with each cable able to cater for the full operational load of its section, provides security of supply to the underground mine.

The permanent network will be expanded once production operations reach -420m, -500m, -580m and -660m levels however, the H.V. substations installed in -420m and -580m levels will be temporary substations; installed and used during the production operations in these areas but removed once production has ceased.

A series of schematic drawings for each phase of the mine development/production are available in Appendix C.

7.6.2.1 Mine Development – Stage One

The following work will be undertaken during the Mine development:

- Commence de-watering of the Blötberget Shaft to -373m level;
- Construction of the Main Decline roadway to the -353m level.

The Underground Substation (SS1) supplying the Main decline development will be located at Industrial Area No.1, in a suitable location close to the conveyor roadway portal. As noted above, the heading substation will be supplied from the Main Surface Substation STV1 using two, 120mm^2 single wire armoured (XLPE) cables rated at 12kV.

The 12kV substation will consist of two incoming switches, a bus section and five outgoing switches. The duties of the outgoing switch are as follows.

- No.1: Supply to local 500kva transformer feeding L.V equipment (nuisance water pumps, lighting, etc);
- No.2: Supply to Main Decline heading transformer rated at 500kva;
- No.3: Spare;
- No.4: Spare; and
- No.5: Supply to Main Decline Ventilation transformer rated at 500kva.

The heading transformer/switchgear will initially be installed at the surface in the vicinity of the decline entrance, until the roadway construction has advance sufficiently to allow the transformer and switchgear to be moved into the roadway itself. Once inside the decline, the transformer and switchgear will be advanced in accordance with the heading development.

Outbye waste water stage pumps will be installed in the decline during the development phase as and when required depending on quantity of waste water experienced, the initial pumps being powered from the local supply at the portal entrance.

7.6.2.2 Mine Development / Initial Production Operations – Stage Two

The Main Decline roadway will have been constructed to a depth suitable for accessing the Sandel orebody at -230m level and the Decline roadway construction will continue to -310m level using the existing heading supply arrangement.

A second power source (UG Feed 2), from Underground Substation SS1, will be installed for the mining development/production activities in the Sandel ore block; this feed will be utilised as the Sandel power source during the initial development of the Main Decline roadway to -353m level. Following the completion of the -310m level Substation (UG 1), the cable will be rerouted from the Sandel area and utilised to provide the second incoming feed to substation (UG1), with Sandel power being provided by a new feed from -310m substation, if mining activities in this area are still ongoing.

Power requirements in the Sandel block will be provided via a 750kva rated transformer.

7.6.2.3 Mine Development / Initial Production Operations – Stage Three

The de-watering of the Blötberget Shaft to -373m level, via the pipe range installed in the shaft, will be complete. A local submersible pump will be utilised to ensure the water level is maintained below -373m level and a permanent pump station will be constructed to pump underground process and ground water to the Surface. The power supply to the redundant Shaft dewatering system (Surface) will be removed from the Blötberget shaft substation. Nuisance water pumps installed in the decline during the development will be removed and the water routed to the permanent sump at -373m level.

The construction of the Main Decline roadway will be completed to -373m level and the main underground ventilation system will be installed at this point (370m level).

The outgoing switches at the Surface/Underground Substation (SS1) will be modified as follows:

- No.1: Supply to local 500kva transformer (local supplies);
- No.2: Supply to UG1 Substation (-310m);
- No.3: Supply to UG1 Substation (-310m);

- No.4: Spare; and
- No.5: Spare.

A new, permanent Underground Substation (UG.1) will be installed at -310m level. The new substation will be supplied using the two, 120mm2, single wired armoured cables both installed in the Main Decline roadway, one previously used for the Decline roadway development activities and the second previously used for the Sandel Ore block development.

The new No.1 Underground Substation will consist of two incoming switches, bus-connector and eight outgoing switches; these will be configured as follows:

- Spare;
- Spare;
- H.V. supply to 12/1.0kV Transformer rated at 500kVa for Access road development (-310m to -420m level);
- H.V. supply to 12/1.0kV Transformer rated at 500kVa for Decline road development (-310m to -420m level);
- H.V. supply to -373m Pump Lodge Substation;
- H.V. supply to local transformer feeding Workshops/Local supplies;
- H.V. supply to Development Ventilation Fans;
- H.V. supply to Sandel Orebody.

The substation will provide power to the production areas, the Decline & Access development roadways (down to -660m level), the underground Main Ventilation System, the Main Pump Lodges at -373m level, No.3 & No.5 substations at -500m & -660m level respectively and the auxiliary substations for ramp conveyors and local supplies.

The two H.V. cables feeding the development headings to -420m level (Access and Decline roadways) will be utilised to power No.2 underground substation (UG2). The development declines to -500m level will then be powered from H.V. switchgear in UG2 substation.

7.6.3 POWER REQUIREMENTS DURING PRODUCTION OPERATIONS

7.6.3.1 Production Operations – Stage Four

The construction of the Main Decline roadway will be completed to -420m level, a ventilation connection to Blötberget Shaft, is available at 370 level and an Access roadway connection will be constructed between -310m and -420m level.

No.1 & No.2 Conveyor system will be installed and commissioned ready for use, providing a Run-of-Mine (ROM) conveying system from -420m level to the Surface.

The Crusher system at -420m level will be installed, commissioned and available for use, providing a crushing facility for the underground production system.

Underground development activities will continue with construction in both the Access and Conveyor declines between -420m and -500m levels.

The Surface substation (SS1) will be modified with the H.V. switchgear allocated the following duties.

- No.1: H.V. supply to Surface Stockpile Conveyors;
- No.2: No.1 H.V. supply to Main UG (UG1) Substation;
- No.3: No.2 H.V. supply to Main UG (UG1) Substation;
- No.4: Spare;
- No.5: Supply to No.1 ROM Conveyor.

The Substation (UG1) at -310m level will be modified with the H.V. switchgear allocated the following duties.

- No.1: H.V. supply to Main Underground Fans;
- No.2: H.V. supply to Conveyor Substation at -353m level;
- No.3: No.1 H.V. supply to -420m Temporary Substation (UG2), via Access decline roadway;
- No.4: No.2 H.V. supply to -420m Temporary Substation (UG2), via Conveyor decline roadway;
- No.5: H.V. supply to Main Pump Lodge (-373m level);
- No.6: Workshops/Local supplies
- No.7: H.V. supply to heading fans; and
- No.8: H.V. supply to Sandel Production area.

The temporary substation (UG2) installed at -420m level will provide power for production operations between -420m to -500m levels; this substation will be removed once the main substation (UG3) has been installed at -500m level and will then be re-used in -580m level. The substation (UG2) will utilise two incoming switches, one bus-coupler and four outgoing switches; outgoing switchgear will be allocated the following duties.

- No.1: H.V. supply to Huggett Production Area;
- No.2: spare;
- No.3: spare;
- No.4: H.V. supply to Crusher Substation (-420m level).

7.6.3.2 Production Operations – Stage Five

The construction of the Conveyor and Access decline will be completed to -500m level. The second crusher system will be installed at -500m level and No.3 Conveyor drive at -420m level will be installed to complete the ROM conveying system between 420 and 500 levels; the Crusher system in -420m level will be removed.

Underground development will continue with construction in both the Access and Conveyor decline roads between -500m and -580m levels, with the equipment fed from UG3 Substation.

The substation (UG3) installed at -500m level will provide power for production operations between -500m to -580m levels. The substation will utilise two incoming switches, one bus-coupler and four outgoing switches; outgoing switchgear will be allocated the following duties.

- No.1: H.V. supply to Production Area No.1;
- No.2: H.V. supply to Heading fans;
- No.3: H.V. supply to Production Area No.2;
- No.5: H.V. supply to Crusher Substation (-500m level).

The substation (UG3) will be supplied from UG1 Substation in -310m level using two, 70mm^2 single wired armoured cables, one installed in each of the Access and Conveyor decline.

Production operations in the Sandel orebody will be complete by this stage and the supply from UG1 Substation will have been isolated and removed.

7.6.3.3 Production Operations – Stage Six

The construction of the Conveyor and Access decline roadways will be completed to -580m level. The Crusher system will be installed at -580m level and No.4 Conveyor drive will be installed at -500m level to complete the ROM transport system; the Crusher system in -500m level will be removed.

Underground development will continue with construction in both the Access and Conveyor declines between -580m and -660m levels using production equipment, when production activities allow.

The substation (UG4) installed at -580m level will provide power for production operations in and around -580m to -660m levels; this substation will have been previously utilised as UG2 Substation at -420m level. The substation will utilise two incoming switches, one bus-coupler and four outgoing switches; outgoing switchgear will be allocated the following duties.

- No.1: H.V. supply to Production Area No.1;
- No.2: H.V. supply Heading fans;
- No.3: H.V. supply Production Area No.2;
- No.4: H.V. supply to Crusher Substation (-580m level).

The substation will be supplied from UG3 Substation in -500m level using two H.V. cables, one installed in each of the Access and Conveyor declines.

7.6.3.4 Production Operations – Stage Seven

The construction of the Conveyor and Access declines will be completed to -660m level. The Crusher system will be installed at -660m level and No.5 Conveyor drive will be installed at -580m level to complete the ROM conveying system; the Crusher system in -580m level will be removed.

The temporary substation (UG4) installed at -580m level will be removed from use.

Substation UG5 will utilise two incoming switches, one bus-coupler and four outgoing switches; outgoing switchgear will be allocated the following duties.

- No.1: H.V. supply to Crusher installation (-660m level)
- No.2: H.V. supply to Production Area No.1;
- No.3: H.V. supply to Production Area No.2;
- No.4: Local L.V. supplies.

The substation will be supplied from UG3 Substation in -500m level using two, 50mm^2 single wire armoured cables installed in the Access and Conveyor decline roadways.

Conveyor No.5 will be supplied from Substation UG3 in -500m level.

7.6.4 Power Cables

Suitable armoured insulated power cables will be employed for the transmission and distribution of electric power around the underground workings, installed as permanent wiring within mine infrastructure facilities. The cables will be installed in purpose-made ducts in fixed installations, run on overhead cable racks or purpose designed hangers of mine service tunnels.

The specified 12kV electric power cables will be in compliance with International Regulations, and certified for use. The cables will have an outer armoured sheath, will use stranded copper conductors, will be covered in adequate insulation and will be suitable for the designated use in the electrical network.

The underground power network will be supplied using a twin-cable system to feed all main substations. The use of two supply cables, with each cable able to cater for the full operational load of its section, provides security of supply to the underground mine. The following sized cables are recommended:

- Surface Substation (STV1) to Surface/Underground Substation (SS1): 120mm^2
- Surface/Underground Substation (SS1) to Substation (UG1) 120mm^2
- Substation (UG1) to Substation (UG3): 70mm^2
- Substation (UG3) to Substation (UG5): 50mm^2

Flexible power cables will be utilised to supply mine machinery and other mobile devices, tools etc, where applications require a cable to be moved repeatedly, i.e. drill/bolting machines, submersible pumps, etc. These cables are generally referred to as “trailing cables" and are specifically designed and constructed to be used with moving mine equipment.

Trailing and flexible cables will contain fine stranded conductors, not solid core conductors, and have insulation and sheaths to withstand the forces of repeated flexing and abrasion; these cables will also comply with international mining standards for use in mines.

7.6.5 Main Supply Cable Routes

Main supply cables for the underground power system, monitoring and communication cables will all be routed through the Conveyor and Access declines.

Initial power for the shaft submersible pumps will be obtained from the local Surface substation at the Blötberget shaft site with the flexible pump cable of the submersible pumps secured within the shaft, as necessary.

A new underground substation will be installed at -310m level following the completion of the Conveyor decline (to -353m level). This will provide power for the main underground pump station at -373m inset and the submersible pump system dewatering the shaft below this level. Once installed, the submersible pumps will be isolated and the cables will be removed from the shaft.

7.6.6 Mine Water Pumping Power Requirements

The initial shaft drainage pumping system will utilise three, 415Kw, 3.3Kv rated submersible pumps and 205 mm ranges to remove water to a level just below the -373m level; these will be powered from the Blötberget shaft substation via a 12kV/3.3kV transformer, rated at 1500kVa.

Once water has been cleared to a level below -373m, a permanent pump station using two, 300kW rated pumps, pumping directly to the Surface, will be installed at the inset of -373m and mine drainage water below -373m level will be pumped up to the permanent pumping lodge using staged pumps. Power will be removed from the Surface pumping system following the successful commissioning of the underground pumping system at -373m level.

A submersible pump (circa 50kW) will also be installed in the shaft at -373m shaft inset to drain waste water from the shaft into the new pump lodge, maintaining the water level below -373m.

The permanent pump station in -373m inset will be powered from No.1 underground substation (UG1) at -310m level, via a local 12kV/1.0kV transformer rated at 750kVa.

All pump systems will be automated and monitored to the Surface.

The original power system for the three, 415kW submersible pumps used during the initial shaft dewatering will be removed once the shaft has been emptied and the permanent pumping system has been installed.

7.7 Communication Systems

The Surface main offices will incorporate a server room as the main communications hub of the mine. The system will be connected to the Process Plant, workshops and other buildings of Industrial Area No.1, BS Shaft, Industrial Area No.2 and the underground mine via the Main Decline roadway; each area will be connected using fibre optic cable. The network will be utilised for main communications, administration, production operations, environmental monitoring systems, alarms, radio communication, security systems, access control, etc.

An automatic telephone exchange, will be located at the mine control room and the System will be capable of connection to the National Telephone Network through landlines. The automatic telephone exchange will be capable of providing automatic switching and connection of both surface and process telephones without the attendance of an operator.

Telephones will be installed at operational and safety related locations throughout the surface site, i.e. the process plant, main conveyor drives, offices, workshops, etc. and telephones will be provided throughout the underground mine at all necessary operational and safety related locations, i.e. Muster points, workshops, substations, main pump house, conveyor drives, crusher units, etc.

The underground type mine telephones will be of robust construction suitable for mine use. They will be installed at muster stations, conveyor drives/transfer points, the mine crusher, pumping stations, fan installations, workshops, substations and production units, etc.

Amplifier units will be installed alongside conveyors to provide a pre-start warning and verbal communication facility between the conveyor roadway and Surface and pre-start warning during the conveyor starting process.

The communication system will also include a leaky feeder type radio system and amplifier system (conveyor decline roadways) in addition to the telephone system

7.7.1 Radio System

The underground radio communication system will be designed to meet the present and future requirements of the mine. For example, the system can be based on a single antenna (leaky feeder cable) installed throughout all underground working roadways and linked to the surface. Once the system is installed then mobile two-way radios can be used for voice communications throughout the mine. It is also possible to utilise this type of system for personnel location, traffic management, surface telephone-cellular link, personnel and asset tracking system and data transmission.

An underground leaky feeder radio system is designed to operate with VHF radio frequencies which are transmitted and received in underground roadways and tunnels up to a maximum of 25 km in length. The high frequency radio signals are transmitted and received by use of a radiating cable with 2 way repeater booster amplifiers

The radio telephones will be of a rugged construction with a robust loud speaking transceiver which can be adapted for mounting on mobile vehicles (haulage trucks and mobile mine vehicles).

7.7.2 Monitoring & Control System

The underground monitoring and control system will be designed to allow specific items of mobile and fixed mine equipment to be remotely controlled and monitored from a central control room at the surface of the mine, i.e. conveyors, pumps, etc. It will also allow environmental information to be collected throughout the mine and to be displayed and monitored in the same surface control room.

The system will consist of a central station connected via a high-speed data highway to remote outstations positioned at strategic points throughout the mine including all conveyor drives, crusher installation, Main Fans, Main Pumps, Auxiliary fans, substations, etc. Field devices and transducers will be connected to the appropriate outstation, which will be configured for the specific application requirement.

The outstations will be designed to allow equipment to be controlled and monitored locally if required, without connection to a data transmission system.

The system will facilitate real time monitoring, logging and trending of essential information and allow configurable shift reports to be produced.

The system will be designed to allow for future expansion with additional outstations being added to the transmission system if or when required.

7.8 Centralised Blasting System

A centralized blasting system will be allocated for the underground blasting process.

The blasting system will utilise the existing communication infrastructure (analogue phone networks, LAN, WLAN or Leaky Feeder), at the mine to safely initiate blasts from either a designated Safe Firing Point underground or from a surface coordination and control point.

8 PROCESS PLANT

8.1 Historical Review of Blötberget Process Plant

8.1.1 Introduction

The Blötberget mine is located in an area of Central Sweden renowned for the mining and smelting of iron ore, Bergslagen, and close to the town of Ludvika. There are two other closed mines (Grängesberg and Håksberg) in the close vicinity of Blötberget that recently have been the subject of re-opening studies and these have some relevance to this study.

Besides general interest, the history of these mines is informative in gaining an understanding of the response of the ore to beneficiation and affirming the process design developed later in this chapter. The Grängesberg mine is of particular interest because the ore is considered to be remarkably similar to Blötberget ore and it may be that the Blötberget orebody is an extension of Grängesberg. The Blötberget and Håksberg mines closed in 1979 and Grängesberg mine closed in 1990; however Nordic Iron Ore has been able to access some of the records of the old mining and processing operations.

8.1.2 Blötberget Mine

Amongst the earliest available data for Blötberget are the results of a programme of sampling conducted by Minpro on two dates during 1968.

A description of the processing plant flowsheet at that time is given. The crushed ore was fed onto a screen with apertures of approximately 1 mm and the oversize passed into three (3) rod mills arranged in parallel. The milled ore and screen undersized formed the feed to wet, low intensity magnetic separators which recovered a magnetic concentrate. The non-magnetic tailings created the feed to rougher spiral concentrators which produced a pre-concentrate of hematite and final tailings. The spiral pre-concentrate was cleaned by cleaner spirals to produce the final hematite 'gravity' concentrate. The concentrates were sold separately or mixed as 'sinter fines' nominally sized < 1 mm and there was no re-grinding of concentrates.

This process concept of grinding followed by magnetic separation to recover a magnetite concentrate and then gravity concentration to recover the non-magnetic hematite content of the ore has been common to the mines in the area.]

Table 8-1: Results of sampling Blötberget processing plant on 16 January 1968

Flow stream	Weight distribution %	Iron content % Fe	Phosphorus % P
Feed to mills	100	36.7	0.53
Magnetite concentrate	31.1	62.1	0.23
Hematite concentrate	25.0	54.7	0.67

Flow stream	Weight distribution %	Iron content % Fe	Phosphorus % P
Blended concentrate	56.1	58.8	0.38

Table 8-2: Results of sampling Blötberget processing plant on 22 August 1968

Flow stream	Weight distribution %	Iron content % Fe	Phosphorus % P
Feed to mills	100	37.6	0.59
Magnetite concentrate	25.0	64.7	0.22
Hematite concentrate	29.7	59.1	0.77
Blended concentrate	54.7	61.6	0.52

At the time, the authors of the report dismissed the earlier sampling programme as 'typical' although the reason is not clear. The higher grades of concentrate observed during the later sampling period could be explained by the finer grind observed during this period as illustrated by Table 8-3.

Table 8-3: Particle size analyses of flow streams in Blötberget processing plant in 1968

Flow stream	Particle size distribution , d_{80} mm	
	January 1968	August 1968
Feed to mills	20	16
Rod mill product	0.59	0.42
Magnetite concentrate	0.55	0.40
Hematite concentrate	0.47	0.40

Table 8-1 and Table 8-2 indicate that historically magnetite and hematite concentrates were in roughly equal proportions. This contrasts with the recent DMT resource model for the unmined Blötberget mineralised zone which indicates that magnetite is marginally the dominant iron mineral (estimated approximately 60% proportion of Total Fe) .The zoning of hematite-rich ore has been identified in the Hugget-Flygruvan orebody at Blötberget and was noted at the neighbouring Grängesberg where hematite-rich ore occurred near the surface and to the north. It has been suggested that the oxidation of magnetite to martite may be associated with intrusions of pegmatite.

Thus, the historical data identifies the need for some flexibility in the flowsheet so that the processing plant can treat ore having variable proportions of hematite and magnetite.

Table 8-1 and Table 8-2 also clearly identify that phosphorus is strongly associated with hematite which became a major issue for the mine in later years. These levels of phosphorus in the concentrates would be unacceptable today when upper limits of 0.05 % P are commonly specified and certainly less than 0.1 % P. Early BF/steelmaking processes in Sweden used the Thomas Process, where the higher levels of P could be accommodated; however the introduction of higher efficient steelmaking LD process rendered the high P levels unsuitable. The indication is that the ore would need to be ground to a particle size significantly less than 1 mm to achieve liberation of the phosphorus mineral which, at the same time, would provide potential for improvement of the iron grade which is particularly low (64.7 % Fe) for a magnetite concentrate. This becomes of greater importance, of course, if it is contemplated selling the hematite concentrates separately. However, NIO does intend to sell a combined concentrate only, where the lower levels of P can, to some degree, be blended out.

A summary of the Blötberget production statistics for the period 1973-1979 is given in Table 8-4 below.

Table 8-4: Summarised production statistics for Blötberget mine 1973-1979

	Feed	Magnetite concentrate	Hematite concentrate
Weight distribution %	100	20.3	26.0
Iron assay % Fe	36.5	67.8	60.9
Phosphorus assay % P	-	0.1	0.52

A significant improvement in the quality of the magnetite concentrate has been achieved both with respect to iron and phosphorus contents but the association between hematite and phosphorus is even more noticeable. Again, the hematite forms the greater proportion of the product but it is understood that the mine was deliberately ‘high-grading’ the orebody and that higher grades were associated with hematite-rich ore.

The processing flowsheet have been modified during the years including consideration being given, but not implemented, to pre-concentration of the crushed run-of-mine (ROM) ore using dry magnetic separators, cobbing, or dense medium separation. A description of the final flowsheet in 1978, when approximately 415,000 tonnes were being processed each year, is given as;

“Crushed ore was fed to rod mills to grind the ore to a particle size of less than 1 mm so that a magnetite pre-concentrate could be recovered by wet LIMS. The non-magnetic tailings of LIMS passed to rougher spirals which rejected final coarse tailings and recovered a hematite pre-concentrate. The magnetite pre-concentrate was reground in rod mills and treated by cleaner wet LIMS to produce a final magnetite concentrate.”

The hematite pre-concentrate was treated by cleaner spirals to produce the final hematite concentrate and tailings. The tailings themselves were re-treated by scavenger spirals to recover any remaining hematite as a scavenger concentrate which joined the final hematite concentrate. These scavenger spirals also received the non-magnetic tailings from the cleaner wet LIMS and produced the final fine tailings. It is notable that this flowsheet included regrinding of the magnetite pre-concentrate but not of the hematite pre-concentrate as is evident from the relatively low iron assay (60.9 % Fe) and high phosphorus content (0.52 % P) of the hematite concentrate.

8.1.3 Grängesberg Mine

The resource reports state that the abandoned Grängesberg mine contains 121 Mt of iron ore grading 47 % Fe and about 1 % P as fluorapatite (3,4). The ore contains varying proportions of the main iron minerals with up to 25% hematite and an average of 80 % iron present as magnetite, 20 % of iron present as hematite. The orebody is considered to be of magmatic origin and of the 'Kiruna' type. In general, it is apparently very similar to Blötberget ore.

The mine has a long history but the principal operations were between 1890 and 1990 during which period 180 Mt of ore was extracted. The first beneficiation plant was constructed in 1910 and flotation to remove phosphorus was introduced in 1935 as one of many modifications over the years to improve product quality. The principal product was 'sinter fines' but the company also produced 'lump' ore and a 'phosphate' concentrate for fertiliser manufacture.

In terms of iron content (% Fe), the quality of the concentrates was high but the removal of phosphorus to the level of < 0.05 % P required fine grinding of the ore. The fine particle size of the concentrates excluded the market for 'sinter fines' and the concentrate became similar to a 'pellet feed'. Eventually, the company was forced to install a pellet plant during the period 1968-70. The company had developed the 'Grangcold' process for cold bonding of pellets using a cement binder but the poor quality of these pellets created problems in marketing. As will be seen later in this chapter, initial metallurgical testing of Blötberget ore by NIO also attempted to produce a coarse 'sinter fines' product.

The assays of typical products of Grängesberg mine are shown below in Table 8-5.

Table 8-5: Typical assays of products of Grängesberg mine

Product	% Fe	% P	% S	% Magnetite
Lump ore	60	0.88	0.01	82
Pellet feed	70.4	0.013	0.003	94.7
Super concentrate	71.3	0.001	0.003	97.9

The description of the flowsheet of Grängesberg mine in 1988, just before it closed, is given as;

- Crushing of the 900 t/h of ore to less than 80 mm followed by pre-concentration using cobbers (dry magnetic separators) which rejected 35 % by weight of the ore and produced an enriched material (60-62 % Fe) some of which could be sold as 'lump' ore and 'fines'.
- Grinding of the enriched ore to a particle size of 90 % passing 0.2 mm and 60 % passing 0.05 mm followed by reverse froth flotation of apatite and silica. The apatite froth was cleaned by flotation to produce a phosphate concentrate containing 17 % P.

The cell underflow from flotation passed to wet LIMS to produce separate concentrates of magnetite and hematite. Both these concentrates were sized by classifiers into an overflow (fine) fraction and an underflow (coarse) fraction which were combined to produce a mixed coarse (90 % passing 0.2 mm) concentrate and a mixed fine (90 % passing 0.075 mm) concentrate. Typical assays of these concentrates are presented in Table 8-6.

Table 8-6: Typical assays of the concentrates produced by Grängesberg mine processing plant

Concentrate	% Fe	% P	% SiO_2
Coarse	69,5	0.017	1.8
Fine	70.0	0.013	1.3

Assuming that the ore is, indeed, similar to Blötberget, the information from Grängesberg mine indicates that it is necessary to grind the ore substantially finer than 1 mm in order to produce a satisfactory iron ore concentrate with respect to phosphorus content. It also introduces the concept of froth flotation to remove phosphates and even silica.

8.1.4 Håksberg Mine

The process flowsheet for the old Håksberg mine is not clear from available records but appears to be coarse grinding followed by two stages of wet LIMS to produce only a magnetite concentrate in the form of ‘sinter fines’. In this respect, it is notable that the ore itself contained only 0.06 % P. The processing plant treated 500,000 to 770,000 tonnes per year to produce an average concentrate grade of 61.4 % Fe, 0.03 % P at a yield of 36 % by weight.

In 2011, on behalf of NIO, PROing prepared a Preliminary Economic Assessment (PEA) for the Blötberget-Håksberg Project which studied the concurrent development of the Blötberget and Håksberg mines at a rate of 5.5 Mt/y with the potential to access iron ore beneath Lake Väsman (Väsman orebody) in later years. The processing plant comprised two lines; one for Blötberget ore (34.5 % Fe, 1.5 % P) and one for Håksberg ore (30.6 % Fe, 0.06 % P). It is interesting to note that it was recognised that both ores would require grinding to a particle size distribution having about 50 % passing 0.044 mm in order to produce high grade concentrates (% Fe > 67 %, % P < 0.05 %).

The flowsheet for Blötberget included grinding by an ABC type circuit (Autogenous, Ball mill and Crusher) after the ROM ore had been crushed to 150 mm. The product of the Autogenous milling circuit had a particle size analysis of <1 mm and was fed to wet rougher LIMS to produce a magnetite pre-concentrate grading 62 % Fe. This pre-concentrate was reground in a ball mill before cleaner LIMS produced a concentrate grading 68 % Fe, 0.17 % P. The non-magnetic tailings from rougher LIMS were also reground in a ball mill before the hematite was recovered by wet high intensity magnetic separation (WHIMS) to produce a hematite pre-concentrate. The hematite pre-concentrate was up-graded to 63 % Fe and 0.03 % P by spiral concentrators.

The combined concentrates were finally up-graded as required by reverse froth flotation of apatite to produce a final concentrate containing 67 % Fe and 0.05 % P.

The flowsheet and metallurgical balance proposed by PROing contrast with the historical practice at other mines in that the magnetite pre-concentrate is shown to carry most of the phosphorus, not the hematite pre-concentrate. The metallurgical testwork from which the flowsheet had been developed was very limited in scope, see 8.3.1.1 for details. Nevertheless, this study introduced the possible applications of Autogenous milling and wet high intensity magnetic separation, neither of which technologies were available to earlier operations.

8.2 Design Basis

8.2.1 Process Plant Feed

The process flowsheet has been developed with the objective of achieving a high recovery of iron into concentrates from ore of variable iron content and varying ratios of iron present as magnetite and hematite. The range for the magnetite: hematite ratios used for development of the process flowsheet were, approximately, between 100:0 and 60:40.

The mass and water balance, see Section 8.5, as well as the discussion of equipment sizing and selection, see Section 8.6, refer to the base case[2] when sub-level caving is employed, and to feed to the plant which has a head grade of 33.6 % Fe of which 76 % is present as magnetite. Reference is made to the mass balance for the years when sub-level open-stoping is employed and during which the ROM head grade is expected to be greater at 36.1 % Fe.

Table 8-7: Composition of process plant feed

Process Plant Feed		
Sub-level Caving (SLC)		
Iron, Fe	%	33.6
Average Ratio Magnetite: Hematite	%	76:24
Sub-level Open Stoping (SLOS)		
Iron, Fe	%	36.1
Average Ratio Magnetite: Hematite	%	76:24

8.2.2 General Design Criteria

The basic mineral separation process was developed by comparison with modern practice and supported by metallurgical testwork. It comprises the following stages:

1. Liberation of the minerals by a comminution (size reduction through a combination of crushing and grinding to a particle size less than approximately 0.1 mm (100 µm)

[2] It should be noted that this is higher than stated in Section 8 of the report, for which results based on a pilot test bulk sample of 33.6% are reported. This difference in ROM feed grade of 2.5% supports using 45% for this Study. Updates of this report will include for a revised mining method to SLOS and improved headgrade.

2. Separation of the magnetite concentrate by low intensity magnetic separation (LIMS)
3. Separation of the hematite concentrate by gravity concentration
4. Removal of phosphorus from the iron ore concentrates by froth flotation

The process plant will receive feed in the form of crushed Run-of-mine (ROM) Ore sized 70 mm or less from the underground mine at a proposed rate of 3.0 Mtpa.

The crushed ore is conveyed directly to the surface by conveyor during only two shifts of the working day; the third shift being reserved for conveying of waste rock and clearing of blasting fumes.

Average nominal (peak) throughput is 627 t/h (690 t/h) for the primary crusher located underground and 431 t/h (475 t/h) downstream of the secondary crusher, the crushed ore silo and the crushed ore emergency stockpile, respectively, see Table 8-8.

Table 8-8: Process plant design criteria

Basic Design Parameters		
Secondary Crushing		
Ore Conveying Days	Days	350
Ore Conveying Shifts (8hrs/shift)	Shift	2
Crusher and Conveyor Availability	%	95
Crusher and Conveyor Utilisation	%	90
Average Nominal Throughput	Tonnes/hour	627
Secondary Crushing		
Additional Peak Flow	%	10
Peak Throughput	Tonnes/hour	690
Processing Plant		
Annual Hours	Hours/Year	8,400
Plant Availability	%	92
Plant Utilisation	%	90
Plant Available Hours	Hours/year	6,955
Annual Ore (ROM) Tonnage	Tonnes	3,000,000
Particle size of ROM	mm	<250
Average Nominal Throughput	Tonnes/hour	431
Additional Peak Flow	%	10
Average Peak Throughput	Tonnes/hour	475

For detailed information, refer to Appendix D.

8.3 Review of Metallurgical Test Data

8.3.1 Core and Bench scale Testing

8.3.1.1 Minpro, PEA, August 2011

The metallurgical development work carried out at Minpro AB, Sweden as part of the preparation of the PEA was limited in scope and comprised a series of Davis Tube (DT) tests on drill core composites from both Blötberget and Håksberg with head grades of 34-36 % Fe and c.30 % Fe respectively. Additional work was undertaken to investigate the potential for the recovery of a 'sinter fines' product on material crushed to less than 5 mm by dry low intensity magnetic separation ('dry LIMS').

The DT testwork investigated the production of both magnetite and hematite concentrates at two different grinds containing 34-43 % and 65-75 % passing 0.045 mm (45 µm) respectively. The magnetite concentrates produced from the Blötberget samples were of high grade assaying in excess of 71 % Fe and containing phosphorus (P) of 0.05 % or less.

The use of a DT in combination with a pyrometallurgical process (which first reduces hematite to magnetite and, upon recovery by the DT, converts the magnetite back to hematite for assaying) to study the recovery of a hematite product is uncommon and the results, particularly with respect to concentrate grade, must be interpreted with great care.

Attempts to recover 'sinter fines' were unsuccessful owing to the insufficient liberation between the iron oxide minerals and the principal phosphorus carrier, apatite, at coarse size. The dry LIMS product obtained from the Blötberget composite assayed 64.8 % Fe at a product yield of 27 % but contained 0.25 % P which would far exceed the typical limit for metallurgical applications.

8.3.1.2 Minpro, Fines and Heavy Aggregate Testwork, August 2012

In 2012, further testwork was undertaken at Minpro to investigate the extraction of a 'sinter fines sized' product for non-metallurgical applications, e.g. the manufacture of 'heavyweight concrete'. This 'heavy aggregate' could, as was stipulated at the time, be produced in the early phases of production 'using simple and low cost mining and processing technology'. Its marketability would principally be determined by certain physical attributes such as size distribution and specific gravity (S.G.) rather than chemical composition.

Two samples of Blötberget ore were tested, namely;

1. a 'surface magnetite sample' from near surface occurrences near the Blötberget mine grading c.44 % Fe, 0.93 % P, and
2. a drill core composite grading c.45.5 % Fe, 0.33 % P.

Minpro employed conventional crushing and screening to three different finenesses, nominally -8, -4, and -1 mm, followed by dry magnetic separation.

Table 8-9 summarises the physical and chemical characteristics reported for the final products obtained at a peripheral drum speed of 4.4 m/s.

Table 8-9: Summary of 'heavy aggregate' testwork at Minpro, 2012

Process Stream	Weight	Fe	P	S.G.	No of stages
	wt%	%	%	%	
Surface ore, -8 mm	28.9	63.4	0.37	4.70	2
Surface ore, -4 mm	39.4	62.7	0.36	4.66	4
Drill core, -8 mm	19.0	65.2	0.12	4.72	2
Drill core, -4 mm	27.2	64.2	0.15	4.67	2
Drill core, -2 mm	26.4	67.3	0.10	n/a	4

Both the size distribution and S.G. of the products were viewed as satisfactory but no further work, such as customer testing, was undertaken at the time.

8.3.1.3 GTK Mintec, 2013-2014

The Geological Survey of Finland Mineral Processing Laboratory (GTK Mintec) in Outokumpu, Finland was contracted to carry out two phases of bench scale beneficiation testwork, namely;

1. Phase I: Initial flowsheet development work, comprising testing of Flygruvan composite sample originating from drillhole BB12015-MET003
2. Phase II: Bench scale 'Variability testing'

The sample tested in Phase I was taken from a single diamond hole drilled specifically to provide material for metallurgical testwork. The drillhole intersected both the Flygruvan and Kalvgruvan deposits, although the drillhole intervals selected for the formation of the composite sample were taken from the Flygruvan intercepts only.

The composite was formed with the objective of matching what, at the time, was believed to be the orebody average in terms of % Fe, % P and ratio of magnetite: hematite.

The results of this initial metallurgical testwork – which included mineralogy, comminution testwork, dry LIMS, wet (low, medium and high intensity) magnetic separation, gravity concentration, and reverse flotation for apatite removal – indicated that a substantial proportion of the iron oxides could be recovered as a relatively coarse product at a particle size of less than 1 mm. For the purpose of recovering such a 'coarse concentrate', the preliminary flowsheet included spiral concentrators after the initial stage of grinding to a particle size less than 1 mm (see Figure 8-1).

However, it was noted at the time that the testwork undertaken to develop the conceptual process flowsheet was limited in scope and used a single ore sample from the Flygruvan horizon only.

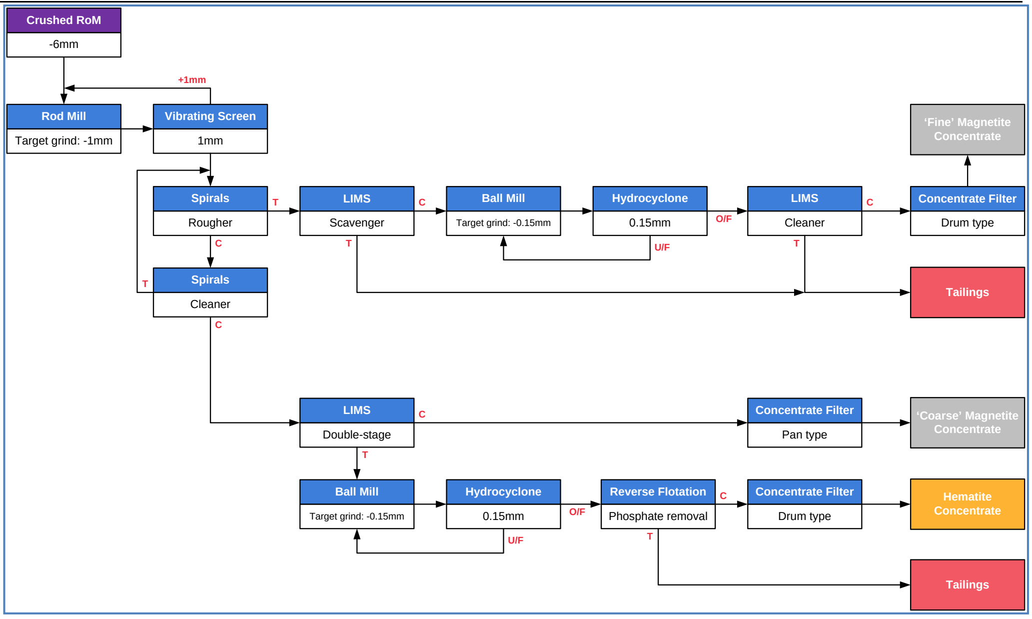


Figure 8-1: Process Flow Diagram (PFD) with option of producing a coarse magnetite concentrate

In Phase II, a further six (6) ore composites, five (5) from Blötberget and one (1) from the Sandell satellite deposit, were subjected to bench scale testing. The scope of work included;

1. Comminution (Crushing and Grinding)
2. Gravity concentration
3. Wet low-intensity magnetic separation (LIMS)
4. Wet high-gradient (or high-intensity) magnetic separation (HGMS or WHIMS)
5. Reverse flotation

The principal objective was the demonstration that satisfactory iron ore concentrates could be produced from a range of different ore types which were anticipated to be mined at different stages of the mine life.

One of the most important findings of this second campaign of testing was that not all samples were conducive to the production of a 'coarse concentrate' at a top size of approximately 1 mm. For some samples, fine grinding to 0.10 mm (100 µm) and finer was required to achieve satisfactory liberation between the iron minerals and the gangue minerals (silica, silicates and phosphates).

Bench-scale wet high intensity magnetic separation using Sala HGMS equipment yielded encouraging results for the beneficiation of hematite but confirmation in a pilot plant environment or/and on a continuous operation was strongly recommended.

A further finding was that all ore samples generally responded extremely well to reverse phosphate flotation using a combination of water glass and Atrac 1563 (produced by AkzoNobel) as dispersant/pH modifier and collector, respectively.

The effective removal of phosphate minerals from iron ore concentrates using reverse flotation should thus be achievable with comparatively short retention times and low collector dosages.

As a result of these findings, a modified flowsheet was proposed as shown in Figure 8-2.

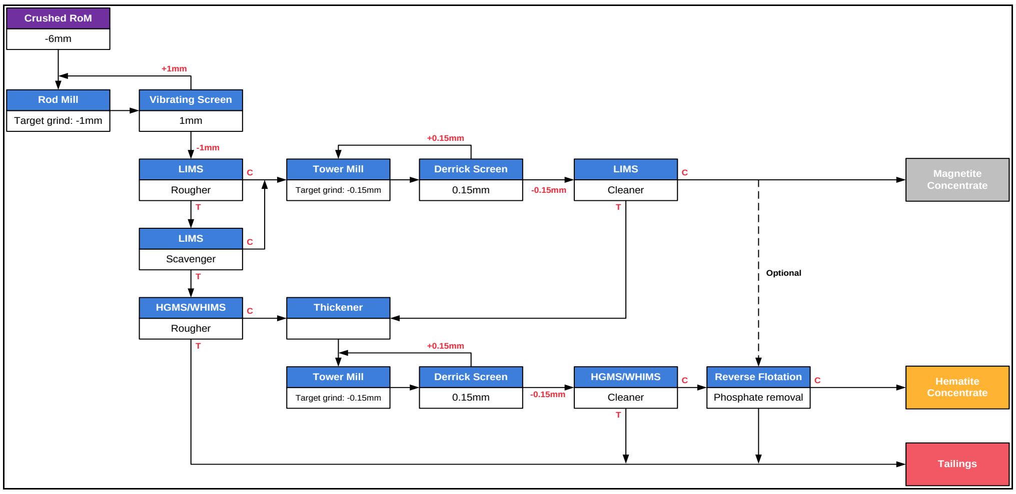


Figure 8-2: Modified' Process Flow Diagram (PFD) proposed for pilot plant operation

8.3.1.4 Bureau Veritas, Davis Tube Testwork, 2014

In May-June 2014, 24 pulverised composites of recent and historical drill core sized c.90 % passing 0.075 mm (75 µm) were submitted to Bureau Veritas in Perth, Australia for the purpose of Davis Tube testwork.

The contents of Fe in the feed samples varied widely (10.3 % to 61.7 %) as did the percentage of iron bound to magnetite (19.8 % to 93.1 %).

Whilst the DT methodology has obvious limitations when investigating ores containing iron oxide minerals other than magnetite (NB: the Davis Tube only recovers the strongly magnetic minerals i.e. magnetite), important conclusions could be drawn upon the recovery of a magnetite product at fine particle size (see also Table 8-10);

1. The DT concentrates were of consistently high quality, with the contents of iron (Fe) exceeding 70 % in all but one cases.
2. Product yield varied widely and, as expected, was principally a function of the contents of magnetite in the feed (coefficient of determination R^2=0.94).
3. % P averaged 0.014 % in the products albeit individual DT concentrates contained as much as 0.11 % P. There was weak to moderate correlation (R^2=0.48) between the levels of phosphorus in the feed and those in the DT concentrates.
4. A significant number of concentrates contained elevated levels of vanadium (expressed as % V or % V_2O_5) and particularly those produced from ore composites originating from the Kalvgruvan orebody (0.29-0.39 % V_2O_5, or 0.16-0.22 % V).

5. The concentrates obtained from Kalvgruvan composites also contained high levels of titanium (0.15-0.65 % TiO_2).
6. There was moderate to good correlation (R^2=0.77) between the levels of vanadium (expressed as % V or % V_2O_5) in the feed and those in the DT concentrates.
7. There was poor correlation (R^2=0.32) between the levels of titanium (% TiO_2) in the feed and those in the DT concentrates.

8.3.1.5 Bureau Veritas, Davis Tube Testwork, 2015

A further 16 pulverised composites of drill core sized c.80-90 % passing 0.075 mm (75 µm) were submitted for Davis Tube testwork to Bureau Veritas in early 2015. The composites contained between 22.4 % and 49.1 % Fe of which 33.5 % to 94.3 % was present as magnetite. The results demonstrated the following (Table 8-11):

1. The DT concentrates were of exceptionally high quality, with the contents of iron (Fe) exceeding 71 % in all cases.
2. Product yield varied widely and was again strongly correlated with the contents of magnetite in the feed (coefficient of determination R^2=0.99).
3. The DT concentrates contained 0.1 % P or less (0.006 % on average).
4. 7 out of 16 composites produced concentrates with elevated levels of vanadium (0.20-0.38 % V_2O_5) and titanium (0.19-0.43 % TiO_2).
5. There was good correlation (R^2=0.87) between the levels of phosphorus (expressed as % P or % P_2O_5) in the feed and those in the DT concentrates.
6. There was good correlation (R^2=0.84) between the levels of vanadium (expressed as % V or % V_2O_5) in the feed and those in the DT concentrates.
7. There was moderate correlation (R^2=0.65) between the levels of titanium (% TiO_2) in the feed and those in the DT concentrates.

Figure 8-3 plots % TiO_2 vs. % V_2O_5 for all magnetite concentrates obtained from the 2014 and 2015 Davis Tube Testwork campaigns.

