TECHNICAL REPORT ON THE HANNUKAINEN IRON-COPPER-GOLD PROJECT, KOLARI DISTRICT, FINLAND, JANUARY 2014

Prepared For

NORTHLAND MINES OY

Report Prepared by

srk consulting

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UK4985

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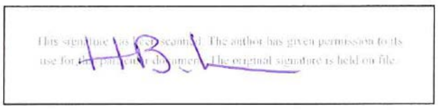
Qualified Person:	 Howard Baker (Principal Consultant, Mining Geology)
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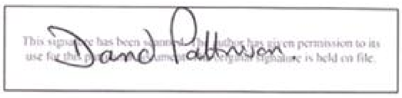
Qualified Person:	 Dr David Pattinson (Corporate Consultant, Mineral Processing)

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EXECUTIVE SUMMARY
TECHNICAL REPORT ON THE HANNUKAINEN IRON-COPPER-GOLD PROJECT, KOLARI DISTRICT, FINLAND, JANUARY 2014

1 INTRODUCTION

This technical report has been prepared for Northland Mines OY ("Northland") by SRK Consulting ("UK") Ltd ("SRK") in connection with the publication by Northland of the Hannukainen Feasibility Study ("HFS"). Northland is a public limited liability company, domiciled in the Grand Duchy of Luxembourg, and governed by the Luxembourg law of August 10, 1915 on commercial companies, as amended. Northland Resources S.A. is the parent company, which in turn holds operating subsidiary companies in Sweden ("Northland Resources A.B") and Finland ("Northland Mines OY"). The common shares of the Company were listed on the Toronto Stock Exchange ("TSX") up to 15 March 2013, and are now primary listed on the Oslo Stock Exchange ("Oslo Børs").

Northland is principally an iron exploration and production company with properties in northern Sweden (the Kaunisvaara project comprising the Sahavaara and Tapuli deposits, currently in production) and Finland (Hannukainen comprising the Hannukainen and Kuervitikko deposits). This report describes the results of a review of the HFS which comprises the exploitation of the Hannukainen iron-copper-gold ("IOCG") project ("the Project") in the Kolari District, Finland ("the Mineral Asset").

1.1 Location/Licence

The Project is located in the Kolari District of Northern Finland, centred on X: 249837, Y: 7498171 using the Finnish National Coordinate System, Projection zone 2 ("FIN-KKJ2 – Kartastokoordinaattijärjestelmä").

1.2 Geology and Mineral Resources

Hannukainen is composed of several separate bodies of magnetite skarn mineralisation relating to a contact zone between a Precambrian intrusive igneous unit and Precambrian metavolcanic and metasedimentary units. SRK created a geological model based on a statistical review of the validated drillhole data. Seven separate iron ("Fe") mineralisation zones, and five separate copper ("Cu") mineralisation zones were defined by SRK and were based on cut-offs determined by Fe Total, sulphur ("S"), Cu and magnetic susceptibility data.

The data used in the estimation and the associated quality control quality assurance ("QAQC") data was given by Northland to SRK. It is the opinion of SRK that the results of the blanks, certified standards, the results of the laboratory duplicates and inter-laboratory duplicates show that a reasonable level of confidence can be attributed to the recent drill samples used in the Mineral Resource Estimate.

A 2 m composite file was used in a geostatistical study (“Variography and Quantitative Kriging Neighbourhood Analysis - QKNA”) that enabled Ordinary Kriging (“OK”) to be used as the interpolation method. The interpolation used an elliptical search following the predominant dip and dip direction of the geological domains via dynamic anisotropy. The results of the variography and the QKNA were utilised to determine the most appropriate search parameters.

The interpolated block model was validated through visual checks, a comparison of the mean composite and block grades and through the generation of section validation slices. SRK is confident that the interpolated grades are a reasonable reflection of the available sample data.

The Mineral Resource Statement generated by SRK is split into an open pit and underground Mineral Resource. The open pit Mineral Resource has been determined by constraining all material through an open pit optimisation excersise and the underground Mineral Resource has been determined through the application of an eqauivalent cut-off grade.

The open pit Mineral Resource has been restricted to all material falling within a Whittle Shell representing metal prices of USD1.50/dmtu for Fe, USD3.35/lb for Cu and USD1,375/oz for gold (“Au”), and above an Fe equivalent of 13.3%. The underground Mineral Resource has been restricted to all material above an Fe equivalent cut-off grade of 35.6%.The underground cut-off grade calculation used the same metal prices as the Whittle optimisation where:

Fe equivalent=(FE/100+(CU_PPM/1000000*82.8833)+(AU_PPB/1000*0.13237512))*100

Processing costs, mining costs, slope angles, mining recoveries and revenue assumptions were also used to demonstrate economic viability in the Whittle optimisation and underground cut-off grade calculation. The open pit and underground material defined in the Mineral Resource Statement represents the material which SRK considers has reasonable prospect for eventual economic extraction. Table ES 1 shows the resulting Mineral Resource Statement for Hannukainen.

The statement has been classified in accordance with the Definitions and Guidelines of the CIM, by the QP, Howard Baker (MAusIMM(CP)), who is an independent consultant with no relationship to a Northland employee and has never been employed by Northland. It has an effective date of 24 October 2012 and incorporates all drilling undertaken to date.

The quantity and grade of reported inferred Mineral Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these inferred Mineral Resources as an indicated or measured Mineral Resource; and it is uncertain if further exploration will result in upgrading them to an indicated or measured Mineral Resource category.

Table ES 1: Hannukainen Mineral Resource Statement

Deposit	Resource Category	Tonnes (Mt)	%Fe Total	%Cu	Au(g/t)	%S
Open Pit						
Hannukainen	Measured	120	32.25	0.18	0.083	2.47
	Indicated	3	24.80	0.19	0.064	2.15
	Meas+Ind	**123**	**32.08**	**0.18**	**0.082**	**2.46**
	Inferred	0.9	27.16	0.19	0.024	2.86
Kuervitikko	Measured	34	23.22	0.19	0.217	2.49
	Indicated	3	23.36	0.15	0.174	1.99
	Meas+Ind	**36**	**23.23**	**0.19**	**0.213**	**2.45**
	Inferred	0.1	19.37	0.15	0.129	2.48
TOTAL	Measured	154	30.28	0.18	0.112	2.47
	Indicated	6	24.09	0.17	0.118	2.07
	Meas+Ind	**159**	**30.06**	**0.18**	**0.112**	**2.46**
	Inferred	1.0	26.24	0.19	0.036	2.82
Underground						
Deposit	**Resource Category**	**Tonnes (Mt)**	**%Fe Total**	**%Cu**	**Au(g/t)**	**%S**
Hannukainen	Measured	14	32.13	0.18	0.150	2.4
	Indicated	8	32.44	0.16	0.072	1.9
	Meas+Ind	**22**	**32.24**	**0.17**	**0.123**	**2.2**
	Inferred	61	32.33	0.15	0.044	2.3
Kuervitikko	Measured	3	17.79	0.19	0.140	2.8
	Indicated	3	20.27	0.17	0.169	2.7
	Meas+Ind	**6**	**19.15**	**0.18**	**0.156**	**2.7**
	Inferred	1	23.21	0.15	0.203	2.3
TOTAL	Measured	17	29.88	0.18	0.149	2.4
	Indicated	11	28.82	0.17	0.101	2.2
	Meas+Ind	**28**	**29.46**	**0.18**	**0.130**	**2.3**
	Inferred	62	32.14	0.15	0.047	2.3
Combined Open Pit and Underground (Total Resource)						
Deposit	**Resource Category**	**Tonnes (Mt)**	**%Fe Total**	**%Cu**	**Au(g/t)**	**%S**
Hannukainen	Measured	154	32.24	0.18	0.090	2.5
	Indicated	6	30.37	0.17	0.070	2.0
	Meas+Ind	**159**	**32.17**	**0.18**	**0.089**	**2.4**
	Inferred	61	32.25	0.15	0.044	2.3
Kuervitikko	Measured	36	22.82	0.19	0.210	2.5
	Indicated	6	21.69	0.16	0.172	2.4
	Meas+Ind	**42**	**22.66**	**0.19**	**0.205**	**2.5**
	Inferred	1	22.87	0.15	0.196	2.3
TOTAL	Measured	171	30.44	0.18	0.113	2.5
	Indicated	17	25.88	0.17	0.122	2.2
	Meas+Ind	**187**	**30.04**	**0.18**	**0.114**	**2.4**
	Inferred	63	32.05	0.15	0.047	2.3

(1) The effective date of the Mineral Resource is 24 October 2012

(2) The open pit Mineral Resource Estimate for the Hannukainen deposit was constrained within grade based solids and within a Lerchs-Grossman pit shell defined by the following assumptions: selling price of USD1.50/dmtu for iron, USD3.35/lb for copper and USD1,375/oz for gold; slope angles of 45º (Hannukainen South and Central), 47º (Hannukainen North) and 48º (Kuervitikko); a base case mining cost of USD1.78/t and an incremental cost will be applied to reflect the haulage at various depths - the incremental cost above and below the reference level will be USD0.02/t/block height, where the block height is 5m; onsite process operating costs of USD6.78/t ore feed; transport costs for iron concentrate of USD 19.22/t and copper concentrate of USD23.24/t; G&A costs of USD1.33/t ore feed;

royalty of 0.15%; copper selling cost of USD0.27/lb.
(3) The underground Mineral Resource Estimate for the Hannukainen deposit was reported above an Fe-equivalent cu-off grade of 35.6% for everything beneath the Whittle shell. The Fe equivalent cut-off calculation is defined by the assumptions above, but with an underground mining cost of USD14.4/t.
(4) Mineral Resources for the Hannukainen deposit has been classified according to the "CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines (December 2005) by Howard Baker (MAusIMM(CP)) an independent Qualified Person as defined by CIM.