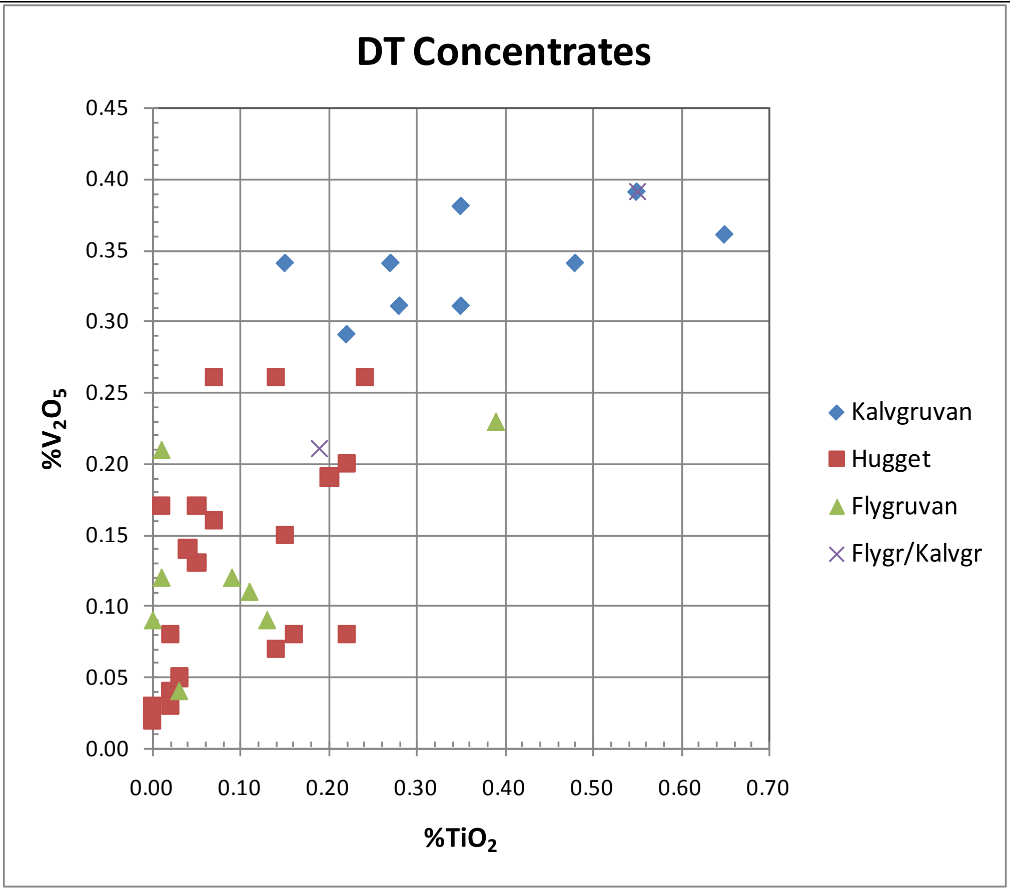


Figure 8-3: Plot of % TiO_2 vs. % V_2O_5 for magnetite concentrates

Table 8-10: Summary of 2014 Davis Tube testwork on 24 drill core composites at Bureau Veritas, Perth/Australia

Sample				DTR Feed							DTR Concentrate					
				Fe	SiO_2	TiO_2	P	V_2O_5	Fe_3O_4	Fe bd. to Mgt	wt	Fe	SiO_2	TiO_2	P	V_2O_5
Hole ID	Sample ID	Comments		%	%	%	%	%	%	%	%	%	%	%	%	%
BB_76-374	L295424	Hugget	Upper-mid level	19.5	54.4	0.40	0.13	0.03	19.0	70.6	20.8	71.0	0.77	0.16	0.002	0.08
BB_76-383	L295425	Hugget	Upper-mid level	23.7	47.0	0.32	0.08	0.02	18.8	57.4	16.0	71.3	0.49	0.22	0.001	0.08
BB_74-310	L295426	Hugget	Upper-mid level	21.4	43.9	0.56	0.40	0.09	25.0	84.5	27.7	71.6	0.19	0.07	0.014	0.26
BB_74-310	L295427	Hugget	Upper-mid level	32.8	34.1	0.31	0.15	0.03	25.0	55.2	29.2	71.4	0.69	0.03	0.003	0.05
BB_66-138	L295415	Hugget	Mid level	45.4	25.7	0.14	0.31	0.04	32.3	51.6	35.3	71.1	0.65	0.02	0.008	0.08
BB_66-138	L295416	Hugget	Mid level	46.5	21.3	0.19	0.98	0.19	22.2	34.6	24.1	71.5	0.30	0.01	0.038	0.17
BB_74-002	L295408	Hugget	Mid-lower level	31.7	40.6	0.41	0.13	0.02	17.8	40.6	18.5	71.8	0.22	0.02	0.002	0.03
BB_65-157	L295419	Hugget	Mid-lower level	44.6	18.2	0.64	1.07	0.16	14.3	23.1	15.5	69.2	1.51	0.05	0.109	0.17
BB_75-003	L295422	Hugget	Lower level	18.5	49.2	0.59	0.30	0.09	11.8	46.0	13.4	71.3	0.31	0.14	0.003	0.26
BB_75-003	L295423	Hugget	Lower level	10.3	61.6	0.39	0.16	0.03	11.5	81.3	11.1	70.8	1.03	0.14	0.004	0.07
BB_75-001	L295401	Hugget	Lower level	37.8	29.6	0.40	0.68	0.16	31.2	59.8	25.0	70.8	1.07	0.20	0.007	0.19
BB_67-168	L295411	Flygruvan	Upper level	36.9	32.1	0.45	0.68	0.13	10.1	19.8	13.1	71.1	0.23	0.01	0.010	0.21
BB_67-168	L295412	Flygruvan	Upper level	20.6	51.6	0.35	0.16	0.03	15.4	54.0	21.5	70.7	0.55	0.13	0.002	0.09
BB_66-166	L295417	Flygruvan	Mid level	37.1	36.8	0.19	0.25	0.06	46.2	90.2	31.4	70.4	1.19	0.09	0.021	0.12
BB_67-167	L295421	Flygruvan	Mid level	29.4	43.9	0.27	0.32	0.05	34.5	84.9	33.1	71.6	0.29	0.11	0.003	0.11
BB_75-002	L295409	Flygruvan	Mid level	61.7	6.89	0.01	0.76	0.17	26.5	31.1	27.9	71.7	0.18	0.01	0.021	0.12
BB_74-001	L295405	Flygruvan	Lower level	34.8	37.5	0.47	0.42	0.09	42.3	88.2	39.3	71.3	0.43	0.39	0.005	0.23
BB_67-168	L295413	Flygr/Kalvgr	Upper level	50.9	16.1	0.74	0.51	0.28	61.9	87.9	54.3	70.6	0.67	0.55	0.010	0.39
BB_67-168	L295414	Kalvgruvan	Upper level	51.3	17.7	0.35	0.41	0.24	63.6	89.6	65.7	70.8	0.63	0.48	0.008	0.34
BB_66-166	L295418	Kalvgruvan	Mid level	48.2	19.5	0.31	0.42	0.20	58.7	88.2	62.9	70.2	1.21	0.22	0.026	0.29
BB_67-167	L295420	Kalvgruvan	Mid level	43.2	23.3	0.63	0.61	0.21	53.4	89.3	55.9	70.6	0.60	0.65	0.010	0.36
BB_75-002	L295410	Kalvgruvan	Mid level	53.2	15.2	0.22	0.61	0.24	68.4	93.1	74.1	71.5	0.26	0.15	0.010	0.34
BB_74-001	L295406	Kalvgruvan	Lower level	53.3	13.6	0.63	0.59	0.28	67.8	92.1	60.0	71.0	0.32	0.55	0.013	0.39
BB_74-001	L295407	Kalvgruvan	Lower level	40.2	26.6	0.47	0.51	0.19	50.3	90.5	47.0	70.8	0.97	0.27	0.008	0.34

Table 8-11: Summary of 2015 Davis Tube Testwork on 16 drill core composites at Bureau Veritas, Perth/Australia

Sample				DTR Feed							DTR Concentrate					
				Fe	SiO_2	TiO_2	P	V_2O_5	Fe_3O_4	Fe bd. to Mgt	wt	Fe	SiO_2	TiO_2	P	V_2O_5
Hole ID	Sample ID	Comments		%	%	%	%	%	%	%	%	%	%	%	%	%
BB_14-001	L295450	Hugget	Upper level	32.8	38.3	0.33	0.40	0.11	41.7	87.3	40.3	71.3	0.34	0.24	0.006	0.26
BB_14-006	L295446	Hugget	Mid level	27.5	46.3	0.20	0.12	0.02	22.5	60.6	23.1	71.7	0.34	0.02	0.002	0.04
BB_14-009	L466555	Hugget	Mid level	31.7	42.0	0.20	0.07	0.02	21.3	49.7	21.9	71.6	0.26	0.00	0.002	0.03
BB_14-010	L466556	Hugget	Mid level	31.4	38.3	0.53	0.19	0.09	19.6	46.8	20.7	71.4	0.30	0.15	0.004	0.15
BB_14-013	L466561	Hugget	Mid level	40.4	29.0	0.22	0.61	0.14	18.6	33.5	19.1	71.8	0.27	0.05	0.007	0.13
BB_14-011	L466564	Hugget	Mid level	22.4	51.6	0.26	0.19	0.04	26.4	82.3	23.8	71.5	0.42	0.04	0.003	0.14
BB_14-004	L295448	Hugget	Mid level	24.4	50.5	0.17	0.04	0.01	25.0	74.1	24.8	71.8	0.61	0.00	0.002	0.02
BB_14-003	L466552	Hugget	Mid level	27.3	45.9	0.29	0.24	0.07	34.1	91.4	33.7	71.3	0.39	0.22	0.007	0.20
BB_14-007	L466553	Hugget	Mid- lower level	28.3	41.8	0.35	0.37	0.09	15.5	39.5	16.1	71.2	0.55	0.07	0.007	0.16
BB_14-010	L466557	Hugget	Mid- lower level	20.8	53.9	0.16	0.05	0.01	18.6	66.5	19.4	71.6	0.46	0.02	0.001	0.04
BB_14-005	L466560	Flygruvan	Upper level	40.5	30.7	0.13	0.46	0.08	20.0	36.1	20.3	71.8	0.26	0.00	0.005	0.09
BB_14-004	L295447	Flygruvan	Mid level	30.8	42.5	0.35	0.08	0.02	25.7	62.1	26.7	71.8	0.22	0.03	0.002	0.04
BB_14-002	L295449	Flygr/Kalvgr	Upper- mid level	34.6	36.7	0.34	0.31	0.08	33.6	71.4	35.0	71.5	0.28	0.19	0.005	0.21
BB_14-008	L466563	Kalvgruvan	Upper- mid level	38.7	28.9	0.40	0.43	0.20	48.7	92.8	50.7	71.0	0.57	0.35	0.009	0.38
BB_14-004	L466551	Kalvgruvan	Mid level	44.0	24.3	0.45	0.59	0.19	55.7	93.5	57.9	71.1	0.46	0.35	0.012	0.31
BB_14-005	L466562	Kalvgruvan	Mid level	48.6	19.1	0.38	0.49	0.22	60.5	92.1	64.3	71.0	0.75	0.28	0.011	0.31

8.3.2 Pilot Plant, November 2014

8.3.2.1 Introduction

In support of the development of the mineral separation process, the Geological Survey of Finland Mineral Processing Laboratory (GTK Mintec) in Outokumpu, Finland was contracted to operate a small-scale processing plant or pilot plant on a blend of ore samples supplied from Blötberget and an adjacent satellite deposit (Sandell). The principal objective was the demonstration, on a semi-continuous basis, that the plant could produce satisfactory iron ore concentrates, and to obtain substantial quantities of such concentrates. Other, lesser objectives of the pilot plant included generation of samples for vendor testing and collection of design data.

8.3.2.2 Material for Testing

Since the amount of drill core (2,000-3,000 kg) collected during resource definition drilling was insufficient for the purpose of the pilot plant, the bulk sample of Blötberget iron ore was obtained by blasting and excavating outcrops of the orebodies including the adjacent Guldkannan/ Sandell orebodies.

Specifically, the bulk sample material was taken;

6. from the footwall of an existing open pit known to be a surface extension of the Blötberget orebody, as well as
7. from the Guldkannan historical open pit which is a parallel and similar orebody to that of Sandell satellite deposit..

The materials excavated from the outcrops were processed to remove dilution derived from the footwall rocks before blending in proportions necessary to generate a composite bulk sample that was believed to be typical of the ore to be mined, specifically with respect to the principal assays of the ore namely % Fe, % P_2O_5 and the ratio of magnetite to hematite. For the composite bulk sample, these principal assays were;

32.5 % Fe, 0.56 % P_2O_5, and 79 % of the iron bound to magnetite.

At the time, the assay of the Run-of-Mine (ROM) ore was predicted to be; 35 % Fe and 1.0 % P_2O_5 with 80 % of the iron bound to magnetite.

It is evident that the phosphorus content of the bulk sample was lower than that of the projected ROM average. With the sole exception of the Sandell/Guldkannan sample, the materials used to form the bulk sample contained very low levels of phosphorus, 0.06 % P or less. As a consequence, almost all of the phosphorus in the blend originated from material from the Sandell/ Guldkannan deposit which, firstly, is small relative to the overall resource size and, secondly, will be exclusively mined in the early years of production.

Also, the levels of vanadium (V) and titanium (TiO_2) in the composited bulk sample were somewhat lower that those seen in many of the core samples tested to date and particularly from deeper areas of the Kalvgruvan orebody.

8.3.2.3 Testwork Results

The small-scale plant operated for about 24 hours over a 4 day period and treated approximately 22 tonnes of the composite bulk sample.

The plant demonstrated that the process could produce very high grade concentrates albeit on a semi-continuous basis. For example, the best result in terms of concentrate grade was a bulk concentrate (magnetite + hematite) containing 70.3 % Fe, 2.15 % SiO_2 and 0.13 % P_2O_5 without flotation. This concentrate was obtained at a yield of 45.7 wt%.

The magnetite concentrate alone was of exceptionally high grade containing as much as 70.5 % Fe but only 0.1 % P_2O_5 and therefore did not require flotation. Without flotation, the hematite concentrate had a grade of up to 67.0 % Fe but a very high content of phosphorus, up to 0.95 % P_2O_5. Nevertheless, small-scale continuous froth flotation was able to produce a final hematite concentrate containing only 0.135 % P_2O_5 (0.06 % P).

A major finding of the operation was that the wet high intensity magnetic separators (WHIMS), in this case SLon equipment, were considerably less effective than predicted by bench-scale metallurgical test work using Sala HGMS equipment. The reasons have not been fully explained and there was insufficient time and sample to investigate this problem thoroughly. Nevertheless, the replacement of the SLon separators by spiral concentrators to recover hematite proved to be very successful.

The plant operation demonstrated that the process was very effective in recovering magnetite from the ore with recovery in all cases exceeding 98.5 %. Recovery of hematite was less effective and estimated at 50 % or less even when spiral concentrators replaced the SLon separators. The ultimate losses of hematite were attributed to the generation, during regrinding, of very fine hematite particles sized less than 37 µm, too finely sized to be recovered by spiral concentrators.

Although not entirely successful, an attempt was made to determine the effect of a coarser primary grind in order to reduce grinding power consumption and produce a coarser tailings for dam construction and to facilitate disposal. The results indicated that the quality of the concentrate would be reduced because the bulk concentrate contained 68.8 % Fe and 0.23 % P_2O_5 and the yield decreased marginally to 43.6 %.

The small-scale plant was periodically sampled for the purpose of calculating mass balances and also the design of a full-scale processing plant.

The plant produced 7 tonnes of magnetite concentrate and 300 kg of hematite concentrate.

The average moisture content of the magnetite concentrate after filtration using a Choquenet filter press was 6.9 wt%. In comparison, a bench-scale Larox filter yielded a filter cake containing an average moisture level of 4.65 wt%.

8.3.3 Mineralogical Investigations

Berg och Gruvundersökningar AB (BGU) was contracted to undertake mineralogical examinations of a wide variety of samples, specifically;

8. Twenty (20) polished thin sections prepared from resource drilling core material and 'grab' samples by polarised light microscopy,
9. Six (6) polished thin sections and five (5) polished sections from ore samples tested at GTK Mintec (NB: Before preparing the polished thin sections the samples were also studied under a stereomicroscope).

In addition, selected sections were submitted for Scanning Electron Microscopy (SEM) analysis.

The principal finding from the mineralogical investigation is that there is considerable variation in terms of mineral composition, liberation and texture between and within the orebodies of Blötberget (Flygruvan-Hugget, Kalvgruvan, Sandell).

The variation was found to be most pronounced in Sandell, 'significant' in Flygruvan-Hugget, and least pronounced in Kalvgruvan.

Particular emphasis was placed upon the study of the occurrence of phosphate minerals. Apatite is the principal carrier of phosphorus albeit Monazite, a rare earth element (REE) phosphate principally containing cerium (Ce) and lanthanum (La), was also identified. The report states that;

"Most … apatite grains occur as free grains in the gangue matrix or at hematite-magnetite grain boundaries. Only a minor part of apatite occurs as small micro inclusions in predominantly hematite but it also occurs less common in magnetite.

The apatite … in (the) Kalvgruvan orebody is more fine-grained compared with the apatite in Flygruvan-Hugget. The Kalvgruvan apatite grains have irregular popcorn shape which will affect the grinding liberation grain size negatively."

Apatite liberation was estimated to occur at 0.125 mm or less in Kalvgruvan magnetite ore and 0.25 mm or less in the Hugget–Flygruvan ore.

Rare earth elements (REE) occur principally in inclusions of monazite and xenotime in apatite. REE also occur in other minerals such as bastnäsite (REE carbonate), especially in Sandell.

Hematite is of secondary nature i.e. occurs as alteration product of magnetite. The hematite alteration is said to be "tectonically related and follows ductile shear zones cross-cutting magnetite-rich ore zones".

The study also concluded that the K- and Na-feldspars in the shear zones of Flygruvan-Hugget showed no signs of hydrothermal alteration which raises questions as to the cause of the alteration of magnetite to hematite and the introduction of titanium (Ti) into hematite.

Below follows a brief description of the main ore zones identified at Blötberget.

Flygruvan-Hugget is a felsic, disseminated to massive magnetite ore rich in apatite with secondary hematite along shear planes. The gangue is dominated by quartz, biotite, K-feldspar and albite. The ratio of magnetite: hematite varies in wide ranges. The average grain size of magnetite and hematite in Flygruvan and Hugget was reported as 0.1-0.4 mm and 0.2-0.5 mm respectively, that of apatite as 0.2-0.3 mm.

Kalvgruvan is a mafic intrusive massive to banded disseminated apatite-magnetite ore. Principal gangue minerals are actinolite-augite, orthopyroxene, biotite, quartz and anorthite. The average grade of magnetite was estimated at 20-80 wt% and that of apatite at 5-8 wt%. The average grain size of magnetite is 0.1-0.2 mm. Apatite occurred in irregular-popcorn shape grains of 0.1-0.2 mm.

No inclusions of apatite in magnetite were observed. Magnetite was reported to be free of Ti and V, a finding which is contradicted by the results of the Davis Tube testwork undertaken at Bureau Veritas. Magnetite was found to hold small amounts of micro-inclusions of apatite.

Guldkannan-Sandell is an apatite, amphibole-pyroxene rich skarn iron ore which shows strong hydrothermal alteration with presence of magnetite as the principal iron oxide mineral in a smectite-chlorite matrix. Hematite was reported to occur as a minor constituent. This ore horizon also contains calcite and bastnäsite, a REE bearing carbonate mineral.

The ore was found to contain between 20-65 wt% magnetite and 4-13 wt% apatite. The average grain size is 0.1-0.3 mm for magnetite and 0.1-0.2 mm for apatite.

8.3.4 Vendor Testwork

8.3.4.1 High Pressure Grinding Roll (HPGR) Testwork

1,070 kg of < 70 mm composite bulk sample material were sent to Weir Minerals (KHD) laboratories in Cologne, Germany for the purpose of a 'scouting' test to investigate the use of High Pressure Grinding Roll (HPGR) technology on Blötberget ore.

The test procedure comprised HPGR grinding of the iron ore followed by dry screening at 5 mm and wet screening at 1 mm. The report issued by Weir Minerals was well written and followed the format of tests conducted for similar projects in terms of the parameters determined for HPGR.

The results were promising in that they identified the potential benefits of including High Pressure Grinding Roll technology in lieu of the alternative methods of comminution, namely:

1. Conventional crushing and grinding comprising a tertiary cone crusher and rod mills,
2. Semi-autogenous grinding mill (if required in combination with a ball mill).

Specific grinding energy for the size reduction from 70 mm or less, a typical product of secondary crushing, to 1 mm or less (d_{80} of 0.5 mm) was estimated at c.3.0 kWh per tonne of fresh feed.

The formation of an autogenous wearing layer was seen as indicating a long operating wear life of the rolls, an indicative figure of 13,000 hours of operation was stated by Weir Minerals. Flakes (NB: a considerable portion of the product may emerge from the compression zone as compacted cake or ‘flake’) generated from Blötberget ore were weak to moderately strong and are not anticipated to pose any major operational challenges.

Whilst the findings of the scouting test were generally encouraging, the report by Weir Minerals also states clearly that caution must be exercised when interpreting the results as the parameters in a closed circuit will differ from those generated in an open circuit / single pass scouting test. It is recommended that the initial testwork be followed up by a locked-cycle operation with oversize recycling and input parameter variation to increase confidence in the design data and provide a basis for equipment performance guarantees.

8.3.4.2 Semi-autogenous Grinding Mill Comminution (SMC) Test

180 pieces were selected from the >19 mm composite bulk sample before crushing and sent to SGS Lakefield Laboratory, Ontario, Canada for the SMC test.

A summary of the results is presented in Table 8-12 below.

Table 8-12: Summary of SMC testwork

Sample	Relative Density	JK Parameters	
	g/cc	A x b	DWI
Demonstration plant composite feed	3.43	87.6	3.9

The sample was categorised as ‘soft’ with respect to resistance to impact breakage.

8.3.4.3 Filtration Testwork

Samples of magnetite and hematite concentrate produced during the pilot plant campaign were sent to the laboratories of Metso Minerals in Sala, Sweden for filtration testwork.

The results can be summarised as follows;

- Vacuum filtration:
    - A rest moisture of 11 % w/w can be achieved at a filtration rate of 6,000 kg/m_{eff}^2/h.
    - A rest moisture of 9 % w/w can be achieved at a filtration rate of 2,850 kg/m_{eff}^2/h.
- Pressure (VPA) filtration:

- 6.5 % w/w can be achieved using an estimated total cycle time of 3 min comprising 1 min chamber filling, 30 sec cake compression and 1.5 min air blowing.
- It is possible to reduce the moisture content to 4.5 % w/w by extending the air blow period to 6 min but this would incur substantially greater capital and operating costs.

8.3.4.4 VertiMill Testwork

Samples were also despatched to Metso Minerals, York, Pennsylvania, USA for a pot milling test related to the measurement of grindability with respect to the design and selection of tower mills ('VertiMill' is a trademark name for tower or stirrer agitated media mills manufactured by Metso).

The principal objective of the testwork was to determine a 'grindability' parameter, similar to the Bond Ball Mill Work Index, which can be used to calculate the power input and model number of the appropriate tower mill.

Table 8-13: Predicted specific 'VertiMill' energy

Sample	Feed d_{80}	Product d_{80}	Media	Predicted specific VTM Energy
	µm	µm	mm	kWh/mt
Magnetite	364	100	19.1	4.91
Hematite	237	100	19.1	5.75

The testwork results (see Table 8-13) can be extrapolated to calculate the specific energy requirement for a size reduction from 500 µm (d_{80}) to 80 µm (d_{80});

- Magnetite - 6.9 kWh/mt
- Hematite - 11.0 kWh/mt

8.3.5 Chemical Composition of Products

Table 8-14, which was prepared on the basis of the testwork described above, provides an estimate of the product specification which can typically be expected for iron ore products sized 0.1 mm (100 µm) or finer from Blötberget.

As will be discussed in Section 8.4, Process Description and Flowsheet, provision must be made for treatment of either or both concentrates by froth flotation to remove the mineral apatite (calcium fluoro-phosphate) into the froth by reverse flotation.

Table 8-14: Estimate of average chemical composition of concentrates

Analyte %	Magnetite Concentrate	Hematite Concentrate
Top size (mm)	**0.10 mm**	**0.10 mm**
Fe(tot)	69.5-70.5	65.5-66.5
FeO	29-31	<1
SiO_2	2.0-3.2	2.0-3.6

Al_2O_3	0.2-0.6	0.3-0.8
CaO	0.03-0.2	0.1-0.3
MgO	0.2-0.4	0.1-0.4
P	<0.05	<0.05
S	<0.01	<0.01
Na_2O	0.01-0.03	<0.08
K_2O	0.02-0.04	<0.05
MnO	0.03-0.21	<0.01
SrO	<0.001	<0.001
BaO	<0.02	<0.02
TiO_2	0.01-0.60	0.7-1.0
V	0.03-0.22	<0.05
Cr	<0.025	<0.05
Co	0.002-0.015	<0.01
Ni	0.002-0.012	<0.01
Cu	<0.003	<0.003
Zn	0.002-0.010	0.002-0.010
Pb	<0.001	<0.001
As	<0.001	<0.001
Cd	<0.001	<0.001
Tl	<0.001	<0.001
F	0.01-0.03	0.01-0.03
Cl	0.01	0.01
LOI	-2.6 to -3.3	n/a
Moisture (% w/w)	6.5	6.5

8.4 Process Description and Flowsheet

8.4.1 Mineralogy and Mineral Separation

The Basis of Design (BoD) of the processing plant has been explained in Section 8.2 above.

The mineralogy of the iron ore is described in Section 4.2 and 8.3.3. Essentially, the ore comprises variable quantities of the valuable iron minerals, magnetite and hematite, together with gangue minerals which are predominantly calcium magnesium alumina-silicates. It is notable that negligible quantities of other iron-bearing minerals, especially iron silicates, are present. Thus, the total iron content of the ore is potentially recoverable as iron oxides.

It is also notable that the ore contains very little sulphur and negligible quantities of iron sulphides such as pyrrhotite which can complicate the mineral processing flowsheet because it can be significantly magnetic.

One of the most significant findings of the mineralogical study is the presence of phosphates including apatite and some phosphates of rare earth elements.

The latter are present only in a very small proportion and the great majority of phosphorus is contained in apatite, a calcium fluoro-phosphate.

Through mineralogical study and metallurgical testwork, the mineral grain size of the iron minerals, which controls the liberation size, was estimated to less than 0.10 mm (100 µm).

The mineral magnetite is strongly magnetic (ferromagnetic) and is commonly recovered from ore by low intensity magnetic separation (LIMS) especially by wet low intensity magnetic drum separators. The mineral hematite is only weakly paramagnetic and cannot be separated by LIMS. This mineral may be recovered by exploiting its relatively high density (gravity concentration) or by wet high intensity magnetic separation (WHIMS) or variants of this process (HGMS, SLon).

The mineral apatite is non-magnetic and is commonly separated from iron ore concentrates by reverse froth flotation in which process the apatite is removed to the froth.

Thus, the basic mineral separation process, as developed by comparison with modern practice and supported by metallurgical testwork, comprises the following stages:

1. Liberation of the minerals by a comminution (size reduction through a combination of crushing and grinding to a particle size less than 0.10 mm (100 µm)
2. Separation of the magnetite concentrate by LIMS
3. Separation of the hematite concentrate by WHIMS or gravity concentration
4. Removal of phosphorus from the iron ore concentrates by froth flotation

8.4.2 Pre-concentration

From observation of the texture of the ore and results of metallurgical testwork, it was noted that substantial liberation of gangue occurs at a particle size of less than 1 mm compared to the ultimate liberation size of 0.10 mm (100 µm). Thus, it is possible to eliminate hard gangue minerals at a coarser size thereby saving energy and costs in subsequent fine grinding to the liberation size. This process is known as pre-concentration.

Consequently, the basic process flowsheet becomes;

1. Partial liberation of the gangue by size reduction to a particle size of less than 1 mm.
2. Separation of a magnetite pre-concentrate by LIMS
3. Separation of the hematite pre-concentrate by WHIMS or gravity concentration

4. Regrinding of the magnetite pre-concentrate to the liberation size of 0.10 mm
5. Separation of a magnetite concentrate by LIMS
6. Regrinding of a hematite pre-concentrate to the liberation size of 0.10 mm
7. Separation of a hematite concentrate by WHIMS or gravity concentration
8. Removal of phosphorus from the iron ore concentrates by froth flotation

The enhanced mineral separation flowsheet is presented on the following pages.

8.4.3 Ore Quality

The process flowsheet has been developed with the objective of achieving a high recovery of iron into concentrates from ore of variable iron content (33.6 % to 36.1 % Fe) and varying ratios of iron present as magnetite and hematite. The range for the magnetite: hematite ratio used for development of the process flowsheet were, approximately, between 100:0 and 60:40.

An important aspect of the proposed flowsheet is that the ore is ultimately ground to a particle size of less than 0.10 mm before a final concentrate is produced. In the early development phase of the flowsheet, initial metallurgical testwork upon a drill core sample indicated that it would be possible to produce a relatively coarse concentrate at a particle size of less than 1 mm. For the purpose of recovering such a coarse concentrate, the preliminary flowsheet included spiral concentrators after the initial stage of grinding to a particle size less than 1 mm.

As detailed below, the process has been designed on a precautionary principle. Due to the variability in mineralogy/texture/liberation, it may be possible to recover concentrates at a coarser size during some periods whilst, for other parts of the orebody, finer grinds may be required to produce satisfactory concentrates. A size of less than 0.10 mm is the best estimate at this point in time.

8.4.4 Flowsheet Description

8.4.4.1 Introduction

The figures given in the flowsheet description relate to the base case of processing ROM ore during the period when sub-level caving is employed as the mining method and the feed material contains 33.6 % Fe and 35.6 % magnetite. The flowsheet for processing ROM ore when sub-level open-stoping is employed as the mining method is identical but the figures for % Fe and % weight distribution will be different.

8.4.4.2 Primary crushing and grinding

The ROM ore is crushed underground by a primary crusher to a particle size less than 250 mm before it is conveyed to the surface and crushed by a secondary crusher to a particle size less than 70 mm. The crushed ore is now suitable for feeding to high pressure grinding rolls (HPGR) for reduction to a particle size of less than 1 mm. This process of size reduction achieves partial liberation of the gangue so that the ore may be pre-concentrated.

The first stage of pre-concentration is low intensity magnetic separation (LIMS) to recover a magnetite pre-concentrate containing 59 % Fe. In the second stage of pre-concentration, the non-magnetic fraction passes directly to gravity concentration (spirals) which recovers a hematite pre-concentrate containing 32 % Fe and rejects a final coarse tailings containing 4.1 % Fe and representing 44.5 % weight of the feed. The consequent savings in subsequent processing which are gained by rejecting nearly half of the ore at this relatively coarse size are obvious.

The hematite pre-concentrate is then reground to a particle size of less than 0.10 mm to liberate more gangue before passing to gravity concentration (spirals preceded by a hydrosizer or elutriator) which recovers a hematite concentrate containing 66.0 % Fe and representing 4 % by weight of the feed. Gravity concentration rejects a final fine tailings containing 17.7 % Fe representing 13.6 % weight of the feed.

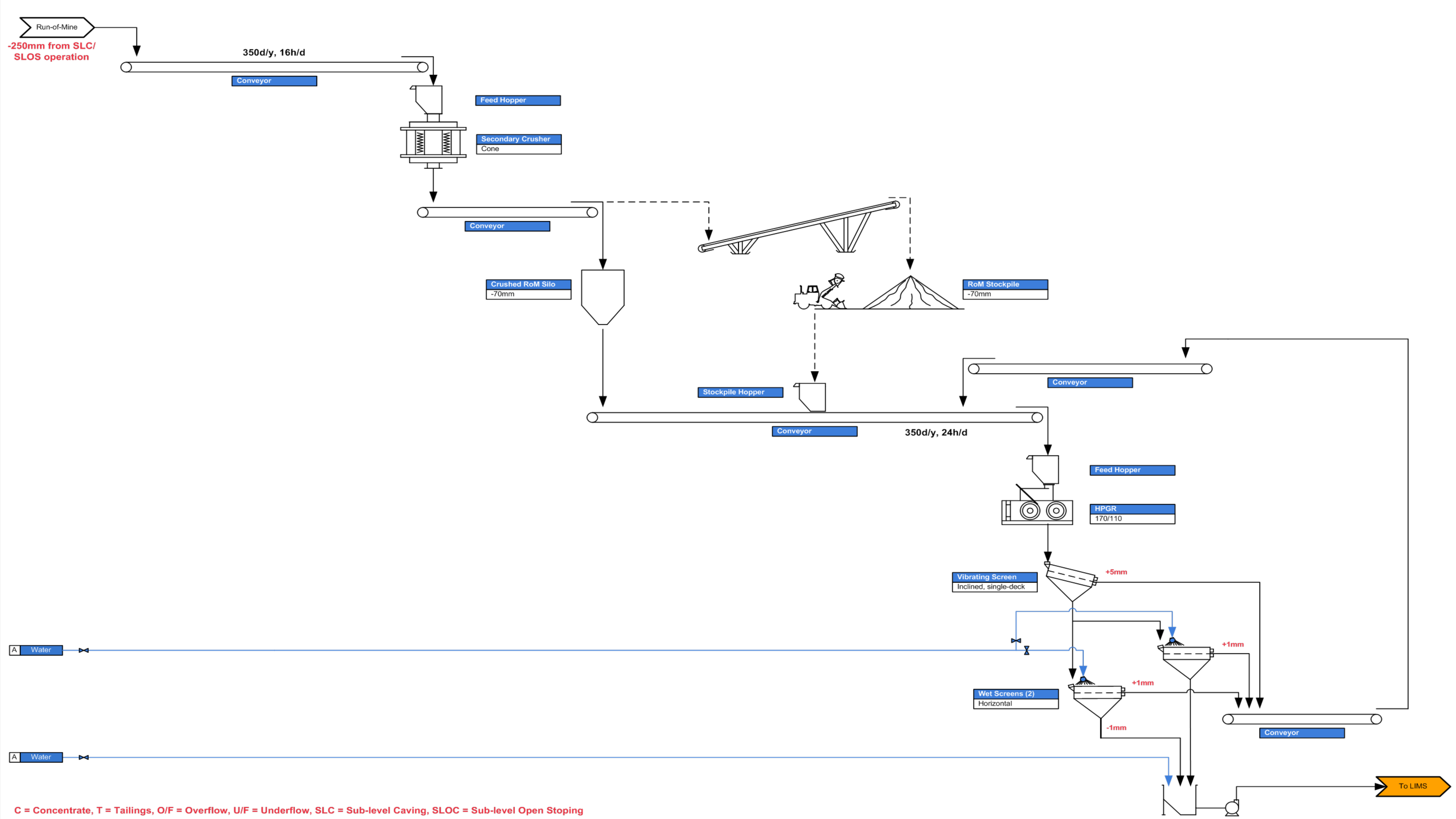
The total tailings, thus, amounts to 58.0 % of the feed and the principal iron loss is to the fine tailings in the form of very finely sized (<37 µm) hematite.

The magnetite pre-concentrate is reground to a particle size of less than 0.10 mm which liberates the remaining gangue but also liberates any hematite that was closely associated with the magnetite. The final magnetite concentrate containing 70.5 % Fe and representing 38 % by weight of the feed is then recovered by cleaner low intensity magnetic separation (LIMS). The cleaner LIMS also produces a non-magnetic tailings containing liberated hematite and 18.3 % Fe which is recycled to the hematite regrinding circuit.

Subsequent processing then depends upon the phosphorus content of the concentrates. Provision has been made for treatment of either or both concentrates by froth flotation to remove the mineral apatite (calcium fluoro-phosphate) into the froth by reverse flotation. During actual operation, it is uncertain whether the magnetite concentrate will require treatment by flotation and the concentrate may by-pass this process.

It is probable that the hematite concentrate will contain excessive phosphorus but it may be possible to blend the relatively small quantity of this concentrate with the magnetite and achieve a satisfactory phosphorus content of the combined concentrates.

Finally, the concentrates are dewatered by pressure filtration to produce the final product as filter cake.



C = Concentrate, T = Tailings, O/F = Overflow, U/F = Underflow, SLC = Sub-level Caving, SLOC = Sub-level Open Stoping

Material Stream
Material Stream (Optional)
Water Stream
Water Stream (Optional)
By-Passed Stream
By-Passed Stream (Optional)

Water Circuit Information:
A Water — Water Circuit 'A', from Tailings Thickener and TMF
B Water — Water Circuit 'B', from Concentrate Thickener

Revision Details	Rev	By	Apprvd	Date
Initial Release	0	MR	MRS	27/02/2015

Title	INTERIM TECHNICAL REPORT Blötberget Iron Ore Project Process Flowsheet 1/3				
By	Matthias Reisinger	Rev.	0	Date	27/02/2015
Client No.	NIO2				
TSC No.	NIO-INT/TECH/REP-PFD-1e				

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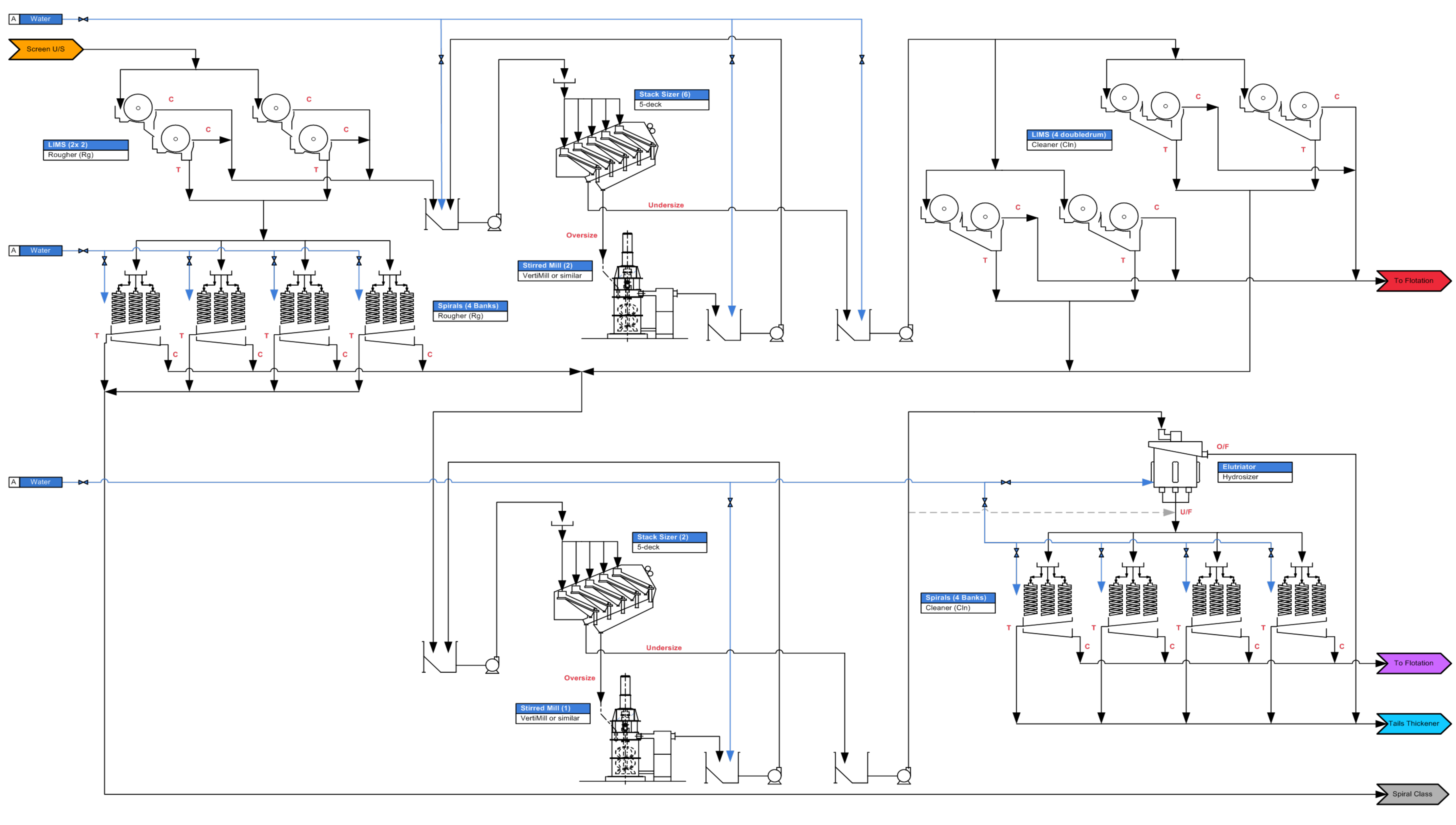

C = Concentrate, T = Tailings, O/F = Overflow, U/F = Underflow, SLC = Sub-level Caving

Material Stream
Material Stream (Optional)
Water Stream
Water Stream (Optional)
By-Passed Stream
By-Passed Stream (Optional)

Water Circuit Information:

A Water — Water Circuit 'A', from Tailings Thickener and TMF

B Water — Water Circuit 'B', from Concentrate Thickener

Revision Details	Rev	By	Apprvd	Date
Initial Release	0	MR	MRS	27/02/2015

Title	INTERIM TECHNICAL REPORT Blötberget Iron Ore Project – SLC Process Flowsheet 2/3				
By	Matthias Reisinger	**Rev.**	0	**Date**	27/02/2015
Client No.	NIO2				
TSC No.	NIO-INT/TECH/REP-PFD-1f				

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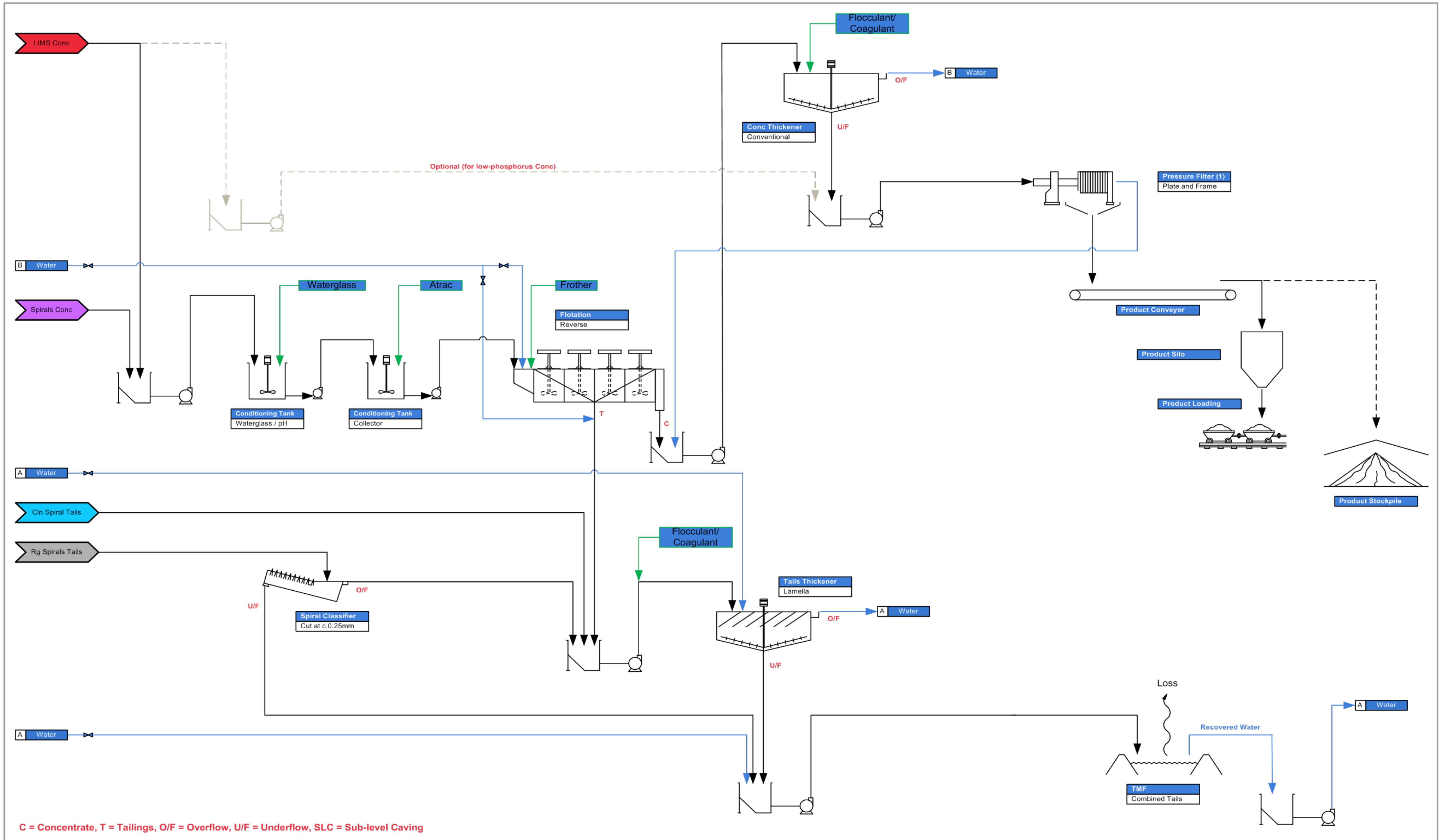

Material Stream
Material Stream (Optional)
Water Stream
Water Stream (Optional)
By-Passed Stream
By-Passed Stream (Optional)

Water Circuit Information:

A Water — Water Circuit 'A', from Tailings Thickener and TMF

B Water — Water Circuit 'B', from Concentrate Thickener

Revision Details	Rev	By	Apprvd	Date
Initial Release	0	MR	MRS	27/02/2015

Title	INTERIM TECHNICAL REPORT Blötberget Iron Ore Project - SLC Process Flowsheet 3/3				
By	Matthias Reisinger	Rev.	0	Date	27/02/2015
Client No.	NIO2				
TSC No.	NIO-INT/TECH/REP-PFD-1g				

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8.5 Mass and Water Balance

This Section presents Process Flow Diagrams (PFD) for the proposed Process Plant design, incorporating full mass and water balances (NB: Fe analyte data is provided for critical streams only) for the 'base case' estimated for the period of mining when sub-level caving is employed, resulting in feed to the plant which has a head grade of 33.6 % Fe of which 76 % is present as magnetite.