The Project has a combined Measured and Indicated open pit Mineral Resource of 159 Mt grading 30.06% Fe Total, 0.18% Cu, and 0.112 g/t Au. Of this, 154 Mt grading 30.28% Fe Total, 0.18% Cu, and 0.112 g/t Au is in the Measured category, and 6 Mt grading 24.09% Fe Total, 0.17% Cu, and 0.118 g/t Au is in the Indicated category. In addition, 1 Mt grading 26.24% Fe Total, 0.19% Cu and 0.036 g/t Au is in the Inferred category.

The Project also has a combined Measured and Indicated underground Mineral Resource of 28 Mt grading 29.46% Fe Total, 0.18% Cu, and 0.130 g/t Au. Of this, 17 Mt grading 29.88% Fe Total, 0.18% Cu, and 0.149 g/t Au is in the Measured category, and 11 Mt grading 28.82% Fe total, 0.17% Cu, and 0.101 g/t Au is in the Indicated category. In addition, 62 Mt grading 32.14% Fe Total, 0.15% Cu, and 0.047 g/t Au is in the Inferred category.

In total, the Project has a joint underground and open pit combined Measured and Indicated Mineral Resource of 187 Mt grading 30.04% Fe Total, 0.18% Cu and 0.114 g/t Au. In addition, there is a total of 63 Mt grading 32.05% Fe Total, 0.15% Cu and 0.047 g/t Au in the Inferred category.

The Mineral Resource Estimate has not been affected by any known environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

1.3 Reserve Estimation and Mine Design

A mining block model has been developed from the Mineral Resource block model for mine planning and design purposes. The mining block model has been developed by regularising the Mineral Resource block model based on the Standard Mining Unit ("SMU").

The use of a diluted regularised block model has the following advantages for the mine planning process:

- dilution in the model is applied locally and does not rely on global factors;
- facilitates the optimiser to distinguish between ore and waste in marginally economic regions of the deposit;
- allows for metallurgical regressions to be coded into the model on a diluted basis;
- simplifies the mine planning and design process; and
- allows the relationship between equipment size and block size to be assessed.

The following mining model has been used for the optimisation, mine design and production planning studies:

- a 6.25 x 6.25 x 5 m sized regularised mining model has been selected for the Project; and

- the mining recovery and dilution for the regularised block model are 97.7% and 6.5%, respectively.

The objective of the pit optimisation has been to select the optimal pit shell to take forward to detailed pit design. A sensitivity analysis has been conducted to identify the key components that affect the value of the Project.

On the basis of a 17 year mine life minimum, the 0.89 revenue factor pit shell (relating to an iron ore price of USD 1.25/dmtu) was selected as the basis for the pit designs. The pit shell contains 117.6 Mt of ore and 416.0 Mt of waste at a variable cut off grade ("CoG") depending on the processing recovery.

The following conclusions can be drawn from the pit optimisation results:

- the pit optimisation is relatively insensitive to changes in Cu concentrate transport and selling costs, Au selling price, and slope angles; and
- the pit optimisation is most sensitive the Fe selling price followed by the Fe processing recovery.

SRK recommends that the following work be undertaken following this study:

- verify the metallurgical recovery parameters for Cu and Au, which have remained unchanged since March 2011; and
- undertake another round of pit optimisation using the updated costs and input parameters from the HFS findings to check for any significant changes.

The final and cutback designs have been completed to verify the engineering feasibility of the optimised pit shells. The objectives of the design work are to:

- provide practical engineered pits and cutbacks for mining;
- incorporate geotechnical design parameters;
- provide a design which can be used as a basis for geotechnical assessment;
- honour equipment operating limitations;
- honour the interface between the cutbacks; and
- provide a basis for detailed mine scheduling.

The resulting tonnages from the engineered pit design are 114.8 Mt of ore with 446.8 Mt of waste (Table ES 2). The engineered pit designs resulted in a 2.9 Mt reduction in ore and 31.3 Mt increase in waste.

Table ES 2: Mineral Reserves

	Quantity Mt	Fe %	Cu %	Au g/t	S %
Hannukainen					
Proven	91.8	32.2	0.186	0.088	2.4
Probable	0.8	32.6	0.148	0.060	2.4
Kuervitikko					
Proven	21.9	23.6	0.183	0.216	2.5
Probable	0.3	23.8	0.177	0.194	2.5
Total					
Proven	113.7	30.5	0.185	0.112	2.4
Probable	1.1	30.0	0.157	0.100	2.4
TOTAL	**114.8**	**30.5**	**0.185**	**0.112**	**2.4**

SRK has reported a Proven Mineral Reserve of 113.7 Mt grading 30.5% Fe, 0.185% Cu, 0.112 g/t Au and 2.4% S and a probable Mineral Reserve of 1.1 Mt grading 30.0% Fe, 0.157% Cu, 0.1 g/t Au and 2.4% S for a total of 114.8 Mt grading at 30.5% Fe, 0.185% Cu, 0.112 g/t Au and 2.4% S. Confidence in the modifying factors has resulted in classifying all Measured material as a Proven Ore Reserve and all Indicated material as a Probable Ore Reserve.

The Qualified Person with overall responsibility for the reporting of Mineral Reserves is Ms Colleen MacDougall, BEng, MAusIMM(CP), who is a Senior Consultant (Mining Engineering) with SRK. All work has been reviewed by Mr Rick Skelton, CEng, MSc (Mining), MIMMM, MSAIMM, who is an employee of SRK. Rick Skelton is a mining engineer with over 30 years' experience in the mining industry and has been involved in the review and reporting of Mineral Reserves on various iron ore properties in Europe, Africa and South America during the past five years.

The engineered pit designs:

- are the basis of the mineable inventory for the schedule;
- are used a basis for the hydrological and hydrogeological studies; and
- are used to develop the haulage network for the production schedule.

SRK recommends that the following work should be undertaken during the detailed engineering phase of the Project:

- The pits and cutbacks have been designed for 181 t class trucks, a re-evaluation of the mine schedule has determined that larger trucks (227 t) are more economical for this Project and these have been used for this HFS. Therefore, a re-design of the pits will be required to ensure the ramp width is suited to the 227 t class trucks; that said, the current design of 27 m ramp width is within the standard tolerance limits of 3 to 3.5 times the truck width should a 227 t truck be used.
- SRK does not believe this will have a significant impact on the mineable tonnages or the economics of the study.
- The footwall slopes should be laid back following the ore contact to decrease ore loss.

The waste dump outlines were based on mining, geology, environmental, hydrological, and infrastructure inputs.

The waste dump designs were designed based on the following criteria:

- contain sufficient capacity for the waste inventories within the engineered pit design;
- honour waste dump design parameters;
- provide three distinct sections for the different material types: overburden (“OVB”), potentially acid forming (“PAF”) and non-acid forming (“NAF”);
- ensure PAF material is contained within the groundwater catchment area;
- ensure waste dumps are located within mining lease boundary;
- locate as much NAF material to be within the groundwater catchment area as possible;
- minimise haulage distances;
- develop a basis for equipment destination scheduling for waste
- develop a basis for the road layouts and infrastructure design; and
- establish a basis for dump scheduling to demonstrate the dump development.

The results for the waste dump design are summarised below:

- three waste dumps have been designed: East, West and West Overburden;
- the waste dumps have been designed based on the waste inventory from the pit designs; and
- the dump capacity has been limited by the groundwater catchment area for the PAF material and to a lesser extent the NAF material.

SRK recommends that the following work should be undertaken during the detailed engineering phase of the Project:

- the waste dumps have been designed for 181 t class trucks, a re-evaluation of the mine schedule has determined that larger trucks (227 t) are more economical for this Project, therefore a re-design of the waste dumps will be required to ensure the ramp width is suited to the 227 t class trucks;
- evaluate the potential to store PAF and NAF material outside of the groundwater catchment area, which would decrease haulage cycle times; and
- the potential for in-pit dumping should be re-evaluated to decrease haulage distances.

Groundwater and surface water inflows to an open pit mining operation can create saturated conditions and standing water within the pit. The objectives of the mine dewatering are to:

- prevent loss of access to areas of the pit;
- reduce explosives failure or the need to use more expensive explosives due to wet blast holes;
- reduce equipment wear;
- prevent inefficient loading and hauling; and

- allow for safe working conditions.

The main conclusions and outcomes from the study are as follows:

- There are currently two pits located at the Project which will need to be dewatered before operation mining can commence and will be completed in approximately 6 months.
- Pre-operational pit dewatering requirements have been incorporated into the operational dewatering design such that equipment, including both pumps and pipe-work which can be utilised during both phases of the mine operation.
- The predicted transient state inflow from groundwater to the pit based on the numerical model was 30 to 300 m^3/h at Hannukainen and 110 to 320 m^3/h at Kuervitikko. These values represent flow throughout the lifetime of the mine and illustrate the increase in flow rate as the pits expand. It should be noted that these values are considered the most likely given the estimates of hydraulic properties available at the time of study but pit inflows could be higher or lower depending on the properties of the rock mass. This is highlighted as both a potential risk and opportunity.
- Surface water pumping requirements have been assessed on the basis of two key factors; firstly, average precipitation and the seasonal effects of snow-melt, and secondly, the impact of large rainfall events. The maximum monthly rainfall plus snowmelt volume occurs in May when a capacity of <410 m^3/h may be required at Hannukainen and <110 m^3/h at Kuervitikko (about four times that produced by precipitation within other months). Significant additional volumes of water can be produced from 24-hour rainfall events.
- These flow rates are illustrative of the transient state inflows that may be expected in the pit and form the basis on which a flexible dewatering scheme has been designed and cost estimated at a conceptual level.
- Pump requirements have been assessed and cost estimated based on the inflow results from the numerical modelling. Flexible systems have been outlined and example electrical sump and in-line booster pump options presented.
- An engineered surface water diversion system should be put in place comprising of bunds and ditches around the perimeter of the pit and will provide additional protection from surface water inflows.
- If properly managed, the predicted groundwater inflows are unlikely to cause significant operational problems. Surface water inflow (including direct precipitation) will be significant and will need careful management to ensure that the inflow volumes associated with the spring thaw do not affect mine operations.

SRK recommends that the following work should be undertaken during the detailed engineering stage of the Project:

- conduct a cost benefit analysis of various sump pumps;
- estimates of hydraulic properties of the area can be improved by long duration pumping tests, which will enable the predicted pit inflows to be better constrained;

- additional field testing could be undertaken in the exploration wells to identify zones of high permeability and map the variability of hydraulic conductivity, particularly at Kuervitikko where the pit inflow predictions are sensitive to the estimation of the Hannukainen Thrust Zone hydraulic conductivity;
- investigate the potential for advanced dewatering at Kuervitikko pit;
- additional groundwater monitoring during advanced dewatering to confirm the hydraulic properties of the fault zone and constrain the likelihood of connection between the pit and the Äkäsjoki River;
- during operations, a flexible pump dewatering system enables overall pump capacity to be varied according to demand by adding (or removing) pumps to the system if required;
- the mine operators should consider horizontal drain investigations into fault zones prior to push backs in order to identify zones of potentially high pit inflows;
- spring melt water peak may be higher than that predicted or coincide with a storm event, resulting in greater pumping time required, pumping costs and possible disruption to mining schedules; however, an engineered surface water diversion system will ensure that all ex-pit surface water runoff is prevented from entering the pit and therefore minimise this risk; and
- use of flexible pump system allows variation in pumping capability as required. This could allow prioritisation of north (or south) sump pumping at Hannukainen or Kuervitikko if required.

The mine production schedule for the Project has used the pit inventories from the pit designs. The objectives of the production schedule are to:

- develop a planning model suitable for modelling the development constraints of the Project;
- achieve annual quantity and quality targets;
- honour crusher capacity constraints;
- determine pre-stripping requirements;
- strip waste to ensure sufficient quantities of ore are available to maintain production targets; and
- develop a production schedule suitable for developing HFS mining capital and operating cost estimates.

The results of the mining schedule are shown in Figure ES 1 and are summarised below:

- a two year pre-stripping period (including 350 kt of ore) is scheduled to develop the initial stages down to large ore areas;
- the Hannukainen Central (“HC”) region is targeted in the early parts of the schedule to access high Fe grades, it is blended with Hannukainen South (“HS”) and Hannukainen North (“HN”) ore to lower the strip ratio in the early years;
- the Kuervitikko (“KU”) region is left until the end of the schedule due to the low Fe grades and is mined with the final HC stage to increase the Fe grades;

- Fe grades ramps up to approximately 32% to 34% for the first 10 years and then slowly decrease with the introduction of lower quality ore, especially when KU ore starts to be fed in 2029;
- S grades fluctuate between 2.0% to 2.8% depending on the ore source, with the lowest S grades in HC region;
- Cu and Au grades follow the same trend ranging from 0.11% to 0.26% and 0.03 to 0.27 g/t, respectively;
- the Fe concentrate ranges between 1.9 to 2.2 Mtpa for the initial 13 years and then drops as ore feed quality decreases;
- the Cu concentrate fluctuates with the Cu grade in the ore feed;
- the product recoveries follow the trends of the ore feed grades;
- a topsoil clearing schedule has been split into three occasions beginning in 2015 Q3, 2019 Q1 and 2020;
- contractors will be used for all material movement until 2016 Q3 and all topsoil clearing;
- 26 m^3 face shovels will mine waste material, a maximum of two units will be required, to be purchased in 2016 Q4 and 2018 Q1;
- a 19 m^3 front end loader (“FEL”) will supply ore feed and mine excess waste material which will be required from 2017 Q4;
- it has been assumed that all Run of Mine re-handle will be handled by a small FEL;
- a maximum of 12 haul trucks are required (year 2025);
- the haulage fleet follows the general trend of the total material movement profile, with an increase in trucks required towards the end of the schedule due to the deepening of the pit and the increasing height of the waste dumps; and
- mine labour requirements for the mine operations, mine technical services and mine maintenance groups are 155, 14, and 38 employees, respectively, for a total of 207 employees at maximum production.

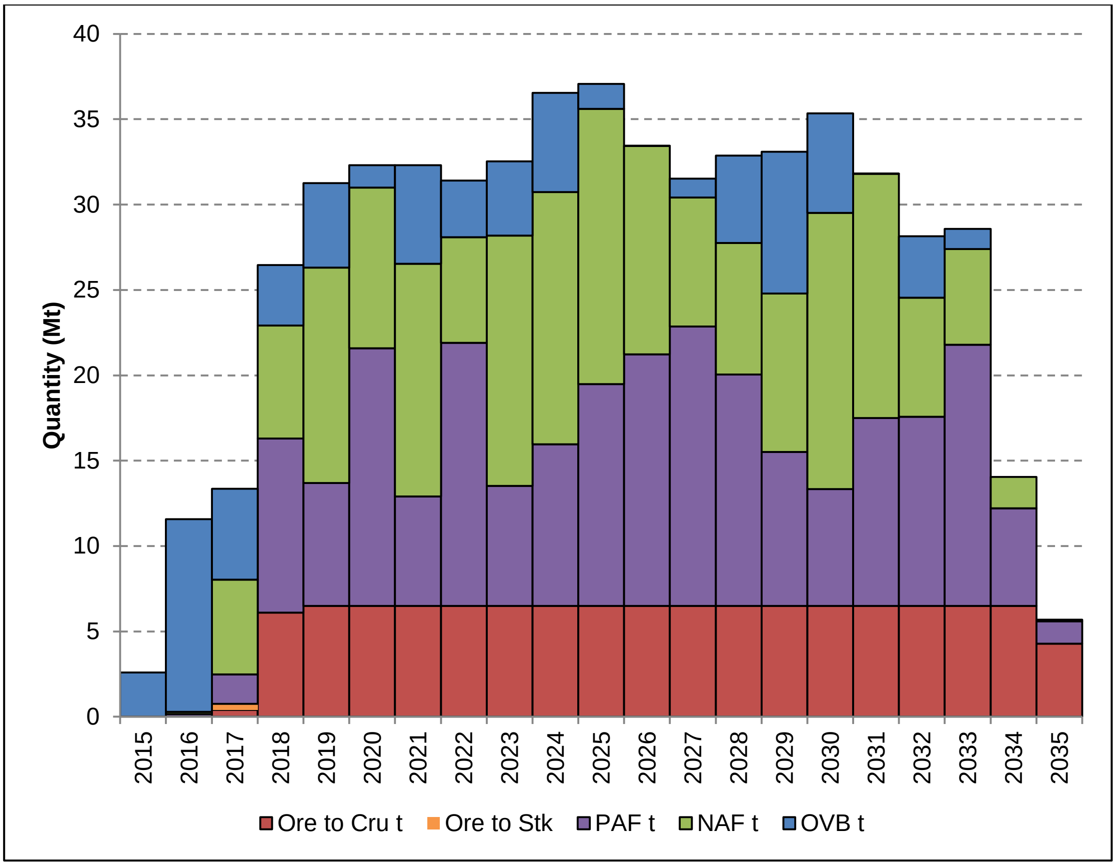


Figure ES 1: Total Material Movement Profile

SRK recommends that the following work should be undertaken for the production schedule during the detailed engineering phase of the Project:

- review the potential for ore stockpiling for extended periods to enable high quality ore feed to be targeted in the early years while stockpiling the lower grade ore.

A general operating strategy is required for the equipment selection and estimated productivities in order to justify the production schedule. The objectives of the operating strategy are summarised below:

- justification of a truck and shovel match;
- development of shovel loading productivities;
- development of equipment operating times; and
- development of a drill and blast strategy.

The findings of the operating strategy are summarised below:

- the mining loading fleet consists of 26 m^3 face shovels and a 19 m^3 FEL;
- the haulage fleet comprises 227 t capacity haul trucks;
- waste will be mined in full 10 m benches, while the ore is flitch mined in 5 m benches;
- a mine dispatch system is required to manage the fleet as well as recording and reporting purposes;
- overburden material will be mined by free digging, with all other material requiring drilling and blasting; and

- a mixed fleet of 229 mm and 172 mm blasthole drills will be used for production, trim and in-fill drilling.

SRK recommends that the following work be undertaken during the detailed engineering phase of the Project:

- it has been assumed that the FEL will mine all ore, due to the inflexibility of the face shovels; further investigation should be undertaken to determine whether the FEL can achieve selectivity to the SMU size;
- benchmark operating time estimates with production data for similar operations;
- benchmark operating delays due to weather with operations in similar climates;
- benchmark the estimated drill production rates with supplier information; and
- benchmark the blasting parameters with an explosives supplier.