The mass balance for the alternative scenario, when sub-level open stoping is employed and resulting in feed to the plant which has a head grade of 36.1 % Fe of which 76 % is present as magnetite, is presented in simplified (tabular) form only.

The calculations for the base case, which are discussed in detail in Section 8.5.1, are predominantly based upon data collected during the pilot plant campaign undertaken by GTK Mintec in November 2014 and specifically upon the runs on 19 November and 20 November 2014.

The design must be considered a 'best approximation' and endeavoured to reflect the observations of the demonstration plant as closely as possible, specifically in regards to;

1. the overall recoveries of magnetite (which in all cases exceeded 98.5 %) and hematite (which was less effective and estimated at approximately 50 %) and,
2. the concentrate grades, which were assumed at 70.5 % Fe (magnetite) and 66.0 % Fe (hematite).

8.5.1 Design Data and Assumptions

Below follows a detailed description of the data used and assumptions made in order to calculate the mass balance for the base case (head grade of 33.6 % Fe of which 76 % is present as magnetite). Reference to data collected during the pilot plant exercise is shown as *[italic]*.

- Rougher LIMS

Weight recovery = % Fe_3O_4 (S9) x 1.37 [1.34 for Flowsheet 2a, 19/11/14; 1.40 for Flowsheet 2b, 20/11/14]

99.4 % Fe_3O_4 (magnetite) recovery

- Cleaner LIMS:
    - Weight recovery: Calculated using concentrate grade (70.5 % Fe) and magnetite/hematite recovery figures
    - 99.1 % magnetite recovery (resulting in overall recovery of magnetite in Cleaner LIMS concentrate of 98.5 %)
    - 18.0 % hematite recovery [26.1 % for Flowsheet 2a, 19/11/14; 11.4 % for Flowsheet 2b, 20/11/14]

- NB: A significant proportion of hematite is recovered by association with magnetite in the Rougher LIMS concentrate.

- Rougher Spirals:
  - Weight recovery: Calculated using 45.0 % Fe in concentrate [40.5 % for Flowsheet 2a, 19/11/14; 53.4 % for Flowsheet 2b, 20/11/14] and assuming a recovery of Fe of 63 % [65 % for Flowsheet 2a, 19/11/14; 61.5 % for Flowsheet 2b, 20/11/14]
- Cleaner Spirals:
  - Weight recovery: Calculated using 66.0 % Fe in concentrate and assuming a recovery of Fe of 52.5 % *[42.2 % for Flowsheet 2a, 19/11/14; 63.0 % for Flowsheet 2b, 20/11/14]*
- Reverse Flotation of Phosphates:
  - Weight recovery 99.5 %
  - Total hematite recovery: 50.4 %
  - Total magnetite recovery: 98.5 %
  - Total Fe recovery: 87.3 %
  - [87.4 % for Flowsheet 2a, 19/11/14; 88.0 % for Flowsheet 2b, 20/11/14]
  - Overall product yield: 41.8 wt%

The material balance for the base case is presented in Table 8-15.

Table 8-15: Mass and metallurgical balance, base case (sub-level caving, SLC), 33.6% Fe, magnetite: hematite 76:24

Process Stream	Weight	Fe	Fe Rec	Fe_3O_4	Fe_3O_4 Rec	Fe_2O_3 *	Fe_2O_3 Rec
	wt%	%	%	%	%	%	%
Feed (S9)	100.0	33.6	100.0	35.6	100.0	11.3	100.0
Rougher LIMS Tails (S11)	51.3	9.5	14.5	0.4	0.6	13.2	60.0
Rougher LIMS Conc (S10)	48.7	59.0	85.5	72.6	99.4	9.2	40.0
Cleaner LIMS Tails (S15)	10.7	18.3	5.8	3.0	0.9	23.1	22.0
Cleaner LIMS Conc (S21)	38.0	70.5	79.6	92.2	98.5	5.3	18.0
Rougher Spirals Tails (S17)	44.5	4.1	5.4	0.4	0.5	5.4	21.4
Rougher Spirals Conc (S16)	6.8	45.0	9.1	0.7	0.1	63.6	38.6
Cleaner Spirals Tails (S23)	13.6	17.7	7.1	2.4	0.9	22.7	27.4
Cleaner Spirals Conc (S22)	4.0	66.0	7.9	1.0	0.1	93.3	33.2
Flotation Tails (S25)	0.2	42.0	0.2	1.0	0.0	59.0	0.8
Final Concentrate (S31)	41.8	70.2	87.3	83.8	98.6	13.6	50.4
Process Tailings	58.2	7.3	12.7	0.9	1.4	9.6	49.6

Process Stream	Weight	Fe	Fe Rec	Fe_3O_4	Fe_3O_4 Rec	Fe_2O_3 *	Fe_2O_3 Rec
(S32)							

* All % Fe not bound to magnetite was assumed to occur in the form of hematite (Fe_2O_3).

The material balance for the case of sub-level open stoping is presented in Table 8-16.

Table 8-16: Mass and metallurgical balance, sub-level open stoping, SLOS, 36.1% Fe, magnetite: hematite 76:24

Process Stream	Weight	Fe	Fe Rec	Fe_3O_4	Fe_3O_4 Rec	Fe_2O_3*	Fe_2O_3 Rec
	wt%	%	%	%	%	%	%
Feed (S9)	100.0	36.1	100.0	38.2	100.0	12.1	100.0
Rougher LIMS Tails (S11)	47.7	10.9	14.5	0.5	0.6	15.1	60.0
Rougher LIMS Conc (S10)	52.3	59.0	85.5	72.6	99.4	9.2	40.0
Cleaner LIMS Tails (S15)	11.5	18.3	5.8	3.0	0.9	23.0	22.0
Cleaner LIMS Conc (S21)	40.7	70.5	79.7	92.2	98.5	5.3	18.0
Rougher Spirals Tails (S17)	40.4	4.8	5.4	0.4	0.5	6.4	21.4
Rougher Spirals Conc (S16)	7.3	45.0	9.1	0.7	0.1	63.6	38.6
Cleaner Spirals Tails (S23)	14.5	17.6	7.1	2.4	0.9	22.7	27.4
Cleaner Spirals Conc (S22)	4.3	66.0	7.9	1.0	0.1	93.3	33.2
Flotation Tails (S25)	0.2	42.0	0.2	1.0	0.0	59.0	0.8
Final Concentrate (S31)	44.9	70.2	87.3	83.8	98.6	13.5	50.4
Process Tailings (S32)	55.1	8.3	12.7	1.0	1.4	10.8	49.6

* All % Fe not bound to magnetite was assumed to occur in the form of hematite (Fe_2O_3).

8.5.2 Process Water

Process water is supplied to the process plant from two (2) above ground tanks by horizontal pumps (one operating and one stand-by pump per tank);

1. Tank for Water Circuit ‘A’, the principal supply of which will be tailings thickener overflow (O/F) and reclaim water from the tailings management facility (TMF), and

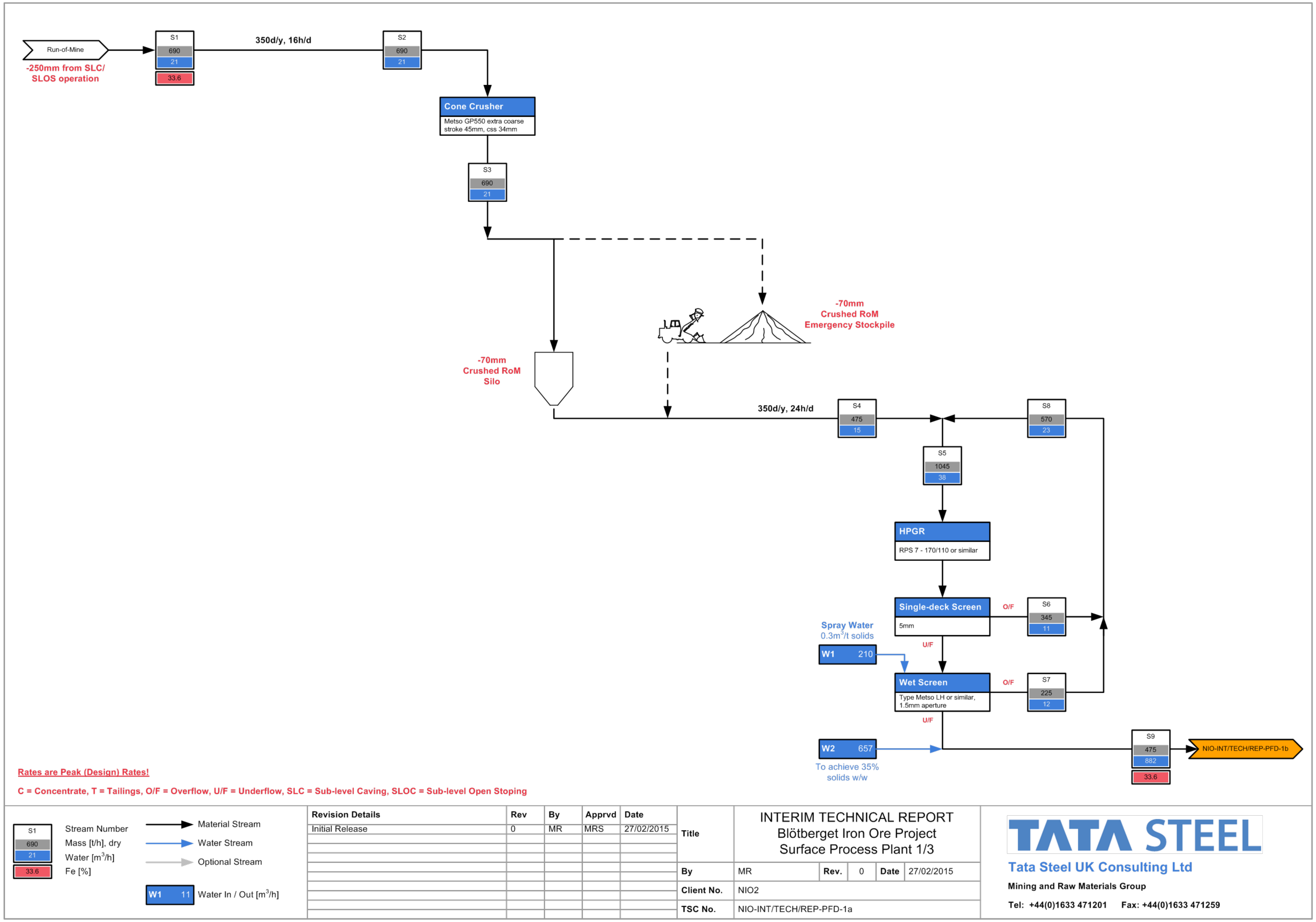
2. Tank for Water Circuit ‘B’ which receives ‘reagentised’ water in the form of concentrate thickener O/F.

The make-up water demand for the process plant will be met by a combination of raw (rain) water, industrial surface water, mine water as well as abstraction from Lake Väsman, which is permitted to be up to 100 litres per second over a maximum period of any four (4) months per year.

Water reservoirs of approximately 400 m^3 (ø10 x 5 m) and 80 m3 (ø6 x 3 m) for Water Circuits ‘A’ and ‘B’, respectively, will be installed in the process plant. The capacity of the process water reservoirs is based on fifteen (15) minutes of normal flow capacity.

8.5.3 Process Flow Diagrams (PFD)

The material flows (solids, water) presented in the PFDs below are peak flows (nominal +10 %) and accurate to within ±1 (rounding errors).


Run-of-Mine
-250mm from SLC/
SLOS operation
S1
690
21
33.6
350d/y, 16h/d
S2
690
21
Cone Crusher
Metso GP550 extra coarse
stroke 45mm, css 34mm
S3
690
21
-70mm
Crushed RoM
Emergency Stockpile
-70mm
Crushed RoM
Silo
350d/y, 24h/d
S4
475
15
S8
570
23
S5
1045
38
HPGR
RPS 7 - 170/110 or similar
Single-deck Screen
5mm
O/F
S6
345
11
Spray Water
0.3m³/t solids
U/F
W1 210
Wet Screen
Type Metso LH or similar,
1.5mm aperture
O/F
S7
225
12
U/F
W2 657
To achieve 35%
solids w/w
S9
475
882
33.6
NIO-INT/TECH/REP-PFD-1b
Rates are Peak (Design) Rates!
C = Concentrate, T = Tailings, O/F = Overflow, U/F = Underflow, SLC = Sub-level Caving, SLOC = Sub-level Open Stoping
S1 690 21 33.6
Stream Number
Mass [t/h], dry
Water [m³/h]
Fe [%]
Material Stream
Water Stream
Optional Stream
W1 11 Water In / Out [m³/h]
Revision Details | Rev | By | Apprvd | Date
Initial Release | 0 | MR | MRS | 27/02/2015
Title
INTERIM TECHNICAL REPORT
Blötberget Iron Ore Project
Surface Process Plant 1/3
By MR Rev. 0 Date 27/02/2015
Client No. NIO2
TSC No. NIO-INT/TECH/REP-PFD-1a
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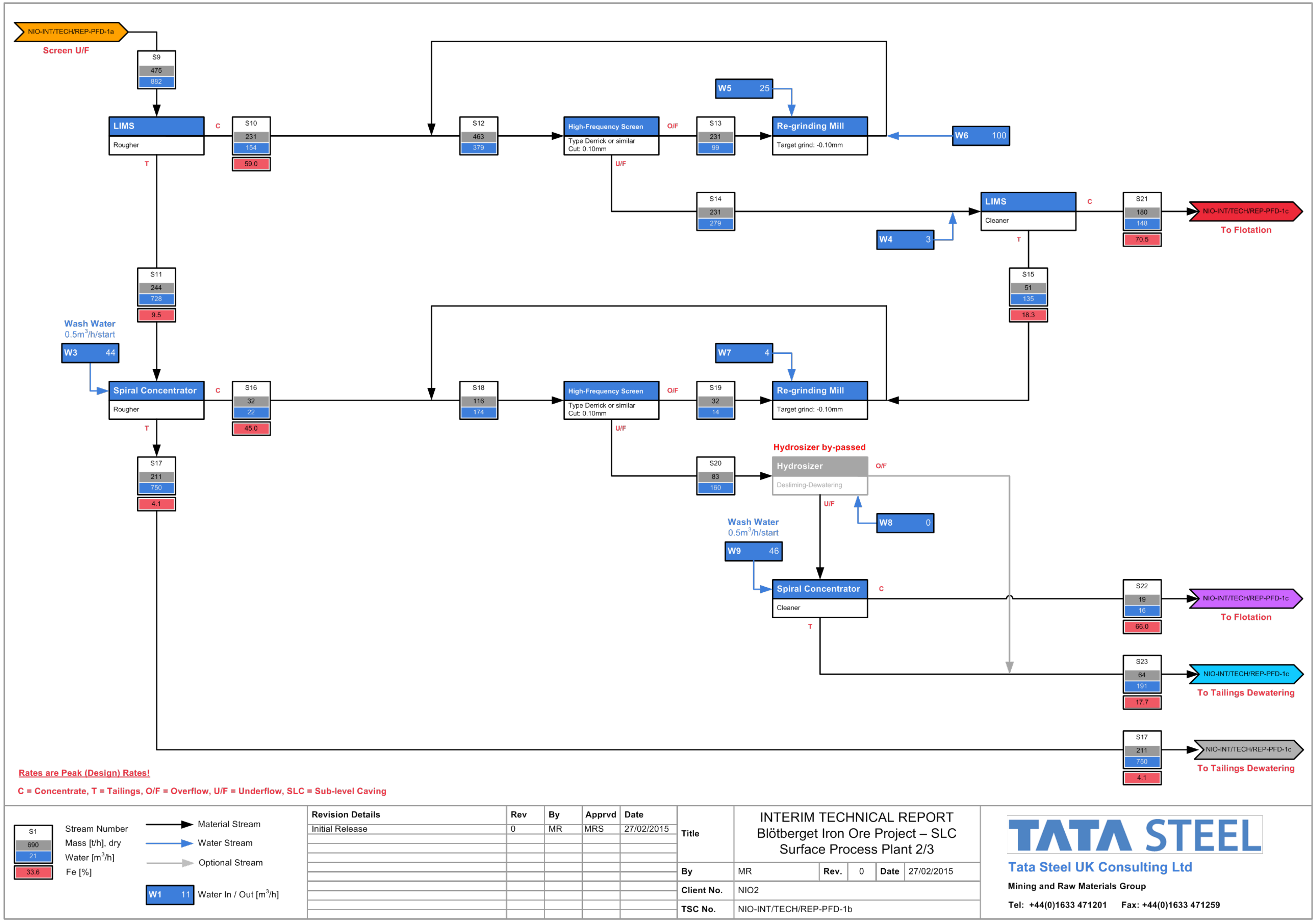
NIO-INT/TECH/REP-PFD-1a
Screen U/F
S9 475 882
LIMS
Rougher
C
T
S10 231 154 59.0
S12 463 379
High-Frequency Screen
Type Derrick or similar
Cut: 0.10mm
O/F
U/F
S13 231 99
W5 25
Re-grinding Mill
Target grind: -0.10mm
W6 100
S14 231 279
W4 3
LIMS
Cleaner
S21 180 148 70.5
NIO-INT/TECH/REP-PFD-1c
To Flotation
S15 51 135 18.3
S11 244 728 9.5
Wash Water
0.5m³/h/start
W3 44
Spiral Concentrator
Rougher
S16 32 22 45.0
S18 116 174
High-Frequency Screen
Type Derrick or similar
Cut: 0.10mm
S19 32 14
W7 4
Re-grinding Mill
Target grind: -0.10mm
S17 211 750 4.1
Hydrosizer by-passed
S20 83 160
Hydrosizer
Desliming-Dewatering
W8 0
Wash Water
0.5m³/h/start
W9 46
Spiral Concentrator
Cleaner
S22 19 16 66.0
NIO-INT/TECH/REP-PFD-1c
To Flotation
S23 64 191 17.7
NIO-INT/TECH/REP-PFD-1c
To Tailings Dewatering
S17 211 750 4.1
NIO-INT/TECH/REP-PFD-1c
To Tailings Dewatering
Rates are Peak (Design) Rates!
C = Concentrate, T = Tailings, O/F = Overflow, U/F = Underflow, SLC = Sub-level Caving
S1 690 21 33.6
Stream Number
Mass [t/h], dry
Water [m³/h]
Fe [%]
Material Stream
Water Stream
Optional Stream
W1 11 Water In / Out [m³/h]
Revision Details Rev By Apprvd Date
Initial Release 0 MR MRS 27/02/2015
Title
INTERIM TECHNICAL REPORT
Blötberget Iron Ore Project – SLC
Surface Process Plant 2/3
By MR Rev. 0 Date 27/02/2015
Client No. NIO2
TSC No. NIO-INT/TECH/REP-PFD-1b
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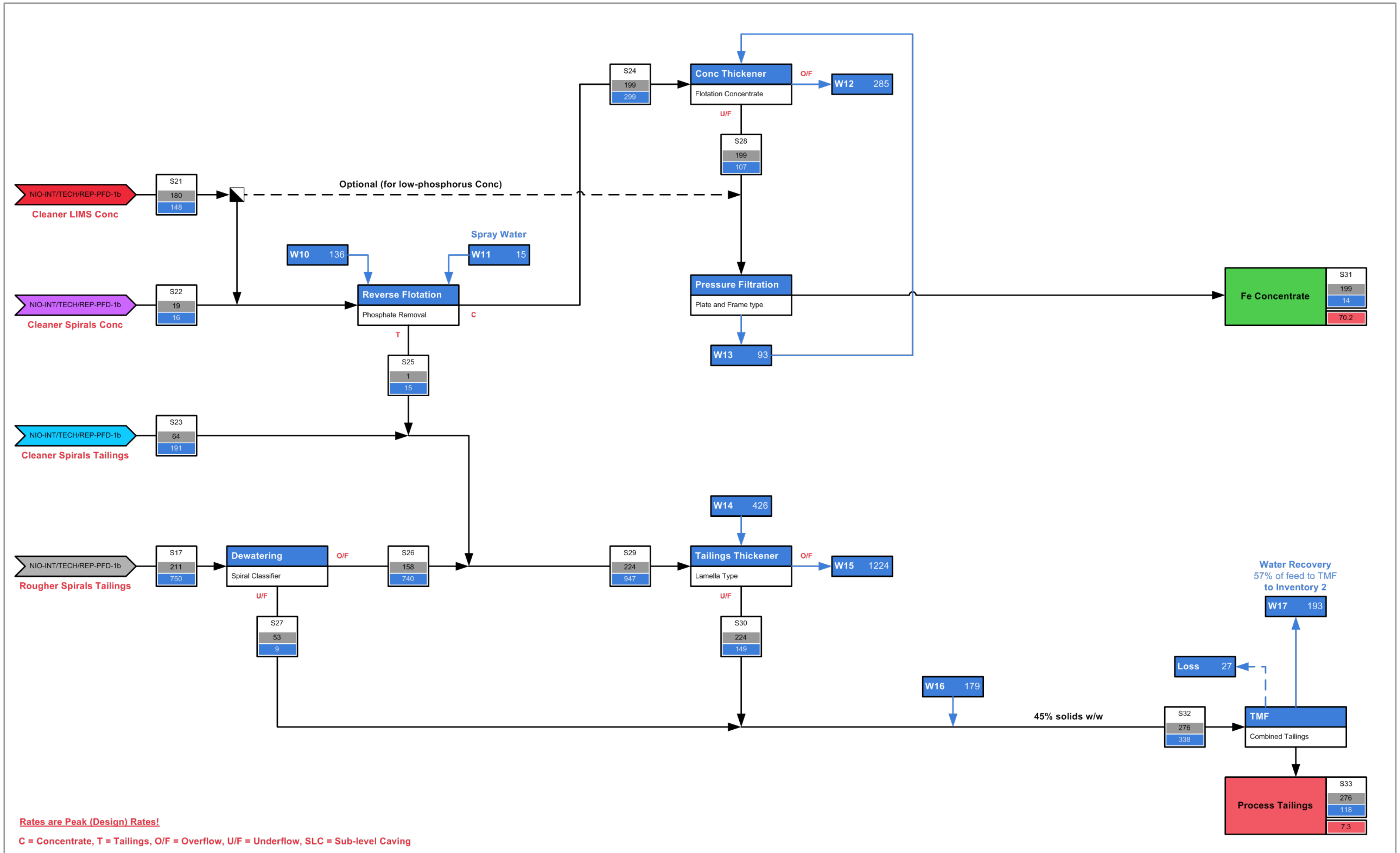

S1 690 21 33.6 — Stream Number; Mass [t/h], dry; Water [m³/h]; Fe [%]

Material Stream
Water Stream
Optional Stream
W1 11 Water In / Out [m³/h]

Revision Details	Rev	By	Apprvd	Date
Initial Release	0	MR	MRS	27/02/2015

Title	INTERIM TECHNICAL REPORT Blötberget Iron Ore Project - SLC Surface Process Plant 3/3				
By	MR	Rev.	0	Date	27/02/2015
Client No.	NIO2				
TSC No.	NIO-INT/TECH/REP-PFD-1c				

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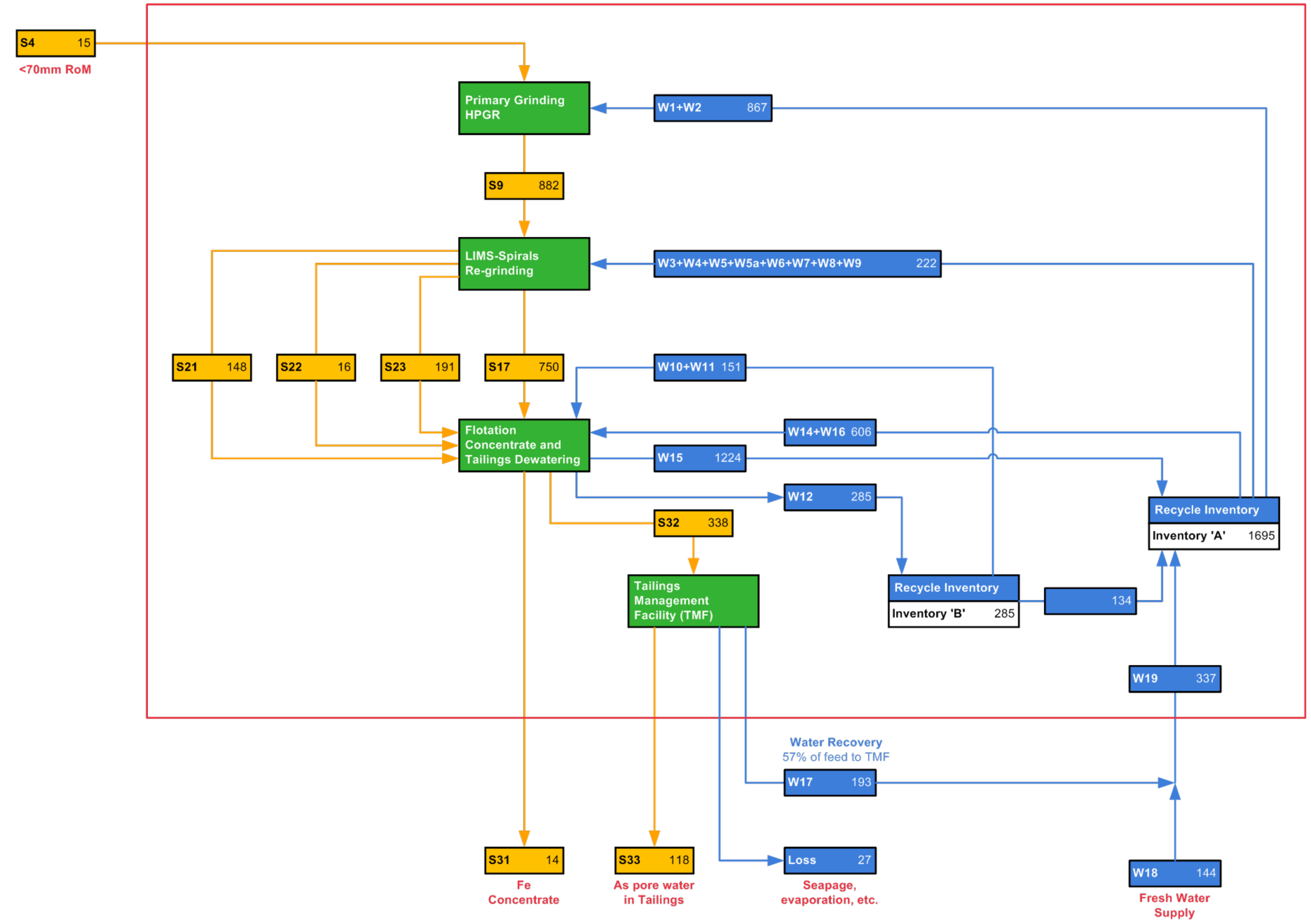

Rates are Peak (Design) Rates!

W1 20 Water Only Stream [m³/h]

S1 43 Solids / Water Stream, Water Volume Only [m³/h]

Please note: balance does not include any water losses that may be incurred through washing, spillage etc.

Revision Details	Rev	By	Apprvd	Date
Initial Release	0	MR	MRS	27/02/2015

Title	INTERIM TECHNICAL REPORT Blötberget Iron Ore Project - SLC Process Plant Water Balance				
By	MR	Rev.	0	Date	27/02/2015
Client No.	NIO2				
TSC No.	NIO-INT/TECH/REP-PFD-1d				

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8.6 Equipment Selection

8.6.1 Introduction

The discussion of equipment sizing and selection refers to the base case of mass balances estimated for the time when sub-level caving is employed, see Section 8.5, and to feed to the plant which has a head grade of 33.6 % Fe of which 76 % is present as magnetite. Reference is made to the mass balance for sub-level open-stoping and during which time the ROM head grade is expected to be greater at 36.1 % Fe, see also Section 8.5, where this is relevant to equipment selection.

The equipment has also been selected in order that the process plant can successfully recover concentrates from ROM ore having different ratios of magnetite: hematite where it affects equipment selection and sizing.

The inevitable consequence of the above selection criteria is that some equipment may appear to be over-sized with respect to the base case and that there is some redundancy of equipment during periods of ‘normal’ operation.

Process flow diagrams (PFD) for the base case (when SLC is employed) are presented in Section 8.5.

In the following sections, figures for nominal mass flow rate of solids are given as t/h and (t/h) for peak flow rates (+10 %).

8.6.2 Primary crushing

The mined ore is crushed underground by a jaw crusher to a particle size of less than 250 mm, see Chapter 5.3, and a machine similar to the Metso C160 jaw crusher has been selected.

The crushed ore is conveyed directly to the surface by conveyor during only two shifts of the working day; the third shift being reserved for hoisting of waste rock. The nominal hoisting rate is 627 t/h during 16 hours per day. The peak rate is 690 t/h, equivalent to 11,000 t/d.

8.6.3 Secondary crushing

At surface, the primary crushed ore is further crushed continuously to a particle size of less than 70 mm by a secondary cone crusher. A cone crusher of the type Metso GP550 (extra coarse) operating at a setting of 34 mm would be suitable.

The secondary crushed material is stored in a silo of capacity 5000 t or transported to an emergency stockpile of capacity 10,000 t. The processing plant has a peak capacity of 475 t/h and the silo is designed to store more than sufficient crushed ore (475 x 8 = 3,800 t) to feed the processing plant during the third shift when waste rock is being hoisted.

The emergency stockpile must be reclaimed by mobile plant but provides additional capacity to feed the plant for 24 hours in the case of unplanned stoppages in ore hoisting.

8.6.4 High Pressure Grinding Rolls (HPGR)

At a nominal rate of 431 t/h (475 t/h), the <70 mm crushed ore is fed from the silo to a mass flow bin and then to a high pressure grinding roll (HPGR) which crushes the ore before it passes to a dry, vibrating screen having apertures of 6 mm. The dry oversize is carried by the recirculation conveyor to the HPGR feed bin while the undersize passes to two (2) horizontal, vibrating, wet screens having apertures of 1.6 mm. The undersize of these screens form the feed to the rougher magnetic separators and the oversize is collected by the recirculation conveyor and returned to the HPGR feed bin.

A Weir Minerals RPS 7-170/110 HPGR (1,700 kW) was recommended for the duty following testwork conducted by Weir Minerals (KHD) at their facility in Cologne which determined an estimated specific energy consumption of 3 kW/t of product having a particle size less than 1 mm.

The HPGR will operate with a recirculating load of 120 % and suitable screens are as follows;

1. Dry screen with apertures of 6 mm accepting a nominal feed of 950 t/h (1,045 t/h peak rate) of crushed ore; one (1) off Metso RF3061-1, 3.0 m x 6.1 m or similar
2. Wet screens with apertures of 1.6 mm accepting the undersize product of the above screen at a nominal rate of 637 t/h (700 t/h peak rate); two (2) off Metso LH3061-1, 3.0 m x 6.1 m or similar

Alternatively, Weir Minerals recommended three (3) off MF2461-2 Banana Screens, 2.4 m x 6.1 m, double deck, with the lower deck equipped with water sprays.

The selected equipment taken from the ‘Modified Whyalla Design’ of HPGR circuit named after the operation of Arrium at Whyalla steelworks in Australia. Successful screening on an industrial-scale of the HPGR product at a fine particle size of a few millimetres requires wet screening but recirculation of the wet oversize material may incur severe wear of the roll surfaces (as a consequence of slippage).

Thus, the principal objective is to produce a finely sized product without the problems associated with recirculation of an excessively wet screen oversize material. Therefore, the circuit incorporates two stages of screening; the first at a relatively coarse particle size of 6 mm producing a dry oversize material and a second stage of wet screening at the final control size of 1-2 mm which produces a moist oversize material. The subsequent combination of the two oversize materials with new feed produces a feed to the HPGR which does not contain excessive moisture.

Two alternative methods of comminution were also considered and evaluated;

1. Conventional 3-stage crushing and grinding comprising an additional Metso GP550 tertiary cone crusher and two 900 kW, 3.5 m diameter x 5.5 m rod mills,
2. Semi-autogenous grinding (SAG) mill, 6.1 m (20 ft) diameter, 1,700 kW installed motor power, followed by ball mill (BM), 4.1 m diameter x 6.7 m, 1,600 kW installed motor power.

The HPGR was selected for the following reasons;

- Simplicity of the comminution circuit compared to use of rod mills,
- Elimination of equipment needed for handling and sorting worn rods in rod mills,
- Smaller footprint of equipment within a restricted area which had been defined by an earlier development permit,
- Lower operating cost, especially the elimination of media wear in the form of steel rods or steel balls in SAG mill,
- Similar capital costs at significantly reduced power consumption (see Table 8-17 and Table 8-18
- Potential reduction of power consumption in regrinding which has been observed as a benefit of HPGR saving up to 10 % power consumption,
- Potential enhanced liberation of minerals which has been observed and reported as a benefit of HPGR by a number of HPGR operators

Table 8-17and Table 8-18 provide information upon the specific energy and CAPEX requirements for the SAG-BM and Secondary Crusher-HPGR Options.

Table 8-17: Summary of SAG mill - BM option

Duty	Specific Energy	CAPEX
	kWh/t ore	US$
SAG Mill, 6.1 m x 5.94 m, 1,700 W inst. F(100) = 250 mm; P(80) = 3.0 mm	3.00	4,540,000*
Ball Mill, 4.1 m x 6.7 m, 1,600 W inst. F(80) = 3.0 mm; P(80) = 0.5 mm	2.90	2,790,000*
Total	**5.90**	**7,330,000**

* excluding motor starters, initial mill charge, discharge trommel housing and primary starters

Table 8-18: Summary of Secondary Crusher - HPGR option

Duty	Specific Energy	CAPEX
	kWh/t ore	US$
Secondary Cone Crusher F(100) = 250 mm; P(90) = 70 mm	0.57	1,487,500
HPGR	2.97	5,811,125**

Duty	Specific Energy	CAPEX
	kWh/t ore	US$
F(90) = 70 mm; P(80) = 0.5 mm		
Total	**3.54**	**7,298,625**

**including set of drives and set of STUD-PLUS lined rolls

8.6.5 Rougher Wet Low Intensity Magnetic Separation (LIMS)

The finely crushed product of the HPGR circuit is mixed with water at a rate of 431 t/h (475 t/h) to create a slurry of 35 % weight solids which is pumped to the top of a steel structure supporting the rougher magnetic separators which are of the low intensity wet drum type. Four (4) single separators in rougher-scavenger configuration having a drum measuring 1.2 m diameter x 2.4 m length and capable of treating 240 t/h were selected for the duty.

Based upon the performance of the pilot plant and laboratory equipment it is estimated that the feed will be separated into 211 t/h (231 t/h) of magnetite pre-concentrate (48.7 % wt) containing 59 % Fe and 222 t/h (244 t/h) of non-magnetic tailings (51.3 % wt) containing 9.5 % Fe.

The magnetic product of the rougher LIMS will be effectively dewatered to 60 % solids by weight by the separation process and will require some dilution by water addition so that it will flow by gravity to the screens which close the regrinding mill circuit, see 8.6.7.

8.6.6 Rougher Spiral Concentrators

The non-magnetic tailings of the rougher LIMS flow by gravity into pulp distributors positioned above 64 double start spiral concentrators arranged in four banks of 16 spirals. The number of spirals has been calculated based upon a unit capacity of 2.8 t/h per start plus an additional 20 % to allow for maintenance and cleaning. The spiral selected for this duty is the HC33WW.

The spirals recover the hematite from the tailings of the rougher LIMS to produce 29 t/h (31 t/h) of hematite pre-concentrate containing 45 % Fe.

Initially, and based upon the results of laboratory testwork, the flowsheet included wet high intensity magnetic separators (WHIMS or HGMS or SLon) at this roughing stage to recover weakly magnetic hematite. However, the unexpectedly poor performance of the SLon wet high intensity magnetic separators during the GTK pilot plant operation led to their replacement by spiral concentrators which proved to be superior especially at this roughing stage of gravity concentration.

These rougher spirals produce a final coarse tailings at a rate of 192 t/h (211 t/h) containing only 4.1 % Fe.

8.6.7 Regrinding Magnetite Pre-concentrate

In order to produce a high grade magnetite concentrate the pre-concentrate is reground to a particle size of less than 100 µm (d_{80} = 80 µm) to liberate the remaining gangue using vertical stirred media or 'tower' type of grinding mills of which type 'VertiMill' is the patented trade name of Metso.

Vertical stirred media mills were selected because (i) the mills occupy less floor space than conventional ball mills and it was necessary to lay out the equipment within a restricted space (ii) experience has shown a 20-40 % reduction of power consumption compared to ball mills (iii) experience has also shown a consequent 20-40 % reduction in grinding media consumption compared to ball mills. The previous limitation of the application of 'tower' mills has been the power draw and unit capacity but this has now been extended to +3,000 kW, more than sufficient for the envisaged duty of regrinding the pre-concentrates.

Based upon the results of the jar millings testwork (see Section 8.3), the necessary power to regrind 211 t/h (231 t/h) was calculated as 1,600 kW. In order to obtain commonality of equipment between the regrinding mills, flexibility in terms of varying ratios of magnetite: hematite and a future increase of ROM head grade, two (2) off 1,500 HP (1,120 kW) vertical stirred media mills were selected for this duty. These two mills will have the capacity to regrind the quantity of magnetite pre-concentrate generated by ore containing 100 % of the iron as magnetite. The Metso 'VertiMill' VTM-1500 would be suitable.

Normally, the vertical stirred media mills operate in closed circuit with classifiers, typically hydrocyclones. However, because the grind size is relatively coarse and the valuable minerals are much denser than the gangue there is a danger of over-grinding the pre-concentrates. Thus, it is proposed to use fine screens to close the circuit instead of classifiers. The mill product will be diluted with water to a pulp density of 50 % solids by weight and pumped up to the screens which will be mounted above the feed entry point of the grinding mill so that screen oversize flows by gravity into the mill. These same screens will also receive the magnetic product of the rougher LIMS, that is, the magnetite pre-concentrate. The screening action will remove most of the water from the oversize product such that the oversize will have a suitable pulp density for recirculation directly to the grinding mill.

At an assumed recirculating load of 100 %, the feed rate to these screens having a cut-point of 100 µm will be 421 t/h (463 t/h). Six (6) Derrick 5 deck, 'Stack-sizer' screens are proposed for this duty based upon applications in other iron ore projects and the advice of Derrick.

8.6.8 Cleaning the Magnetite Pre-concentrate

The closed circuit screen undersize which forms the reground magnetite pre-concentrate is pumped up to wet low intensity magnetic drum separators supported on a steel structure at a rate of 211 t/h (231 t/h). Four (4) double-drum, wet, low intensity magnetic separators have been selected for this duty. The drums have a diameter of 1.2 m and length 2.4 m and a rated capacity of 60 t/h each.

The cleaner LIMS produce 164 t/h (180 t/h) of final magnetite concentrate containing 70.5 % Fe which may require further treatment by froth flotation depending upon phosphorus content, see section 8.6.12.

The non-magnetic material is not yet final tailings because it may now contain hematite that has been liberated from association with magnetite. The quantity of this recycled material is 46 t/h (51 t/h) containing 18.3 % Fe but it is in the form of a very dilute slurry. This slurry is pumped to the screen in closed circuit with the hematite pre-concentrate regrind mill so that the hematite may be recovered by gravity concentration (spirals), see 8.6.9 below.

8.6.9 Regrinding Hematite Pre-concentrate

In the same manner as in 8.6.7 above, in order to produce a high grade hematite concentrate it is necessary to regrind the hematite pre-concentrate to a particle size of less than 100 µm (d_{80} = 80 µm).

In the base case, the feed to the regrinding circuit will comprise 29 t/h (31 t/h) of hematite pre-concentrate plus 46 t/h (51 t/h) of cleaner LIMS tailings giving 75 t/h (83 t/h). If the magnetite: hematite ratio of the ore increases, as may be the case in certain parts of the orebody, the load on the magnetite pre-concentrate regrind mills increases but there is more than sufficient installed power to regrind the magnetite pre-concentrate generated by ore containing only magnetite and no hematite. If the proportion of hematite increases, the load on the hematite pre-concentrate regrind mill will increase but that of the magnetite pre-concentrate mills will commensurately reduce. At a magnetite: hematite ratio of 60:40, which may be encountered in the upper parts of the orebody, the load on the hematite pre-concentrate mill would increase to an estimated 91 t/h (100 t/h).

Based upon the results of the jar millings testwork (see Section 8.3), the necessary power to regrind 75 t/h (83 t/h) of pre-concentrate was calculated as 910 kW. However, in order to obtain the ability to process ore containing as much as 60 % of the iron as hematite plus an increased ROM head grade and maintain commonality of equipment between the regrinding mills, one (1) off 1,500 HP (1,120 kW) vertical stirred media mill was selected for this duty. Again, the Metso 'VertiMill' VTM-1500 would be suitable as it is capable of regrinding up to 100 t/h of hematite pre-concentrate although, under normal conditions, it will operate at a turn-down ratio of about 30-40 % of full capacity.

Again, the vertical stirred media mill will operate in closed circuit with a fine screen.

At an assumed recirculating load of 100 %, the nominal feed rate to this screen, having a cut-point of 100 µm, will be 90 t/h (100 t/h). Two (2) Derrick 5 deck, 'Stack-sizer' screens are proposed for this duty.

8.6.10 Hydrosizer

A hydrosizer or elutriator uses an upward current of water to separate minerals by a combination of size and specific gravity. Heavy and large particles preferentially settle against rising water and are discharged via spigot valves at the bottom of the conical vessel whilst light, fine particles report to the overflow stream. Hydrosizers can also be operated as 'teeter bed separators' in which case the heavy particles settle through a bed acting as an artificial heavy medium.

For the present case, the hydrosizer serves the principal purpose of eliminating excess water ahead of the cleaner spiral concentrators during periods in which the solids loading in the Derrick Stack Sizer screen underflow is low (NB: This will occur when the ore contains only little hematite). However, finely sized, liberated gangue (of low S.G.) will report to the overflow as well which is anticipated to benefit the subsequent cleaner spiral concentrator operation.

It is expected that, during much of the operation of the process plant, the hydrosizer will be by-passed as the solids contents in the Derrick Stack Sizer screen underflow stream will be sufficiently high (25-35 % solids by weight) for the subsequent cleaner spiral concentration stage.

Hydrosizers or elutriators can be considered a well-proven and established mineral processing technology for the treatment of iron ore fines; amongst the main advantages are that they are relatively capital inexpensive and not significantly affected by variations in feed rate. Occasional operational problems are chiefly associated with process disturbances caused by oversize material (> 3-5 mm in size) in the feed, resulting in cyclical discharge patterns of the installed units.

One (1) MEP Ø 2.4 m TBS Hydrosizers would be suitable for this duty.

8.6.11 Cleaner Spiral Concentrators

The reground hematite pre-concentrate will be treated by spiral concentrators to produce a hematite concentrate.