The manufacturer and supply quotes have been compiled by Arundon Mining Solutions Oy ("Arundon") under direction from the Company and provided to SRK.

The results from the cost estimate are summarised below:

- an average life of mine ("LoM") mine operating cost of USD2.00/t moved;
- a total LoM mine operating cost of USD1,125m;
- a total LoM mine capital cost of USD198m, including sustaining capital costs;
- haulage costs, maintenance costs, diesel prices and labour rates are the largest contributors to the operating cost; and
- the cost estimate is most sensitive to changes in the haulage costs.

SRK recommends that the following work should be undertaken during the detailed engineering phase of the Project:

- evaluate the sensitivity of the currency exchange rates; and
- verify the availability of the truck and FEL tyres from the manufacturers; and
- benchmark truck and FEL tyre life with similar operations using the same tyre manufacturers.

The key risks for the Project are summarised below:

- the ore must be mined to a selectivity of the SMU block size to achieve the mining recovery and dilution predicted; if the SMU size cannot be achieved with the selected FEL the mining dilution and ore loss will increase;
- the pit optimisation is most sensitive to Fe selling price, a decrease would result in ore loss and a decrease in Project value;
- the pit optimisation and scheduling are based on preliminary metallurgical parameters from March 2011 for Cu and Au recovery, changes to these could affect the pit size and have an impact on the Cu concentrate product;
- there is potential for some ore loss and waste gain in the redesigns required for the pit and cutback designs due to the change in truck size;

- the cost estimate is most sensitive to changes in the haulage costs, increases to these costs will significantly impact the operating costs;
- equipment manufacturers have highlighted the potential decrease in availability of truck and FEL tyres in the coming years, costs may escalate should this occur;
- SRK has estimated three non-production days in the schedule due to adverse weather, a change in non-production days will impact productivity and operating costs;
- there is currently a global shortage of skilled workers available in the mining industry, a shortage of skilled workers will decrease productivity and possibly delay the start up of the Project; and
- the global mining industry is experiencing an increase in mine start up activity and lead times for mining equipment is increasing as build spots are being booked over 12 months in advance. This creates a risk for the start up of the operation if mining equipment cannot be purchased and delivered in a timely manner.

The key opportunities for the Project are summarised below:

- there is potential to lay back the footwall slopes to follow the ore contact directly to decrease ore loss and excess waste mining;
- evaluate the potential to store PAF and NAF material outside to the groundwater catchment area, which would decrease haulage cycle times;
- the potential for in-pit dumping should be re-evaluated to decrease haulage distances; and
- review the potential for ore stockpiling for extended periods to enable high quality ore feed to be targeted in the early years while stockpiling the lower grade ore; this would increase the value of the Project with high grade ore being produced earlier.

1.4 Mineral Processing/Plant Design

Metallurgical testwork has been performed in a number of campaigns using a total of 36 metallurgical samples including two pilot plant tests and has demonstrated that high quality iron and copper concentrate can be produced at the Project with the developed flowsheet.

A geometallurgical model has been established for the the Project. The model has been used in selecting samples for variability and metallurgical testing and for pit optimisation. Variability testing included more than 200 samples. The ore bodies have been classified in terms of metal recovery with respect to iron, copper and sulphur content. There is sufficient understanding of the main ore types to be able to define the expected metallurgy and to predict the metallurgical recoveries of iron and copper in to saleable magnetite and copper concentrates.

Sufficient testwork has been performed to define the metallurgy and the selected process flowsheet is considered to be appropriate for the different ore types at Hannukainen.

Comminution testing has been completed in laboratory and pilot scale testing. The results have been incorporated in to the grinding circuit design.

The selected flowsheet includes two stage grinding using a Semi Autogenous Grinding ("SAG") ball mill configuration with flash flotation of fast floating chalcopyrite. Hydrocyclone overflow, nominally 80% minus 90 microns passes to copper rougher flotation followed by pyrite flotation. The tailings from pyrite flotation are treated by Low Intensity Magnetic Seperation ("LIMS") to produce a combined magnetic concentrate containing magnetite and pyrrhotite. This is cleaned to remove the sulphur bearing pyrrhotite by flotation. Copper concentrate is cleaned in three stages of flotation incorporating a regrind circuit. Final tailings are stored as high and low sulphide tailings in two separate impoundment areas.

The magnetite and copper concentrates contain acceptable levels of impurities.

Pelletizing characteristics of the magnetite concentrate have been studied by COREM and show that the product has excellent properties.

Satisfactory recovery functions have been developed for iron in terms of % iron and % sulphur levels in the feed and for copper recovery in terms of copper in the feed. Recovery functions for gold require further work if considered significant in terms of the Project economics. These recovery functions represent the projected performance of the latest flowsheet.

Sufficient engineering has been performed to establish the plant capital cost.

The process operating costs assumed for the HFS were estimated from first principles and SRK considers the underlying assumptions and overall costs to be reasonable.

The implementation schedule and the plant ramp up time for the Project are considered realistic. The potential effect of weather windows on the overall schedule should be reassessed if the Project start date changes significantly.

1.5 Concentrate Transport

Concentrate transport for the Project will be achieved using two principal methods depending upon the concentrate type being shipped.

Approximately 66,500 tpa of Cu-Au concentrate will be produced at the Project and this will be transported using conventional 60 t heavy goods vehicles with a 40 t payload, to avoid the requirements for special permitting by the Finnish and Swedish roads authorities and the HFS proposes that up to four trucks per day will be required. Several potential destinations exist for this concentrate within Sweden and Finland including facilities at Gällivare (Sweden); Skellefteå (Sweden) or Harjavalta (Finland).

Fe concentrate will be shipped using rail transportation from the Rautuvaara rail head to ports in Sweden or Finland.

The HFS base case for Fe concentrate transportation is from the Rautuvaara rail head to the Port of Kokkola (approximately 520 km south of Rautuvaara) by train. The system developed comprises several discrete components:

- warehouse / stockpiles at Rautuvaara;
- reinstated Rautuvaara - Kolari rail line;
- existing Kolair-Kokkola rail Line;

- Port of Kokkola; and
- Ocean Going Vessels for shipment to customers

It is SRK's opinion that the proposed method and route for the export of concentrates from the Project has been established in principle and in sufficient detail to determine the feasibility of the selected route and associated capital and operating costs.

The options selected have been developed to a reasonable degree of detail that is considered suitable for the HFS. The capital costs required for the infrastructure and the transportation of concentrates have been assessed in detail and where the designs are conceptual, reasonable allowances have been made.

The costs presented give a total transportation cost of USD17.75 per tonne ore shipped to the port of Kokkola including rail transportation, port operations and project management.

1.6 Mine Site Infrastructure

The proposed mine development is split between two key operational areas; Hannukainen and Rautuvaara, which are separated by approximately 3 km and the Muoniojoki River.

To facilitate the export of Fe concentrate and Cu-Au concentrate, the materials handling infrastructure will be required to handle 6.5 Mtpa of Run of Mine ("ROM") material with 30.6 Mtpa of overburden and waste rock stockpiled at the Project.

Mine trucks will transport ROM material from both pits to the crushing station at the ROM pad. Two ground level dump positions will feed the underground primary crusher. A steel frame building covering the crushing station is proposed at ground level which will contain a ground mounted hydraulic breaker and 40 t overhead crane. Beneath the crusher a 50 m deep, 14.5 m wide feeder hall is proposed to house a storage silo and conveyor feeder.

A 700 m long transfer conveyor, of which 545 m lies within a proposed 5.3 m wide by 5.55 m high shotcrete tunnel, is proposed to transport crushed ore at an inclination of 3.36° (5.8%) from beneath the storage silo to an elevated transfer tower at ground level.

Cu-Au concentrate will be exported by road trucks, whilst Fe concentrate will be exported by rail.

Power will be supplied to the Project site using existing Tornionlaakson Sähkö Oy ("TLS") overhead power infrastructure, the diversion of approximately 5 km of 110 kV overhead power line and construction of a new switchyard near Rautuvaara.

Separate solid fuel and oil fired boilers are proposed at both Hannukainen and Rautuvaara to generate heat for ventilation, room heating and warm potable water in buildings as heat output from process machinery is anticipated to be insufficient.

Surface water from surrounding areas will be diverted to a proposed process water pond, created through the construction of a dam across the Kivivuopionoja River. Water will be pumped to a water treatment plant before reticulation to the process plant. All excess and treated process water will be pumped via a 10.6 km long pipeline to the Muonionjoki River near Kolari.

Potable and foul water demands will be provided through new connections to the existing Ylläksen Yhdyskuntatekninen Huolto Oy ("YYTH") water and sewage water networks.

SRK has reviewed the proposed mine infrastructure and in general considers it to be in accordance with an HFS level of detail, however there are some aspects where the design basis and process followed requires clarification.

SRK has recommended the following technical studies:

- design of terracing bulk earthworks;
- design of an above ground crushing facility and associated infrastructure to reduce Capital Costs ("CAPEX"), improve accessibility and safety during operation/maintenance activities and reduce construction/schedule risk;
- develop a waste management strategy in accordance with accepted international environmental practice to define associated infrastructure;
- definition of anticipated power consumption to determine annual operating costs through negotiation with the power authority; and
- definition of surface water management pollution control infrastructure based on the SRK Waste Rock Geochemical Characterisation Report and Northland Hydrological Impact Assessment as presented in the HFS.

1.7 Tailings/Waste Rock Management

The initial phase of the Project considered depositing the tailings at the Hannukainen site. This site was located within an environmentally sensitive area and implied high costs. The Rautuvaara site was subsequently selected as an alternative site as it had already been disturbed from previous mining activities and provided an opportunity for lower cost.

Tailings deposition system consists of two tailings streams: Tailings from LIMS and high-sulphur ("High-S") tailings. Total tailings production over the life of mine will be 65.2 Mt (33.4 Mm^3) for the LIMS and 11.1 Mt (5.6 Mm^3) for the High-S. The High-S tailings will need to be encapsulated in a fully lined impoundment and include a base drainage system. The key feature is that the LIMS tailings will not need to be lined as for the High-S tailings. The configuration enables minimal initial CAPEX.

The LIMS and High-S tailings will be pumped from the mill to the Tailings Management Facility ("TMF") via lined carbon steel pipelines. The pipeline route will enable the use of unreinforced plastic pipe material over the last portion of the pipeline. Piston pumps were chosen for LIMS and High-S tailings pumping.

The LIMS tailings will be reclaimed by placing a cover that will consist of a bentonite mat placed directly on the deposited tailings and covered with 1.0 m of soil to protect the bentonite liner and to support vegetation. The High-S tailings will be reclaimed by placing a qualified cover over the consolidated tailings. The cover will include a bentonite mat on top of the tailings, then a HDPE liner, a protective geotextile and a 1.0 m thick soil cover at the surface. A network of ditches will maintain gravitational water flow at post-closure.

The initial CAPEX for the TMF has been estimated at USD18.39M. The estimated total Sustaining CAPEX is: USD58.17M for the TMF. The costs for engineering and construction management were allocated to Project Support while the costs for the tailings pumps were assigned to the Process Plant costs. The costs for design, field investigations and construction management are included in the costs mentioned above. The operating cost was estimated as per the Project cost template and included labour, fuel and power, maintenance and lubricants, spares and consumables, and equipment rentals. Cost for environmental monitoring is not included.

The total OPEX for the TMF has been estimated at USD24.14M. It includes the tailings pumping costs for LIMS tailings pumping at USD255k per year for the first 6 years of operation. The energy consumption subsequently increases because the tailings will be deposited with a longer pipeline that increases the pressure drop in the pipeline. After year 6, the annual energy consumption of LIMS tailings pumping is estimated at USD422k per year. The annual tailings pumping costs for High-S tailings are estimated at USD89k per year and remain constant over the entire life of mine. Pipeline maintenance costs are based on annual maintenance of 3% of capital costs. The annual cost for spare parts for tailings pumping is estimated at USD134k per year for the entire life of mine.

1.8 Environmental Management

Northland is in the process of undertaking an Environmental Impact Assessment ("EIA") for the mine site at Hannukainen and the Rautuvaara processing site and tailings storage facility. A number of issues exist that will need to be evaluated proactively to ensure additional material costs are not incurred, in particular:

- the acid rock drainage potential associated with some of the mine waste rock and tailings and the potential to contaminate aquifers and water courses via contact with base flow;
- dewatering of aquifers and impacts on base flow of water courses;
- the impacts on ecologically protected areas and protected plant and animal species;
- land acquisition;
- the relationships with the local reindeer husbandry cooperative and its members; and
- other parties who are responsible for undertaking the necessary assessments and gaining permits for the railway and port upgrade.

Some water courses in Project areas are Natura 2000 sites and breeding habitats for the protected Arctic Salmon and Sea Trout of critical ecological importance. Water quality standards and mixing zones have yet to be agreed with authorities, who may impose stricter standards than those used for modelling impacts in the EIA. As a consequence, costlier mitigation may be required than that proposed in the EIA.

The necessary permits have still to be obtained but there is a strategy in place to obtain these. SRK considers there is a significant risk Einkeino, Liikenne-ja Ympäristöministeriö ("ELY") will require Northland to revise its EIA report and this could affect the subsequent joint application for the environmental and water permits. It is also possible the permit process will also be subject to delays due to the significant public interest and sensitivity of the water issues (see above). In addition, both the EIA and the permit process may be subject to appeals (by either Northland or members of the public), further delaying the approval process.

1.9 Capital and Operating Costs

The operating costs estimated as part of the HFS have been incorporated into SRK's financial model with no material adjustments. SRK has reviewed these costs and considers them to be reasonable for the Project. Figure ES 2 illustrates an overall breakdown of the operating expenditure over the life of mine, split between the major cost centers and excluding contingency. These are also summarized in Table ES 3. An overall contingency of 5% has been assumed for operating costs.

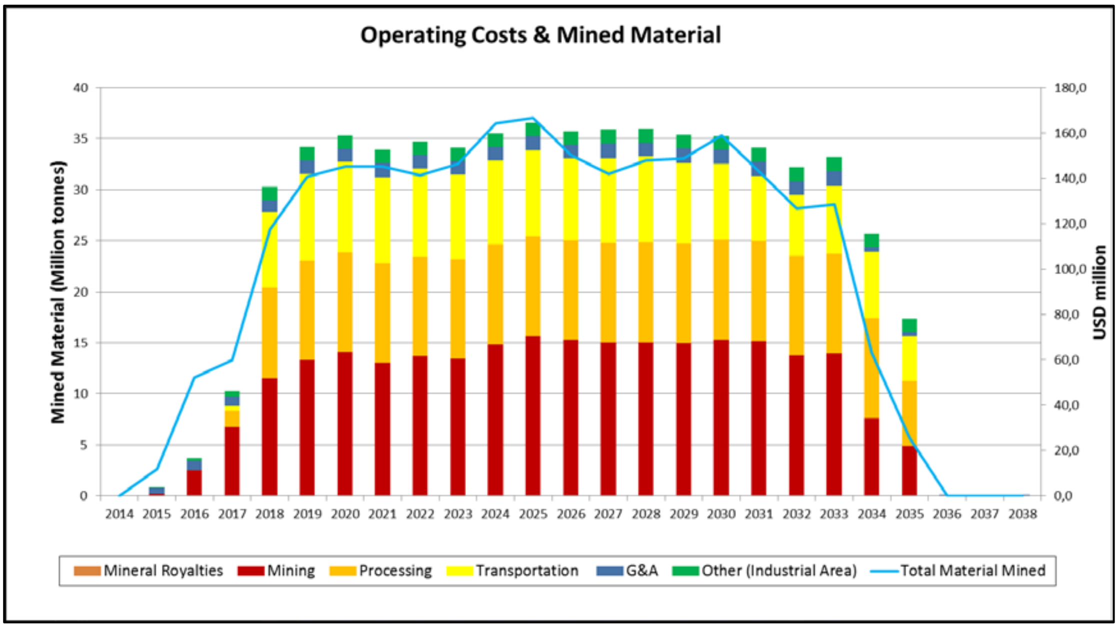


Figure ES 2: Summary of operating costs over the LOM, by major cost centre

Table ES 3: Summary of unit operating costs

	Unit Operating Costs per tonne total material (USD / tonne)	Unit Operating Costs per tonne milled (USD / tonne)	Unit Operating Costs in US cents per dmtu for concentrate sold
Mining	2.00	9.80	49.3
Processing	1.39	6.79	34.2
G&A	0.20	0.99	5.0
Other (Industrial Area)	0.20	0.98	4.9
Transportation	1.10	5.37	27.0
Mineral Royalties	0.01	0.07	0.4
Total Operating Expenditure (pre-contingency)	4.91	24.02	120.8
Total Operating Expenditure (incl 5% contingency)			126.9
TCRC's (Cu/Au Concentrate)			3.6
By-product credits (Cu/Au Concentrate)			-58.7
C1 Cash Costs*			71.8

** C1 costs include mining, processing, site admin, transportation, smelting and refining, net of byproduct credits*

1.10 Capital Cost Summary

The capital costs estimated as part of the HFS have been incorporated in to SRK's financial model with no material adjustments. These costs total USD 736 million, pre-contingency. A contingency of 10% is applied. Overall, SRK considers these costs to be reasonable for the Project.

Figure ES 3 gives an overview of the envisaged capital expenditure over the life of mine, excluding contingency. Figure ES 4, Figure ES 5 and Table ES 4 present a breakdown of initial and sustaining capital between the major cost centres.