During the pilot plant operation at GTK, the spiral concentrator proved to be superior to the SLon high intensity magnetic separators which had been proposed for this duty on the basis of results of bench-scale metallurgical testwork. Nevertheless, although the performance of the spiral concentrator was better, it did not recover the particles of hematite with a size less than 37 µm. Consequently, the recovery of hematite is only approximately 50 % compared to 98 % recovery of magnetite. It is expected to increase the recovery of hematite through further investigation and metallurgical testwork. Such testwork is currently being planned.

The number of spiral concentrators has been calculated as 100 based upon a unit capacity of 0.9 t/h per start and allowance of an additional 20 % for maintenance and cleaning.

It is proposed to install 128 spirals arranged as four (4) banks of 16 double-start spirals, similar to section 8.6.6.

8.6.12 Phosphate Reverse Flotation

Although much of the bench-scale, metallurgical testwork (Davis Tube Tests) upon drill core samples recovered from various parts of the orebody indicated that the magnetite concentrate will contain a very low level of phosphorus and it is uncertain whether some magnetite concentrates will require treatment by reverse flotation of apatite, the principal phosphorus mineral, to reduce the phosphorus to an acceptable level.

On the other hand, it is almost certain that the phosphorus content of the hematite concentrate will exceed the typical specification for a saleable iron ore concentrate if marketed alone.

Thus, the processing plant has been designed with sufficient flexibility to;

1. Combine both concentrates and treat by froth flotation to remove phosphorus,
2. Process the hematite concentrate alone by froth flotation to remove phosphorus,
3. Blend both concentrates to achieve an acceptable phosphorus content of the combined concentrates (without the need for froth flotation).

This is achieved by installing sufficient flotation capacity to treat the combined concentrates at a rate of 180 t/h (199 t/h) but also providing a by-pass and arrangements such that not all the flotation cells in the bank need to be active. The mass recovery of iron ore concentrate by flotation is expected to be 99.5 % thereby yielding 180 t/h (199 t/h) of concentrates.

The flotation cells will be preceded by two conditioner tanks for addition of pH modifier, depressant and collector reagents. These will have volumes of 45 m^3 and 19 m^3 to provide ten (10) minutes and five (5) minutes residence time respectively.

It is proposed to install 4 x 20 m^3 cell-to-cell machines in series to provide at least 15 minutes residence time. The Metso RCS20 flotation cell would be a suitable machine.

8.6.13 Dewatering of Concentrates

In the circumstance that flotation is required, the flotation cell underflow, with a solids content of about 40 % w/w, forms the final combined concentrate but will require thickening before filtration. A conventional thickener of 4 m diameter was selected for the duty based upon jar settling testwork and a specific settling rate of 30 t/h/m^2.

In the circumstance that flotation is not required, both the magnetite concentrate produced by cleaner LIMS and the hematite concentrate produced by spiral concentrators will have sufficiently high pulp densities (weight percent solids) to be pumped directly to the filters.

Based upon the testwork carried out (see Section 8.3), one (1) VPA 2050-56 or similar is recommended to produce a filter cake containing 6.5 % w/w moisture.

Estimated total cycle time of three (3) minutes comprising one (1) minute chamber filling, 30 seconds cake compression and 1.5 minutes air blowing. It was shown as possible to reduce the moisture content to 4.5 % w/w by extending the air blow period to six (6) minutes but this would incur substantially greater capital and operating costs.

Alternatively, three (3) off Metso top-fed TFF3030 (3.0 m diameter x 3.0 m) rotary drum filters would provide sufficient 'active' filtration area to filter the combined concentrate at the rate of 180 t/h (199 t/h). In this case the filter cake is expected to contain 9 % moisture.

8.6.14 Dewatering of Tailings

The processing plant will produce 251 t/h (276 t/h) of tailings which will be a combination of (i) coarse tailings of the rougher spiral concentrators (ii) fine tailings of the cleaner spiral concentrators and (iii) froth of the reverse flotation process. The tailings will arise as pulps containing approximately 30 % solids by weight.

As mentioned several times previously, a design criterion for the processing plant was that the equipment should be accommodated within a restricted area that had been consented for a processing plant under a previous permit granted to Nordic Iron Ore. Thus, a high-rate type thickener was selected over a conventional circular tank thickener to minimise the footprint.

The advice of a supplier of Lamella thickeners, Metso, based upon their experience with the design of the concentrator for the similar Northland project at Kaunisvaara, Sweden, was that the thickener should be preceded by a classifier to remove coarse, sand-sized particles.

Thus, a Metso 120FF screw or spiral rake classifier is proposed followed by a Metso LTE 800 Lamella thickener of 10.5 m diameter. The screw classifier will have a nominal size cut-point of 250 µm and recover between 15 and 25 % by weight of the solids as a moist sand. The coarse sand product of the screw classifier will be conveyed to a pump sump where it will be mixed with the underflow of the Lamella thickener and the pulp density adjusted to 45 % weight solids by addition of water before pumping to the tailings disposal facility see Section 8.9.

8.7 Plant Layout

The plan layout of the process equipment is shown in Figure 8-4 below.

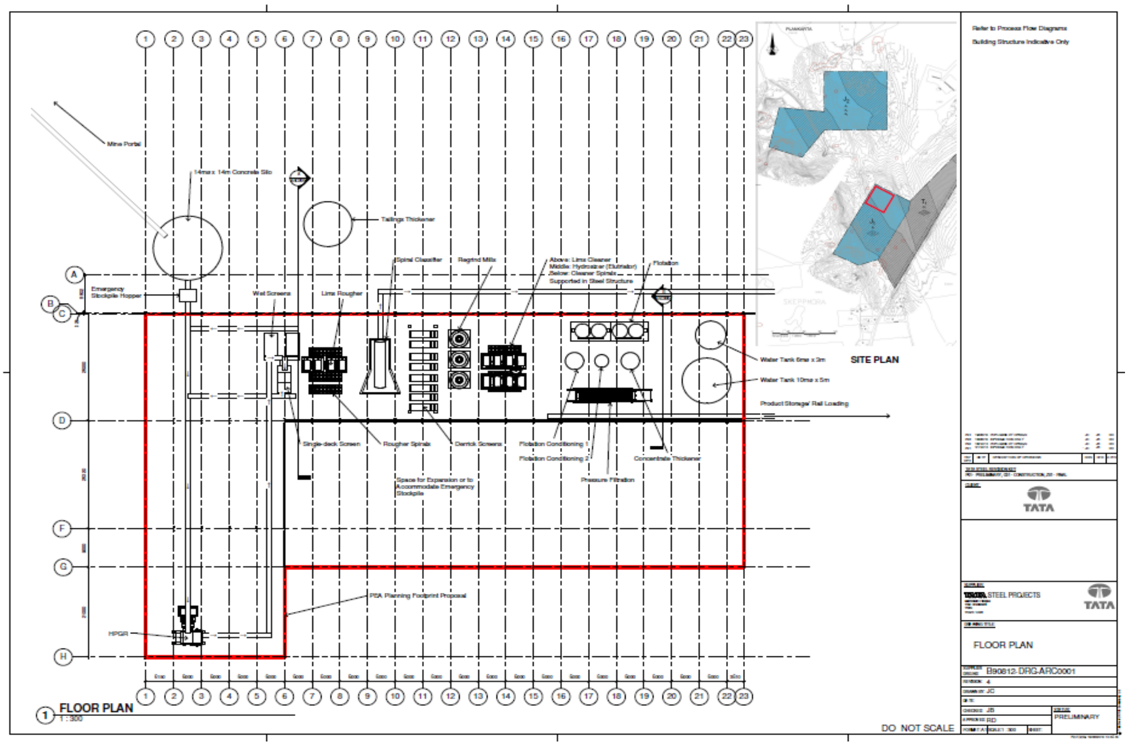

Figure 8-4: Plan layout of process equipment

The orientation of the process plant is arbitrary and the orientation shown in Figure 8-4 assumes that the secondary crushed ore will be conveyed from the mine portal located to the upper left of the drawing.

Figure 8-4 also shows the outline (in red) of an area previously consented for the construction of a processing plant associated with the proposed development of the Blötberget, Håksberg and Väsman deposits. The plant equipment has been laid so that the plant building is within this consented area.

In fact, it has been possible to lay out the process equipment within approximately half of the consented area by selecting equipment, wherever alternative equipment was available and practical, which had the smaller footprint. This applies particularly to:

- use of HPGR in place of a conventional two-stage crushing and screening plant and rod mills;
- use of vertical stirred media mills in place of conventional ball mills;
- use of double-start spiral concentrators;
- use of high-rate ‘Lamella’ tailings thickener and, finally,
- use of vertical arrangement of the magnetic separators and gravity concentration equipment.

The remainder of the consented area may be reserved for additional processing line(s) in case for future expansion.

However, it must be noted that this did not compromise the process plant in terms of efficiency or capital cost.

Indeed, the adoption of HPGR and vertical stirred media mills significantly reduced the operating costs and vertical elevation of the magnetic separators allowed the products of these separators to flow by gravity to the spiral concentrators thereby avoiding pumping as shown by the elevations of the processing plant presented in Figure 8-5, Figure 8-6 and Figure 8-7 respectively.

In general, the process equipment is laid out in the same order as the process flowsheet with a few notable exceptions.

The 'L' shape of the process plant building is the result of the need to elevate crushed ore and recycled oversize material to the feed bin above the HPGR and to the sizing screens themselves. A cautious inclination of 15 degrees (gradient 1 in 7) was selected for the conveyors in the HPGR circuit necessitating a length of approximately 80 m to raise the material to the feed bin. Consequently, the HPGR circuit could not be conveniently accommodated longitudinally in the building.

The other major exception is the location of the spiral classifier which receives the rougher spiral tailings. The classifier was positioned immediately below the rougher spirals to avoid pumping of this coarse material.

The processing plant building also accommodates the two, large water storage tanks of the site infrastructure, the positions of which are arbitrary at present.

Overall, the compact layout achieved creates a substantial, free and consented area which could be used for future expansion of the processing plant or other facilities. Further compaction would be possible but the layout shown meets the principal objective and permits access to all major equipment by overhead travelling cranes.

Isometric views of the processing plant are shown in Figure 8-9 and Figure 8-10.

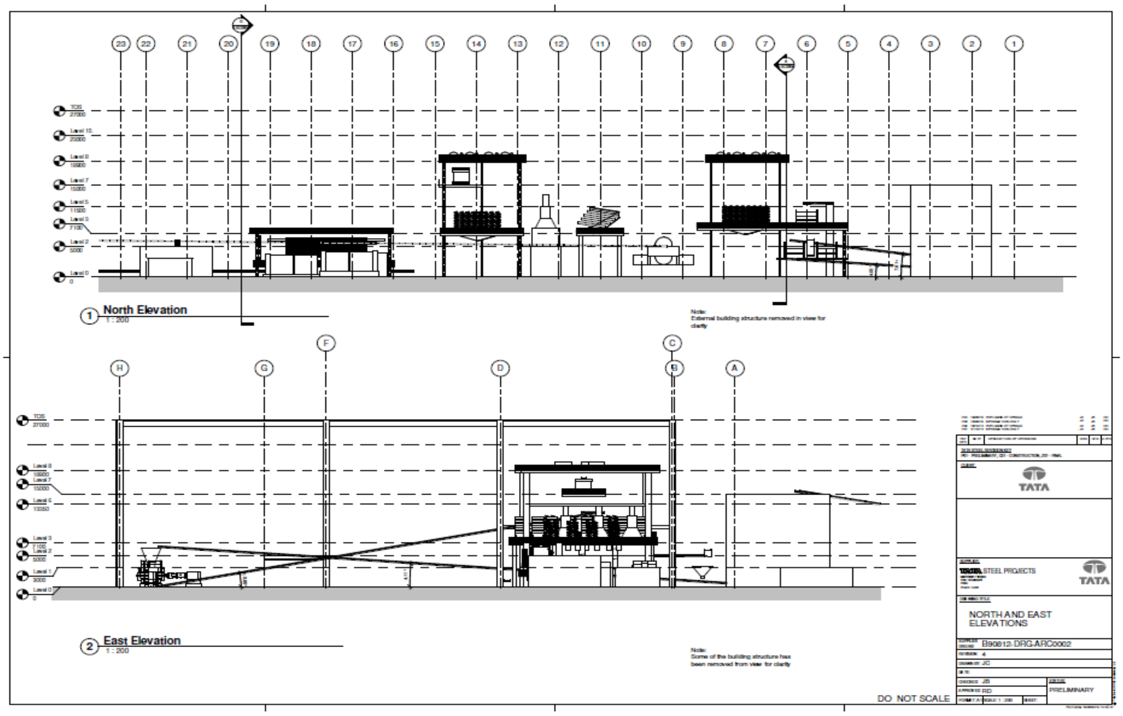

Figure 8-5: North and east elevations of process plant

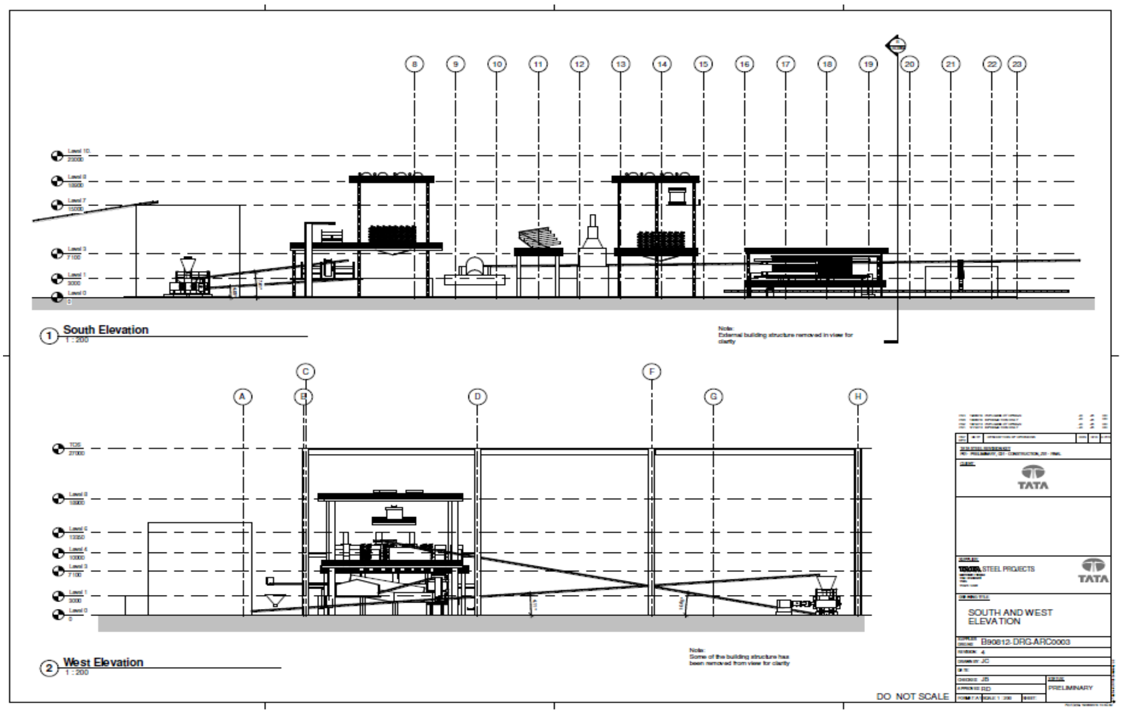

Figure 8-6: South and west elevations of process plant

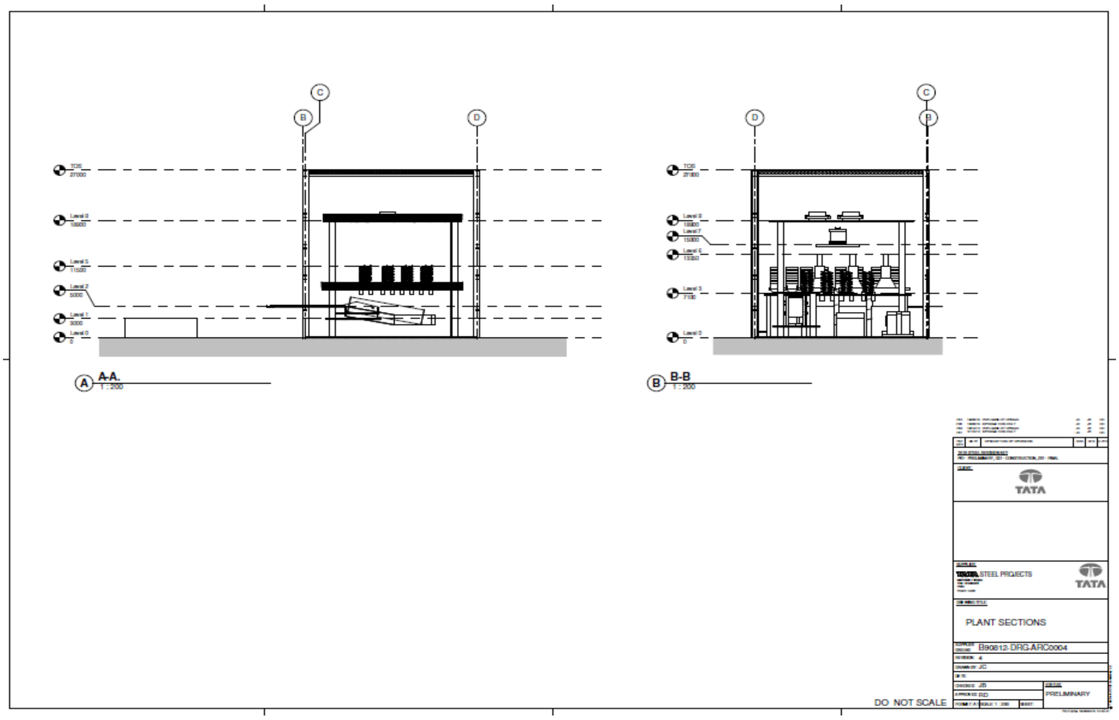

Figure 8-7: Process plant sections A-A and B-B

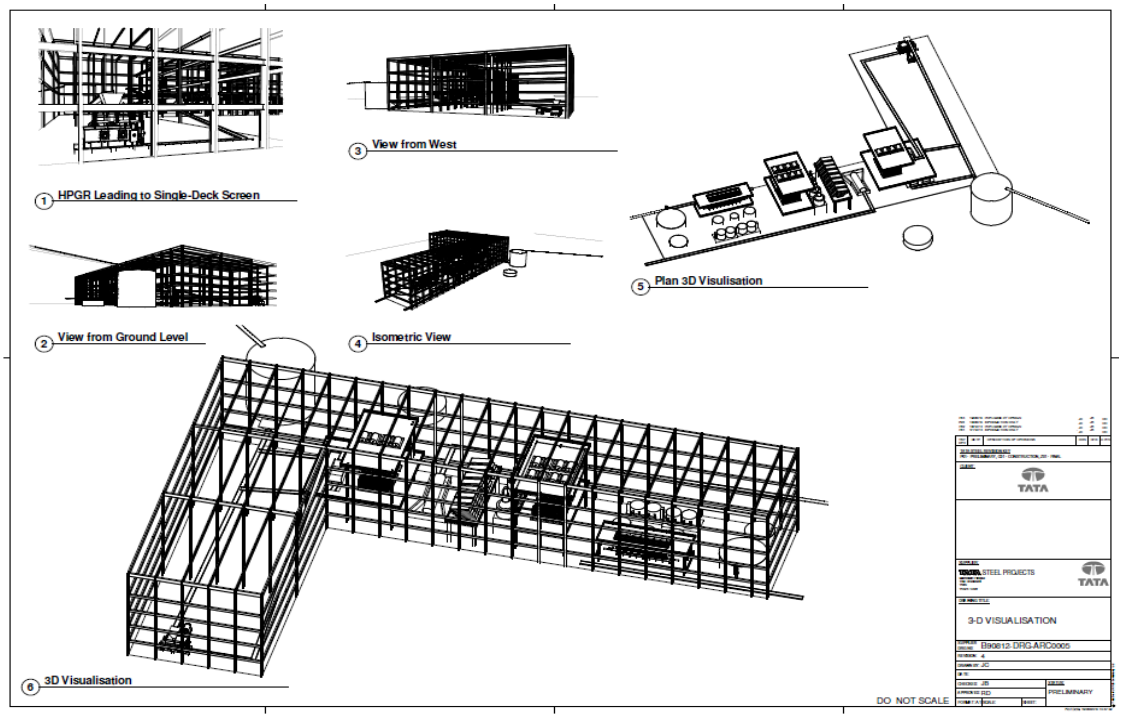

Figure 8-8: Process plant isometric view A

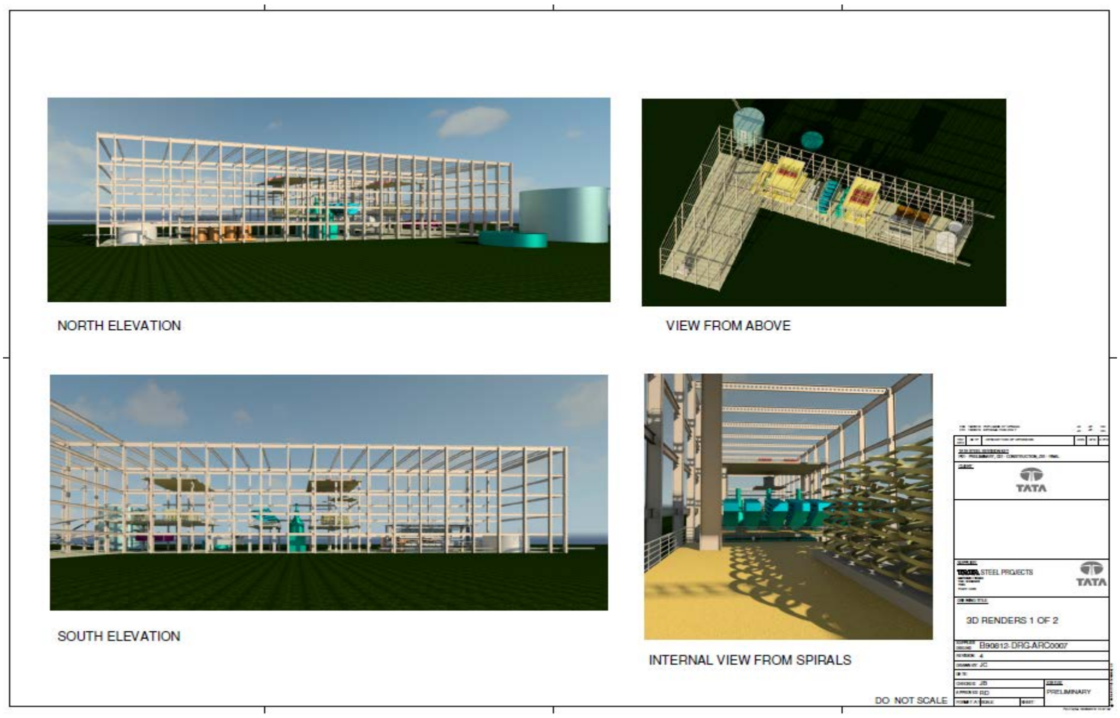

Figure 8-9: Process plant isometric view A

Figure 8-10: Process plant isometric view B

8.8 Tailings Pipeline to/from TMF

8.8.1 Design Duty

As can be seen in Section 8.5, 'Mass and water balances', under normal operating conditions the processing plant will produce 251 t/h (276 t/h) of solids as tailings. It is required that a slurry containing 45 % w/w of these solids having an assumed density of 3,000 kg/m^3, should be delivered to the Tailings Management Facility (TMF) where the slurry will be deposited through spigots onto the beach of a conventional tailings dam. Samples of tailings collected during the operation of the pilot process demonstration plant at GTK indicated that the solids would have a wide particle size distribution of 100 %wt passing 1.0 mm and a d_{50} of about 0.15 mm.

The slurry was calculated to have a density, ρ, of 1,430 kg/m^3.

8.8.2 Route and Profile

The most direct route between the processing plant and the TMF has a length of approximately 1,600 m but includes an initial rise to a height of about 20 m above the processing plant at a distance of 800 m followed by an initial descent to an elevation of -10 m, see Figure 8-11 and Figure 8-12 below.

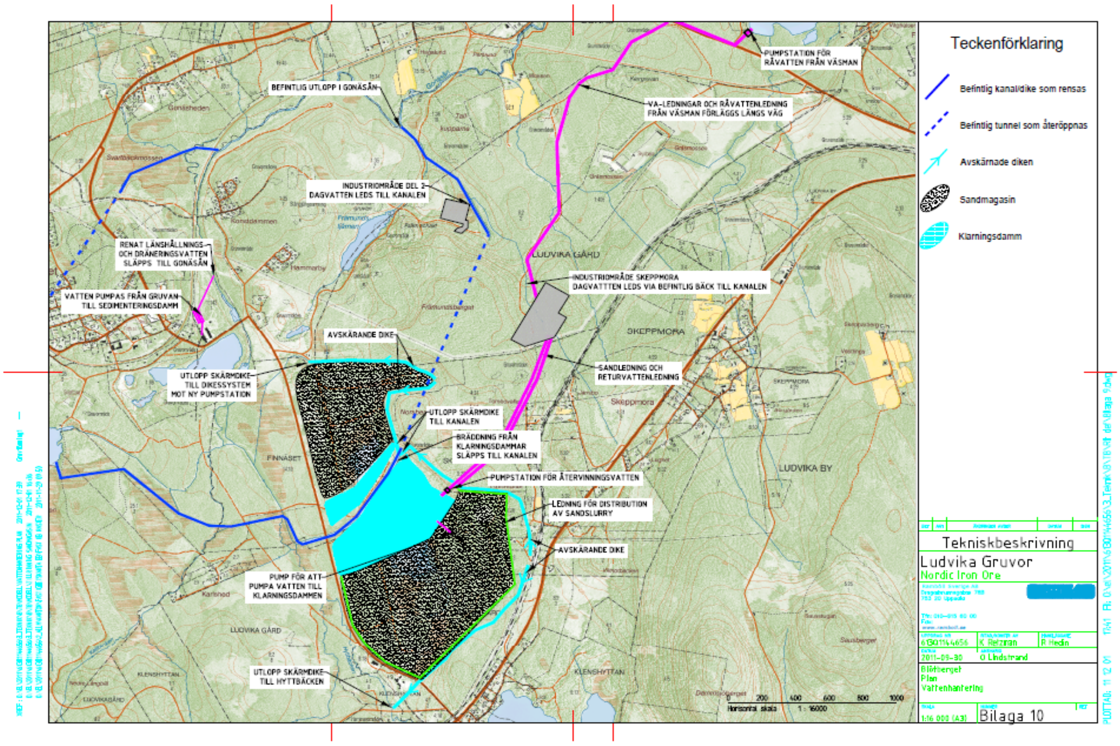

Figure 8-11: Proposed route for tailings pipeline

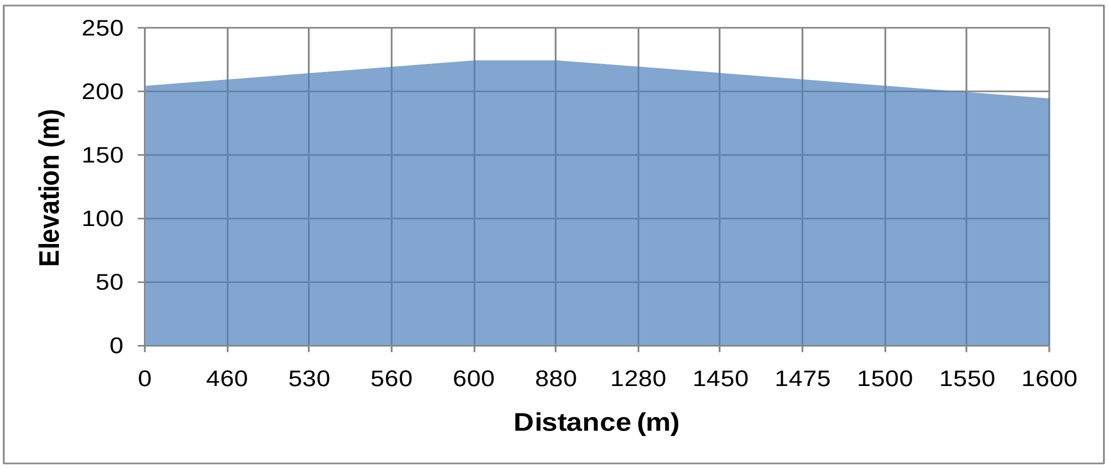

Figure 8-12: Profile of proposed tailings pipeline

It has been assumed that there are no constraints upon the route imposed by land-ownership or environmental issues.

An alternative route avoiding the hill was considered but the increased length incurred a significantly greater pressure loss so it was discarded.

8.8.3 Minimum Slurry Velocity

The principal consideration in the design of the tailings pipeline is to ensure that the solids will remain in suspension and not settle out causing blockages and excessive wear. Use was made of the empirical nomogram devised by Durand to estimate the Limiting Settling Velocity, V_L. This requires iterative calculations of the slurry velocity, V, from assumed values of pipe diameter, D, and the values of mass flow rate and solids % w/w given above until the values of V and V_L converge. The minimum settling velocity was estimated to be 3.4 m/s giving a pipe diameter of 200 mm and a value of V = 3.8 m/s (+10 %) was taken for design purposes.

8.8.4 Pressure Losses

The pressure loss through a pipeline comprises head loss owing to flow plus static head loss. In this system, the flow is actually fully turbulent and Reynolds Number exceeds 10^5 so viscosity and friction losses are not relevant.

In fact, the static head loss is negative (-1.4 bar) and favours flow but calculations revealed that this is relatively unimportant compared to pressure loss due to flow. It is, however, important that the pump has sufficient power to raise the slurry to a height of +20 m when the pipeline is empty on start-up; equivalent to a static head loss of 2.8 bar plus pressure loss incurred by flow through 800 m of pipe.

The pressure loss was estimated using values of the Darcy friction factor, f_D, taken from the chart devised by Moody and assuming that the pipeline would be made from ‘plastic’ such as HDPE or lined with smooth abrasion resistant material. In fact, at such high values of Reynolds Number, the pressure loss is not very sensitive to the assumed roughness of the pipe surface. Pressure loss was calculated from the following equation;

Pressure loss, Pa = $f_D(L x \rho x V^2)/2D$ where L is pipeline length, m.

Thus, the pressure loss owing to flow was estimated to be 11.1 bar (1 bar = 10^5 Pa).

Allowing for the static head loss (gain), the total head loss was estimated at 9.8 bar.

8.8.5 Pump Power

The theoretical power required to pump the slurry against the calculated total head is given by;

Power, kW = (Head (Pa) x Flow rate (L/s))/1000 = 117 kW

However, the presence of solid particles in the slurry significantly reduces both the head generated by a centrifugal pump and the efficiency when compared to pumping water. This effect is quantified by the 'Head Ratio'(HR) and 'Efficiency Ratio' (ER) which are used to reduce the performance of a pump when measured pumping water, that is, the typical published pump characteristic curves. Using the nomograms presented in the Weir Slurry Pumping Manual, the HR and ER were estimated to be equal and 0.85.

Thus, assuming an efficiency of 70 % for a centrifugal slurry pump when pumping water, the overall efficiency becomes 0.7 x 0.85 = 64 % and pumping power becomes 187 kW.

The required head becomes 9.8/0.85 bar = 11.5 bar.

This head is also more than sufficient to raise the slurry to the highest point of the pipeline during start-up (+2.8 bar) plus pressure loss owing to flow through 800 m (7.1 bar).

8.8.6 Pump Duty and Selection

In summary, the estimated duty of the pump is;

Power:	187 kW	
Flow rate:	119 L/s	1,800 USgpm
Head loss:	11.1 bar	500 ft H_20

The pumps were selected from information published by Weir Minerals, formerly Warman International, for the AH range of centrifugal pumps suitable for handling highly abrasive slurries. A low speed of rotation of the impeller was deliberately chosen in order to prolong wear life and maximise availability. Similarly, the pumps were selected so that the duty fell well within the capability of the pump and not at an extreme.

It was assumed that a complete set of spare pumps would be required.

Thus, the selection was six (6) off 8/6 AH Weir pumps arranged as two sets of three (3) pumps in series.

8.8.7 Return Water Pump

It is estimated that, on average, 337 m^3/h of water will require pumping back to the processing plant. The duty of the return water pump was calculated in exactly the same manner as that of the tailings pump with the results being;

- Power: 90 kW
- Flow rate: 94 L/s 1,415 USgpm
- Head loss: 7.2 bar 206 ft H_2O

A centrifugal pump suitable for pumping water or mildly abrasive slurry was selected such as Weir (Warman) 150QC-DWU which is also a 200 mm/150 mm or 8”/6” frame pump where the dimensions refer to the diameter of inlet and outlet respectively.

8.8.8 Pipeline

The tailings pipeline has an estimated diameter of 200 mm and nominal length of 1,600 m. It must withstand a maximum pressure of 11.5 bar or 23 bar with a factor of safety of 2. It is assumed to be made of HDPE and butt welded.

The return water pipe is almost identical but the wall thickness can be less since it must withstand a maximum pressure of only 7.2 bar.

The pipelines will be buried and/or lagged and hot-wire traced to provide protection from freezing during the winter months.

8.9 Tailings Management Facility (TMF)

To store all of the tailings produced over the LOMP, two separate facilities are needed, referred to as the Northern TMF and the Southern TMF.

The Northern TMF has been designed with a capacity of 5.5 Mm^3 and the Southern TMF has a capacity of 12.5 Mm^3 giving a total capacity of 18 Mm^3. The estimated total tailings produced for the LOM is 17 Mm^3 therefore the designed TMFs have enough capacity.

The location of the TMFs has been selected with consideration of the existing tailings so that tailings deposition can continue in the same location. The CPs are located adjacent to the TMFs and also utilise the presence of low laying wetlands which can be used as settlement ponds with easy discharge into the canal leading to the River Gonäsån. It is also a relatively short distance to the processing plant for pumping.

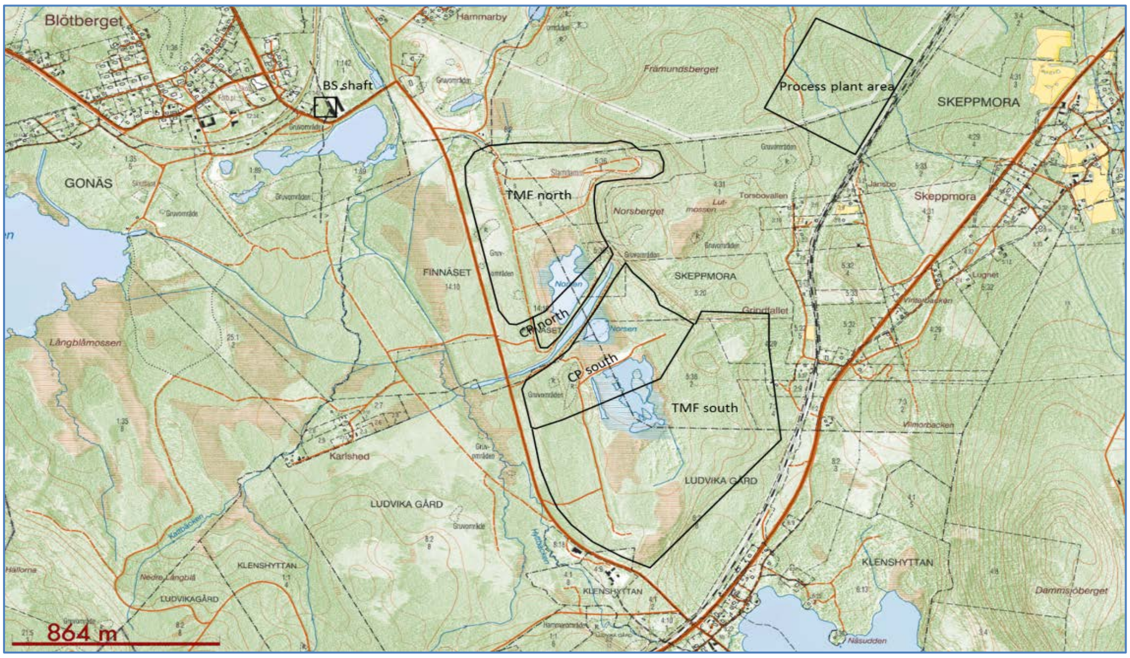

Figure 8-13: Location of TMFs and CPs

The Northern TMF will be constructed first to keep Capex to a minimum during the early stages of the project.

Part of the water management system will incorporate two clarification ponds (CP) that will handle decant water from the TMFs, surface water run-off and ground water from the mine site.

The Northern CP will be constructed together with the Northern TMF and will be sufficient to handle tailings and water for the first seven (7) years of production. Construction will start on the Southern TMF and CP in year five (5) so that tailings storage can continue after year seven (7).

The clean water from the clarification ponds will be pumped back to the process plant with excess water discharged to the River Gonäsån.

Following testwork using samples from the pilot test plant it has been determined that the target solids content of 45% w/w provides good segregation.

The Northern TMF dam will be constructed utilising an initial starter dam made from the localised moraine material found in the area. The tailings will be pumped to the spigot of the starter dam such that coarse material will settle raising the height of the dam in stages using conventional hydraulic deposition.

The design of the dam structures has been carried out according the Swedish Hydropower Industry guidelines, RIDAS. This covers the construction, operation, maintenance and surveillance for water retention dams. The Mining Industry also has guidelines, GruvRIDAS, which are based on RIDAS and cover a number of additions that are specific to tailings management facilities.

A comprehensive technical study has been carried out by Ramböll Sverige AB that details the material selection, design, testing, construction and management of the TMF and water management for the Blötberget mine. This report can be viewed in full within the Appendix H of this report.

8.9.1 Site Water Balance

All process water will be pumped from the clarification ponds with a small amount coming from the Blötberget shaft if required.

Excess water will be discharged into the canal flowing into the River Gonäsån.

The exception to this will be during four winter months when the drainage water will be frozen. It is estimated that 0.8Mm^3 of water will need to be provided by Lake Väsman via an extraction licence. This is an over-estimation in the calculations and may be considered as a backup supply. In general there is an overall net excess of water.

8.9.2 References

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9 SURFACE INFRASTRUCTURE AND UTILITIES

9.1 Introduction

For Phase 1 of the Project, NIO intends to develop an Industrial complex at Skeppmora, adjacent to the main railway line between Ludvika, Göteborg and Orebro.

Two land parcels have been permitted for this development namely; Industrial Area 1 and Industrial Area 2. See Figure 9-1.

The Industrial Area 1 will be placed close to the railway. Within the industrial Area 1 the process plant and associated facilities, vehicle work shop, filling station, office/locker room, stores, incoming transformer, switchgear, drill core archive, emergency stockpile, general maintenance are placed.

The Industrial Area 2 will be a gravel area that will be used by contractors for portable cabins etc. There will be also a temporary stocking area for waste rock.

The structures and facilities planned at the Skeppmora industrial complex include:

- Site access road from the public Gonäs road
- roads within and between the industrial areas
- service buildings for the mining operations
- the surface portal of the main access decline
- Various stockpiles and storage facilities for ore, waste and iron ore concentrate
- transfer conveyors for ore, waste and concentrate product
- buildings housing the concentration plant, tailings pipeline, offices etc.,
- fire and fresh water facilities
- electric power installations and distribution network to surface and underground facilities
- communication network
- Railway terminal and concentrate load out facility.

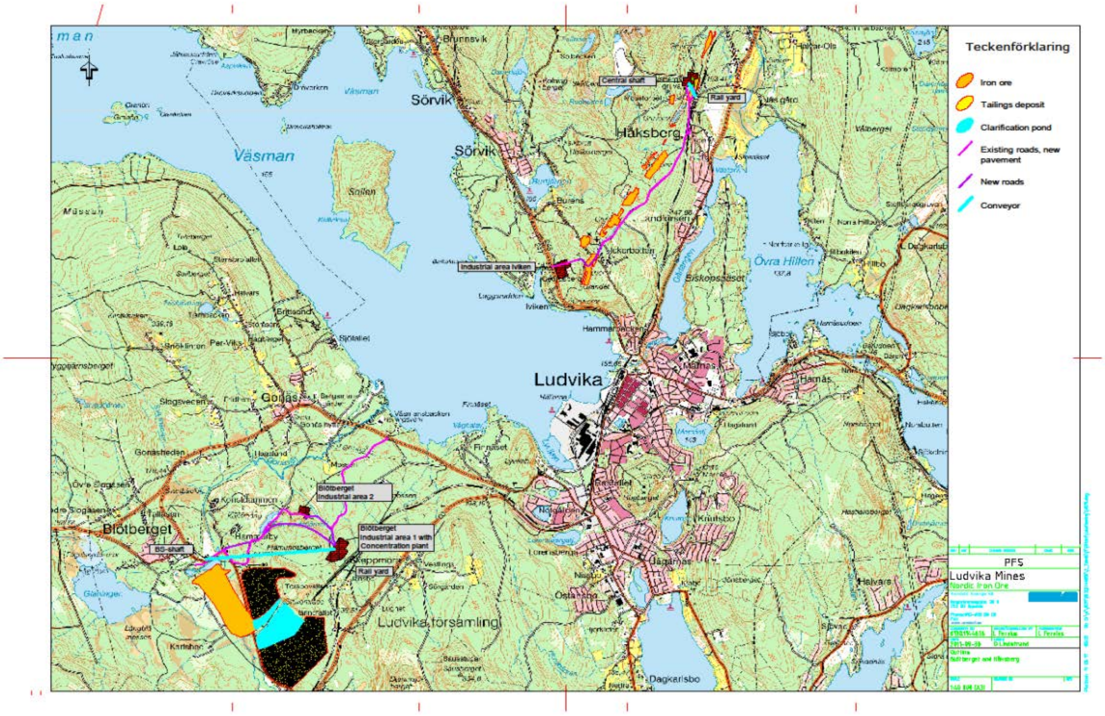


Figure 9-1: Project Areas

9.1.1 Industrial area at Skeppmora

This area will be the site of the rail terminal and the connection to the main railway line, which is discussed in detail in Section 10.2.

9.2 Industrial Area 1 Layout and Building Infrastructure

9.2.1 Site Clearance and Levelling

The selected site area for Industrial Area 1 is covered by various depths of moraine overlying the rock head. The area is also heavily forested which will be felled as part of the site preparation works.

It is intended to establish the main site platform at a level of approximately 200 masl.

A limited geophysical survey has been undertaken to establish the depth of moraine and overburden and the level of rockhead across the intended industrial site area. No intrusive site investigation survey has been undertaken to date.

Based on the rockhead contour map generated from the survey data, an estimate of the cut and fill requirements for the industrial site, the tailings dams and the track embankment has been prepared. It is calculated that 200,000 m^3 of waste rock will arise from the underground development work during the construction period.

It is estimated that 32 000 m^3 of rock fill is needed for site levelling, the rail embankment requires 225, 500 m^3 and the TMF dams will require 140, 000 m^3. The total rock extracted during the two years of construction will be approximately 416,000 m^3 while the rock fill and construction demand is approximately 365,000 m^3.

This preliminary assessment suggests that there should be sufficient rock arising from the various sources to satisfy the rock fill needs of Phase 1 of the Project with a positive balance of about 51,000 m^3.

9.2.2 Buildings and Structures

This section concerns the surface buildings and structures that are located within the Industrial Area 1. The location, type of building/structure, size and cost have been considered for each of the buildings that will be required to support all underground mining activities.

The buildings and structures are listed in the table below;

Table 9-1 Building & Structure Reference Drawings

Description	Quantity	Size	Ref Drawing
Office & Locker Room	1	1151m^2	A-40.1-101 A-40.2-111 A-40.3-121 A-40.3-122
Drill core Archive	1	885m^2	A-40.1-201 A-40.2-211 A-40.3-221 A-40.3-222
Stores Building	1	1800m^2	A-40.1-301 A-40.2-311 A-40.3-321 A-40.3-322
Process Plant Building	1	8965m^2	A-40.1-501 A-40.1-502
General Maintenance Workshop	1	473m^2	A-40.1-701 A-40.2-711 A-40.3-721 A-40.3-722
Vehicle Workshop	1	1015m^2	A-40.1-801 A-40.2-811 A-40.3-821 A-40.3-822
BS Shaft Air Supply Building	1	374m^2	A-40.1-901

A Drawing List is provided in the Appendix A to this report for further reference.

9.2.3 Office Building and Locker Room Facility

This is a single storey composite steelwork building with a sloping pitched roof; constructed using traditional concrete base and strip type foundations with a concrete floor slab. The steel frame will be clad using a combination of brick/blockwork cavity walls and SIPs. Internal walls will comprise non structural studwork partitions.

The building will contain both the mine site office facilities and the shift change-over facilities incorporating its own plant room for services control and distribution. The building forms an ‘L’ shape with each wing serving different functions. The total floor space is approximately 1,151 m^2

The building will be the focal point of the mine site surface support area comprising the main entrance and office facilities for all the site management and technical support staff. There are 13 offices, a conference/meeting room and a medical centre. In addition there is an open-plan room that will contain the technical support staff for Mine Planning and Geology departments.

The building will also contain a self serviced canteen/dining area for around 54 personnel per sitting.

The rest of the building will contain the changing rooms, washing facilities and lockers for the shift changeover personnel. This area will be divided into clean and dirty areas and will be accessed via personnel doors to the rear of the building giving access to the car parking facility and to the front of the building giving access and egress to the site. A separate lamp room is provided with access/egress externally from the site side of the building.

Drawing number A-40.1-101 (See Appendix B) shows the layout of the floor plan and drawings A.40.2.111, A.40.3.121 and A.40.3.122 show sections through the building and the elevations.

9.2.4 Drill Core Archive Store Building

This is a single storey open plan building that will contain the core boxes and associated materials in a protected environment.

There will be approximately 885 m^2 of storage space available, for the core boxes, which can be stacked to over 6m high. The core boxes can be moved around within the building and stored using pallet trucks. Two 5m high access doors will make access/egress easier for vehicles such as forklift or telehandlers. The building can be accessed via several personnel doors located adjacent to the vehicular access doors and on the opposing elevation.

The building will be of similar construction to the office block, i.e. a composite brick/block steel frame, constructed on reinforced base and strip type foundations and clad with SIPs panels. The building will be provided with electrical power for lighting and power for equipment. Additional light will be provided by the provision of full height opaque windows to both gable ends of the building.

Please refer to drawings listed in Table 9-1 above for details of layout and elevation which are contained in the Appendix B of this report.

9.2.5 Stores Building

This building is exactly the same as the drill core archive building however it is much larger at 1,800 m2 to accommodate parts, materials and equipment for the underground conveying equipment, mobile/fixed plant and general sundries.

Any heavy pieces of equipment that require internal storage can be contained within the building via access through two 5m high 4.5m wide doors. Once inside parts and equipment can be moved around within the building either using forklift trucks or pallet trucks.

The exact internal layout of the building can be tailored to suite the requirements of the mine, since the space has been left as flexible as possible so that stacking or shelving can be installed.

Refer to Table 9-1 for the list of Stores drawings showing plan, section and elevation contained in the Appendix B.

9.2.6 Process Plant Building

This building contains the processing plant and equipment with a conveyor feeding ROM ore via an external storage silo and a delivery conveyor exiting the building carrying concentrate. The process plant room is approximately 6,465 m^2 plus the addition of around 2,500 m^2 of floor space for offices and storage.

The building also contains other facilities that are required to support the processing of ore. These include offices and a control room for the process staff and managers which are located on an upper lever overseeing the plant room floor. The upper level also has a meeting room and toilet facilities for the staff.

Access from the ground floor to the upper level is via a stairwell located externally to the plant room floor but within the ancillary complex of the building structure. This also gives access to an upper level gantry platform which runs the internal length of the plant room and gives access to the electrical storage, ventilation, switchgear and air machinery plant rooms.

The lower level contains a laboratory room, workshop, cable cellar and reagent preparation room. Reagent product is kept separately in its own facility.

In addition there are offices and toilet facilities on the lower level with internal access from the plant room floor.

Layout plans can be seen within the Appendix B of this report with reference to Table 9-1 above for the Process Building.

9.2.7 General Maintenance Workshop

A single storey composite steel framed building of approximately 473 m^2.

The building foundations and construction is the same as for the offices and other surface buildings.

This workshop facility is intended for maintaining and repairing electrical and mechanical equipment and also for the conveyor.

Three separate areas are provided along with a stores area and offices including toilet facilities. A specific area is provided for any hot works.

Access to each of the three specific working areas plus the hot working area is via 5m high 4.5m wide doors so that large pieces of equipment can be worked on undercover within the building. Overhead gantry cranes for each area are also provided. The building will be provided with its own power for lighting and equipment with additional lighting coming from natural light through a number of windows to the gable ends of the building.

Potable water will be provided to the building along with all the other buildings including the drill core storage facility.

In addition the waste product drainage pipes will be included for all buildings that have toilet and or cooking facilities. The waste pipes will be connected to the municipality foul water and sewerage system.

For plans and elevations refer to Table 9-1 and the Appendix B for details.

9.2.8 Vehicle Workshop

This building will provide maintenance and repair facilities for servicing the mobile plant on site.

This building is a single storey composite steel framed, of approximately 1015 m^2. It will have clear internal height of over 8m to accommodate LHD's with raised skips or other mobile equipment that require additional height for working on.

There are four service pits that can accommodate mobile plant plus additional areas for wash down and general working/storage. The service pits will provide safe access to the underside of vehicles without the need to elevate the machinery.

There is also a separate area containing the offices/welfare and changing facilities.

A liquefied petroleum gas station is included as an annex to the main building with external access for safety reasons.

See Table 9-1 for details of relevant drawings showing the layout and elevations which are included in the Appendix B of this report.

9.2.9 BS Shaft Air Supply Building

This structure is constructed over the existing BS Shaft and forms an annex to the existing shaft building. It is not included in the support buildings/structures that are contained within the Industrial Area 1.

The building contains the air intakes and heat exchangers that pump warm air down the shaft into the underground workings.

Air heating is typically done with large heater units or blowers burning propane, diesel, natural gas or oil.

The heat exchanger unit is built as part of the mine ventilation system to the required flow and connected to the air ducts and fan units.

Details of the system can be found within the ventilation section of this report.

9.2.10 Roads

Three new roads will be constructed to provide access to the industrial areas 1 and 2 and for access to and from the site. The site access road will be asphalt and be 8m wide. This includes a section of road to the office complex area. The remainder of the road between the industrial area 1 and industrial area 2 will be gravel. There is also a section from the industrial area 2 to the BS-Shaft which will be re-surfaced. The section of gravel road between industrial area 1 and the industrial area 2 will be dimensioned for ore traffic.

In addition there is an access road from industrial area 1 to the road rail load out terminal of gravel construction for the conveyor and light vehicle traffic.

The areas will be gravelled, except from parts around the office and the relevant parking areas, which will be paved with asphalt. All sections belonging to the industrial area will be enclosed with industrial fencing.

The number of parking lots is calculated based on the assumption that there will be 100 dressing rooms and a 3-shift working schedule.

The industrial area of Blötberget is designed to support traffic of both haulers and trucks with trailers. All the buildings in the area are sited so as there is sufficient space for this.

9.3 Surface Electrical Power

The Mine transmission and distribution network will be derived from a 50kV overhead power line that is currently located on the project site; the mine site will be powered via a single 50kV/12kV transformer rated at 40MW feeding a main substation (STV1) located at Industrial Area No.1.

The surface power network will utilise a main distribution network rated at 12kV, with the main supply transformed down to a variety of supply voltages for use in Industrial Area No.1 and other remote areas of the mine, i.e. Industrial Area No.2 and BS-Shaft. The surface system comprises of one main 12kV substation and a series of auxiliary substations and distribution boards, operating on a 3phase 50Hz system.

The low voltage substations and distribution boards will be located in close proximity to the large load consumers and large kW drives to limit cable feed lengths, to provide good voltage regulation and reduce cable costs. The Process plant and Surface infrastructure will utilise voltages of 3.3kV, 1.0kV and 0.4kV, the voltage of each system determined by plant and equipment size, and voltage regulation economics.

9.3.1 Grid Supply

The mine will be powered using a distribution/transmission system operating on a three-phase, three-wire, 50Hz, supply via a local connection to an overhead power supply from VB-Energi; the system will provide a maximum capacity of 40MW.

The 50kV overhead power line that is currently located on the project site will be repositioned outside of the mine boundary. A 50kV spur connecting the overhead power line will be provided into the mine Incoming Grid Substation located adjacent to the main Surface Substation in Industrial Area 1. The mine supply will then be transformed down to 12kV using a 50kV/12kV transformer rated at 40MW and used to feed the main Surface Substation STV1. The overhead line into the Grid Substation will be provided with an overhead shield conductor and will include associated rated surge arrestors, to provide lightning protection. This work is to be undertaken by the owner of the power system, VB-Energi.

The mine site power supply network comprises the following major electrical systems:

- Main intake substation;
- Electricity transmission and distribution networks;
- Mine Area substation including mine equipment pumping, stockpiles crushing feed conveyors;
- Process beneficiation plant including blending beds and product stockpiles;
- Administration, major workshop facilities, mine services and welfare facilities.

9.3.2 12kV Mine Surface Substation (STV1)

The Mine Surface Substation will be located adjacent to the Incoming Grid Substation and will be powered using an incoming 12kV feed from the 50/12kV transformer in the Main Grid Substation.

The substation will utilise fully IEC certified, air insulated, composite switchgear designed with full metallic segregation of its modular compartments, including bus-bars cabinet, low voltage protection/monitoring cabinet, circuit breaker cabinet, cabling/wiring, etc. The switchgear will provide control and protection for the mine distribution system cables, transformers and motors.

Each substation switchboard will be designed for its full rated load with individual switches specified for their allocated duties; vacuum contactors will be used throughout.

The 12kV switchboard will comprise of an incoming and fourteen outgoing switches used to supply the process plant, underground power system, BS Shaft Area, Industrial Areas and various surface conveyor systems. The duties of the outgoing switches are as follows.

- No.1: Power factor correction;
- No.2: Spare;

- No.3: Supply to Industrial Area No.2 (TR9);
- No.4: Supply to BS-Shaft Substation (STV3);
- No.5: Underground Supply No.1 (via Surface/UG Substation SS1);
- No.6: Underground Supply No.2 (via Surface/UG Substation SS1);
- No.7: Supply to Industrial Area No.1 (TR1);
- No.8: Spare;
- No.9: Supply to Process Plant (TR8);
- No.10: Supply to Process Plant (TR7);
- No.11: Supply to Process Plant (TR6);
- No.12: Supply to Process Plant (TR5);
- No.13: Supply to Process (TR2) and
- No.14: Power factor correction.

Switchgear for a second power source/emergency generator supply has not been included in the design of the substation however provision has been made in the main substation to allow for the installation of switchgear to be installed at a later date.

9.3.3 Auxiliary Substations/Distribution Boards

The Main substation supplies power at 12kV to various auxiliary substations and dedicated distribution boards at the Mine Surface in addition to two feeds to the underground workings.

9.3.3.1 Surface / Underground Substation (SS1)

The Surface/Underground Substation (SS1) is supplied from the Main Surface Substation (STV1) via switch No's 5 & 6 and a 120mm^2 XLPE Single Wire Armoured cable (SWA). The Substation consists of two incoming switches separated by a bus-connector and five outgoing switches. The switchgear will initially be utilised to supply the underground development with the following duties.

- Switch No.1: Local supplies and pumping;
- Switch No.2: Main Decline supply;
- Switch No.3: Sandel Development supply;
- Switch No.4: Spare;
- Switch No.5: Decline ventilation.

Switchgear functions will be modified once development operations are complete, prior to the commencement of production. The duties of the outgoing switches will then be as follows.

- Switch No.1: Supply to Surface Conveyor System;

- Switch No.2: Underground Feed No.1 (to -310m Substation);
- Switch No.3: Underground Feed No.2 (to -310m Substation);
- Switch No.4: Spare;
- Switch No.5: Supply to No.1 Conveyor (Main Decline).

The substation will normally operate with both feeders powered and the bus-connector locked in the open position. The incoming switchgear and feed cables will be suitably rated to cater for full operating load in the event that one of the feeds/switches becomes faulty. Should this occur, the faulty section will be isolated and locked in the open position and the bus-connector will be closed, allowing the system to function whilst the faulty unit is repaired; in effect providing a standby system.

The Surface Conveyor system comprises the Secondary Crusher (and feeder), ROM Conveyor (to Surge Bunker), ROM Conveyor (Surge Bunker to Process Plant), Reclaim Conveyor (from stockpile) and Waste Ore Conveyor. This system will be supplied from Switch No.1 via a 12/1.0kV transformer rated at 750kVA located at the Decline Conveyor delivery tower. The feed cables from SS1 Substation to the tower will be routed through underground cable ducts. The conveyors/crusher will be powered and controlled using a motor control centre positioned within the conveyor tower. Suitably rated SWA cables will be utilised to supply system motors. These cables will be installed along the conveyor structure using cable hangers.

The Main Decline Conveyor will be powered from Switch No.5 via a 12kV/3.3kV transformer rated at 2000kVA located at the Decline Conveyor Delivery Tower; feed cables from SS1 Substation to the tower will be routed through underground cable ducts. The conveyor will be powered and controlled by stand alone switchgear incorporating soft-start controls.

9.3.3.2 BS-Shaft Substation (STV3)

The BS-Shaft Substation (STV3) is supplied from the Main Surface Substation (STV1) via switch No.4, using 150mm^2 XLPE, Single Wire Armoured cable (SWA); the cable will be routed along the roadway connecting Industrial Area No.1 to the BS-Shaft. The Auxiliary Substation consists of an incoming switch and three outgoing switches with the following duties.

- Switch No.1: BS Shaft general power;
- Switch No.2: BS Shaft Heating;
- Switch No.3: BS Shaft drainage pumps.

General power at the BS Shaft will be supplied from Switch No.1 via 12kV/0.4kV transformer (TR17) rated at 1.0MW and a local distribution board. Shaft heating facilities at the shaft will be powered using 12kV/0.4kV transformer (TR11) rated at 2.0MW. The BS Shaft drainage pumps will be supplied using 12kV/3.3kV transformer

(TR10) rated at 1500kVA; pumps will be powered, controlled and monitored via a motor control centre (MCC) located in the BS Shaft area.

9.3.3.3 Process Plant

The Process Plant is supplied from the Main Surface Substation (STV1) via switches 9~13, using SWA cables routed in cable ducts. The five feeds supply two 12/1.0kV transformers (TR6 & TR7) rated at 2MW and three 12/0.4kV transformers (TR2, TR5, TR8) rated at 3.0MW, 2.0MW & 0.8MW respectively. Transformers TR6, TR7 & TR8 will provide power to the various equipment/plant located within the process plant and will utilise local MCC and distribution boards to power/control & monitor the plant from within the Control Room.

Transformer TR2 will provide general power to the plant via local distribution boards.

Transformer TR5 will provide power to the conveyors supplying the rail out-load system.

9.3.3.4 Industrial Area No.1

General power for Industrial Area No.1 is supplied from switch No.7 in the Main Surface Substation (STV1); this supplies 12/0.4kV transformer (TR1) rated at 1.0MW to provide a power source to the various local distribution boards.

9.3.3.5 Industrial Area No.2

General power for Industrial Area No.2 is supplied from switch No.3 in the Main Surface Substation (STV1); this supplies 12/0.4kV transformer (TR9) rated at 0.6MW to provide a power source to the various local distribution boards. The transformer will be powered using 25mm^2, SWA cable (XLPE) routed via the road between Industrial Areas No.1 & No.2.

9.3.3.6 Motor Control Centres

Motor Control Centres (MCC) will be used primarily to centralise the distribution of power and control of motors in the process plant and surface conveyor system where multiple systems are in close proximity. The unit comprise a steel framed sheet enclosure complete with central bus bar system for the power supplies and multiple compartments for the different control/monitoring functions. The enclosures will contain an incoming supply section, a segregated bus bars systems rated and certified for the relevant motor voltage applications, multiple starters and motor drive sections designed and rated for each motor operating duty. Other cubicle sections may include SCADA, Telemetry, PLC, small power distribution, as required.

Incomer sections will be fitted with vacuum circuit breakers, Air circuit breakers, Moulded Case circuit breakers, and fuse switches as required. Starter compartments contain all the protection and control equipment necessary for a single motor or multi motor operation, and will be configured for either fixed or withdrawable contactors. The

MCC cubicles will be installed in air-conditioned rooms within close vicinity to the equipment to be controlled.

9.4 Water Supply

A potable local water supply is available from the municipality system and the mine will draw water from this system. This water supply will not be used for process water.

9.5 River Gonäsån System

Prior to the dewatering of Blötberget mine and mine water discharging into the local water courses, it will be necessary to re-route the River Gonäsån.

At present the River Gonäsån carries water from Lake Glaningen, Lake Dammsjön, the brook Öräbäcken and rain run-off from the mining area in Blötberget. The average flow per month, over the last 15 years, varied between 17 litres per second and 5,250 litres per second. Peak flow occurs in spring and low flow in late summer.

The River Gonäsån has been rerouted once before in connection to previous mining operations in the area, i.e. 1950-1979. Two water courses were built in the 1950s to divert the River Gonäsån from the mining area, that included two tunnels, one under Blötberget and the other under Främundsberget and four diversion channels. Two channels run from Lake Glaningen to the tunnels and two from the tunnels to River Gonäsån, by-passing the mining area

The northern route through Blötberget became the main outlet from Lake Glaningen. The level of the lake was regulated, and when it became too high the southern route through Främundsberget served as an emergency outlet.

9.5.1 River Gonäsån re-routing

All of the existing channels and tunnels will be used in the new diversion of the River Gonäsån. A new emergency outlet from Lake Glaningen will be excavated towards the duct that leads to the southern channel. The existing outlet will be sealed.

Also the pumping station for rain run-off from the mining area will be rebuilt at the same location as the previous station and the rain run-off will be pumped to the southern channel. It is envisaged that the capacity of the new pumping station will be 300 litres per second. This capacity is based on a 2-year rainfall and also corresponds to typical snow melting run-off rates in the area. Monitoring systems will be installed to assist with the operation of the pumping station.

Furthermore, the surplus water from the new tailing pond located at Gravgruvan will also be directed into the southern channel.

9.5.2 River Gonäsån Rerouting Options

It is envisaged that most of the diverted flow from Lake Glaningen will be through the Blötberget tunnel and the remaining flow will pass through the Främundsberget tunnel.

Rain run-off from the mining area will be pumped to the channel connecting to the Främundsberget tunnel.

The operational arrangements recommended to facilitate the rerouting operation are:

9.5.2.1 Northern Route

Works include the clearing of the channel downstream of the Blötberget tunnel and re-opening of the Blötberget tunnel by removing the soil dyke currently sealing both ends of the tunnel.

A non-permanent dyke will be constructed upstream of the tunnel to allow working in the dry. After the tunnel is reopened and inspected the dyke will be removed and the tunnel will serve as the main outlet from Lake Glaningen. A new fixed weir/level control will be constructed at the new outlet to control the level at the lake.

9.5.2.2 Southern Route

Works include the clearing of the channel downstream of the Främundsberget tunnel and re-opening of the Främundsberget tunnel by removing the soil dykes at its entrances and inspecting the tunnel.

An emergency outlet will be constructed east of the sealed outlet and the existing duct connecting it to the southern channel cleared. Construction of a new pumping station to pump run-off water from the mining area to the channel will be carried out.

This re-routing operation will need to ensure that it complies with all the Environmental requirements prior to commencement of any work.

9.6 Ore loading terminal

9.6.1 Ore loading terminal

The concentrate rail loading terminal has been designed to handle the concentrate product from the processing plant which will be produced at a variable rate of 199tph maximum (180tph nominal) and conveyed by a series of conveyors to the product storage silos. The initial design concept is to relocate the silos adjacent to the loading track and fill the wagons utilising feeders and a conveyor discharging into a surge bunker and loading pocket respectively. Figure 9-2 below illustrates the proposed rail loading arrangements for transhipment of the concentrate product to Oxelösund port.

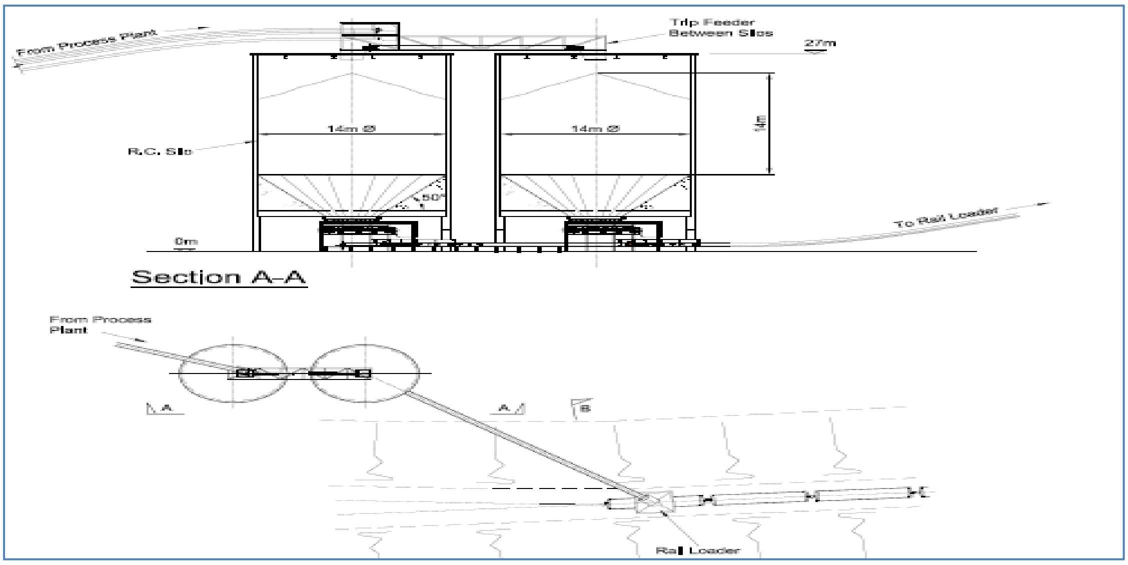


Figure 9-2 Schematic layout of product loading facility

9.6.2 Conveyors/Silos

The concentrate product will be conveyed to the loading facility by a number of conveyors installed adjacent to the twin rail track. The additional conveyor carrying capacity will allow for a potential increase of process plant capacity at a later stage of the project. The conveyers will deliver the concentrate into the first silo which has a capacity of approximately 7,500 tonnes of concentrate at an average SG of 2.0-3.2 tonnes/m^3. The design of the concentrate silo's is based on the ROM silo in order to standardise the equipment and civil design requirements for the project. When the primary silo is filled to capacity a trip feeder conveyor arrangement installed on the top of both silos will direct the ore into the second silo which also has a capacity of 7,500 tonnes.

9.6.3 Discharge and Filling

Either product silo can discharge the concentrate at a predefined rate controlled by a vibratory plate feeder onto the loading conveyor which will deliver the material into a surge bunker located above the rail track.

Directly below the surge bunker will be a loading pocket with the predetermined tonnes of concentrate ready for discharging into the wagon below. The surge bunker has a capacity of approximately 175 tonnes of concentrate to be stored ready for dispatch. The loading pocket has a capacity of approximately 87.5 tonnes of concentrate available to be loaded into the rail wagon below. The design of the rail loading facility will allow for variable pay loads to be handled based on the capacity of the railway rolling stock.

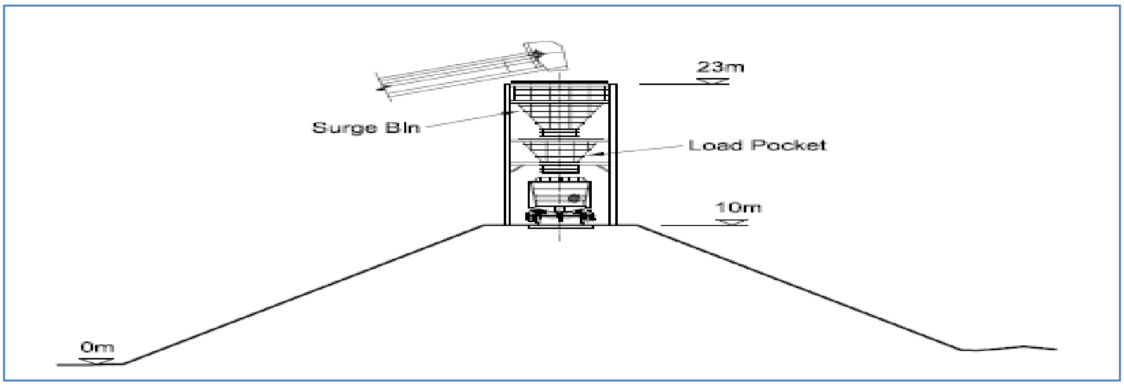


Figure 9-3 Schematic layout of rail loading facility

The capacity of the in feed conveyor to the rail surge bunker will be designed for a train filling cycle of 2/3 hours.

The loading facility will be controlled by a PLC/SCADA operating system and the logistical requirements for the system can be set to achieve the most efficient and practical loading times to best suit the operation.

10 PRODUCT TRANSPORTATION AND LOGISTICS

10.1 Introduction

Arguably NIO's most significant advantage over many mining projects is the existence of the logistics and services from the previous mining operations. The original rail lines were built connecting the former iron ore mining area around Ludvika to the ports on the east coast of Sweden and most specifically for the port of Oxelösund. Each of the mines in the region had its own rail connections and terminals, including mines at Blötberget, the subject of the NIO redevelopment. The railway built primarily for the export of the regions iron ores and it is now planned that NIO will take advantage of the purpose built rail line and port. The railway is now owned by the national Swedish rail authorities, and as such has been brought up to modern standards, and indeed is the beneficiary of a planned investment in upgrading the railways in the region to aid the industrial development in central Sweden.

Significantly, NIO is able to purchase the services from the owners and operators of the railways and the ports purely as an OPEX cost and without NIO contributing to any major investment. The only capital required by NIO is for the proposed rail terminal at the mine head at Skeppmora. The Swedish authorities, along with EU rules, have opened up the Swedish rail network and operations to competition for the provision of services, which ensured that NIO received highly competitive bids for the potential contract to move NIO's products to the port of Oxelösund,

From Oxelösund, with the planned development of the port, NIO will be able to supply the global steel industry buyers in ships up to "baby-Cape capacity" potentially as high as 90,000-120,00DWT; though initially this will be maximum Panamax 90,000dwt vessels Furthermore there is a very competitive and efficient fleet of smaller vessels and barges that operate in the Baltic regions, providing an alternative efficient supply chain to reach many of the potential European customers of NIO.

10.2 Loading terminal – Skeppmora

10.2.1 General

Nordic Iron Ore is working together with the Swedish Transport Administration (STA) in the project "New Terminal Skeppmora". A cooperation agreement has been signed where the assignments and responsibilities and costs will be provided for each party.

Under this agreement STA, with NIO´s approval, contracted the Swedish consultancy company WSP to design the Skeppmora terminal to a final procurement document standard, ready for assignment of a contractor to commence building.

In the vicinity of the small village Skeppmora there is a section of approximately 1,000 meter of straight track on the main railway line, which is considered suitable for the placement of the main switching yard. However; the adjacent main line is inclined by 9-10 % towards the south and similarly towards Grängesberg. The proposed main switch leading in to the rail yard is planned on the south side of the straight main track, facing to the south, but as far as possible from the Skeppmora village. The switching yard will consist of 2 terminal tracks approximately 630m long and a loading track approximately 400 m long and can potentially be extended; see drawing in Appendix B.

The complete loading facility will be electrified and controlled by a composite signalling system to ensure an efficient logistical operation. The terminal is planned to operate 3 unit trains simultaneously;

- 1 (incoming) arriving,
- 1 (outgoing) departing and
- 1 at the loading track

This proposed operating system for the terminal will have sufficient capacity to transport Blötberget iron ore product volumes up to 1.4 Mt/y. The initial loading terminal is capable of future expansion through a planned track loop connecting back to the main line facing south, increasing the planned capacity. The estimated cost for the terminal, at this Phase 1 stage, is approximately 99 million SEK, +/- 15%.

10.2.2 Planned Phases

The Skeppmora loading terminal is planned to be developed in three phased stages to enable NIO to access the main track as early as practicable (2016). The proposed lead time can be achieved by STA, initially establishing a “temporary” main track connection called Linjeplats using manual interlocking facilities and at a later date (possibly, 2017) build the 2nd stage to a fully remote-controlled system called Driftsplats.

The terminal will be regulated by signals (required by the Swedish Transport Administration), which means that the progress of the terminal is divided into three phases:

- Stage 1 Main Switch (2016): The main switch can be installed and operational in year 2016 with a shorter loading track.
- Stage 2 Linjeplats (Swedish) - operated by a line controller (2016). The terminal is developed according to the schematic layout plan for the track route but is not controlled by a signalling system. This means that there will be limited train departures / arrivals at the terminal. Additional staffing will be necessary to operate the terminal.
- Stage 3 Driftsplats (Swedish) – fully remote operation and control of the freight terminal (year2018)

The terminal is developed according to the track plan and all monitoring switches are fully installed, remotely controlled and the tracks on the terminal are fully automated signal controlled.

The Stages of the installation enables the terminal to be commissioned in 2017 (Q4) with limited operating parameters which are planned to be completed in 2018 (Q3).

10.3 Ownership

Nordic Iron Ore will be the “infrastructure manager” responsible for operations when the terminal is completed. The interface to the Swedish Transport Administration track will be the connecting gear at the southern end of the terminal where the Swedish Transport Administration is responsible for connecting and operating the switch to the main railway and Nordic Iron Ore for all other rail infrastructure belonging to the terminal. This means that Nordic Iron Ore, must be an approved “infrastructure manager” authorized by The Swedish Transport Agency before the terminal is commissioned and able to operate freight traffic. The Swedish Transport Agency has been informed about the project and its scope.

10.4 Functional requirements for the terminal

Functional requirements for traffic:

- Rational solution for rail yard, operations with as minimal shunting as practicable .
- In principle, the empty trains arrive from the main line and stop at the northern end of the arrival track. The loco will uncouple from the wagons and manoeuvre to the opposite end of the train via the passing track.
- Sequenced signalling regulates entrances and exits to the terminal yard from the main line, which is remotely controlled from the STA dispatcher central room in Hallsberg/ remote dispatcher.
- Terminal loop are proportioned in length to the longest option.
- Rail yard will be installed level or limited to minimal inclines as practicable.
- (Track model) 630 m, meeting length in the yard tracks.

Rail terminal area to be electrified to the extent that the locomotives can do the necessary shunting.

Other features:

- Stationary brake test equipment.
- Heating devices for main-line locomotive.
- Protective Switch zones or track barriers that allow switching to be undertaken safely when moving trains.
- Heating in all points switches; ensure safe operation and reliability during winter periods.

10.4.1 Train movements

The rail loading terminal is designed to avoid excess shunting and reduce the need to uncouple train sets. An empty train set arrives on track, and the wagons are disconnected from the locomotive.

The locomotive will utilise track 2 for shunting past the empty wagons and re-couple at the opposite end of the train set. This enables the loco to shunt the whole empty train-wagons into the loading track. When the train has been loaded, it can depart from the loading track.

The planned maximum train length for Blötberget mine would be 400m and the train maximum gross weight is expected to be around 3,000t at a maximum planned rate of 2 trains per day.

10.5 Main-line Skeppmora-Oxelösund

The Swedish Transport Administration has allocated 450M SEK for the upgrading of the main line between Skeppmora-Oxelösund. The upgrade consists of track reinforcement, track replacement; signal boxes, new pass byes points etc. These improvement measures will ensure that the main line becomes more robust, reliable and safer for heavy goods transport. The work is planned to start 2015/16 and continue for the next 7-8 years.

10.6 Infrastructure to the Port of Oxelösund

The distance to the Port of Oxelösund is approximately270 km and the trains will operate from (Ludvika) Skeppmora through Ställdalen – Frövi – Jädersbruk – Valskog – Rekarne – Eskilstuna – Flens övre – Nyköping till to Oxelösund. No changing or recirculation of the locomotives is required on the route. Minimum passing track length for the planned train set is 604 meters long. The route allows 80 km / h for Stax E (25 tons axle load) and for normal trains varies between 80 and 160 km / h (some short speed reductions may occur at some stations). The route is in medium to good condition.

Table 10-1: Technical Specification of planned railway

Technical standard railway Ludvika-Oxelösund	
Maximum Permissible Axle Load	25 ton
Stvm (metre weight)	8 ton/m
Sth* (speed)	80-160 km/h
Distance	270 km
Train weight (highest so far)	3000 ton
Inclination (varies along the route)	10-14 per mill

Technical standard railway Ludvika-Oxelösund	
Train Length (shortest passing track)	604 m
*Highest permitted speed with Maximum Permissible Axle Load E (25 tons axle pressure) is 80 km/hour regardless of whether or not the line permits higher speeds for other trains.	

Parts of the main rail network are quite heavily utilised (regional passenger traffic) and some areas are less utilised. The planned NIO route is the same as Grängesbergbolaget previously used for the transport of ores from the mines in Grängesberg to the port in Oxelösund. The proposed rail route has a very favourable profile for loaded trains with only a few major gradients.

The areas with high rail traffic operating are mainly part Jädersbruk – Valskog – Rekarne - Eskilstuna – Flens övre but also occasionally in the section Ludvika – Ställdalen – Frövi. On the latter route there also increased freight traffic that utilises some of the rail capacity. On the proposed main line (entire route), Green Cargo operate its “ Steeltrains” between Oxelösund – Domnarvet – Oxelösund using heavy trains (2000 - 2400 tons) at 80 km/hr, and currently two train pair/day.

Section Eskilstuna – Flen is really not heavily utilised; however travelling trains have basically rigid schedules with fixed meetings in Hälleforsnäs. Station distances in this line section are variable and it is difficult at certain periods of time to introduce additional trains. The abandoned deleted pass bye point Harsjön resulted in a 12 km length of rail track without a pass bye in comparison with the other rail track distances varying between 4-8 km.

In Oxelösund the train is required to shunt from the station uphill to the port Höjdbangården. The section is partly controlled by signals and both rail yards at the station and at the port are electrified.

Advantages of the main rail line is that it already accepts trains with Stax E and 80 km/hr and the route is in a reasonably good standard and has favourable inclinations for the loaded trains.

10.6.1 Re-routing

Relatively good opportunities are available for rerouting traffic in the event of service disruption. Some of diversion routes, however conflict with the normal commuter traffic. Some of distances are not normally accessible for rail freight with largest axle load of 25 tonnes.

Details of the re-routing options from Skeppmora to the Port of Oxelosund are shown below in the map.

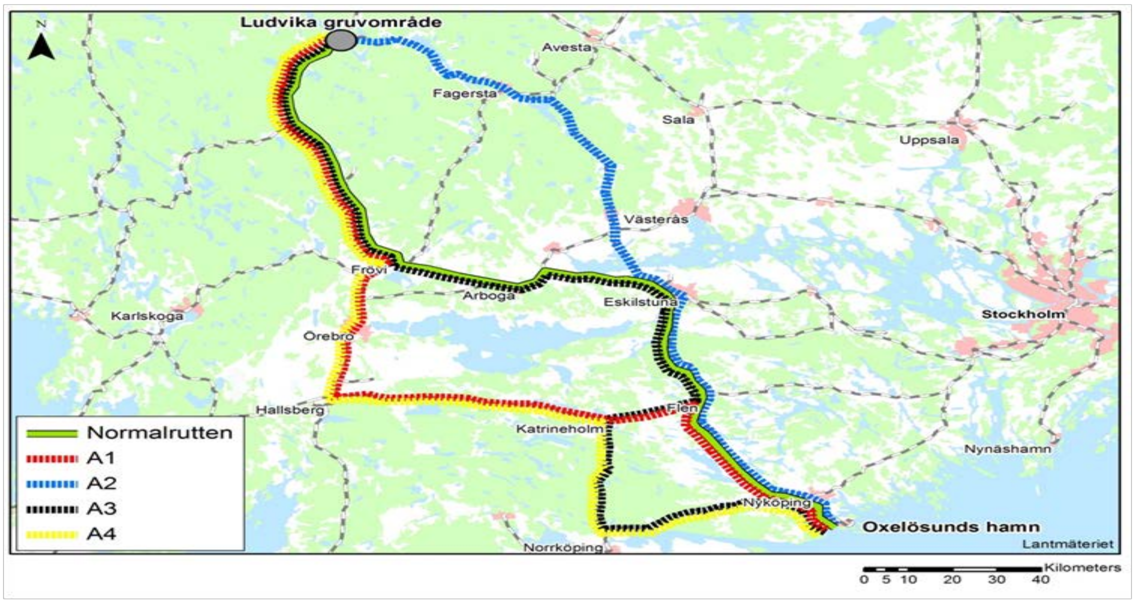


Figure 10-1: Alternative Routes

10.7 Port of Oxelösund

The Port of Oxelösund is 50% owned by the municipality and 50% by SSAB Oxelösund AB.

Oxelösund has previously been a major shipping port for iron ore from Bergslagen and has been designed as a dry commodity bulk handling port, with iron ore and coal as the main commodities. A large proportion of the shipping is being transferred from large ships to smaller, so-called feeder boats.

Size-wise, the port is one of Sweden's largest, depending on which mechanism the operations are measured in. In 2010 the Oxelösund port handled approximately 7.6Mt of cargo. Current nominal capacity for the dry bulk cargo vessels is Panamax, up to 90,000dwt. Up to 120,000-DWtonne ships can be handled in the port in the future and the port has several tugs available to assist manoeuvring larger vessels. Oxelösund is located south of the insurance limit for winter classed vessels (ice).

Nordic Iron plans to have a storage capacity in the vicinity of the ship loader of Blötberget product of approximately10,000 tonnes in an open stock pile and a further 5,000 tonnes in the loading silo. The planned main storage capacity in Oxelösund Port assigned to NIO is approximately 250,000 tonnes capacity if required; which approximately corresponds to 1-3 large vessel loading requirements.

The Port of Oxelösund has a number of distinctive advantages:

- Harbour depth - 16.5 meters
- Entering conditions - piloting not required
- Ice-free all year
- Experienced bulk handling port

- Technical performance of the infrastructure between Ludvika and Port of Oxelösund
- Only small investments are needed (by the port owners) to facilitate the NIO product.
- Environmental permission (MKB) in place

In view of the above benefits Nordic Iron Ore has chosen to establish a Letter of Intent (LOI) with the port of Oxelösund.

The current loading capacity of the port is estimated at a average rate of 2,000tph via the loadout crane via the conveyor belt from the storage area. The existing loadout conveying and crane have a nominal annual capacity of 2.5Mt/y. This is more than sufficient for the Phase 1 development at Blötberget.

10.8 Logistics chain from Mine to Port

The tendering process was initiated during 2014 for the procurement of the logistical operations from the iron ore terminal Skeppmora to port of Oxelösund. The invitations to the potential operators included the entire logistics operations including costs for all railway transportation to the port from the mine.

Competitive tenders have been received from national and international train operators.

10.8.1 Content of Logistics Costs

The overall logistics cost were divided into three main areas:

10.8.1.1 Access charges and fees

This includes all fixed costs and fees regarding access to the state owned railway line. Cost of electrification usage, charges for using the railway line which includes both a track path (fee for the timetable slots for journeys) charge and passage charge for the set route. Additional charges that will affect the logistics setup include an insurance (accident) charge and an emissions (environmental) charge which is relative to diesel consumption.

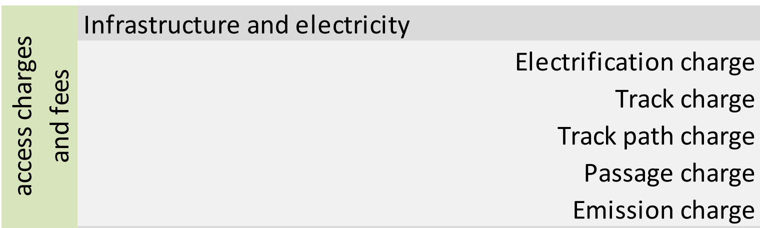


Figure 10-2: Access charges and fees.

10.8.1.2 Rolling Stock – Financing Cost and Maintenance.

The bidding operators have submitted prices for the operation and maintenance of that equipment, as part of the total tariff to be charged. NIO has also been supplied with prices of the locomotive alternatives and is able to compare the capital requirements (financing costs) to the operating costs for any given locomotion

Equipment specifications are also important as it has a direct correlation to the overall performance of the logistics chain from mine to port. The proposed wagons also have a large impact on the equipment needed in the loading and unloading terminals, including a impact on overall lead time (ie loading/unloading speed, journey time), as well as the train load size.

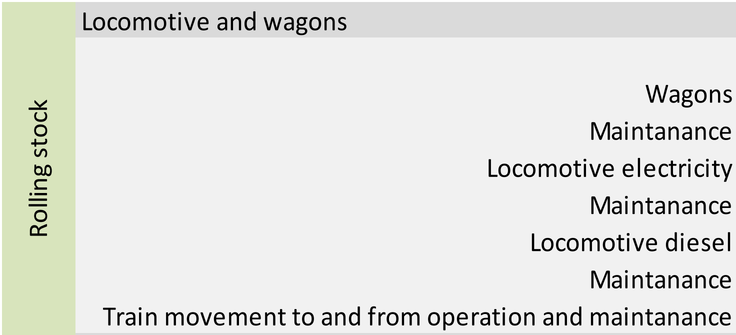


Figure 10-3: Rolling Stock

10.8.1.3 Operations – Variable Costs

The figure below provides an overview of the variable costs that have been included in the tenders for the logistics operations.

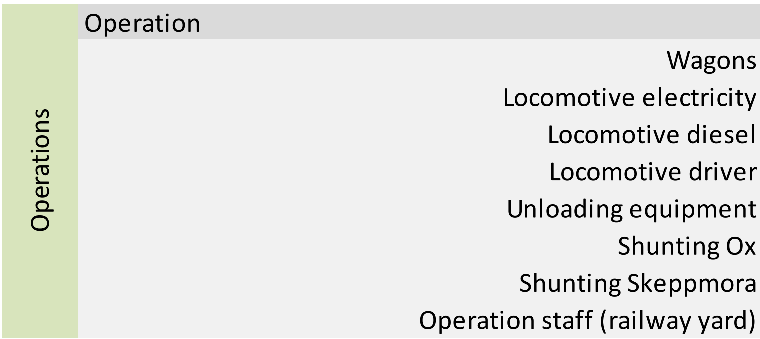


Figure 10-4: Operations – variable costs

After review of all offers NIO consider that five of the companies offered competitive tenders.

- All five companies with competitive tenders will update their tenders during May 2015.

Based on the preferred business model of the transport submissions the following actions will take place:

- Meetings and negotiations with the potential companies based on the tender returns
- Contract negotiations and signing a LOI
- Binding contract for the negotiated period

10.9 Shipping

As described above the Port of Oxelösund has a draft of 16.5m, capable of handling vessels up to baby Cape size. The port will initially handle Panamax vessels, expanding to larger units when the port upgrades the mooring size and loading equipment. The major restriction in the region is the draft available through the Straights of Denmark at 15.7m, limiting vessel size to Max vessels and baby Capes of around 120,000Dwt.

However, the major advantage of the Port of Oxelösund is in the potential cost effective handling of vessels to the major European markets, especially those in the Baltic Region. Here flexibility could be the key as well as cost effective ballast trips, where it is possible to use even barges for deliveries to certain markets in Poland and Germany. LKAB uses barges (around 13,000t dwt) from Lulea for deliveries of pellets to SSAB, Rukki and Ovako. The Baltic also uses a very efficient fleet of Handy and Handymax size vessels ranging from around 20,000-50,000dwt.

The use of barges opens up the possibility of direct deliveries by canal/rivers into Germany and other parts of Europe. NIO's use of Oxelösund will prove to be highly competitive to these markets, with a significant advantage compared to Europe's main suppliers in Brazil, S Africa and Canada; who would have to either unload the iron ore in Rotterdam for transhipping by barge or rail, use Panamax vessels or partially unload in Rotterdam to deliver into the heart of Europe. Even Voest Alpine in Austria have iron ore transportation coming in from Rotterdam on small trains. NIO has significant competitive advantage over major competitors, even as today when the shipping costs are at historic lows.

NIO is also competitive shipping to customers in the Atlantic and MENA regions compared with those competitors in S & N Americas; however the advantage is less and the competitive position would be beprimarily on product quality.

10.10 Remaining work

The following Agreements and Permits are required to finalise the logistics arrangements.

10.10.1 Logistics

Based on the preferred business model of the transport setup the following actions will take place:

- Meetings and negotiations.
- Contract negotiations and signing LOI.
- Binding contract for the negotiated period.

10.10.2 Loading terminal:

Assuming a time schedule for the connection to the Main railway in 2015/08 (Linjeplats) and (Driftplats) in 2016/08 the timetable given in Table 10-2 is currently envisaged.