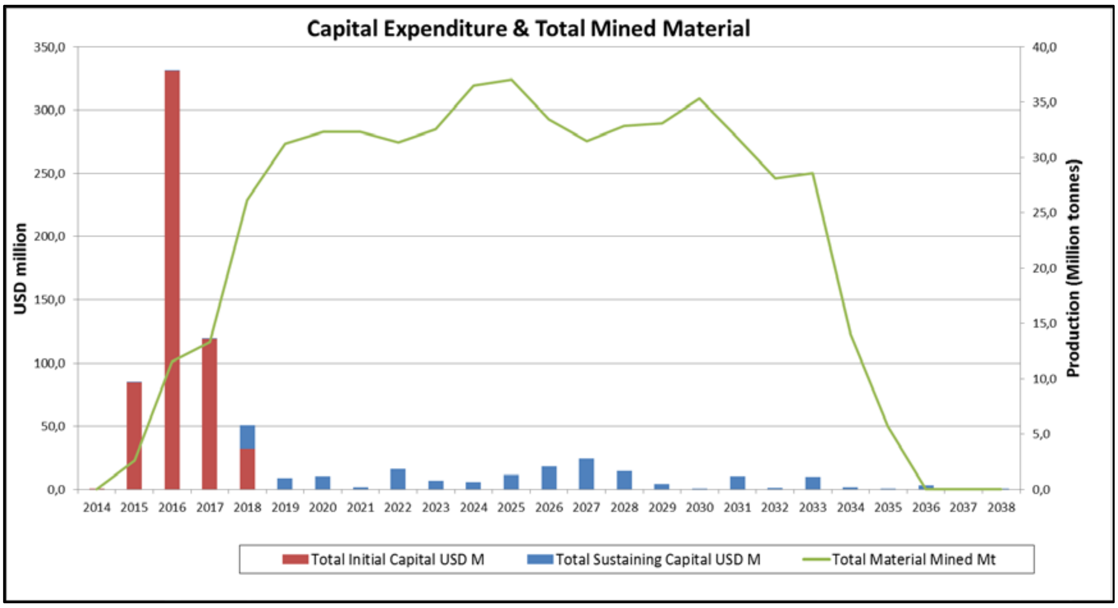


Figure ES 3: Forecast capital expenditure against total material movement

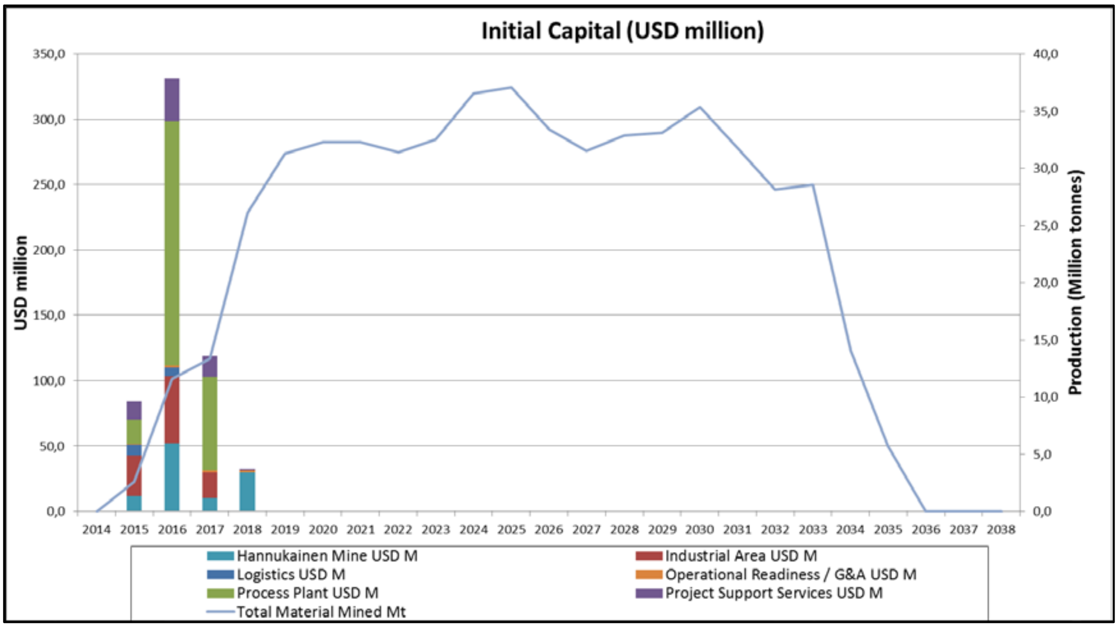


Figure ES 4: Initial capital expenditure by major cost centre and total material movement

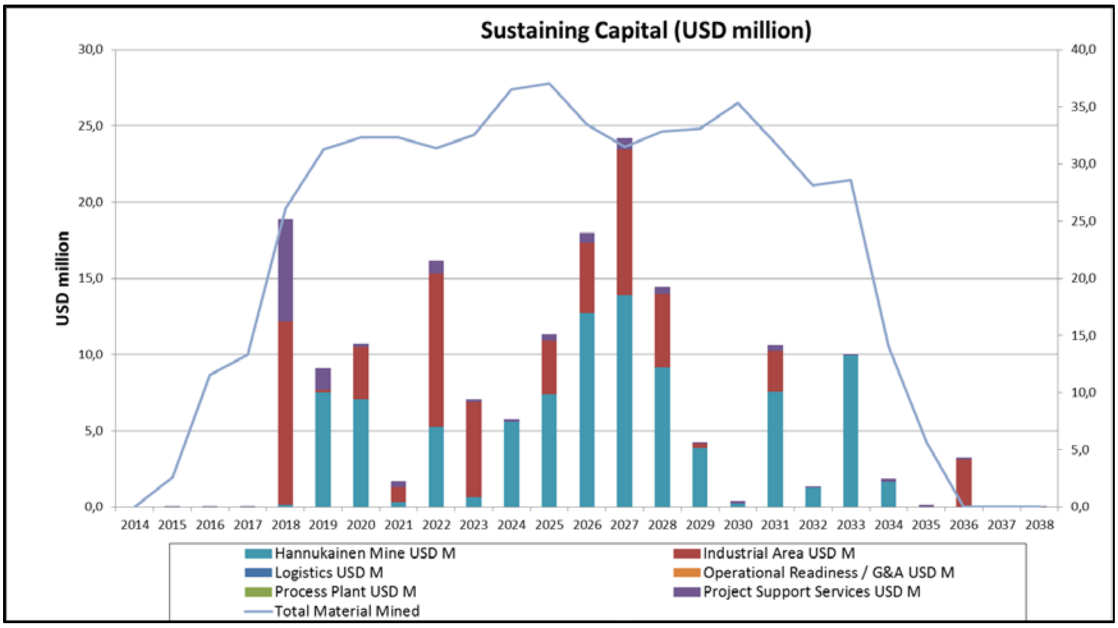


Figure ES 5: Sustaininig capital expenditure by major cost centre and total material movement

Table ES 4: Breakdown of initial and sustaining capital expenditure by major cost centre

Initial Capital	USD million	Sustaining Capital	USD million
Hannukainen Mine	104	Hannukainen Mine	94
Industrial Area	102	Industrial Area	62
Logistics	15	Logistics	0
Operational Readiness / G&A	6	Operational Readiness / G&A	0
Process Plant	277	Process Plant	0
Project Support Services	64	Project Support Services	14
Total Initial Capital	567	Total Sustaining Capital	169
Total Capital Expenditure (pre-contingency)	736		
Capital Contingency (10%)	74		
Total Capital Expenditure	**810**		

1.11 Economic Analysis

As part of this technical report, SRK's role has been to construct a financial model in order to derive a post-tax, pre-finance Net Present Value ("NPV") for the Project and independently verify (or otherwise) the valuation derived by Northland for the HFS. SRK has constructed its independent financial model using the cost data from Northland's cost templates, as well as extracting the underlying technical assumptions, macro-economic assumptions and life of mine plan from the HFS financial model itself.

The model is based on production from two open pit mines (Hannukainen and Kuervitikko), feeding a single process stream with a combined maximum annual throughput of 6.5 Mtpa and housed within a single processing plant. The plant produces a magnetite concentrate of 70% Fe, and a copper-gold concentrate of 25% Cu and 7.1 g/t Au. These concentrates are planned to be transported by rail from site to the port of Kokkola, Finland.

The valuation currency is USD, with any EUR or SEK derived capital and / or operating costs being converted at the following rates, which are assumed to be consistent over the life of mine:

- SEK:USD exchange rate of 6.9:1; and
- EUR:USD exchange rate of 0.7813:1

SRK notes that for operating expenditures, some 87% is denominated in EUR and 2% in SEK, with the remaining 11% denominated in USD. For capital expenditures some 94% is denominated in EUR and 1% in SEK, with the remaining 5% denominated in USD.

SRK also notes that the spot closing exchange rates as at 19 December 2013 are: SEK:USD exchange rate of 6.59:1 and EUR:USD exchange rate of 0.73:1. Single parameter sensitivities of the Project valuation to variations in the EUR:USD exchange rates is presented below in Section 21.8.

Net Present Values ("NPVs") as presented in this Technical Report are on a post-tax and pre-finance basis and assume a base case discount factor of 8%. All figures are presented in real terms.

Working capital assumptions are as follows:

- Debtor days = 30
- Creditor days = 30
- Inventory days = 8

Straight-line depreciation has been applied to the sum total of capital expenditures over the LOM to derive profits before tax. A useful economic life of 5 years is assumed along with zero salvage value. A corporate income tax rate of 24.5% is applied to pre-tax profits to arrive at a post-tax cashflow.

Labour rates provided to SRK by the Client and incorporated into the financial model are inclusive of on-costs. These on-costs include employers social contributions which SRK understand may range between 26.5% and 34.2%.

A summary of tax assumptions and mineral royalties are presented in Table ES 5 below.

Table ES 5: Summary of tax and mineral royalty assumptions

Type	Value
Mineral Royalty	0.15%
Corporation Tax	24.50%
Depreciation	5 years
Employers Social Contributions	Between 26.5% and 34.2%

Commodity price forecast data was provided to SRK by Northland. Northland developed this forecast internally, based primarily on an independent third party report by Raw materials Group (“RMG”), dated October 2013. RMG developed the base case iron ore price forecast model, with the base case price forecast subsequently adjusted by Northland in consideration of a Value-In-Use premium of USD3 dmt.

Figure ES 6 presents the iron ore price forecast for the life of mine as incorporated in SRK’s financial model.