Table 10-2: List of identified Agreements/Permits required for building and operating the Skeppmora Terminal

Item	Responsible	When
Execution Agreement design	Trafikverket	Ready
Execution Agreement production	Trafikverket	2015/09
Traffic Control Agreement	Trafikverket	2015/10
Procurement documents, design Terminal	WSP	2015/11
Connection Agreement Linjeplats 2015	Trafikverket	2015/12
Procure Contractor building Terminal	NIO/Trafikverket	2016/03
Ground preparation Terminal	NIO	2016/04
Infrastructure Owner Agreement	NIO	2016/05
Operation Instructions Linjeplats 2015	NIO/Train operator	2016/05
Permit to connect Linjeplats 2015	Transportstyrelsen	2016/06
Building Linjeplats 2015	NIO/WSP/Contractor	2016/08
Track working periods =5 weekends	"	"
Connection Agreement Driftplats 2016	Trafikverket	2017/04
Building Linjeplats 2015	NIO/WSP/Contractor	2016/08
Track working periods =5 weekends		
Connection Agreement Driftplats 2016	Trafikverket	2017/04

11 ENVIRONMENTAL

11.1 Regulatory

The Swedish Environmental Code (SFS 1998:808) provides the legal environmental framework for environmental matters. The Environmental Code comprises 33 different chapters dealing with different aspects as provisions concerning management of land and water areas, environmental quality standards, environmental impact statements, protection of nature and species, provisions concerning environmentally hazardous activities and health protection, contaminated land, water operations, chemical products, waste and producer responsibility, consideration of cases and matters, supervisions and charges, and penalties. As part of the framework there are also a number of ordinances issued under the Environmental Code. Such ordinances mainly specify or clarify the rules or objectives stipulated in the Environmental Code.

Regarding mining operations the most important chapters and related ordinances are as follows.

Chapter 2 - General rules of consideration

General rules on environmental knowledge, protective measures, avoiding hazardous chemical products, materials and energy conservation and site selection. When applying for an environmental permit the applicant must show that the obligations arising out of this chapter has been complied with.

Chapter 3 - Provisions concerning the management of land and water areas

Basic provisions related to the use of land and water as well as rules regarding areas of national interest.

Chapter 5 - Environmental quality standards. Ordinance on air quality (SFS 2010:477)

Guidance on matters to be specified in EQS, compliance, action programs and measurements. The ordinance contains specific EQS values for certain substances as nitrogen dioxide, sulphur dioxide, carbon monoxide, ozone, benzene, particles, benzo(a)pyrene and some metals. The EQS for surface waters are published in a regulation issued by the Swedish Agency for Marine and Water Management (SAMWM) with the title Regulations on classification and EQS regarding surface water (HVMFS 2013:19). Corresponding EQS for groundwater are published in a regulation (SGU-FS 2008:2) issued by the Geological Survey of Sweden.

Chapter 6 - Environmental impact statements and other decision guidance data. Ordinance on environmental impact assessments (SFS 1998:905)

Rules on compulsory consultation activities prior to performing the EIA work and rules on the content in an EIA report. The ordinance contains rules and guidance on when significant environmental impacts can be expected from certain activities.

Chapter 7 - Protection of areas. Ordinance on protection of areas (SFS 1998:1252)

This chapter covers the protection of and certain rules related to national parks, nature reserves, culture reserves, habitat areas, shores and water supplies (surface water or groundwater). It also contains rules related to the protection and conservation of special areas of interest by the Commission of the European Communities (conservation of wild birds [2009/147/EG] and conservation of natural habitats and of wild fauna and flora [92/43/EEG with adjustments in 2006/105/EG]). The ordinance covers inter alia additional rules on the protection of areas with certain biotopes and information about the content in an application for exemption under chapter 7.

Chapter 8 - Special provisions concerning the protection of animal and plant species. Ordinance on species protection (SFS 2007:845)

The important rules on species protection are published in the ordinance in which a list of protected species is presented in an appendix.

Chapter 9 - Environmentally hazardous activities and health protection. Ordinance on environmentally hazardous activities and health protection (SFS 1988:899) and Ordinance on environmental permitting (SFS 2013:251)

Definitions and basic rules on environmentally hazardous activities and health protection. The ordinance SFS 1988:899 contains inter alia rules on environmental permitting when handling and storing certain hazardous substances, rules related to discharge of waste water and rules on notifications that has to be performed for certain activities. For instance clean-up activities of contaminated land has to be notified to the supervising authority prior to commencement. The ordinance SFS 2013:251 describes the kind of operations and measures that cannot be performed without an environmental permit or a notification. Two permitting levels are described in the ordinance: A-level which implies that the Environmental Court is the permitting authority, and B-level where the County Administrative Board deals with the permit application. Mining operations and processing of ore are covered by chapter 4 in the ordinance and such activities belong to the A-level. Also see chapter 11 below.

Chapter 10 - Operations that causes environmental damages

This chapter sets out responsibilities for investigation and clean-up of polluted areas and applies to land and water areas, buildings and structures that are so polluted that they may cause damage or detriment to human health or the environment. The Environmental Code invokes the “Polluter Pays Principle” in so far as operators who have performed an activity or taken a measure that is a contributory cause of the pollution shall be liable for clean-up, to the extent reasonable. Section 4 states that when the extent of liability is determined, account shall be taken of the length of time that has elapsed since the pollution occurred, whether the person liable was obliged to prevent future damage and any other relevant circumstances. Where an operator can show that he was only responsible for the pollution to a limited extent, this shall also be taken into account when determining the extent of liability.

Chapter 11 - Water operations. Ordinance on water operations (SFS 1998:1388)

Water operations require in most cases an environmental permit granted by the Environmental Court. Water operations in chapter 11 cover inter alia the constructions, alterations, repair and removal of dams or other water structures in water areas, filling and piling in water areas, blasting and cleansing in water areas, as well as other measures in water areas whose purpose is to change the depth or position of water and the diversion of groundwater and the erection of structures for this purpose. Chapter 11 is applicable on a number of planned measures at the mine site and the environmental permitting has covered such measures. During 2014 additional rules regarding safety classification of dam constructions has been implemented in chapter 11.

Chapter 14 - Chemical products and biotechnical organisms. Ordinance on chemical products and biotechnical organisms (SFS 2008:245), Ordinance EG 1907/2006 “Reach” and Ordinance EG 1272/2008 “CLP”

This chapter covers inter alia definitions and general rules regarding handling of chemical products. Additional rules regarding particularly hazardous chemical products are present in SFS 2008:245. Most of the detailed rules on reporting, classification and labelling are provided in the Reach and CLP Ordinances. The dominating part of the rules on chemical products are only applicable on chemical producers and principally only handling and storage matters will have to be considered by Nordic Iron Ore.

Chapter 15 - Waste and producer responsibility. Ordinance on waste (SFS 2011:927) and Ordinance on mining waste (SFS 2013:319)

This chapter includes waste definitions and responsibilities related to handling of waste. The ordinance SFS 2011:927 contains additional rules on waste including information on waste classification. The ordinance 2013:319 covers rules on characterization of mining waste, classification of waste storage facilities (waste rock storage and tailings storage facilities), handling of mining waste, monitoring activities and duties related to closure. A waste management plan has been presented as part of the permit application (appendix to EIA report). Performed characterization of waste rock material and tailings from processing tests shows that the waste material can be regarded as inert waste. However additional characterization needs to be performed when the processing starts and the waste management plan should be updated.

Chapter 22 - The procedure for application cases in the environmental courts

In this chapter the permitting procedure is outlined together with rules regarding the contents in a permit application.

Chapter 26 - Supervision. Ordinance on operators self-monitoring (SFS 1998:901)

Rules related to supervision and environmental reporting. The ordinance include duties for operators to document environmental responsibilities within the organisation, to implement procedures for control of equipment, to repeatedly assess risks associated with the operations and to keep records of chemical products.

In addition to the Environmental Code there are other environmentally related acts that will become applicable on the planned mining activities. One is the Flammable and Explosives Act (SFS 2010:1011) which applies on storage and handling of inter alia diesel fuel and liquefied petroleum gas (LPG). The planned storage of up to 100 m3 fuel and approximately 60 tonnes of LPG at Blötberget imply that a permit is needed. Such a permit is granted by the local municipality and will not involve any efforts or costs of concern. The permitting threshold limits are presented in the Regulation on license for handling of flammable gases and liquids (MSBFS 2013:3) issued by the Swedish Civil Contingencies Agency (SCCA).

The Act on measures to prevent consequences from severe chemical accidents (SFS 1999:381) is potentially applicable on the storage of PLG at Blötberget. If the stored amount of PLG exceeds 50 tonnes, but is below 200 tonnes, the act is applicable in such a way that an action programme needs to be prepared. The storage should also be notified to the supervising authority.

The Swedish Environmental Protection Agency (SEPA) publishes with support of the Environmental Code regulations and guidelines on specific matters like industrial and construction noise, waste management and treatment, discharge of industrial waste water, protection of species and key biotopes, environmental reporting and control/monitoring activities.

11.2 Environmental Permitting Procedure

The first step in the permitting process is consultation with the County Administration Board (CAB), the Local Environmental Department (LED) and potentially affected private individuals. This consultation is most often done as two separate meetings, one with the CAB and LED and one with the locals.

Prior to the consultation the applicant has to prepare a consultation document that covers planned localizations for the activities, the extent of the planned operations, preliminary designs and the foreseen environmental impacts from all activities.

If the planned activities are expected to impose significant impacts, consultation shall also be carried out with other national authorities, municipalities, environmental organizations (NGO's) and the public. The purpose of the consultation is to obtain viewpoints to consider in the EIA work. The CAB has a key role to guide the applicant regarding the extent of the EIA. Nordic Iron Ore has performed the consultation in accordance with the Environmental Code. All performed consultation work including viewpoints from the authorities, organizations and the public was presented in an appendix to the EIA report.

After finalizing the EIA report and the technical description of the activities and facilities, a formal permit application (legal) is prepared. All reports, drawings and documents are thereafter submitted to the Environmental Court.

In the next step the Environmental Court sends the full application to the consultation bodies for viewpoints if supplementary information or data is required to regard the application as complete. The public gets the same opportunity to respond as the consultation bodies. Requests for additional information and/or clarification are thereafter sent to the applicant who gives the opportunity to submit additional information to the Court. This process with adding more information to the application can in complex projects be repeated many times. In the Ludvika mines project supplementary information were submitted to the Court at four occasions.

When the Court has judged the permit application to be complete, announcement of the application and the EIA report is done.

After announcement of the outcome, consultation bodies, experts, organizations and private individuals are asked to provide their opinion on the complete application within a certain time. Opinions are sent by the Court to the applicant who gets the opportunity to response to the provided opinions. Thereafter the Court decides upon a date for court hearings after consultation with the applicant.

The hearings normally takes place in local premises in the area of the project site, making it possible for local stakeholders to attend and giving the Court a possibility to perform a site visit together with the applicant, consultation bodies, stakeholders and others.

In mining projects the hearings often extend over 3-5 days. At the end of the hearings the Court informs when a decision on the application can be expected. The decision made by the Court can be appealed to the Environmental Court of Appeal.

11.3 Summary of Environmental Permit

NIO has a permit for mining work etc. at both Blötberget and Håksberg in Ludvika. For now NIO will only claim the Blötberget part of the permit. Below are the conditions to the permit that have been stated:

The Land and Environmental Court has granted NIO a permit, according to Chapter 9 of the Environmental Code:

- to construct and operate Blötberget and Håksberg mines for mining a maximum of 3 million tonnes of ore per annum and mine,
- to deposit waste sand (tailings) from the concentrating plant, and
- to erect, construct and operate the installations required for the work, including, among things, shafts, drifts, ramps and other rock work required for the mining, installations for the transport and storage of mined ore and waste rock above and below ground (tunnels, roads, shafts, conveyor belts, hauling devices, mucking arrangements, intermediate stores etc., plant for crushing above and below ground, a concentrating plant, plant for transporting and loading raw ore and finished products (railway terminals etc.), an installation for transporting waste out of the concentrating plant and a tailings storage facility including clarification dam, all in the municipality of Ludvika and mainly in accordance with what was stated in the application and generally in the case.

The Land and Environmental Court has granted NIO a permit, according to Chapter 11 of the Environmental Code s follows:

- to remove the water present in the mines and to erect the installations which this removal requires (pumps, settling basins, pipes to the River Gonäsån and Lake Övre Hillen respectively),
- via the above-mentioned installations, to discharge the surface water and groundwater leaking into the mines during regular operation,
- on part of the section between Lakes Glaningen and Väsman, to divert the River Gonäsån from its natural bed to excavated channels and rock tunnels under the communities of Blötberget and Främundsberget respectively, and for this purpose to cut off the natural outlet of Lake Glaningen with a barrage, open the inlets to the channels and tunnels, block the river at the properties Gonäs 1:35,Gonäs 1:142 and Finnäset 14:10, block off Hyttbäcke (stream) at the property Finnäset 14:10, demolish existing embankments at the properties Gonäs 1:35 and Ludvika gård 8:2, carry out cleaning work in the channels and rock tunnels,
- construct a culvert over the southern channel at the property Gonäs 25:1, construct a culvert under Klenshyttevägen at the property Finnäset 14:10 and construct and demolish settling dams respectively during the construction period,
- to drain land in an area south east of the Blötberget mine and for this purpose erect a pumping station at the property Finnäset 14:10,
- to construct and operate dams and associated devices for the tailings disposal and settling plants,
- to construct the ditch around the tailings storage facility and clarification dam, and
- to abstract up to a maximum of 100 l/s of water from Lake Väsman and erect the installations required for this
- all mainly in accordance with what was stated in the application and otherwise in the case.

11.3.1 Compulsory Right

The Land and Environmental Court grants Nordic Iron Ore AB the right, according to Chapter. 28, § 10, first paragraph, points 2 and 6 of the Environmental Code, to make use of land and water areas that belong to the properties Gonäs 1:35, Gonäs 25:1, Finnäset 14:10, Skeppmora 2:9, Skeppmora 4:29, Skeppmora 4:31, Skeppmora 5:20, Skeppmora 5:36, Skeppmora 7:3, Ludvika gård 8:2 and Klenshyttan 4:1, as well as the communities OUTR:1 and Skeppmora SAMF:2.

The compulsory right now applies to licensed installations and measures for dams for the tailings disposal and clarification, and for land drainage etc.

11.4 Costs related to Compliance with Environmental Permit and Minerals Act

The environmental permit (EP) issued by the Land & Environmental Court requires the Company to instigate certain control programmes prior to the start of construction activities on site. Costs of these have been included Capex estimate and are described below.

11.4.1 Compulsory Rights

The EP defines the process for compensation to the land owners. Most of the land is owned by Bergvik Skog AB who participated in the courts proceedings and accepted the standard method and price calculation for compensation. This compensation method and price will also be used for compensation in connection with the designation of land proceedings.

11.4.2 Compensation for loss of flora and fauna

A plan for compensation of lost flora and fauna, mainly in the tailing dam area, has been submitted to the County Administrative Board (CAB) in line with the requirement in the EP. The plan will incur initial costs during the preparation of the proposed new area and a yearly maintenance cost during the life of mine.

11.4.3 Water table control programme

The EP provides for a water table control programme in connection with water and energy wells around the concession area. The EP also stipulates a emergency planning for households that may get drained water wells. Compensation to the well owners is also stipulated in the EP.

Additionally a compensation for disturbance and detected energy losses must be paid.

11.4.4 Designation of Land – land acquisition

Within the Minerals Act there is a structured expropriation process stipulated for access to any land needed for mining operations. The land affected is the industrial areas 1 & 2 and the train terminal and the southern tailings facility after approximately three years of production when the starting dams in the southern tailing dam area will be constructed.

The total cost of acquisitions according to the established principles for the compulsory rights in the EP and has been included in the capex estimates.

11.4.5 Compensation to other land owners

Separate agreements with house owners in the village Ickorrbotten will be made as a result of discussions in connection with the environmental court proceedings. These will result in further payments at the start of the construction works. Similar agreements are expected to be made with the house owners in Torsbovallen and Jansbo.

11.5 Mine Reclamation and Closure

The EP stipulates that a Land Reclamation Deposit Fund must be deposited at the CAB as from the commencement of the activities at site. In total the company is required to make deposits up to a sum of SEK 53,4 million. Upon commencement of construction activities the company must make a first deposit and thereafter a yearly deposit.

12 CAPITAL COST ESTIMATE

12.1 Summary

The overall capital cost estimate for the option of producing pellet feed concentrate has been compiled by DMT. The cost input parameters and their respective values rely to a significant extent on contributions from the other organisations responsible for the respective sections of this report.

The estimate has been the subject of iterative discussions between NIO and other contributors, and between NIO and DMT.

Unless otherwise stated, all costs are expressed in US dollars, with exchange rates based on rates quoted by Sweden's Central Bank (Riksbanken) at 31st March 2010, with no allowance made for interest or financing during construction.

The base date of the CAPEX estimate is 31st March 2015. The exchange rates used to convert other currencies into US Dollars are:

- SEK / US$ = 8.6232
- SEK / EUR = 9.2869
- SEK / GBP = 12.7441
- EUR / US$ = 1.0770

The estimated cost to bring the mine into production including design, construct, install and commission of the Project operation and facilities described in this report is US$181.2 million. This amount includes the direct field costs of executing the Project, plus shipping and logistics in Sweden.

Sustaining capital expenditure of the life-of-mine is estimated to be US$70.8 million.

Contingency included amounts to US$23.6 M, some 9.4% of the total capital expenditure.

Cost estimates are based on the level of engineering presented in this report and are considered to have an overall accuracy of +/-25%.

Details beyond those contained in this section can be found in Appendix I.

Table 12-1: Breakdown of major elements of estimated capital expenditure

Item Description	Initial Capex US$'000s	Sustaining Capex US$'000s	Total US$'000s
Mining - Underground	49,643	41,877	91,511
Surface Handling	11,444	-	11,444
Rail Terminal	12,055	-	12,055
Port	Incld in port charge	0	Incld in port charge

Item Description	Initial Capex US$'000s	Sustaining Capex US$'000s	Total US$'000s
Site Civils	13,081	-	13,081
Surface Buildings	12,523	-	12,523
Surface Electrical Power	5,371	-	5,371
TMF & pipeline	6,926	14,095	21,021
Process Plant	61,503	3,028	62,715
Environmental & land acquisition	5,143	5,984	11,127
Mine Closure		5,798	5,798
Shipping & logistics	3,479		3,479
TOTAL	**181,167**	**70,773**	**251,940**

12.2 Estimation Methodology

The capital cost estimate is based on the following project data:

- Design criteria
- Flow sheets
- General arrangement drawings
- Single line drawings
- Equipment lists
- Supplemental sketches as required
- Budget quotations from vendors
- Contributors in-house databases and operating experience
- Preliminary mining contractor quotations

The following is excluded from the capital cost estimate:

- Owner's costs before start of construction;
- Permitting costs
- Environmental impact studies
- Any additional civil, concrete work due to the adverse soil condition and location
- Import duties and custom fees
- Cost of further exploration, metallurgical testwork or geotechnical investigation
- Working capital to cover operating cost before any income is received
- Funds invested prior to the estimated capital expenditure
- Permitting expenses

- Future expansion
- Purchase of existing facilities and buildings

12.3 Mining

DMT and NIO have estimated the capital costs for re-opening of the mine including initial dewatering, pre-production development and infrastructure, mobile equipment fleet, minerals handling equipment (primary crusher and conveyors).

Sustaining capital includes on-going capital developments of ramps, declines and ventilation and ore pass raises, new crusher stations, new conveyors and replacement of some mobile equipment units.

Replacement and maintenance costs for the main ore production equipment fleet are included in the mine operating costs.

Life of Mine capital costs total US$91.51 million, of which US$49.64 million is spent during Pre-production, plus US$41.87 million spent on ongoing capital costs.

Table 12-2 provides a breakdown of mining capital costs.

Table 12-2: Mining Capital Costs

Area	Cost (US$'000s)
Mine Development and Equipment	24,078
Mine Dewatering	4,349
Main Decline conveyor	6,100
Permanent Dewatering Equipment	2,319
Underground Crushers, Conveyors, Power	8,160
Ventilation Equipment	4,638
Mine Sustaining Capital	41,867
Total	**91,511**

There are two pre-production years. In Year -2 pre-production construction capital consists largely of mine dewatering, ramp development and surface earthworks. Mobile development equipment is provided by the development contractor.

Mobile mine production equipment costs are based on manufacturer's price lists and preliminary/indicative quotes from Swedish manufacturers obtained by NIO in Q1 2015. NIO also received quotes from Swedish contractors for the development of the surface ramp connecting the existing workings to surface.

Ongoing capital is spent on major overhauls and replacement of mobile equipment every 5 years.

DMT provided the estimated capital costs for the underground crushing and conveying systems based on preliminary quotations received from Sandvik.

12.3.1 Ventilation cost – CAPEX

Total installation cost is estimated to 30.6MSEK.

Table 12-3: Ventilation system capital costs

PosNr	Skede	Beskrivning	Data.beteckning spec.	Enhet	Mängd	Enhetskostnad SEK	Totalt SEK	Notat Anmärkning
	2	Gasolvärmeanläggning 10MW	Maxon	st	1	4 180 000,00	4 180 000,00	Maxon offert
	2	Byggnad för gasolvärme		st	1	1 000 000,00	1 000 000,00	
	2	Gasolcistern inkl installationer	Örebro Gasteknik	st	1	2 750 000,00	2 750 000,00	
	2	Difftrycksportar	EOL Ventsystem	st	6	700 000,00	4 200 000,00	
	2	Väggar mot gamla schakt etc		st	50	100 000,00	5 000 000,00	
	3	Primärfläktar tilluft 420m	EVS 180-76-12-250kW	st	2	500 000,00	1 000 000,00	
	3	Elutrustning och styrutrustning		st	2	500 000,00	1 000 000,00	
	3	Montage (el och mek)		st	2	200 000,00	400 000,00	
	1	Tilluftsfläkt Sandellmalmen	EVS 125-40F-06-22kW	st	2	140 000,00	280 000,00	
	1	Elutrustning och styrutrustning		st	1	140 000,00	140 000,00	
	1	Montage (el och mek)		st	1	100 000,00	100 000,00	
	3	Reglerspjäll tilluft	EOL Ventsystem	st	8	90 000,00	720 000,00	
	3	Elutrustning och styrutrustning		st	8	90 000,00	720 000,00	
	3	Montage (el och mek)		st	8	50 000,00	400 000,00	
	3	Sekundära tilluftsfläktar, diam 900mm	EVS 90-40F-06-37kW	st	10	75 000,00	750 000,00	
	3	Elutrustning och styrutrustning		st	10	75 000,00	750 000,00	
	3	Montage (el och mek)		st	10	20 000,00	200 000,00	
	2	Sekundära tilluftsfläktar "små" diam 630mm	EVS 63-28-12-11kW	st	10	32 000,00	320 000,00	
	2	Elutrustning och styrutrustning		st	10	32 000,00	320 000,00	
	2	Montage (el och mek)		st	10	20 000,00	200 000,00	
	3	Frånluftsfläktar med spjäll	EVS 125-40F-06-22kW	st	12	140 000,00	1 680 000,00	
	3	Elutrustning och styrutrustning		st	12	100 000,00	1 200 000,00	
	3	Montage (el och mek)		st	12	50 000,00	600 000,00	
							-	
	3	Ventilationsduk 1,2m diam	JP752, 20 meter	st	50	3 500,00	175 000,00	
	3	Ventilationsduk 1,0m diam	JP752, 20 meter	st	50	3 000,00	150 000,00	
	2	Ventilationsduk 0,8m diam	JP752, 20 meter	st	50	2 500,00	125 000,00	
	1	Fläktstation för drivning av ramp inkl duk	EOL Ventsystem	st	1	2 300 000,00	2 300 000,00	Elvärmare och container
	1	*Summa för skede 1, Rampdrivning o Sandell*					*2 820 000,00*	
	2	*Summa för skede 2, 240-420m**					*18 095 000,00*	
	3	*Summa för skede 3, full produktion*					*9 745 000,00*	
		Inte med i denna kalkyl är:						
		Bergarbeten						
		Ställverk och högspänningsmatning						
			Summa totalt				**30 660 000,00**	**SEK**

12.4 Process Plant

Major process equipment was priced by TSC from a combination of database and some preliminary vendor quotations. Minor equipment items were based on factor percentages of mechanical equipment supply costs from TSC experience of previous plants.

In establishing the capital expenditure (CAPEX) requirements for the Blötberget process plant, budget quotations were sought from international equipment suppliers for selected capital items, including;

- High pressure grinding rolls (HPGR),
- Vertical stirred media mills for regrinding of magnetite and hematite pre-concentrates,
- Plate and frame pressure filters (type VPA) for dewatering of iron ore concentrate,

Together these account for two-thirds (67 %) of the Main Equipment Costs.

Where no budget quotations were available, information from similar, recent projects and cost databases was used employing typical industry standard costing procedures to arrive at projected investment requirements.

The Main Equipment Costs were used as the basis for generating a Total Investment Cost for the process plant. Following generally accepted industry norms for studies of this kind and accuracy, the estimates for cabling, piping, erection, and civil and structural works were obtained by applying a percentage increment to the Main Equipment Costs.

The overall level of accuracy for the Process Plant CAPEX estimate is ±30 %.

12.4.1 Main Equipment Costs

A Main Equipment List, containing ex-works unit item costs as well as information specifying the basis of the estimate for each of the major items, is presented in Table 12-4.

In presenting their written quotes all of the equipment manufacturers understood that these quotes were budgetary in nature. It would be anticipated that there would be some potential discount to these figures once a full scale tendering process is entered into. However, for the purposes of this Interim Technical Report, these budget quotations were taken as written.

12.4.2 Ancillary Equipment Costs

Following generally accepted industry procedures, a percentage increment of 12 % was applied to the relevant Main Equipment Costs to estimate the capital costs for Ancillary or 'non-major' equipment costs, e.g. those associated with pumps, tanks or access platforms etc. The Ancillary Equipment Costs for the Blötberget process plant are presented in Table 12-4

12.4.3 Erection, piping & cabling cost

A methodology similar to that described above was used to establish an estimate for the cost of Erection, Piping and Cabling. A percentage increment of 15 % was applied to the relevant Main Equipment Costs in this case. The Erection, Piping and Cabling Costs for the Blötberget process plant are presented in Table 12-4

12.4.4 Total Installed Costs

Using the Main Equipment Costs, Ancillary Equipment Costs, Erection, Piping and Cabling Costs and Civil and Structural Costs, as described above, a Total Installed Cost CAPEX figure for the Blötberget process plant was determined.

The estimated Total Installed Cost for the process plant is US$ 47.18, this is summarised in Table

12.4.5 Total Capital Cost

A Total Capital Cost for the Blötberget process plant was compiled using the estimated Total Installed Costs plus percentage elements for shipping, contingencies, EPCM, logistics, taxes and spares.

The process design recommended in this Interim Technical Report requires an estimated initial Capital Expenditure of US$ 61.5 million. This figure is only considered to be appropriate for the design presented within this Interim Technical Report.

An additional figure of 1% of the Total Equipment Costs (approximately US$ 252k per annum) is included for Sustaining Capital over the life of the project.

Table 12-4: Main equipment list, process plant

Position		Units	Type	Design duty (t/h)*	Maximum capacity (t/h)*	Installed power (kW)	Weight basic unit (kg)
Secondary crusher	S2	1	Metso GP550 extra coarse (cone), stroke 45mm, css 34mm (or similar)	690	680	355	26,500
-70mm ROM storage silo	S3	1	Ø15m x 15m concrete silo, circular section; bulk density 2t/m3 (PEA)	5,000t (2,500m3)	n/a	n/a	n/a
-70mm ROM emergency stockpile	S3	1	6m x 20m x 85m windrow (for example)	10,000t (5,000m3)	n/a	n/a	n/a
Emergency stockpile hopper		1	Free standing on top of conveyor belt	10m3	n/a	n/a	n/a
HPGR	S5	1	RPS 7-170/110 (or similar)	1,045	1,350	2x 850	26,500
Conveyors		var	Assume max. 20 degree inclination; 15-18 degrees preferred	n/a	n/a	n/a	n/a
Single-deck screen	S5	1	Inclined vibrating screen, Metso RF3061-1 3.0m x 6.1m (or similar)	1,045	1,140	2x 22	12,000
Wet screens 1		2	Horizontal vibrating screens, Metso LH3061-1 3.0m x 6.1m (or similar)	700	700	2x 55	2x 12,000
Rougher (Rg) LIMS	S9	4	2x 2 drums, Metso WS-1224CR or similar (1,200 x 2,400mm)	475	480	4x 7.5	4x 4,700

Position		Units	Type	Design duty (t/h)*	Maximum capacity (t/h)*	Installed power (kW)	Weight basic unit (kg)
Wet screens 2	S12	6	5-deck Derrick Stack Sizers, 100 micron urethane panels, assumed 90ts/unit from 100 to 350 t/h. This is roughly equivalent to 3 or 4 of the older style Multifeed screens discussed a	463	540	6x 1.9	6x 5,000
Regrind Rg LIMS Conc	S13	2	Metso VTM-1500-WB (or similar)	231	320	2x 1,120	2x 378,000
Cleaner (Cln) LIMS	S14	4	4x double-drums Metso WS-1224CTC or similar (1,200 x 2,400mm)	231	240	4x 15	4x 9,400
Rougher (Rg) spirals	S11	128	4 banks of 16 twin start 7-turn WW spirals, 2.8t/h/start, 20% redundancy	244	358	n/a	4x 4,200
Wet screens 3	S18	2	5-deck Derrick Stack Sizers, 100 micron urethane panels, assumed 90t/unit	116	180	2x 1.9	2x 5,000
Regrind Rg Spiral Conc	S19	1	Metso VTM-1500-WB (or similar)	32	110	1,120	378,000
Hydrosizer (Elutriator)	S20	1	TBS Ø2.4m	83	192	n/a	n/a
Cleaner (Cln) spirals	S20	128	4 banks of 16 twin start 7-turn WW spirals, 0.9t/h/start, 20% redundancy	83	115	n/a	4x 4,200
Flotation - conditioning 1	S21+S22	1	Metso Ø4m x 4m; 45m3, 10 min residence	199 ($206m^3$	n/a	7.5	n/a
Flotation - conditioning 2	S21+S22	1	Metso Ø3m x 3m; 19m3; 5 min residence	199 ($206m^3$	n/a	5.5	n/a

Position		Units	Type	Design duty (t/h)*	Maximum capacity (t/h)*	Installed power (kW)	Weight basic unit (kg)
Flotation	S21+S22+W10	1	4 x Metso RCS20, 80m3; 15 min flotation time; I 2 I 2 I arrangement	199 (342m^3	n/a	4x 37	4x 7,300
Concentrate thickener	S24	1	30t/m2h; 12m2; Ø4m x 3m conventional	199 (342m^3	360	~10	n/a
Pressure filtration	S28	1	Metso VPA 2050-56; 6.5% moisture; 1.5min air blow; 3min total cycle time	199 (150m^3	n/a	1x 200 2x 450	1x 80,000
Spiral classifier	S17	1	48'' (120) Metso 120Lo	211 (+750m^3	n/a	7.5	8,000
Tailings thickener	S29	1	Metso LTE800/C Ø10.5m x 3m Lamella (or similar)	224 (+1374m^3 H2O)	(1,600m^3/h)	~10	n/a

Table 12-5: CAPEX summary for process plant

Pos.	Item	Operating hours	Nominal Throughput	Design Throughput	Imported		Indigenous	Total
		hrs/y	t/h	t/h	EUR	US$	SEK	US$
1	Secondary crusher	4,788	627	690		1,487,500		1,487,500
2-A	HPGR	6,955	950	1045	3,218,646			3,466,375
2-B	HPGR-Set of drives				287,801			309,952
2-C	HPGR-Set of spare rolls				1,374,898			1,480,719
3	Single-deck screen 5mm	6,955	950	1045	232,000			249,856
4	Wet screens 1mm	6,955	636	700	464,000			499,713
5	Rougher (Rg) LIMS	6,955	432	475		292,000		292,000
6	Wet screens 100microns 1	6,955	421	463		1,308,000		1,308,000
7	Regrind Rg LIMS Conc	6,955	211	231		6,723,500		6,723,500
8	Cleaner (Cln) LIMS	6,955	211	231		604,000		604,000
9	Rougher (Rg) spirals	6,955	222	244		522,240		522,240
10	Wet screens 100microns 2	6,955	105	116		436,000		436,000
11	Regrind Rg Spiral Conc	6,955	29	32		3,361,750		3,361,750
12	Hydrosizer (Elutriator)	6,955	75	83		90,400		90,400
13	Cleaner (Cln) spirals	6,955	75	83		522,240		522,240
14	Flotation - conditioning 1	6,955	181	199		111,265		111,265
15	Flotation - conditioning 2	6,955	181	199		72,590		72,590
16	Flotation	6,955	181	199		1,392,000		1,392,000
17	Concentrate thickener	6,955	181	199		35,000		35,000

Pos.	Item		Operating hours	Nominal Throughput	Design Throughput	Imported		Indigenous	Total
			hrs/y	t/h	t/h	EUR	US$	SEK	US$
18	Pressure Filter		6,955	181	199	1,484,400			1,598,649
19	VPA Compressor		6,955	181	199		300,000		300,000
20	Spiral classifier		6,955	192	211	81,432			87,700
21	Tailings thickener		6,955	204	224		280,642		280,642
22	**Total Main Equipment**								**25,232,091**
23	Ancillary equipment cost				12%				3,027,851
24	Erection, piping & cabling cost				15%				3,784,814
25	Civil and Structural Costs				60%				15,139,255
26	**TOTAL INSTALLED COST OF EQUIPMENT**								**47,184,011**
28	Contingency (% of Total Installed Cost)				10%				4,718,401
29	EPCM (% of Total Installed Cost)				15%				7,077,602
32	Spare Parts (% Main Equipment Costs)				10%				2,523,209
33	**TOTAL INVESTMENT COST**								**61,,503,223**
34	Sustaining Capital	1%	of Total Main Equipment						3,027,851

12.4.5.1 Tailings Pipeline

The total estimated capital cost of the pumps and pipelines is US$550,000. A summary is presented in Table 12-6.

Table 12-6: Summary of Tailings Pipeline Capital Costs

Item	Unit cost, US$	Number	Cost. US$
8/6 AH tailings pumps	60,000	6	360,000
1,600 m HDPE 200 mm Ø pipeline	120,000	1	120,000
8/6 DWU water pump	15,000	2	30,000
1,600 m HDPE 114 mm Ø pipeline	40,000	1	40,000
Total			**550,000**

12.4.6 TMF and Water Management

Table 12-7 shows the capital costs associated with the TMF and CP dams.

Table 12-7: Dams – Capital Costs

Activities	Capex Northern [kSEK]	Sust Capex Northern [kSEK]	Sust Capex Southern [kSEK]
Detailed Geotechnical Investigation	1 000	-	1 500
Detailed Design and Construction	1 000	-	1 500
Cut and clean trees	1 077	918	4 122
Excavation drainage ditch	361	570	1 757
Excavation for Dam Foundation	4 197	-	7 009
Construction activities in wet areas	2 680	-	2 680
Processing/Crushing incl trsp < 3km	12 650	10 877	32 870
Placement and Compaction	18 247	9 633	41 221
Instrumentation	600	600	1 200
Detailed Geotechnical Investigation	1 000	-	1 500
Detailed Design and Construction	1 000	-	1 500
Spillway	6 000	295	4 795
Total cost	**45,810**	**22,893**	**95,654**

12.5 Infrastructure

Ramböll estimated the civil construction and building costs based on unit rates and preliminary cross-sectional slab areas.

Structural steel requirements for the various major equipment items and buildings were estimated from unit rates from similar equipment installations.

Unit costs for steel, including installation labour and equipment requirements, are based on Ramböll in-house database information.

12.5.1 Civil and Structural Costs

Experience from projects of similar nature was used to arrive at an estimate for Civil and Structural Costs; a percentage increment of 60 % was applied to the relevant Main Equipment Costs.

The Civil and Structural Costs for the Blötberget process plant are presented in Table 12-5.

The capital cost estimate for the buildings and structures is included as an Appendix I to this report.

The capital cost estimate has been prepared considering the following assumptions;

- RC concrete foundations
- Insulated concrete floor slabs
- Prefabricated concrete bases
- Steel frame and steel lattice trusses for the roof
- Insulated sheet metal roof
- External walls made from SIPs
- Internal stud work partition walls

The estimate does not include;

- Equipment fit out
- Geotechnical Surveys
- Furnishings
- Special details specific to processing

A summary of the capital cost for each building can be seen in Table 12-8

Table 12-8 Capital Cost Summary – Buildings

Description/Building	Quantity m^2	Cost SEK
Office/Locker Room	1151	17,700,000
Drill core Archive	885	7,500,000
Stores	1800	13,200,000
Process Plant Building	2550	27,100,000
General Maintenance Workshop	473	8,100,000
Vehicle Workshop	1015	15,100,00
BS Shaft Heating/Ventilation Building	374	4,900,000

Description/Building	Quantity m^2	Cost SEK
	Total	**93,600,000**

12.6 Logistics

12.6.1 Skeppmora Rail Terminal

The capital investment for the Skeppmora rail terminal is estimated to be a total of US$12.05 M made up as follows:

- Main Switch (year 2016): estimated cost US$3.47 M
- In Swedish terms “Linjeplats” - operating by a line controller (year 2016) Estimated cost: US$4.64 M
- In Swedish terms “Driftsplats” – full remote control terminal (year 2017) Estimated cost US$3.36 M
- Contingency (5%): US$0.57 M

12.6.2 Oxelösund Port

The necessary expansion works at the port will be funded by the Port Authority and have been factored into the port handling charges already agreed with NIO.

12.7 Environmental and Land Owner Compensation Costs

The environmental permit (EP) issued by the Land & Environmental Court requires the Company and instigate certain control programmes prior to the start of construction activities on site. Costs of these have been included in the Capex estimate as outlined below.

12.7.1 Compulsory rights

The EP defines the process for compensation to the land owners. Compensation is due to land owners and in total SEK 1 million will be paid to the land owners and has been included in the costs.

12.7.2 Compensation for loss of flora and fauna

A plan for compensation of lost flora and fauna, mainly in the tailing dam area, will incur costs during the preparation of the proposed new area and is estimated to SEK 2 million with a yearly maintenance cost of SEK 250k during the life of mine.

12.7.3 Water table control programme

The EP requires a water table control programme in connection with water and energy wells around the concession area to be implemented. The running cost of the programme is estimated to be SEK 2 million during the construction phase and until the mine is finally drained.

Additionally a compensation for disturbance and detected energy losses SEK 1.4 million is to be paid.

12.7.4 Designation of Land – Land Acquisition

The Minerals Act provides for a structured expropriation process for access to any land needed for mining operations. The land affected by the Project is the industrial areas 1 & 2 and the train terminal. Some of the land will be needed after approximately three years of production when the starting dams in the southern tailing dam area will be constructed.

The total cost of land acquisitions is approximately SEK 20 million.

12.7.5 Compensation to other land owners

Separate agreements with house owners in the village Ickorrbotten was offered by NIO in connection with the environmental court proceedings and these will result in a payment of SEK 2.3 million at the start of the construction works. Similarly, payments estimated at SEK 1.0 million are expected to be made to the house owners in Torsbovallen and Jansbo.

12.8 Closure & Reclamation

The Environmental Permit requires that the fund for land reclamation shall be deposited at the County Administrative Board (CAM) as from the commencement of the activities at site. In total the company shall make deposits up to a sum of SEK 53.4 million. Upon commencement of construction activities the company must make a first deposit of SEK 15.8 million and thereafter a yearly deposit of SEK 3.2 million.

A further lump sum for closure and reclamation costs after operations are completed of SEK 50 million has been included.

12.9 Shipping and Logistics in Sweden

An allowance of 6% of capital cost for shipping of imported equipment (principally process equipment) totalling US$1.7 million has been made.

Logistics within Sweden of the main items of equipment (process equipment, conveyors and crushers and mobile equipment) has been allowed for at 3% of the capital cost amounting to US$1.8 million.

12.10 Contingency

Contingency accounts for unforeseen costs within the Project scope.

Percentages were assigned by area and category, including indirect capital costs. Contingency ranged from 10% on well-defined areas (quoted mobile equipment) to 15% on processing equipment, infrastructure and earthworks.

Totalling US$23.59 million, contingency amounts to 9.6% of direct capital costs.

13 OPERATING COST ESTIMATE

Average Total Operating costs are estimated to be US$19.64/tonne of ore, equivalent to US$43.96/dry metric tonne of concentrate, based on an average ROM production of 3.0 Mtpa and a process weight recovery (yield) of 45%.

The average LOMP operating costs have been estimated for each main area of the operation and are summarised in Table 13-1.

Table 13-1: Average Opex Costs

Item/Description	Operating Expenditure $US\$/t_{ore}$	Operating Expenditure $US\$/t_{conc}$
Mining Opex	9.54	21.19
Processing Opex	2.05	4.88
Underground Infrastructure Opex	0.19	0.43
Surface Infrastructure Opex	0.10	0.23
Power (excluding Process)	1.10	2.44
TMF Opex	0.40	0.88
TMF Pipeline & pumps	0.06	0.13
Railway Opex	2.86	6.35
Port handling	1.78	3.95
G&A Costs	1.56	3.48
TOTAL	**19.64**	**43.96**

13.1 Mine Operating Costs

The mine operating cost estimate includes:

- Stoping ore development
- Production drilling & blasting
- Ore & waste haulage
- Mining Labour
- Power
- Overhaul & Maintenance
- NIO management & services

A contingency allowance of 15 % was applied to the total estimated mine operating costs.

Table 13-2: Breakdown of Mining operating costs

Item/Description	Operating Expenditure	Operating Expenditure

	US$/tore	US$/tconc
Production drives incl. rock support	1.38	3.06
Access drives incl. rock support	0.27	0.60
Foot wall drives incl. rock support	0.48	1.07
Long-hole drilling & blasting	1.69	3.76
Charging & blasting	0.65	1.45
Loading	2.32	5.15
Horizontal haulage	0.19	0.43
Running cost - power, heating & gasoil surface	0.04	0.08
Heating UG (gasoil)	0.39	0.86
Maintenance of ventilation, electrical, water	0.56	1.25
Geology, engineering departments	0.21	0.47
Overhead	0.12	0.26
Contingency	1.24	2.76
TOTAL	9.54	21.19

13.2 Process Operating Costs

The operating cost estimate presented hereinafter covers:

- Power
- Grinding media and liners
- Overhaul & Maintenance
- Reagents
- Labour

13.2.1 Power

The Main Equipment motor list (see Table 13-3) was used to estimate average power consumption. The cost of power was estimated at SEK 0.5/kWh.

Other assumptions included;

Table 13-3: Process Plant Main Equipment Power List

Item	POWER			
	Power Installed kW	Load Factor	Average Power Draw kW	Specific Energy kWh/t
Secondary crusher	355	0.85	208	0.57
HPGR	1,700		1,283	1.35
Single-deck screen 5mm	44	0.85	37	0.05
Wet screens 1mm	110	0.85	94	0.17
Rougher (Rg) LIMS	30	0.85	26	0.07

Item	POWER			
	Power Installed kW	Load Factor	Average Power Draw kW	Specific Energy kWh/t
Wet screens 100microns 1	11.4	0.85	10	0.03
Regrind Rg LIMS Conc	2,240		1,453	6.92
Cleaner (Cln) LIMS	60	0.85	51	0.29
Rougher (Rg) spirals	0	0.85	0	0.00
Wet screens 100microns 2	3.8	0.85	3.2	0.04
Regrind Rg Spiral Conc	1,120		320	11.00
Hydrosizer (Elutriator)	0	0.85	0	0.00
Cleaner (Cln) spirals	0	0.85	0	0.00
Flotation - conditioning 1	7.5	0.85	6.4	0.04
Flotation - conditioning 2	5.5	0.85	4.7	0.03
Flotation	148	0.85	126	0.82
Concentrate thickener	10	0.85	8.5	0.06
Pressure Filter	200	0.30	60	1.11
VPA Compressor	900	0.50	450	4.97
Spiral classifier	7.5	0.85	6.4	0.04
Tailings thickener	10	0.85	8.5	0.05
Pumps and other (+10%)	696	0.85	592	
Total	**7,659**		**4,746**	

The total installed power for the Process Plant is estimated at 7,659 kW. The estimated average power requirement is 4,746 kW.

Table 13-4 details the specific consumptions and unit costs as assumed for the OPEX estimate.

Table 13-4: Specific reagent consumption and costs

Position	Through put, t/h	Dispersant g/t	Collector g/t	Frother g/t	Polymer g/t	Cost US$/h
Flotation – cond. 1	181	500				36.2
Flotation – cond. 2	181		50			51.6
Flotation	181			10		3.6
Conc. thickener	181				20	16.3
Tailings thickener	204				5	4.6
Total						112.2

Experience from projects of similar nature was used to arrive at an estimate for Labour costs; a percentage increment of 15 % was applied to the relevant OPEX Costs.

A contingency allowance of 15 % was applied to the relevant OPEX Costs.

A summary of the process plant Opex is presented in Table 13-5.

Opex data for the Tailings Pipeline and Return Water Pipeline and associated pumps are presented separately in Section 8.8.

Table 13-5: OPEX Summary for Process Plant

Item/Description	Operating Expenditure US$/tore	Operating Expenditure US$/tconc
Power	0.64	1.52
Liner & Media Wear	0.28	0.68
Overhaul & Maintenance	0.39	0.94
Reagents	0.26	0.62
Sub-total	1.57	3.76
Labour	0.24	0.56
Contingency	0.24	0.56
TOTAL	**2.05**	**4.88**

13.2.2 Tailings Storage Facility

The following Table 9-20 shows the operating cost estimates for the TMF and CP facilities.

Table 13-6: Dams – Operating Costs

Activities	Annual Opex Northern [kSEK/year]	Annual Opex Southern [kSEK/year]
Tailings disposal	624	624
Raising the dam	1 900	1 444
Processing/Crushing including transport < 3km	4 467	3 541
Placement and Compaction	2 893	2 210
Spillway	172	172
Supervision-Internal	882	1 734
Supervision-External	111	103
Total cost	**12 048**	**10 828**

Estimated operating cost of tailings and return water pumps, excluding power is US$0.03/tonne of concentrate.

Estimated power cost = US$0.10/tonne of concentrate

Total estimated operating cost = US$0.13/tonne of concentrate.

13.3 G & A

General and Administration costs have been developed by NIO. Costs include corporate management, accounting, environmental monitoring, health and safety, training, security, insurance, IT and other site services.

Annual G & A costs are estimated to be SEK 40.5 million, giving operating costs of US$1.56/tonne of ore and US$3.48/tonne concentrate.

13.4 Logistics

13.4.1 Train operation:

The railway operating cost that includes all cost components amounts to US$5.97/wet metric tonne concentrate (51.50 SEK/tonne).

13.4.2 Port handling:

Handling costs at the Port of Oxelösund amount to US$3.71/wet metric tonne concentrate (32 SEK/tonne).

14 PRODUCT MARKETING AND DISTRIBUTION – IRON ORE CONCENTRATE – NIO

14.1 Introduction

Metallurgical testwork programmes on samples of the Blötberget ore carried out by NIO in Sweden and Finland, plus historical records demonstrate that a very high quality iron ore concentrate can be produced from the iron ores of Blötberget. The testwork demonstrates that iron ore concentrates with around 70% Fe(t) content can be produced with low levels of contaminants. (See below for complete analysis).

This section of the report aims to highlight the major dynamics in the supply demand balance, the variability of the products produced by the miners and the geographical global supply to the end users. The vast majority of iron ore mined (probably >98%) goes to the steel industries.

NIO has kept close contact with the market place and players within it. This includes maintaining a constant dialogue with analysts, suppliers and buyers; with recent inputs from leading commodity analysts CRU to help reinforce the fact that the development of the NIO mines is low risk (compared with many developments around the globe) and will provide the market place with low volumes of a niche product to those steel companies wishing to increase their product quality and lower their operating costs.

Anybody who is in the iron and steel industries will be aware that there has been a significant decline in the prices of the iron ore. This has had a major impact on many of the iron ore development projects and an even bigger impact on many operating mines which were developed when prices were high, capital spending less well controlled and operational efficiency less important.. The influences that have driven these prices lower are many and are dealt with in more detail later in this section.

NIO has also collated significant data from banks and other financial institutions regarding the trends in the iron ore market and presents in this section, its own view of the market for iron ore.

14.2 Basics of iron ore

Iron ore is the major raw material used in the production of iron and steel and it occurs naturally in a number of forms. The principal iron-bearing minerals are magnetite (Fe_3O_4), hematite (Fe_2O_3), siderite ($FeCO_3$) and limonite ($FeO(OH)\cdot nH_2O$) whilst other less important iron sources include goethite, chamosite, lepidococite and chalybite.

Pure magnetite and hematite contain approximately 72.4% and 70% iron respectively whist siderite contains only 48% iron. Because of its variable hydrated nature, the iron content of pure limonite can range between 38-51%.

Iron ore deposits almost always occur as an intimate mixture of two or more of the iron bearing minerals plus a number of unwanted or ‘gangue’ constituents. These include minerals and rocks such as quartz, clays, shales, calcite, mafic silicates, granites & intrusives, sulphides and apatite etc, which give rise to a number of unwanted elements and compounds, contaminating and diluting the as-mined ores.

Typically, high-grade iron ore will contain between 60-68% Fe whereas low to medium grade ores contain as little as 20-48% Fe. In recent years in some countries ores below 20%Fe have been mined, but iron ore pricing in the market and economics will ultimately dictate if that is sustainable. The only lower grade ores considered as potentially viable are almost invariably magnetite ores.

Most mining operations employ some degree of beneficiation (i.e. ore upgrading), which is carried out before the ore is transported to market. This may involve a simple crushing, washing and screening operation in the case of high-grade ores. However, as many of the accessible higher grade deposits have been depleted, lower grade materials are entering the market, thereby increasing demand for high grade products.

Magnetic separation is widely used with magnetite ores, whilst other techniques including gravity concentration, heavy media separation and froth flotation are used with other ores. However not all ores are amenable to beneficiation and therefore not all ores can be materially upgraded without losing a significant amount of the total iron content (Fe and mass recovery curves).

Over recent year’s mineral processing, materials technology and process controls have significantly improved and it is now not uncommon to be able to upgrade some ores containing less than 30% Fe to produce a concentrate with a Fe content well in excess of 67% whilst still recovering 75% or more of the total iron.

NIO Blötberget pilot plant testing demonstrated that it was possible to produce a magnetite concentrate containing in excess of 70%Fe (See Section 8.3) with recoveries of total Fe over 98%.

Iron ore may also occur as a bi-product mineral in other mineral deposits (for example as iron pyrite in a copper sulphide deposit or as iron occurring in titanium rich minerals such as ilmenite) and through processing it can be concentrated (as a waste material) and produced in such volumes as to have potential as an economic source of iron. Though, it has to be said, this only occurs on a relatively small scale when compared with the larger iron ore mining operations.

14.3 Iron Ore Products

Iron Ore is supplied to the world’s iron and steel industry in 4 main forms:

- Lump Ore
- Sinter Fines
- Pellet Fines/Feed/Concentrates
- Sinter and Pellets (agglomerated ores)

NB: pellet fines/feed and concentrate and pellets are more or less the same generic product, since, almost all of the iron ore concentrate is used to produce pellets. It very much depends on the location of the pelletising plant (i.e. at the mine or at the steel plant) as to the exact format of the iron ore supply to the steel plant.

14.3.1 Lump Ore ("Direct Shipping Ore")

Lump ore is supplied in the nominal size range >6mm<100mm (although there is normally very little material that is >50mm) and is produced in a range of different iron grades:

- Lower grade lump can vary in iron content from 58-64% and is generally used as feed material to BF iron making operations, together with sinter, in proportions from 0-100%. Though in modern, high productivity BF operations 5-30% lump ore has been used more typically
- High-grade lump typically contains between 64-68% Fe and (assuming the correct metallurgical properties) and is almost exclusively used in direct reduction (DR) iron making, where it usually commands a premium price. Though through lack of reliable source many operations now use predominantly DR pellets

Lump ore is predominantly produced by crushing the as-mined ore to a top-size of around 60mm and then typically screened at both 50mm and 6mm. The >50mm material is recycled to be re-crushed whilst the >6mm<50mm size fraction is extracted as the lump fraction.

In addition to the chemical requirements, lump ore must have sufficient physical strength or competence to resist breakdown in the upper/feed end of the BF or DR shaft during the heating and reduction processes.

Lump ore supply has become increasingly scarce globally and is not used or as valued as much as it was in the 1970's to 2008. Lump has generally been replaced by pellets or sinter products. This is demonstrated by the fact that recent pricing saw the premium of lump over fines almost disappear for a while.

14.3.2 Sinter Fines

The <6mm fraction from a crushing and screening operation is commonly de-slimed or washed (if water is available), during which some of the <1mm material may be removed. The resulting >1<6mm product is known as sinter fines and is supplied to sinter plants at the steel plant where it is fused with fluxes, coke breeze and other carbon and ferrous steel plant "wastes" to produce sinter for use in Blast Furnace iron making operations.

Sinter fines can vary considerably in iron content but generally speaking most steelmakers operate with 58-62% Fe in the product. For reasons of sinter quality the vast majority of sinter fines are hematite/siderite/limonite/goethite ores, though some operators in China are known to use significant amount of magnetite ores in their sinter-making process.

14.3.3 Pellet Fines, Feed and Concentrates

Many ores are not sufficiently high grade to be useable in the iron making processes without further processing. Although there are many different methods of beneficiation, almost all require the crushed product to be finely ground (usually <1mm) before undergoing beneficiation to remove excess gangue material.

The iron grade of the concentrate will primarily depend on the ore type, the mineralogy and the relationship between the iron minerals and the gangue minerals. The particle size of the ore at which significant gangue minerals are liberated is crucial, dictating the quality of the product, the Fe recovery and the equipment/energy required to achieve a saleable product.

In recent years internationally traded concentrates have usually contained in excess of 63% Fe content. In China, where low grade magnetite occurs in great quantities, it is generally difficult to achieve concentrates with Fe levels over 60%Fe. Generally the higher quality concentrates traded internationally trade at base prices similar to sinter fines, but with some form of a premium paid for products in the range of 65-70%Fe (commonly based on value-in-use criteria associated with levels of contaminants, primarily Si, Al, S and P.

The NIO Blötberget product falls into the highest Fe content available in the current market place and as such is considered to be a niche product.

14.3.4 Sinter

Sinter plants combine sinter fines, steel plant wastes, fluxes, screenings from lump and pellet feed to BFs to make a porous feed for the BF that is readily reduced and melted in the BF. Sinter plants are often the “chemical” trim for the BF.

However because the sinter plant productivity performance used to suffer if the quantity of concentrates exceeded certain limits (typically 10-15%wt% in Europe) the quantities of concentrate used were often limited.

In recent years many of the sinter makers have moved to using a proportion of concentrate in their feed to the sinter plant. This is primarily because they can provide a good value source of high Fe content while allowing the operator to add a higher amount of low grade waste or cheap ores into the feed blend without overly affecting the Fe content of the sinter.

The larger quantities of pellet feed and concentrates that have entered the market has resulted in many steel plant operators looking for techniques to allow them to use significantly higher proportions of concentrates in their feedstock. Improvements in pre-agglomeration techniques are amongst the process improvements being tested. This does mean that, in Europe at least, more steel companies will now consider using high grade concentrates in their sinter feed blends.

14.3.5 Pellets

After beneficiation many concentrates tend to have a fine average particle size (invariably<0.25mm) and therefore require some form of agglomeration to allow ease of handling and use within the Blast Furnace or DR shaft, either sintering or pelletising. Often the ore will need further grinding to provide a pellet feed, which typically have a P_{80} of between 40 and 80 microns or Blaine indices of 1200-1800.

Pelletising involves binding the particles together in the balling process using “sticky” clay minerals in small quantities and subsequent drying and firing of the pellet (ball), The pellet produced is a hard, but porous, spherical ball, generally of between 6 and18mm diameter for use in either the Blast Furnace or Direct Reduction iron-making processes. These operations prefer a close size range distribution and consistent physical and chemical properties for optimum performance.

Pellets are considered a high “value-in-use” iron ore product and can carry a significant price premium over fines particularly the high-grade (low - gangue) pellets normally associated with use by the DR iron making processes. Recently this premium has been between US$30-42/t..

There are 3 main types of pellet produced and sold to the steel producers:

- Acid Blast Furnace (ABF) Pellets
- Fluxed Blast Furnace (FBF) Pellets
- Direct Reduction (DR) Pellets

Acid BF Pellets have a basicity (i.e. the ratio of lime +magnesia/silica + alumina) of less than 0.5. Acid pellets generally will have no additions to the concentrate prior to pelletising.

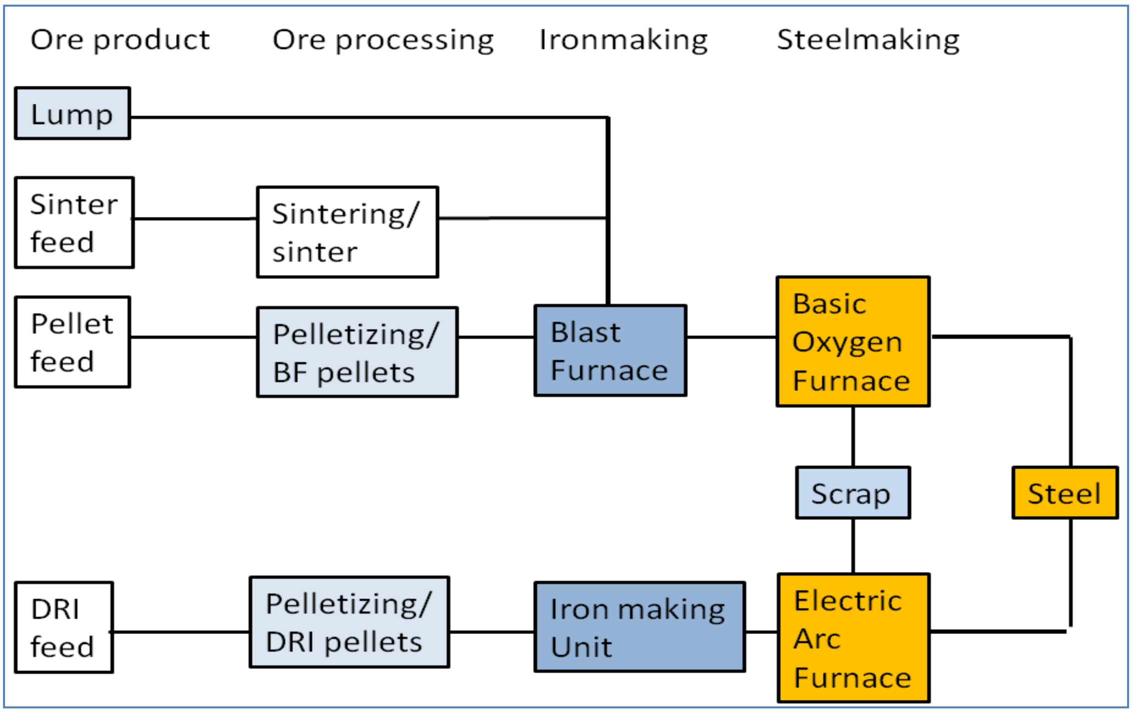
Fluxed BF Pellets exhibit a basicity of more than 0.5. Fluxed pellets are usually produced by adding fine lime or magnesia to the concentrate before pelletising and this, inevitably, lowers the final %Fe content of the pellet.

DR Pellets are characterised by a high %Fe and a low total acid gangue content – generally below 2%. It is not usual to add significant fluxes to DR concentrates prior to pelletising.

14.3.5.1 Requirements for Iron Ore

The following figure shows the various production routes to the final steel product.

Figure 14-1: Different production routes to steelmaking



There are certain minimum physical, metallurgical and chemical properties that are required to ensure that the ore is useable and therefore saleable. The physical and metallurgical properties of each of the ores are very specific to a particular product or source of ore. However, there are some characteristics that are desirable to all products.

- lump ores, sinter and pellets should demonstrate good reduction characteristics
    - high strength
    - suitable low-temperature softening characteristics for use in the BF, Corex and DR processes.
- Sinter, pellet fines and concentrates must exhibit good agglomeration characteristics

For all types of products, however, there are a number of common chemical requirements, most of which are universally demanded by the end-users: both phosphorus and sulphur must be carefully controlled below 0.02% and 0.05-0.06% respectively and ideally lower. Other tramp elements such as Cu, F, As, Pb, Zn, tend to concentrate in the BF process off-gases and can be difficult and expensive to deal with and should therefore be avoided in the feed ores to the BF. These elements do tend to be driven off, at least partially, in the sintering and pelletising operations and can be more easily handled, although at a cost.

Similarly radioactive elements tend to concentrate in the refractory in the hot metal processes and can cause disposal issues.

14.4 Iron Ore Quality, Pricing and Premiums

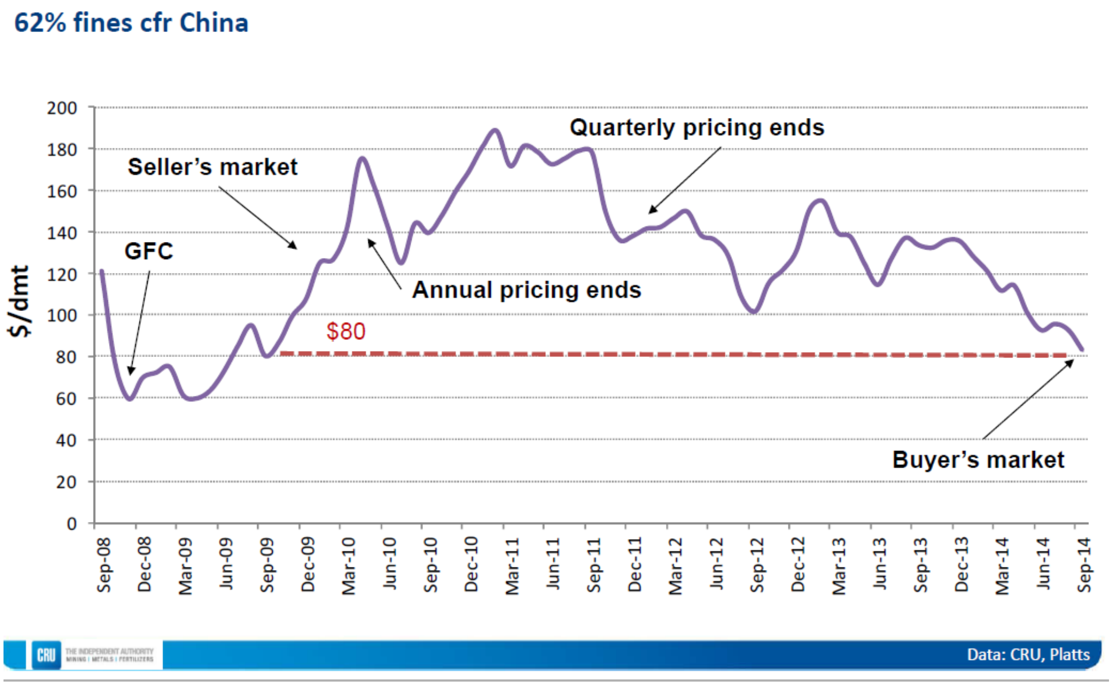
The price paid for iron ore is primarily based on the Fe content in the product sold and the level of certain impurities in the product. However, there are also a host of other factors which determine the price paid for iron ore, most importantly the size of the ore and its physical properties, together with the detailed chemical composition and moisture content.

The requirements of individual steel companies vary, and indeed no one blast furnace operation is identical to another. The steel companies plan to use a certain blend of ores, unless they are integrated into one iron ore supplier (usually captive mines) which in their view suits their economics and their operations, the combination of blends providing their best economics.

A primary concern for the buyer is always the consistency of the quality of the products delivered. Any high levels in the unwanted elements (gangue minerals) will often be mitigated by addition of a additional fluxes or other operational procedures. This is impossible to control if the ore supplier provided a product with variable qualities from one delivery to another

Additionally at the end of the day the effects of the price paid for the specific products and its effects on the steel companies combined processes (not just the blast furnace) will be estimated before purchase

Go back to the times before the explosion in demand from China, the pricing mechanism used was called “the benchmark” pricing system for iron ore. This was a system where yearly prices were set by the large buyers in certain markets (Japan and Europe alternating) and sellers (BHP, Hammersley (Rio) and CVRD (now Vale) during a relatively short negotiation period; though in later years of this system it often took a large part of the year to form a settlement Once finalised the concluded negotiations were then used as a standard for other buyers and sellers to follow. The benchmark price was then adjusted according to the iron content and other gangue minerals; and indeed any trace elements that affected the steelmakers processes.



Source: CRU 9th EU Iron Ore Conference, Prague 2014

Following the start of rapidly increasing iron ore demand from China and a lack of ability of the large international suppliers to meet the sea-borne demand to supply it and the unreliability of the domestic Chinese iron ore producers, started to put pressure on the benchmark system. This lead to, during the 2007-2009 period, rapidly increasing benchmark prices, combined with rising freight rates (there was a lack of large ships to carry the ore to China). This culminated in the breakdown of the benchmark system for the mainstream pricing of iron ore, in 2009-2010

The benchmark system was largely replaced with a mixed system, where prices are established quarterly by two of the most important producers (Rio Tinto and Vale) on the basis of indices, which in turn are based on spot market transactions. Originally the spot market was small and defined by Indian iron ore supplies to Japan and China, but with the increased demand India started to increase supply and it became a more significant component in the international pricing mechanisms and a significant part of the market started to operate with spot pricing. Whist some buyer/suppliers retained longer term agreements based on an agreed version of the benchmark mechanism the vast majority of transactions used versions of the spot pricing mechanisms, often revolving around quarterly price adjustments to reflect the previous quarter iron ore prices.

All pricing systems take into account differences in value in use (i.e. how the product behaves in the blast furnace or effects the steelmaking, depending on grades and other quality parameters), although the size of premiums and penalties, and their method of calculation, may differ from one transaction to another. This is a specific arrangements (contract) between buyer and seller) The most relevant quality references at present are the standards used in the published indices.

The main indices used by the major iron ore companies and buyers are published by Metal Bulletin, The Steel Index, and Platts and differ slightly in how were originally constructed but, by and large, to reflect the markets for low grade (i.e. ~58 % Fe) and medium grade (~62 % Fe) iron ore. These indices are all based on the delivery prices of the ore to major Chinese ports, typically CFR Tianjin or Qingdao. There has in recent years been some expansion of indices to include other grades or iron ore. For example Platts' currently has indices for the following:

- 62%Fe fines, 3.5%Al,
- 58% Fe fines, 3.5% Al
- 62%Fe fines, 2%Al
- 63.5%Fe fines, 3.5%Al
- 65%Fe fines, 1%Al

Note that Al refers to the content of alumina (Al_2O_3)

There is a premium paid for higher grade ore which many of the indices try to reflect, by either publishing Fe differentials (per 1%Fe) and/or a "premium" pricing per 1%Fe above standard indices plus The premium for higher grade ore (i.e. ~66 % Fe or above) is not necessarily directly proportionate to the premiums achieved by medium grade above low grade.

Other iron ore products e.g. lump and pellets generally command a premium to sinter fines, whilst concentrates are generally on small discount or the same as fines. The size of this premium is not constant and has varied from virtually zero for lump up to US$25 /t. For pellets the price differential was as high as US$150 /t in the past, however in the last couple of years the pellet premium varied less, generally between US$25-45 /t; settling recently to a narrower US$35-40 /t.

The higher quality concentrates often command a premium price, in China, according to the Indices. This is variable according to demand and perceived benefits of using high Fe, low Si and Al concentrates, moving between US$1-US$7/t/Fe unit.

14.4.1 Contaminants

The inclusion of even small amounts of some elements in the iron ore can have profound effects on the blast furnace or steelmaking operations, or indeed the quality of the iron ore the steel products. Some chemical elements are deliberately added, such as fluxes, to make a blast furnace operations more efficient or to produce higher quality slag materials (potentially a valuable bi-product for the cements and aggregate industries and an important part of the iron making economics). The steel companies will look carefully at the ferrous burden mix they are feeding to their furnaces in order to ensure that they can make the quality of steels desired in the most economic manner. This will include looking at key elements in the blend (including their recycled materials).

The choice of ore, fuel, and flux also determines how the slag will behave and the operational characteristics of the iron produced. Ideally iron ore contains only iron and oxygen the reality is this never happens and the steel companies have to deal with a variety of contaminants. Typically, iron ore contains a host of elements, often unwanted, in the iron making process, often for the main reason that they require additional fuel to melt for them to enter the slag in the furnace operations. Some examples of such elements are mentioned below.

- SiO2 Silica, silicon dioxide
- Al2O3 Aluminium oxide
- CaO Calcium oxide
- MgO Magnesium oxide
- S, As, Fl Sulphur, Arsenic, Fluorine
- P, Cu, Phosphorous and copper
- TiO2 Titanium dioxide
- V2O5 Vanadium Pentoxide
- Mn, Mo Manganese, Molybdenum
- Zn, Pb, Ag, Zinc, lead, silver
- Hg, Sn, Sb Mercury, tin, Antimony
- Na2O, K2O Alkalis, Sodium oxide (soda), potassium oxide - Potash
- U, Th etc Uranium and other radioactive elements
- Ni, Cr, W, As Nickel, Chromium, Tungsten, etc

Below is a brief description of the effects of each of the elements on the iron and steel making processes.

14.4.1.1 Silica

Silica (SiO_2) is almost always present in iron ores and most of it is removed in the slag during BF operations, where it forms one of the key components. But the silica, along with the alumina require relatively high fuel rates to get into the liquid state and at temperatures above 1300°C some will be reduced and form an alloy with the iron. The hotter the furnace, the more silicon will be present in the hot metal (HM) product.

14.4.1.2 Aluminium

Aluminium is very hard to reduce and in that respect aluminium contamination of the iron is not a problem, however, it does increase the viscosity of the slag and requires a high fuel rate to overcome. This will have a number of adverse effects on furnace operation, the thicker the slag the slower the descent of the BF charge, potentially slowing productivity High aluminium content also makes it more difficult to tap off the liquid slag.

14.4.1.3 Calcium and magnesium oxides

If the natural ore contain sufficient Ca and Mg, then further additions of lime and dolomite may not be necessary. Calcium and manganese oxides are the main fluxes in the BF and help form the slag with silica and alumina, which in turn is used primarily to remove other harmful elements such as sulphur and phosphorus. By controlling the composition of the slag, it is to some extent, possible to influence the distribution of tramp elements between hot metal and slag. Calcium and magnesium oxide are basic slag formers, whereas silica and aluminium oxide form acid slags. The additions of Ca and Mg to the furnace, either directly or via sinter or pellets can influence the productivity and quality of products, as well as helping to have smooth and constant operations.

14.4.1.4 Sulphur, Arsenic, Selenium, fluorine, Chlorine, Bromium etc

Sulphur dissolves readily in iron at the temperatures usual in iron smelting and effects the steel qualities, therefore slags are designed to (hopefully) remove sulphur during the iron making stages. Sulphur increases the flux (slagging) volumes and hence the coke rates (energy) required.

Sulphur converts to sulphur dioxide gases in the flue emissions from a smelter and can cause severe corrosion and environmental emissions problems for the steel companies. Sulphur can be present in the ores usually in the form of sulphides which are often common in iron ores. High sulphur levels in iron ore are likely not be bought by the main steel makers in Europe, N America or Japan (<0.03% commonly required) or potentially they will incur penalties. The inclusion of even small amounts of these elements can preclude sales or incur penalties.

14.4.1.5 Phosphorus, Copper, Nickel, Vanadium, Chromium, Manganese, Molybdenum, Tungsten, etc

Phosphorous and many of the other elements mentioned above occur at some levels in most iron ores and tend to find themselves passing through the iron making stage to end up in steelmaking. Here the effects are many, but typically in the finished steel, elements like phosphorus contribute to the hardening of the steel, but at too high levels can make the steels brittle, especially in the quenching and tempering processes. This results in reduced ductility and impact toughness. These elements are better removed before reaching the iron making stage as high levels could preclude sales or again incur penalties. Minerals to watch for are apatite, copper sulphides, wolfram and oxides of the elements above.

14.4.1.6 Manganese, Vanadium and titanium

Titanium is particularly difficult during the iron making processes Magnesium, Vanadium and Titanium are difficult to remove during iron making stages, however they all may work as alloying elements in the final steel product, but steelmakers like to control these at that stage and do not like high levels in the HM feed.

Operationally titanium is used to help protect the hearth of the furnace with a layer of "refractory" titanium carbonitrides, but this has to be applied in controlled manner. Use of ferrous burdens with high levels of titanium can cause hearths to get blocked.

14.4.1.7 Zinc, lead, mercury and alkali metals

Metals such as zinc, lead, silver and alkali metals evaporate during smelting of the ore but may re-oxidise or condense to form carbonates with the coke in the furnace. The oxides or carbonates deposit on the cold furnace walls causing "scaffolding" or scabs on the furnace walls resulting in disturbance of the gas flow and hence decreasing productivity. The scabs will often slip and disturb the iron making process, again increasing instability and decreasing productivity. Furthermore these "slips" damage furnace lining and can hasten the requirement for an expensive furnace reline.

Furthermore some of these elements pass through into the gas cleaning systems where again they can condense out in cooling towers etc. Recirculation and energy recuperation systems also see a build-up of recirculating loads of these elements, which will cause increased fuel levels and extreme cases catastrophic failure of cooling towers. Removal is both difficult and expensive. So many operators now look closely at these elements in their feed.

The lead and zinc often occur as sulphides, and the alkalis usually in the form of micas or feldspars.

14.4.1.8 Radioactive and RE elements

These radioactive elements can concentrate in the iron making and steelmaking processes, particularly into the refractory linings.

14.4.2 Requirements by steelmakers purchasing iron ore

Primarily the iron-makers are looking for is consistency in the chemical analysis and the physical properties of the iron ores they buy. This is to ensure that HM processes can operate at the optimum planned for many years. Typically the blast furnace buys ore in contracts spanning several years to ensure that they receive the same quality of iron ore for a sustained period. They will generally buy iron ore from several different producers and then blend them themselves to get the optimum properties of the iron ore for their operations.

14.5 Iron Ore Markets

The global iron ore business is dominated by a limited number of large suppliers from Australia and Brazil, along with the main domestic Chinese suppliers, predominantly in to their own market; with smaller roles played by suppliers in Canada, Africa, FSU and Sweden.

In the early to mid-2000's, China's domestic iron ore supply was unable to maintain supply quantity and quality. In the last 10 years steel production in China has risen nearly 800%, and the import of seaborne iron ore has risen to over 850Mt/y. This provided significant increased demand for the global suppliers of iron ores but they were ill prepared for this unprecedented increase in demand as a result of low investment programmes in place during the period of the 1970's to the early 2000's. Together with the huge price rise of basic iron ore products from around US$20/t to as high as US$190/t (2008 and 2012) investment into the iron ore sector increased and new more focused and nimble companies started to develop iron ore production during this period. As a result the big three suppliers of globally shipped iron ore (Rio Tinto, BHPB and Vale) have been challenged by a large number of new iron ore developments from Australia, Brazil, Canada and most notably W Africa.

New companies entering the global iron ore supply business included FMG in Australia, Anglo American in Africa and S America, MMX in Brazil, several smaller W African operations and some Canadian mines with support money from China and India. Additionally there are several new developments in Australia which are still reported to be coming into production during 2015. However with some of the existing large companies in Australia, such as FMG, clearly struggling to cope with Index 62%Fe prices around US$50/t, it seems unlikely that new projects can expect to start operations with anything close to balanced cashflow while they are ramping the projects up to full production (a period of time when operational efficiency is below optimum). Similarly some of the larger projects in Canada, S America and Africa are suffering similar fates.

In addition, the iron ore markets have changed significantly, with some steel companies, such as ArcelorMittal continuing to secure large iron ore assets in Canada and Africa in particular and to develop them at a slower pace than was originally conceived. ArcelorMittal, as an existing owner of many (largely captive iron ore mines) has good experience of mine development, and has done a great deal to make their operations more integrated and more commercially viable. However, many other steel company ventures into the iron ore mining sectors have been less successful, particularly those in Africa and Canada. Even new projects in Australia have been slow to develop to capacity and are now under severe pressure from the current low iron ore pricing with many operators struggling to get their costs below their selling price.

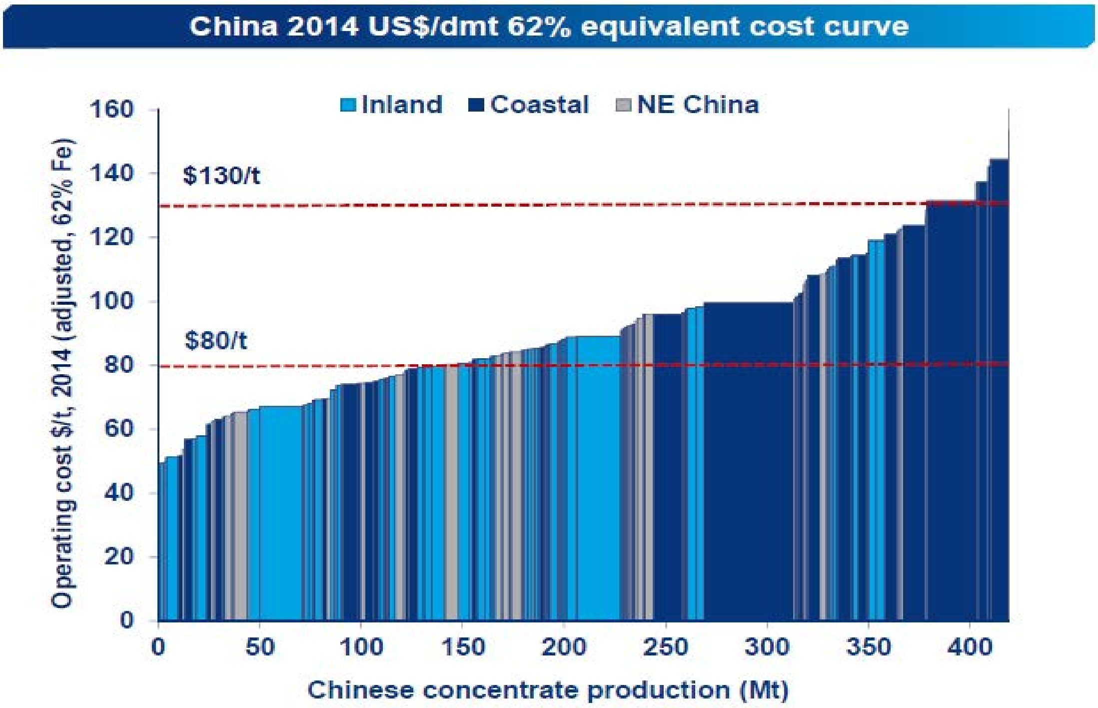
China's promised expansion into ownership of foreign mines has been a slow and painful process, with many of the investments being made at times of high cost. There has been destruction of investor capital in the past 2-3 years as the high prices paid for projects has not paid dividends. The ability of China to continue to invest in projects in which they have ownership is yet to be seen; but what is for sure, the investment and the pace of development of the projects has slowed.

The vast majority of the global market in iron ore is with iron ores in the range of 57%Fe to 63%Fe. The main exceptions are some of the South American products from the likes of Brazil, Venezuela and Chile, where predominantly concentrates products are 65-68%Fe. In contrast, the higher grade sinter fines from S America are trending to lower grades and are also being supplied at finer grain sizes. This is reducing the suitability of these ores for use in conventional sinter plants and is forcing the steel plant operators to look at new sintering techniques (such as pre-agglomeration) to overcome the deterioration in the sinter feed supply quality.

Brazil is the predominant ore supplier into Europe, largely due its geographic position, while imports of iron ore from Australia into Europe have virtually ceased. Unfortunately the largest market remains China, and as a result the current pricing mechanisms (i.e. net-backed calculations based on delivered product to China) are a major disadvantage to the S American suppliers. Consequently, Australia has something close to a US$10/t freight advantage into the largest iron ore market in the world.

Whilst Chinese domestic supply has increased significantly in the past 15 years, it has low iron grade in the ground and requires around 1.4bn tonnes of ore to be mined in order to achieve the equivalent of nearly 400Mt/y of equivalent 62%fe imported iron ore. Domestic ores are expensive to produce and costly to deliver to the largely coastal based steel works. Analysts reviewing the Chinese iron ore sector in China during 2015 have observed that the recent low iron ore prices have had the effect of removing around 130Mt of production (out of a total of around 360Mt (62% basis); but that around 75-80% of the Chinese mines are selling ore at a lower price that their operating costs CIF steelworks. In an article by Jamie Smyth in Sydney and Lucy Hornby in Beijing writing for the Financial Times 9/4/15, they report that the central Chinese recently offered a 40% reduction in the iron ore taxation in order to help support the Chinese iron ore producers. However it has to be said that this, in reality, is little help, only providing minimal support. Whilst local Chinese governments may be forcing some of their local non-profit making miners continue production, one wonders how much longer this can continue.

China has been behind the unprecedented increase in iron or prices during the last 12 years or so and although. some newcomers entered the market, thanks to the low levels of investment being committed by the global payers in the early 2000's, the big suppliers have now caught up and regained the upper hand in the supply of low cost iron ore. The result is that they have been able to regain control of the market supply and squeeze out the higher cost producers both globally and in China,.



Source: Wood Mackenzie – October 2014

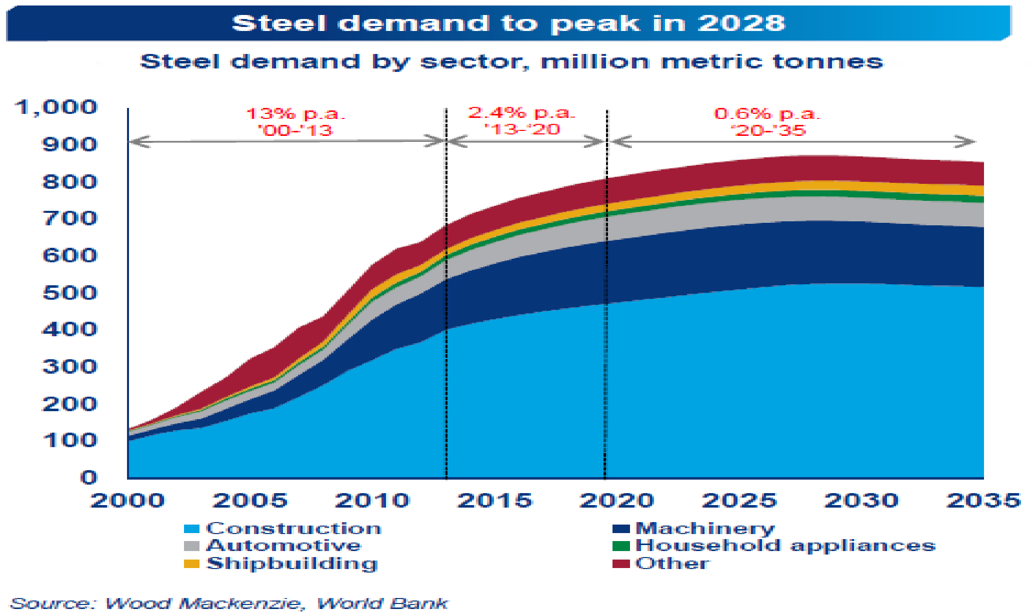
Unfortunately for the global trade in iron ore, this has coincided with the decline in the growth rate in China and doubts are emerging as to the sustainability of the high growth rate in China in the future.

Whilst a lot of analysts continue to perceive further declines in demand and expansions in supply, the reality is that the iron ore market supply and demand balance is cyclical as it always has been The current low prices for iron ore will only continue to be low if there is a continued flow of investment into mine development; feeding the oversupply. However, investment has also declined significantly since 2013 partly because mining companies have been under severe pressure to return dividends to their shareholders and partly because other investors have withdrawn as a result of losing considerable amounts of money as the commodity prices have fallen. The result has been that many operating and developing mines have either had to close or seen their share prices fall, often by 90% or more.

This will inevitably lead to a reduction in the surplus of ores and ultimately to a shortfall in supply in the coming years and inevitably to rising ore prices.

14.6 Iron ore demand forecast

Iron ore demand is driven by demand for crude steel, which in turn is based on population and geopolitical change. Virtually all the additional global crude steel production in the last 10 years has been through the BF route; taking the percentage of global steel produced via the BF route from around 65% to over 80% (crude steel production via the EAF route only occurs in mature markets where scrap is readily recycled. – unlikely to start really in china until 2030 and start to be a major influence around 2050). Whilst China is going through a phase of reduced growth rate in the steel industry (noticeable that 2013/14 is the first time steel industry growth fell to less than the overall economic growth in China); there is still growth for the forthcoming years. See Fig www below



The continued growth in demand is likely to be supported by the continued (and improved) growth of the BRIC nations, as well as rapid development in many smaller nations in Asia, Africa and S America.

But propping up the long term view are some basic economic expectations for the further growth and development of many of the developing nations, including China still. Rod Beddows in his book Steel 2050 sets out some compelling reasons for the continued growth in demand for crude steel from the development of emerging nations and the continued consolidation of many other established developed nations. Using UN statistics and predictions, Beddows states that the steel demand will continue as the major (along with energy supply) foundation (and sign of) industrial and economic development globally. Current global crude steel production is around 1.65bnt/y in 2013 (World Steel Association – Steel Statistical Yearbook 2014) and is expected to grow to around 3bt/y by 2025 and just under 4bt/y in 2050. This assumes modest growth in emerging nations, a forecast global GDP per head of around US$25,000 and an average steel consumption per year of around 330kg per person (compared with 230kg per person currently).

This increase in crude steel demand of around 2.3bnt in the next 35 years will require approximately 3.7bnt of additional iron ore supply. This in turn equates to an average increase annually of over 100mt/y of new supply. This required massive investment required by the iron ore mining industry in the coming years (historically this would require an investment of around US$200/at of production – to pay for the mine and the infrastructure development (Beddows P72). At current levels of investment in the iron ore mining industry it will not be many years before the supply will not be able to match the demand.

14.7 NIO Product Quality and Value-in-Use

14.7.1 NIO Product Quality

As can be seen in the table below, NIO product is made up of two components, the magnetite concentrate, 93Wt%, and the hematite concentrate of 7wt%. It is intended that NIO will produce a single product only.

Products	size	Composition of concentrate	SG g/cm³	Total Iron	Phosphorous	Silica	Alumina	Lime	Sulphur	Magnesium oxide	Vanadium oxide	Titanium oxide		Sodium	Potasium
Blötberget	mm	Wt%		Fet	P	SiO_2	Al_2O_3	CaO	S	MgO	V_2O_5	TiO_2	FeO	Na_2O	K_2O
Blötberget feed composite (measured)		100.0		36.2	0.230	37.70	4.66	1.24	<0.05	3.37	0.03	0.19			
Magnetite Concentrate	<0.15	**93.0**	2.4	69.8	0.051	1.96	0.31	0.22	0.01	0.33	0.08	0.07		0.01	0.052
Hem concentrate	<0.15	**7.0**	2.5	65.9	0.059	3.80	0.46	0.39	0.01	0.31	0.02	0.86		0.04	0.038
Composite concentrate	<0.15	100.00	2.5	69.5	0.052	2.09	0.32	0.23	0.01	0.33	0.07	0.12	87.2	0.01	0.051

Below in the second table are some of the physical characteristics of the concentrate. This product is described as pellets fines, which are coarser than traditional pellet feed concentrates (generally around D_{80} close to 40-70 microns) and would require further grinding before palletisation.

	Hematite conc	Magnetite conc
Solids content (% w/w)	100	93
Solids density (g/cm³)	4.95	4.92
Particle size		
- D_{80} (µm)	94.9	101
- D_{50} (µm)	68.5	53.4
- D_{20} (µm)	48.5	25.6

The NIO product is, however, finer than those materials generally considered as sinter fines, (D_{80}>1mm) and as such is likely to be only useable in limited proportions in feed to sinter plants (in Europe typically 10-16%)

14.7.2 Value-in-Use

The value in use is very customer dependent, though there are some basic rules:

- The higher the Fe content and the lower the gangue minerals – the lower the fuel rate at the iron making furnaces (BF or DRI) (lower fuel normally means lower carbon emissions)
- Lower gangue content (especially acid gangue) requires less addition of fluxes
- Magnetite has a significant cost benefit (i.e. reduced energy requirements) in pelletising (particularly) and sintering processes

- Low moisture content provides lower costs of transportation and handling
- High Fe content minimises logistics costs per Fe unit for buyer

Below are some more detailed estimates of potential benefits from using high Fe content magnetite rich ores in pellet making and the benefits to the steelmakers through use of pellets or sinter.