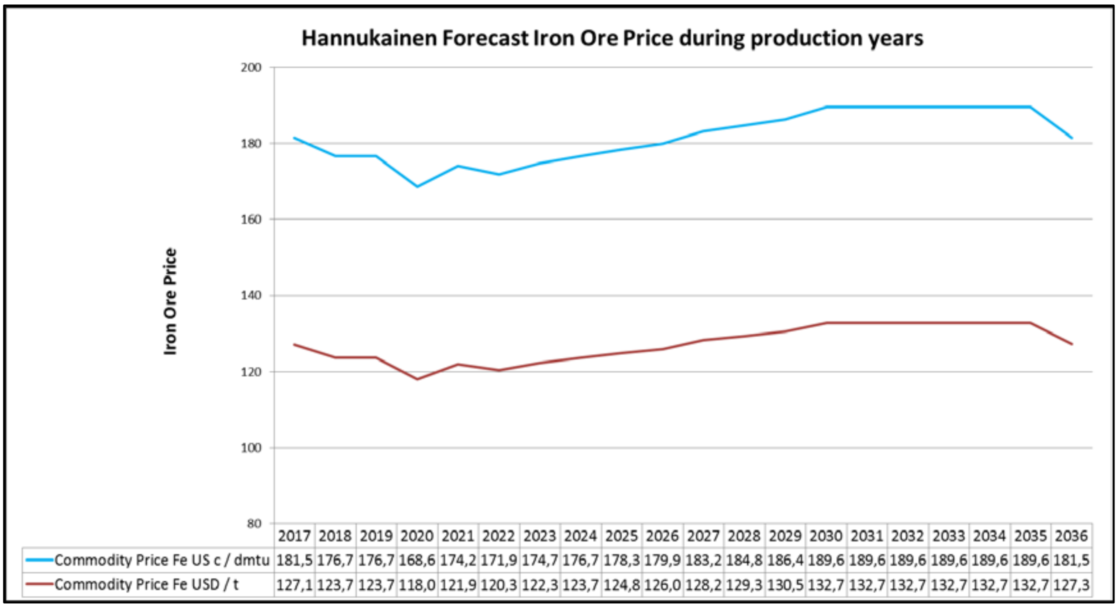


Figure ES 6: Northland forecast iron ore prices for Hannukainen product during production years

SRK are not specialists in metal price forecasting and rely on the Consensus Market Forecasts ("CMF") when considering expected trends in metal prices. The majority of CMF prices considered by SRK for comparative purposes, show an expected decrease in Fe product price until 2019.

Northland has assumed the following forecast metal prices for copper and gold, which for the purposes of the model, are assumed to be consistent during the entire life of mine:

- Copper: 6,950 USD / tonne; and
- Gold: 1,350 USD / troy ounce.

For the avoidance of doubt, Table ES 6 shows the commodity prices have been utilised throughout the HFS.

Table ES 6: HFS Commidity price summary

Commodity	Unit	Resource Reporting	Reserve Reporting	Pit Selection	Economic Analysis
Fe	USD/dmtu	1.5	1.4	1.25	Price profile used – see Figure ES 6
Cu	USD/t	7,385	6,305	5,620	6,950
Au	USD/oz	1,375	1,250	1,116	1,350

SRK notes that Northlands price assumptions fall within the range of available CMF data for Q4 2013 and that these lie at the upper end of this range.

A summary of the combined mass movement of material is presented In Table ES 7. Figure ES 7 illustrates combined ore and waste tonnages mined and Fe% grade over the life of mine. Figure ES 8 illustartes the LOM Cu and Au grades.

Table ES 7: Hannukainen and Kuervitikko combined RoM Ore and Waste movement

Description	Units	Life Of Mine Totals
Ore tonnes	(Mt)	114.8
Ore grade	(% Fe)	30.5
Ore grade	(% Cu)	0.19
Ore grade	(Au g/t)	0.11
Mass waste	(Mt)	446.8
Total Material Mined	(Mt)	561.6
Strip ratio	(W:O)	3.9
Overburden Volume	(Mt)	74.8

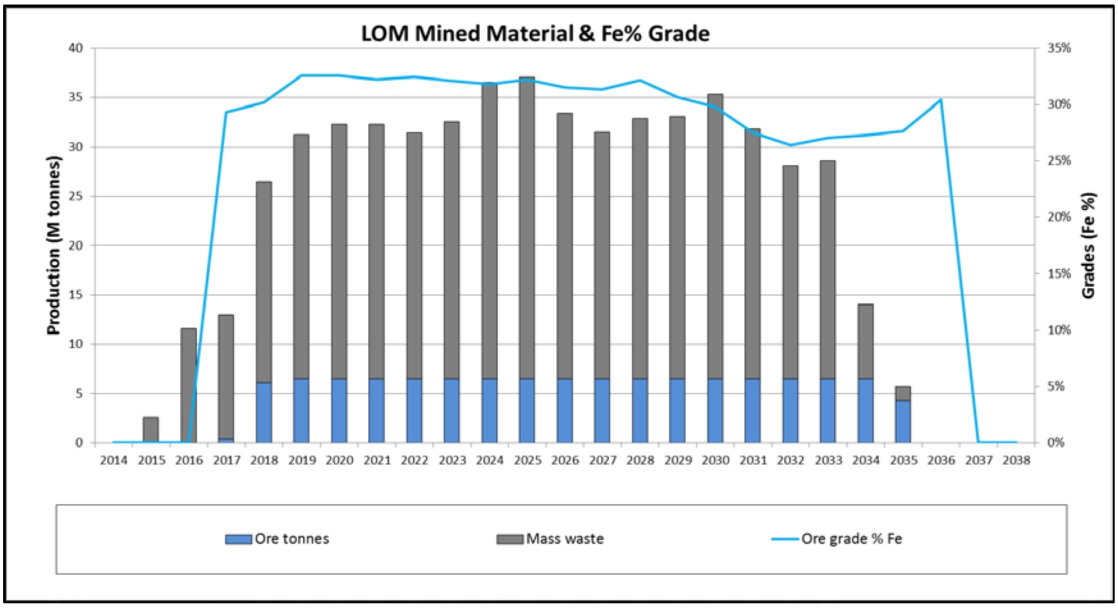


Figure ES 7: Hannukainen and Kuervitikko combined RoM Ore and Waste movement with Fe% grade.

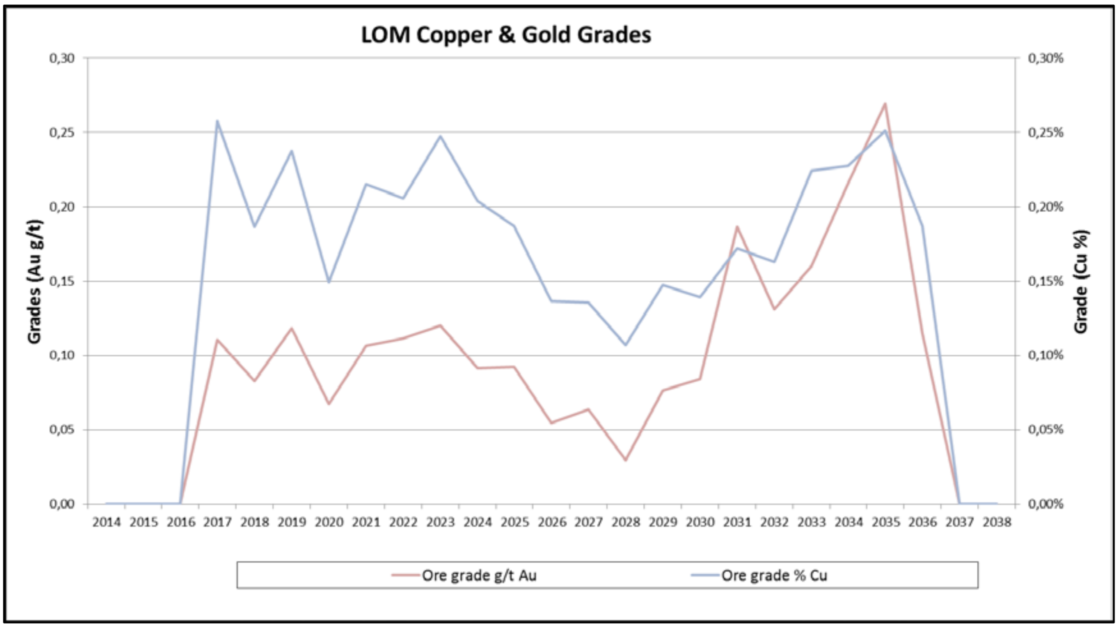


Figure ES 8: Hannukainen and Kuervitikko combined LOM copper and gold grade

Table ES 8 summarises combined recoveries, concentrate grades and concentrate tonnages over the LOM. Figure ES 9 and Figure ES 10 below illustrate concentrate production and plant performance for magnetite and copper-gold concentrate products respectively.