14.7.3 Pelletisers

Pelletising companies have to receive the fines or concentrates which then require grinding to a specified feed size (measured as Blaine) suitable for pelletising those ores. The finer grained the feed delivered the less grinding is required. Many ores used by pelletisers are sinter fines or fines concentrates and often require drying and grinding. Here, NIO concentrate from Blötberget is a pellet fines product, (sub 150 microns) and therefore does need some further grinding. Pelletising with hematite feed requires additional energy compared with magnetite. Usually some anthracite is added to the pellet blend, further adding to handling costs, and additional gas/oil is also required.

Assuming a magnetite feed of around 69% Fe, where the Fe content is 100% magnetite compared with, say, some of the 68%Fe content South American hematite concentrates, it can be expected that there is around 2/3 saving in the energy costs. Assuming costs of anthracite at 0.20/MJ and natural gas at around 0.35/MJ and equivalent energy savings of alone could be around 1-1.2/t of pellet product. For Blötberget, where the magnetite content is expected to make-up around 90% of the Fe units and where the NIO product could potentially make up 25% of the feed it could provide the pellet plant operator with net savings of around 0.3/t pellet – a not insubstantial amount.

Potentially additional savings are to be had depending upon the grinding and drying requirements and the design of the pellet plant.

14.7.4 Iron makers

The Value in use (ViU) for the iron makers can be via the BF route, where the inclusion of the Blötberget concentrate in either the sinter or the pellets will provide savings to the HM production costs, and similarly the use of the NIO product in the pellets fed to the DRI/EAF processes.

14.7.4.1 Sintering

Calculations completed by Tata Steel Consulting, using their Heat and Mass Balance Model for BF operations and assuming a “typical” European blend and operation as their base case, with 763kg/thm sinter and high quality BF pellets at 311kg/thm (along with a proportion of lump – kept constant) in the feed to the BF.

By replacing 25% of the sinter feed with Blötberget concentrate raise the typical sinter Fe content by around 2%*. It is calculated that the coke rate is reduced by around 1.11kg/thm and the productivity of the furnace would increase by around 2.6t per day of HM and importantly there was a reduction in the proportion of pellets necessary – saving around 23kg pellets/thm.

Assuming a pellet differential cost of around US$35/t compared with fine/concentrates, translates into around a saving of about US$0.8/thm. Similarly assuming coke costs of US$200/t coke, then the saving of 1.11kg/thm of coke equates to a further reduction of US$0.22/thm. For a 3Mt/y BF operation this could save US$3M/y. Add to this an increase in productivity potential of 2.6t/d of HM, with a value of around US$300/thm could give close to additional US$0.3M/y. Total benefit in use of close to US$3.3M.

Further benefits can be expected from reduced carbon emissions and the potential reduction in the carbon credits produced. Based on basic calculations it could reasonably be expected that the potential savings in carbon dioxide credits, assuming EUR12.00/t (US$15/t$CO_2$) for carbon credits and a potential saving of around 3kg CO_2/thm, then this would equate to a small saving of close to US$0.05/thm. In certain circumstances in Europe, in a 3Mt/y steel plant operation this could equate to further savings of over US$150,000.

Note: This is theoretical and takes no account of any effects that using the NIO concentrate has on the sintering operations or the physical characteristics of the sinter product.

14.7.4.2 Pelletising – B

Use of the Blötberget concentrate for pellets is estimated to provide similar savings of between US$1-1.2/thm, even more pronounced when the energy savings costs of the pelletising operation are taken into account.

14.7.4.3 Pelletising – DRI

The use of 100% of Blötberget concentrate, replacing hematite concentrates, to make DRI pellets could be expected could be expected to give net savings of close to US$3/t DR product. So the use of say 25% Blötberget feed to the pellet could provide around US$0.80/t savings in the EAF, which added to the savings in pelletising should give over US$1/thm savings.

Once again it should be emphasised that these estimates are assuming “typical” customer requirements, but it will vary considerably from customer to customer depending upon their specific plant and operational flowsheet, as well as other ferrous materials used in their processes.

14.8 The Market for Nordic Iron Ore AB Pellet Fines products

14.8.1 Introduction

Extensive metallurgical testing by NIO has demonstrated that a simple mineral processing flowsheet and inexpensive operations can produce a high quality pellet fines product. (See Section 14.7.1).

This largely magnetite pellet fines concentrate with a high iron content is well suited to find a market either in Europe the MENA-region. In Europe, there pellet plants are limited to Tata Steel in Holland, who could, in theory take most of the Phase 1 production for their pelletising operations. The NIO product is predominantly magnetite and would bring significant benefits to the pelletising process (energy savings) compared with using hematite iron ores.

However, other than Tata Steel, in Europe the most likely use is to utilise the NIO product and mix it in with ”traditional” European sinter feed blends; using the superior iron content to boost help improve fuel rates (i.e. reduce cash costs) and potentially boost productivity. Whilst most European sinter makers would only use limited amounts of in the case of finding a market in the MENA region, the concentrate would most likely find its main use as feed for the production of DRI pellets, or potentially BF pellets in the case of Turkey, who have been discussing the possibility of building a plant for some years.

The development of DRI making in the MENA-region has been one of the few areas outside of China that has continued at some pace and has meant the expansion of DR pellet making in the region specifically is thus of importance.

In the following, we will discuss the pellet market generally and specifically refer to the application of DRI pellet feed under the heading of the “DRI market

14.8.1.1 Competition to NIO

The main suppliers of high grade (magnetite) ores are based in Europe, but the majority of them make pellets from their concentrates (LKAB, LABGOK and Ferroexpo). Some other suppliers in the FIS countries are still captive mines for local steel works, both for sinter and pellet plant supplies.

The vast majority of magnetite projects in Australia are meeting problems with the current prices, as indeed are those high grade concentrate producers in North America (predominantly hematite rich). Mines development and operating mines are closing or partially closing while trying to drastically cut their operating costs. The only new high grade mines coming on stream are Brazilian, and some of those will struggle to maintain supply at the current iron ore prices.

NIO has a real opportunity to supply high quality ore into Europe with limited competition, particularly with the magnetite content. It should be noted that before the demise of Northland Resources, their 69%Fe product was in high demand and now Europe and the ME does not have that ore available. There is a void which could be filled by NIO.

In terms of the competitive position of NIO compared with other developments or even operating mines, there are several reasons why NIO projects are extremely competitive compared with many of the larger projects:-

- NIO has modern high standard national rail network to transport the ore directly to an operating port – apart from a small terminal, no CAPEX for NIO

- The Port of Oxelosund is operating and was designed to export dry bulk cargos
- The logistics solution from mine to ship exists and is competitively priced.
- Gain in lower CAPEX and development time from exploiting the sunken assets of previous mining operations
- NIO product and low moisture content ensure one of the most cost effective transportation of Fe units internationally
- Cash OPEX/Fe unit amongst the lowest globally

14.9 Iron ore price forecast

14.9.1 Introduction

If one thing has been proved in recent years it is that predicting iron ore prices into the future is very difficult, and to tie it into a time frame is almost impossible. What we do know however is that, like oil pricing, it shows a very cyclical nature. In fact two of the few commodities that consistently track each other pretty well are iron ore and oil.

To set the background the iron ore price has fallen (as many predicted) into an over-supply situation, and this has been exacerbated by a fall in the growth rate of the Chinese steel industry and general sentiment. The price is so low now that the only way is up in the medium to long term. As investments in new developments grind to a halt the supply glut will run its course and we will end up with another tight market and prices will be moving upward again. The big questions are when and how far will the prices move up? This is subjective of course and should also assume that the few big suppliers, such as Rio, BHPB and Vale have a much better control on the supply than they did in 2003-2012 and having taken advantage of the situation to help force out higher cost competitors, they won't be so quick in letting competition back into market place.

At the end of the day, it is a pretty simple scenario of supply and demand balance. The questions to be answered are how much iron ore capacity will be removed due to the price squeeze and what level of global production will satisfy the demand. Demand continues to grow, albeit at a slower rate, and will there be another super-cycle like that with the explosion of Chinese growth? Probably not, is the answer; but what is for sure the current levels of Indexed 62%Fe ore at less than US$60/t CIF China will not only close many operating mines, but also stop many mine developments underway.

14.9.2 Iron Ore Pricing Mechanisms

As was discussed elsewhere in Section 14, the global financial crisis and the unprecedented growth in demand for imported iron ore to China led to a change in the traditional annual "benchmarking" price mechanism to a much shorter term spot or Quarter-term pricing mechanisms. Most people in the business believed that these new Index based pricing mechanisms were here to stay, however the current collapse of the iron prices has prompted speculation that some forms of benchmarking could be making a comeback. What is for sure that the small and medium sector of miners and developers, along with many of the steel companies do not like this volatility in the iron ore pricing and margins being severely squeezed in many of these companies. This is dealt with more in Section 14.4.

The dominant methodology for calculating iron ore prices has primarily come from the methods promoted by Vale, which uses the Indices generated by TSI, Platts and MB and a netback calculation to obtain an FOB cost. Clearly an important part of any such equation relies upon the freight rates, and as such provide an advantage to those sellers close to the main market where the indices are based; i.e. China. So in descending order of advantage:

- Chinese domestic (generally)
- Australia
- Africa
- Brazil
- N America

Freight rates have been volatile for the past few years, see Section 14.8.3 below. Currently freight rates are at close to historic lows. This has been brought about because of too many builds, reduced growth rates in China and elsewhere and general sentiment in the market place. We have the situation where Cape vessels are charging below their operational costs per day, which is only sustainable for a while. The thought that there will be a sustained global high level of growth has been shelved for a while and so freight rates are likely to remain low for some time (years). The volatility is shown as the prices for TC4 rates have varied between US$42K/d down to US$4k/d, insufficient to pay the crew.

14.9.2.1 Netback Calculations

This is the favoured methodology of calculating the FOB price achievable for a given supplier, and as described elsewhere is based on using:

- Price index for product in China
- Premiums or grade differentials
- Freight rates to China

This methodology used by Vale, for example, on their 64%Fe sinter fines calculates the FOB value FOB Brazilian Port, based on say a Indices for a 62%Fe of US$80/t CIF. If the premium price is US$2/t/per 1%Fe above 62%Fe, then there would be an additional 2 x 2 dollars to be added. The Net CIF price is US$84/t delivered Chinese port.

Assuming that the freight from Brazilian is US$20/t then the netback price Brazilian port is US$64/t. Then to deliver to Europe the freight to Europe would be added, for example US$10/t to Rotterdam would effectively be US$74/t delivered to Rotterdam.

There are specific discounts and term negotiations held between supplier and buyer. Most buyers like to buy their ore FOB port of discharge and deliver their own ore to the point of use in the most cost effective manner.

Unfortunately there are no indices specifically designed for Europe and so the Europeans particularly believe that the prices they are paying are distorted (upwards) by Chinese influence compared with the way that the benchmarking system worked for them.

14.9.2.2 Calculations for NIO Product.

NIO's product is niche, in that there is very limited supply of similar products and as such there is little traded into China due to lack of availability. However a recently Platts have started to address the issue with an index based on 65%Fe products, with a premium for each 1% Fe above this level. So for example on February 3rd 2015 the Platts IODEX for 62%FeFe mid-range price was US$62.5; whilst the price for the 65%Fe to Quingdao was US$70.2, representing a nominal premium of around US$2.6/1%Fe above 62%Fe. Subsequent differentials were US$1.6/1%Fe/t. Highlighting that NIO needs to look at calculating its net back prices based on the IODEX 65%Fe indices.

The published net-back calculations were as follows:-

FOB netback per route / basis TSI 62% Fe, 3.5% Al fines			
Origin	**Vessel Type**	**FOB ($/dmt)**	**Change**
W.Australia	Capesize	57.90	0.70
India	Supramax	54.30	0.15
Brazil	Capesize	51.65	0.70

The above table show that the prevailing freight rates were around US$4.6/t US$8.2/t and US$10.85/t from Australia, India and Brazil, respectively, to China.

Adjustments will be need to made for the moisture levels in any agreements.

Indexed adjustments – Non Fe: The expansion of the indices has now expanded to include the two key gangue elements SiO2 and Al2O3. Quoted for February 2nd,

Per 1% differentials (Range 60-63.5% Fe), $/dmt			
	Within Min-Max	**$/dmt**	**Change**
Per 1% Fe	60-63.5% Fe	1.25	0.00
Per 1% Alumina	1-2.5% Al_2O_3	0.40	0.00
Per 1% Silica	4.5-6.5% SiO_2	1.15	0.00
	6.5-9% SiO_2	1.40	0.00

Whilst the indices only apply to a limited range it is indicative of the levels of premium that can be sought when negotiating offtake agreements. The level of SiO_2 and Al_2O_3 in the Blötberget concentrate is around 2 and 0.3% respectively. The table above suggests that NIO would be looking for substantial premium over the other indices for lower grade materials. The basis of the quoted 65%Fe index assumes around 1% Al_2O_3 in the product.

An example for calculating the net back price for NIO could look like – IODEX 65%Fe at US$70, plus premiums of say US$2/1%Fe is an additional US$8. Total US$78, less current freight rates from Oxelosund to China at around US$17.50/t is a net-backed price of US$60.5/t FOB.

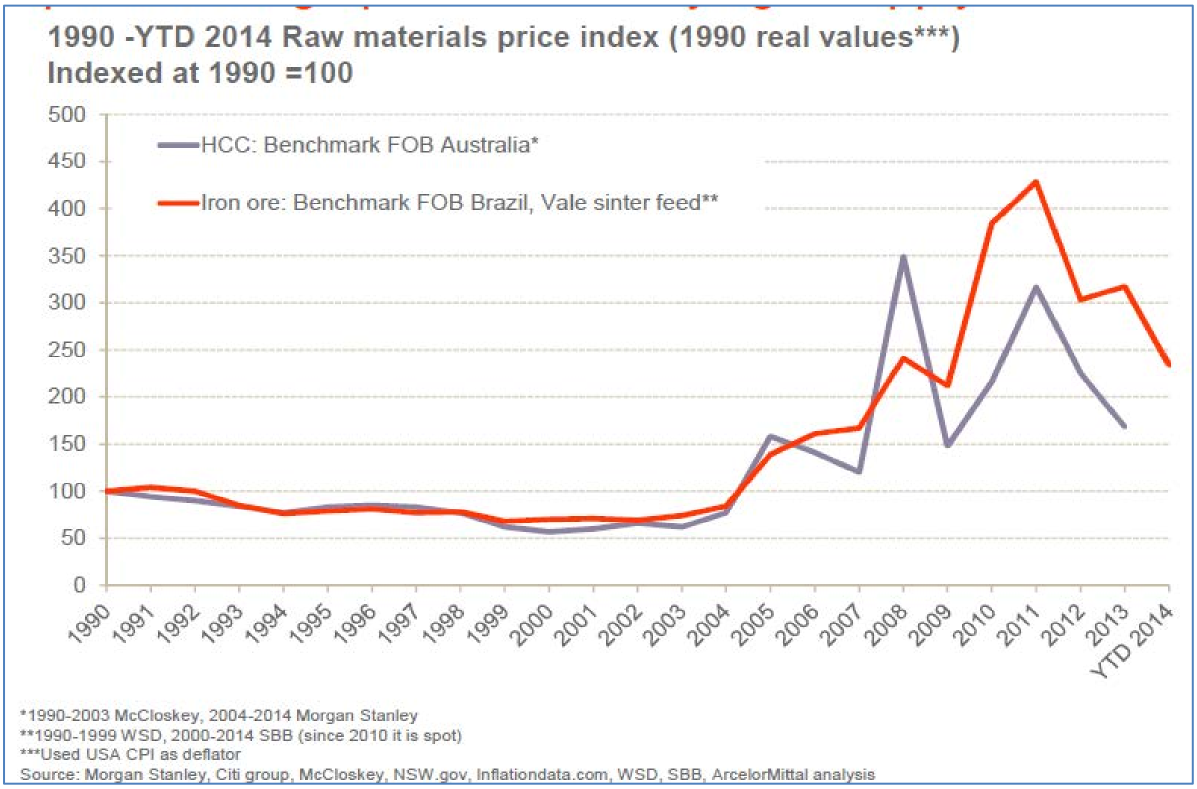
NIO would then need to look at the competition supplying the local European market, particularly those in C & E Europe, Germany, France, UK, and Holland. Here NIO could have a competitive advantage in terms of the shipping or delivered costs, compared with those from Brazil into say Germany, via Rotterdam. NIO should consider looking at CIF deliveries by direct shipping by barges/Handies in the Baltic region or by Barge/rail combinations.

14.9.3 Recent Trends in Iron Prices

Please refer to the two diagrams below, the first of which shows the indexed iron ore or price comparisons for ores from Australia and Brazil, FOB basis. This graph shows two facts:

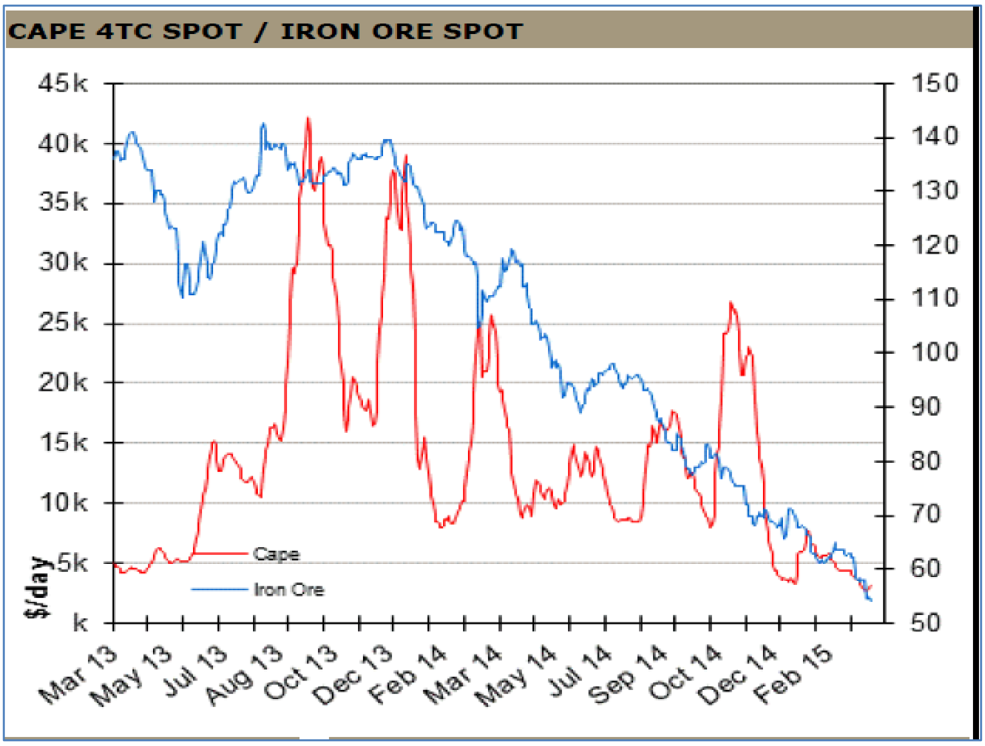
- The rise of the iron ore prices in a period of time of unprecedented fast growth in China and the subsequent decline in prices since 2011 due to the reduced rate of growth in China and the increased iron ore supply
- That since the financial crisis and the introduction of the Spot basis Indices and Quarterly pricing, Brazilian ore has proved to be higher costs FOB, primarily reflecting the higher Fe content in the ore. It also shows that Australia has some significant advantages when taking the shipping to China and Asia into account due to its geographic proximity. This explains why the significant growth in iron ore production in recent years has be in Australia.

The second diagram Figure 14-3shows the volatility in the shipping costs and the rapid decline of the iron ore price in the last two years or so.



Source – ArcelorMittal Presentation – 2014

Figure 14-2: Raw Materials Price Index



Source: SSY Futures - December 2014

Figure 14-3: Spot Prices for Iron Ore – 2 Years to date

The graph above also shows how rapid the decline in the iron ore price has been, from around US$140/t to US$55/t since the beginning of 2014 to the current time March 2015.

14.9.4 Forward Iron Ore Pricing

During the next few years the consensus view in the industry, from analysts and those who work in iron ore production, is that there will continue to be a surplus in supply or iron ores to the market place. What is less clear is the size of the fall-out from the supply chain. There is little doubt high cost capacity is being lost. But there is also little doubt that many iron ore mines are significantly reducing their cash OPEX costs and looking to maintain a profit margin; and many also appear to be successful. However, the effects in the industry that we see now and generally reflecting the responses to earlier index price lows for 62%Fe at US$80/t, the more recent falls to US$60/t and more recently in March 2015 to around US$50/t have yet to reflected in the fallout of capacity. The effects of these price cuts could take the rest of 2015 to become obvious.

Despite increased economic activity in the US and some parts of Europe, they are not expected to reflect in significant increases in steel production and hence iron ore demand. The real focus on the iron ore demand is still China and at the moment there is great uncertainty as to what the growth rate will be sustainable in China; and more immediately what the leaders of China will require from the steel industry in terms of reducing carbon and other emissions. It could lead to the closure of some steelworks and there is no guarantee they will be replaced.

The forecast for the iron ore pricing here is therefore based on a consensus of analysts' views experience from working in the international iron ore supply and steel industries. The forecast tries to reflect the current and near-term situation; but more importantly looks at pricing for iron ore that is more sustainable, to keep up with global economic growth, than the current levels. The long term pricing of the ores is based more on the longer term demand/supply patterns, the ores that will be available in the medium term (in terms of quality), technological developments within the iron ore and steel industries. The long term view (i.e. life of mine and beyond) is discussed in more detail earlier in the Introduction to the section 14.9, where continued iron ore demand and increasing development costs will ensure a higher sustainable price that experienced recently.

Most prices are based on the TSI (Steel Index) pricing mechanism for 62%Fe Fe basis; as it is probably the primarily used indices in the business. The basis will be in terms of real US Dollars based on March 2015 levels of exchange as the basis.

The basis of the price trends assumes the following:

- The price of iron ore has probably reached close to the base level
- Significant capacity will continue to be lost in the next 20 months, both in China and elsewhere. NIO believe that this could be as much as 300mt in the next 2 years
- Supply of low cost iron ore is now in the hands of a few large suppliers, primarily in Australia and Brazil, who look like they are back controlling a record 80% of the seaborne iron ore trade
- Prices will start to rebound when supply becomes tighter and it is apparent that the lack of investment into sustained mine development since 2013 will be insufficient to maintain a surplus. Because of the development cycle, this effect will be seen by 2019/20 and will be reflected in the iron ore pricing a year or two before that.
- Productivity improvements will only get so far in the next year or so. Further investment and technical developments will be required to make another significant step forward.
- Cash OPEX costs will start to rise again when the energy costs start to increase (has this started – March 2015?)
- US, Europe and Japan will see little change in the trends of the steel industry, having little impact on the global pricing trends anymore.
- BRIC nations have also underperformed in the recent past, and as a consequence expectations are now a bit lower (and more realistic) for their growth rates. Political influences play a significant role in these countries, as does the oil price. Things can change rapidly as we have seen in recent years.

- Long term global crude steel requirements suggests continued long term demand for iron ore (Beddows – Steel 2050), suggesting around an average additional requirement for 100Mt.y
- Growing demand for high grade iron ores for improved iron/steelmaking performance and increasing requirements for the palletisation of lower quality ores

14.9.5 Iron Prices 2015-2019

There is little doubt now that the iron ore supply will remain above the demand for a few years. Much depends on the changing fortunes of China and other developing nations and their subsequent demand for iron ore. Currently there is a great deal of uncertainty and this is being reflected in the price for iron ore. What needs to be remembered is that there is significant growth potential in China; and that with the cyclical nature of the economic fortunes, both globally and specifically in China, one can expect these periods of correction and change to the next stages. Once the current economic issues are addressed the economies of China and other nations will pick up and enter into the next upward cycle for a few years.

14.9.6 Iron Ore Prices 2020 – 2030

Iron ore availability (lower cost) in China will continue to diminish and new capacity to compete with global iron ore suppliers will start to impact on the Chinese supply-demand balance. The lack of investments in the iron ore sector since 2013 will also start to impact significantly and there is a real likelihood that there will be a tightening of the overall iron ore supply post 2018. The majority of analysts expect that the long term trend for the iron ore price will not be to the levels achieved during the past 7 years, but are more likely to fall between US$85-US$125.t for 62%Fe indexed ore to China. That is quite a range of prices, but the median is the most likely scenario. Those who are rather more pessimistic are suggesting that the prices will remain below US$80; this is extremely unlikely and would not be sustainable for the next iron ore developments. The cyclical nature of the prices and trends suggest that this scenario is unlikely.

15 PROJECT EXECUTION PLAN

Appendix J provides the outline development and operational schedule for the Blötberget mine project covering the mine establishment period, production, operation and final mine closure and reclamation.

The schedule covers the two years of pre-production (Year-2, and Year-1,) up until the anticipated start of commercial production and then 15 years of ore production, with final mine closure in Year 16.

The schedule was based upon information provided in the production and development schedules and initial construction programmes produced by the contributors to this report.

16 PRELIMINARY QUALITATIVE RISK ASSESSMENT

16.1 Basis of Register

The register of risks and opportunities potentially affecting the success and economics of the project is derived from a review of the uncertainties referred to in the respective main chapters of this report. Qualitative judgments have been made to categorise the likelihood of the risk arising, and the level of potential impact upon the success and economics of the mine.

The diagram below sets out the relative and qualitative risk matrix. The relative risk rank scores placed in the diagram are merely a means of comparing different combinations of likelihood and impact and do not represent numerical risk estimates as such.

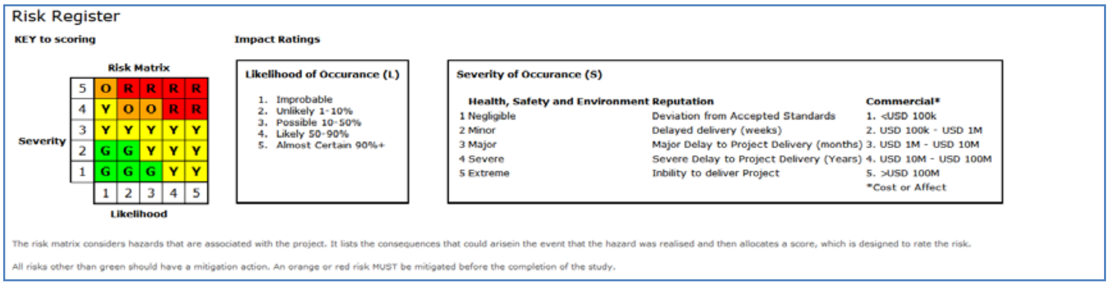


Figure 16-1: Risk Matrix

16.1.1 General level of assessed risk

The risk register provided in Appendix K indicates a range of adjudged risk probability [likelihood] and impact depending upon the content provided in the various report chapters.

Because the mine has previously been operated there are large amounts of data on geology, mining, dewatering and other aspects against which judgments on future activities can be calibrated. This allows the level of uncertainty to be relatively well constrained given that this report is part of the development of a feasibility study.

On the whole the levels of risk in terms of the combinations of likelihood and impact are adjudged to lie within the yellow and green fields of the diagram. This is a function of the level of detail covered in the report, which has been possible by virtue of in-depth assessment of copious historical data, the level of detail provided in the cost estimate.

17 INTERPRETATION & CONCLUSIONS

17.1 Geology

The Hugget and Kalvgruvan/ Flygruvan zones had previously been mined down from near-surface to the 200 m and 240 m levels respectively.

Since acquisition of the property, NIO has undertaken two drilling programmes.

A drilling programme was undertaken by NIO during the summer and winter of 2012 and was completed in November 2012. This 16 hole programme included drilling to confirm the quality of historical drilling data, as well as infill and step-out drilling. NIO completed 16 drillholes totalling 7,430 m of drilling.

The 2014 drilling programme was designed to investigate the area between Flygruvan/Kalvgruvan and Hugget (formally known as "the Wedge" or Betsta area) and to infill the intermediate depth extension of Hugget, in order to improve the confidence of the geological model. 13 drillholes, totalling 7,093 m, were drilled.

'The Wedge' was successfully explored during the 2014 drilling programme and, as a result, Kalvgruvan and Hugget/Flygruvan have now been shown to be continuous zones of mineralisation

Mine maps and historical drilling data have been collected from various sources and digitised, where possible. Drill core from historical exploration drilling in the Blötberget project area has been recovered, re-logged and re-analysed.

In the resource development programme of 2012 and 2014 NIO completed industry standard QA/QC programs to ensure the data is reliable and suitable for resource estimation. The drill density of the resource is adequate for the purpose and is reflected in the JORC compliant resource category classifications of Measured, Indicated and Inferred Mineral Resource.

DMT has relied heavily upon the information provided by NIO, however DMT has, wherever possible, independently verified data provided during the site visits.

DMT was able to overlay licence information on the Mineral Resource estimate area to confirm that the deposit is within NIO's license. DMT has not undertaken a legal review of the licences and assumes that all the required licences are in place.

The geology of the deposit is fairly well understood and DMT has constructed a wireframe geological model for the Blötberget deposit based upon a combination of logged lithologies and analytical and SATMAGAN magnetite results. This has allowed the splitting of the deposit into geological domains comprising, magnetite-rich material of KALV and hematite-rich material of HUGFLY and SAND.

DMT has undertaken a statistical study of the data, which demonstrates adequate splitting of the data into single iron population domains, and undertaken a geostatistical study to investigate the grade continuity and to provide grade estimation parameters for Ordinary Kriging.

A Surpac block model using all the available geological and sample analytical test data has defined an iron ore resource. At this stage of the investigation most of the mineral resources of Blötberget have been classified into the Measured and Indicated categories.

As a result of the site visits, data base verification and validation and the geological and model generated, DMT has estimated the total Measured and Indicated Resources for the Blötberget Project as 47.8 Mt at a grade of 41.5 % Fe (Total) and 0.5 % P at preliminary COG of 25 % Fe. Of the total estimated contained Fe, the magnetite-hematite ratio is estimated at 62:38.

17.2 Mining

The orebodies included in the Blötberget project have been extensively worked in the past down to level 280. A variety of mining methods were used historically including shrinkage stoping, open stoping and sublevel caving.

Although SLC appears to have been successfully applied in the past, NIO has chosen to adopt SLOS as the mining method for this Study. DMT agrees with this approach since it offers more certainty of success at this stage, given the strong hanging wall and the risk that it may not cave successfully. In addition, the use of SLOS allows a lower dilution factor to be adopted, an important element since headgrade to the mill significantly affects iron recovery from the Blötberget ores.

At the time of preparing the LOMP development and production schedule, the new blockmodel was not available. Therefore, this Study is based on assumed in-situ grades although the tonnages are based upon volumes calculated from the latest wireframe.

The mine design is based on utilising as much of the existing underground infrastructure as possible. So for example, the existing shafts (after dewatering) will be used for main intake and exhaust airways. However, main access to the mine and mineral handling of ROM ore and waste rock will be achieved via a new surface access decline driven adjacent to Industrial Area #1 at Skeppmora. The existing BS shaft will not be used for hoisting under the current scheme. All rock from underground will be conveyed to surface via a conveyor system connected to the main access decline, after having been crushed.

The mine design has been advanced to a preliminary level and in DMT's opinion is at a level of definition normal for a scoping study. However, sufficient optioneering studies have been undertaken to allow the mine design to be advanced to feasibility study level, without further option studies being needed. Nevertheless, further work is needed to raise the confidence levels and engineering definition to support a full feasibility study.

Some costs estimates have been developed to a level of accuracy appropriate to a scoping study, while others such as the cost of the rail terminal, are at a more developed level of accuracy. Some costs (notably development and mobile equipment) are based on budget quotations received by NIO from contractors and OEMs, thereby providing some increased confidence in these figures.

17.3 Process

The proposed mineral processing plant described in Section 8 has been designed primarily on the results from the small-scale (1 t/h) pilot plant or pilot plant at GTK which operated for 24 hours over a period of several days processing approximately 22 tonnes of a single, composite bulk sample extracted from surface outcrops of the Blötberget orebodies.

The design is further supported by the results of bench-scale laboratory testwork conducted on drill cores recovered from deeper parts of the orebodies in a programme of metallurgical variability testwork although at the time the process flowsheet and the standard metallurgical test for variability included wet high intensity magnetic separation to recover hematite rather than gravity concentration.

As a result, the preliminary flowsheet used for this study is considered to incorporate sufficient flexibility to ensure that the processing plant would be able to treat ore of wide varying quality.

The flowsheet and equipment has been selected in order that the process plant can successfully recover concentrates from ore of variable iron content and varying ratios of iron present as magnetite and hematite.

Equipment has been reviewed and selected on the basis of utilising current "state of the art" items without taking unnecessary risks with unproven plant or process technology.

17.4 Infrastructure

The main infrastructure needed to support a mining and processing operation will be located at the two Skeppmora industrial sites, which have already been permitted.

The basis of this study is to place the majority of the infrastructure on Industrial Site #1, where the process plant and railway terminal and loadout facility will be located. The portal for the main access decline will also be located on this site area. This avoids transporting of raw ore or concentrate between site areas, reducing the surface environmental impact.

All of the key infrastructure elements necessary for the operation of the Blötberget mine have been included in the Study. With the exception of the railway terminal, the engineering development of these facilities is at a conceptual/preliminary level of definition, as are the associated cost estimates, and will need to be advanced considerably at the next stage of study in order to support a full feasibility study.

The detailing of the tailings storage facility has been advanced to a preliminary level of engineering appropriate to this Study. Reasonable assumptions have been made as regards the available of suitable construction material for the dams and the ground conditions that exist at the dam foundations. However, ground investigation is necessary to support a feasibility level design.

17.5 Environment and Permitting

The Study has been carried paying due consideration to the conditions included in the Environmental Permit already obtained by NIO.

17.6 Capital and Operating Costs

The overall capital cost estimate for the option of producing pellet feed concentrate has been compiled by DMT, with significant contributions from the other organisations responsible for the respective sections of this report.

The estimate has been the subject of iterative discussions between NIO and other contributors, and between NIO and DMT.

DMT considers that the overall accuracy level of the cost estimation presented in this Report is approximately +/-25%. However it is noted that some costs such as the capital and operating costs for the logistics system (railway and port) have been estimated to a higher level of accuracy than this.

The estimated cost to bring the mine into production including design, construct, install and commission of the Project operation and facilities described in this report is US$181.2 million. This amount includes the direct field costs of executing the Project, plus shipping and logistics in Sweden.

Sustaining capital expenditure of the life-of-mine is estimated to be US$70.8 million.

Contingency included amounts to US$23.6 M

Average life-of-mine Total Operating costs are estimated to be US$19.64/tonne of ore, equivalent to US$43.96/dry metric tonne of concentrate at a yield of 45%

17.7 Project Risks and Opportunities

17.7.1 Project Risks

The register of risks and opportunities potentially affecting the success and economics of the project has been derived from a review of the uncertainties referred to in the respective main chapters of this report. Qualitative judgments have been made to categorise the likelihood of the risk arising, and the level of potential impact upon the success and economics of the mine. This is presented in Section 16.

On the whole the levels of risk in terms of the combinations of likelihood and impact are adjudged to lie within a low to medium level of risk. This is largely thanks to the fact that the mine has previously been operated meaning that there are large amounts of data on geology, mining, dewatering and other aspects against which judgments on future activities can be calibrated. This allows the level of uncertainty to be relatively well constrained given that this report is part of the development of a feasibility study.

17.7.2 Project Opportunities

The Blötberget project is Phase 1 of NIO's overall strategy for the development of the former Ludvika mines. Phase 2 of the Project is intended to lead to the development of a much larger operation that incorporates the existing Håksberg mine and the undeveloped and relatively recently investigated Väsman deposit. The development of these later projects will benefit from the establishment of infrastructure provided through Phase 1, the Blötberget Project.

The location of the Project in a major former mining region area of Sweden, within easy access of logistical infrastructure is a major advantage for the project. Support services such as established equipment suppliers and mining service providers are readily accessible in the local region and in Sweden generally. Costs of product logistics are well defined with early agreements at an advanced stage with some of the key parties to this aspect of the Project. Both of these serve to significantly de-risk the project overall and offer the potential for project upside.

With the exception of logistics solution, the level of engineering definition of the project described in this Report is at relatively early stage. This suggests that there should opportunities to optimise the engineering and reduce some of the capital costs. This particularly applies to the process plant and industrial area buildings and structures.

Clearly at the time of undertaking this Study, a major risk to the project economics (which have not been assessed in this Study) is the future price for iron ore concentrates, which is hugely uncertain at the current time. In the medium to long-term, it seems reasonable to assume that there will be a recovery of prices from the currently depressed price level back towards US$100/tonne or above. Given the likely timeframe for the commencement of commercial production from the Blötberget Project, this offers an opportunity to enhance the project economics quite significantly.

However, contrary to the above risk, the low iron ore prices have reduced the capital intensity required for such projects and suppliers are now able to readily accept orders, supply equipment in a much more timely manner and do so at reduced costs. The increase in the value of the US$ has the potential to improve capital costs in non-dollar sourced plant and equipment and increase the value of the revenue SEK.

The current climate is potentially an opportune moment to oversee a "lower" cost development and potentially start production in a rising iron ore market; as suggested above; where the current levels of iron ore prices are unlikely to sustain the dividend payments and future necessary investment in developing iron ore resources, even for the large dominant suppliers.

18 RECOMMENDATIONS

18.1 Geology

18.1.1 Further Drilling

It is considered that there is only limited additional geological information that can be gained from further, expensive, surface drilling programmes. The bulk of the upper levels of the Blötberget deposit that provide part of the proposed mine plan are within the Measured Resource category.

However, surface drilling for rock mechanical/structural and or metallurgical information for detailed mine planning a part of the feasibility should be considered.

Definition and grade control drilling should commence as soon as there is access to the underground areas after dewatering. This close spaced drilling required to support the transfer of Measured Resources into (Proven and Probable) Reserves. The underground drilling should follow a similar drill approach to that used historically, with fan pattern of close spaced drilling into the mine blocks , typically at 35-45 m centres, with wider spaced (100 m) deeper down dip drilling to provide increased confidence in the Indicated area of the resources.

18.1.2 Further Studies

Additional hydro-geological investigations on existing drillholes should be undertaken, as DMT considers that insufficient data exists on the hydrological and hydrogeological conditions for underground mining.

Targeted mineralogical and metallurgical testing is suggested to better understand the ore origins and process response during mining operations.

The properties to the north east of the current project, such as Guldkannan and Sandell offer upside potential to the project. At this stage it may prove worthwhile to carry out an evaluation of the properties in readiness for an application for an extension of the mining concession/environmental permit area that would lead to exploitation of the minerals during the development phase.

18.1.3 Sampling

There was no use of check samples in the historic core re-assay (BGU), this should be addressed as a partial re-run with standards inserted.

The blank samples assayed to date have indicated between 1 % and 2 % Fe. Prior to further analysis being undertaken, the preparation of suitable blanks for insertion into future sample streams should be addressed by NIO.

Check standards have slightly (but consistently) undervalued the results, this should also be corrected ahead of the next phase of core sampling, which will likely be from underground drill locations.

NIO should continue to source historical data and drill core for the purposes of re-assaying, re-logging and integration into the current database.

18.2 Mining

The mine plan and design needs to be further refined based on the new blockmodel to allow definition of stope outlines per level in order to optimise the projected ore recovery and to allow Ore Reserves for the Blötberget deposit to be estimated.

In particular the mine schedule should look at the occurrence of the Magnetite: hematite, levels of P, Ti, V and possible other trace elements throughout the mine programme; in order to better define the ROM characteristics during the mine schedule.

A survey of the existing shafts that will be used to dewater the mine and provide the long-term ventilation intake and exhaust airways must be carried out to remove this risk to the currently adopted mine and ventilation design. It is essential that this is done to support the feasibility study.

18.3 Process

The pilot plant testwork carried out to date has been on bulk samples taken at surface. For the future refinement of the flowsheet and plant design it would be advantageous to continue to use samples from deeper parts of the orebodies for the purpose of refining the knowledge of the metallurgical response throughout the resource.

18.3.1 Confirmation of the process flowsheet

In order to refine and optimise the flowsheet consideration should be given to the following issues:

- Reason for the failure of wet high intensity magnetic separation to produce a high grade hematite concentrate in order that this process may be included /excluded.
- Recovery of fine hematite by spiral concentrators or wet high intensity magnetic separation because the recovery (yield) of hematite is relatively low compared to magnetite and crucially important to the financial viability of the project.
- More detailed review of the occurrence and potential for the removal of some of the trace elements, such as P, Ti and V. Additional lab scale testing and mineralogy should be carried out to get a better estimate of the product(s) to be made during the LoM.
- Locked-cycle tests of HPGR to establish probable superiority over rod mills and SAG milling.
- Further SAG milling tests depending upon outcome of HPGR testwork.

- Pot-milling tests with respect to regrinding of rougher LIMS concentrates and rougher spiral concentrates by vertical stirred media mills to determine the improved grindability after HPGR, if applicable.
- Selection of the most suitable cleaner spiral concentrator or wet high intensity magnetic separator for recovery of fine hematite concentrate.

18.4 Infrastructure

Further engineering and quotations from vendors is required to refine the engineering and the costs estimates of the underground infrastructure.

Basic engineering of buildings and structures will be required to refine the cost estimates for the surface infrastructure.

Detailed ground and topographical surveys are required to develop the cut and fill volumes of the respective industrial sites.

Specific site investigation is required at the Skeppmora industrial sites and at the locations of the North and South tailings storage facilities to confirm foundation requirements and the availability of dam construction material.

18.5 Environment and Permitting

It should be confirmed that all details of the currently envisaged Phase 1 project as presented in this study are within the current planning and environmental permits.

Any variances of the project as presently envisaged to that approved will need to be amended or discussed with the appropriate authorities during the next stage of the Project.

As mentioned above, there are potential advantages to be gained for the overall economics of the project by the extension of the mining concession and environmental permit to areas that include Guldkannan and Fremundberget orebodies. DMT considers that these should be reviewed.

18.6 Capital and Operating Costs

Outline specifications should be developed for the major process plant and equipment items to allow quotations to be sought from vendors and OEMs. This could be completed during an update of this ITR and would require limited resources and costs; but on the upside could potentially improve economics for the project, with likely reductions in CAPEX and OPEX.

This would be a further stage in advancing the project engineering to a sufficient level to allow more refined cost estimates to be developed for the DFS for which typically, engineering completion would be expected to be 15% to 20% complete.

18.7 Overall

In the current difficult economic climate where project financing of mining projects is challenging, NIO's strategy for the development of the Blötberget project as Phase 1 in the re-opening of the former Ludvika mines seems to be appropriate. This will result in a much less capital intensive and potentially financeable Project.

In addition, the reduced activity in the global mining industry offers the possibility to reduce the capital investment needed since suppliers are now prepared to accept orders and supply equipment in a much more timely manner and to do so at reduced costs. The increase in the value of the US$ has the further potential to improve capital costs in non-dollar sourced plant and equipment and increase the value of the revenue in SEK.

The current climate could also be considered an opportune moment to oversee a "lower" cost development with the potential to start production in a rising iron ore market.

The location of the Project in a major former mining region area of Sweden, within easy access of logistical infrastructure is a major advantage for the project. Costs of product logistics are already well defined with agreements at an advanced stage of discussion. Both of these serve to significantly de-risk the project overall and offer the potential for project upside.

Despite the current downturn in iron ore prices at the moment, NIO has an opportunity to advance studies for building a low cost operating mine capable of competing with the other global suppliers.

DMT considers that the Project as described and configured in this Report should be advanced to the next stage of Project development i.e. completion of a DFS.

In advancing the Project to full feasibility a number of steps should be taken towards enhancing this Interim Report. The Life of Mine Plan should be updated to fully reflect the outcome of the April 2015 MRE and the finalised blockmodel with resulting updating of the plant recoveries to reflect the revised headgrades.

The further steps towards feasibility should consist of continued improvement to the engineering design of the mine, the mining schedule, mining infrastructure requirements, process plant and buildings design. Improved specification of the plant and equipment will allow NIO to specify the capital items in much more detail and obtain competitive quotations that provide greater confidence in the overall economics of the project.

Improved engineering details will reduce the overall projects costs as greater confidence allows costs to be defined nearer +-15%.

Appendix A

DRAWING LIST

Appendix B

SURFACE DRAWINGS

Appendix C
UNDERGROUND DRAWINGS

Appendix D

BASIS OF DESIGN

Appendix E

MINERAL RESOURCE ESTIMATE REPORT

Appendix F
GEOTECHNICAL REPORTS

Appendix G

MATERIALS HANDLING OPTION STUDY

Appendix H

TAILINGS STORAGE FACILITIES REPORT

Appendix I

COST ESTIMATE DETAIL

Appendix J

PROJECT EXECUTION SCHEDULE

Appendix K

RISK REGISTER