Table ES 8: LOM Process Physical Assumptions

Description	Units	Hannukainen (combined)
Magnetite Concentrate		
Contained recoverable Fe	(Mt)	35.0
Iron recovery	(%)	65%
Mass yield	(%)	29%
Grade of final magnetite concentrate	(% Fe)	70%
Concentrate tonnage (dry)	(Mt)	32.8
Copper-gold Concentrate		
Contained Cu	(Mt)	0.2
Contained Au	(Million troy ounce)	0.4
Copper recovery	(%)	84%
Gold recovery	(%)	26%
Mass yield	(%)	0.6%
Grade of final Cu/Au concentrate	(% Cu)	25
Grade of final Cu/Au concentrate	(g/t Au)	7.1
Concentrate tonnage (dry)	(Mt)	0.72

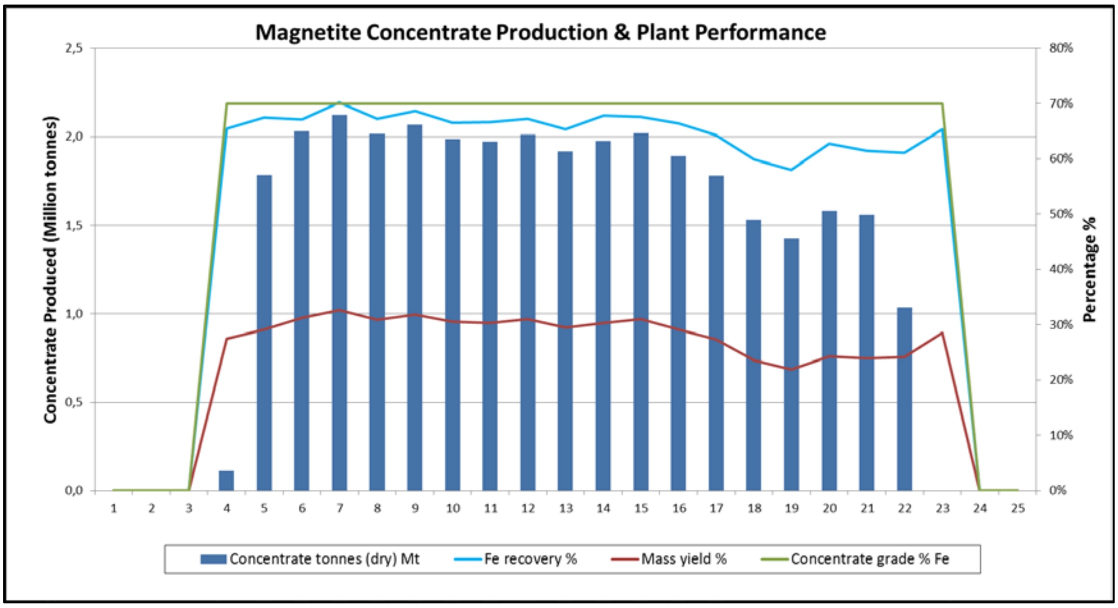


Figure ES 9: Magnetite concentrate production and plant performance (Fe recovery % and mass yield %)

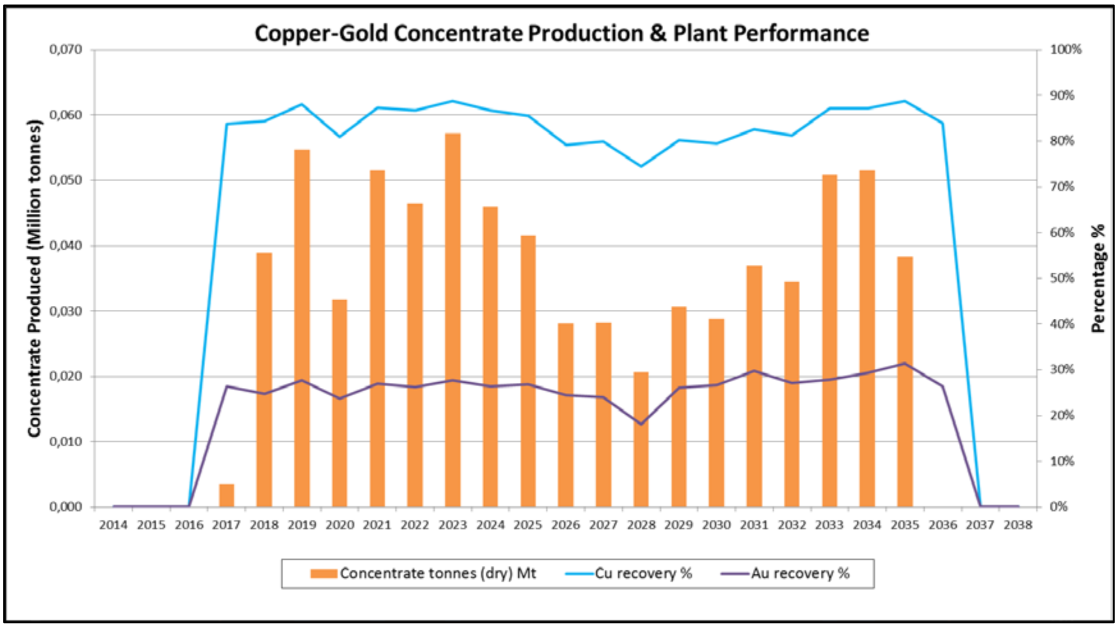


Figure ES 10: Copper-gold concentrate production and plant performance (Cu recovery % and Au recovery %)

Table ES 9 summarises assumed handling losses.

Table ES 9: Assumed handling losses

Description	Unit	Assumed Handling Losses over LOM
Magnetite Concentrate		
Processing	(%)	0.02%
Product to railcars	(%)	0.23%
Product to ship	(%)	0.23%
Ship unloading	(%)	0.23%
Total losses	(%)	0.71%
Total losses	(Mt)	0.23
Concentrate available for sale	(Mt)	32.6
Copper-gold Concentrate		
Total losses	(%)	1.0
Total losses	(t)	7 202
Concentrate available for sale	(Mt)	0.71

Table ES 10 summarises gross revenues, deductions, treatment charges and refining costs (“TCRC’s”) and resulting net revenues by concentrate product.

Table ES 10: Gross Revenues, deductions TCRC's and resulting net revenues by concentrate product.

Description	Unit	Value
Magnetite Concentrate		
Concentrate available for sale	(Mt)	32.6
Gross Revenue	(USD million)	4,120
Copper-Gold Concentrate		
Concentrate available for sale	(Mt)	0.71
Unit deduction copper (4%)	(t)	7,130
Cu metal recovery from concentrate	(%)	96.65%
Payable copper	(t)	165,380
Payable copper	(lb)	364,601,000
Cu gross revenue	(USD million)	1,149
Unit deduction gold (1 g/t)	(kilogram)	713
Payable gold	(kilogram)	4,349
Payable gold	(troy ounce)	139,827
Gross revenue gold	(USD million)	189
Gross revenue copper-gold concentrate	(USD million)	1,338
Total TCRC's	(USD million)	82
Net revenue copper-gold concentrate	(USD million)	1,257
Total net revenue	**(USD million)**	**5,377**

Figure ES 11 presents annual contribution to gross revenue over the LOM, by concentrate product.

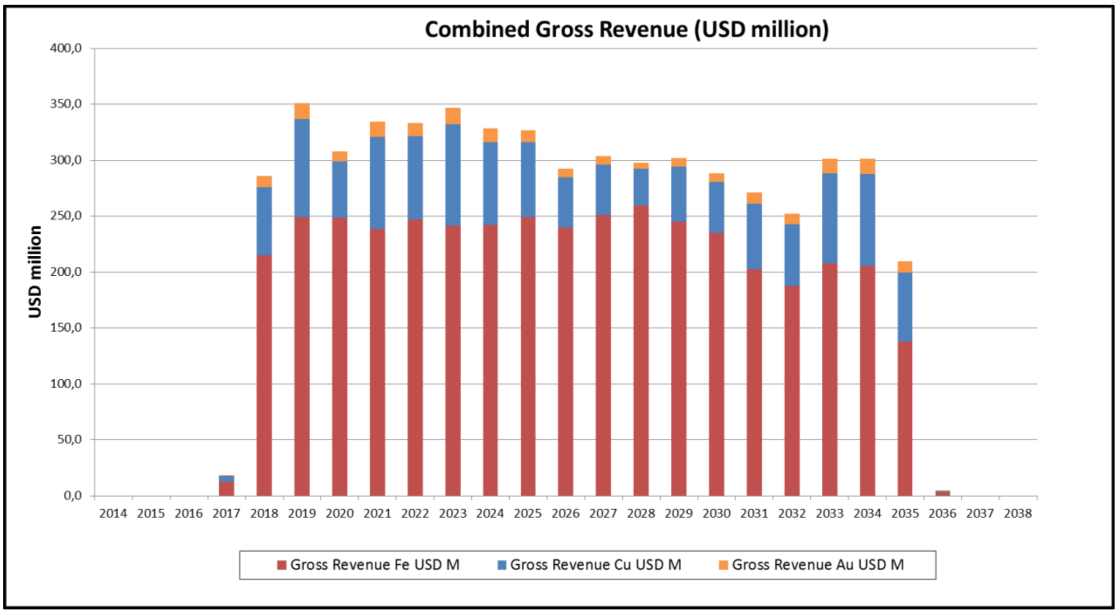


Figure ES 11: Annual contribution to gross revenue over the LOM, by concentrate product

A valuation of the Project has been derived based on the application of Discounted Cash Flow ("DCF) techniques to the post-tax, pre-finance cash flow developed for the HFS.

The resulting post-tax, pre-finance real terms (1 December 2013) NPV derived by SRK is USD 248 million, assuming an 8% discount rate. SRK notes that Northland report a NPV of USD 251 million (post-tax, pre-finance at 8% discount) in the HFS. This difference is not considered to be material.

A summary of the results of the cash flow modelling and valuation are presented in Table ES 11 and Figure ES 12.

Table ES 11: Summary results of cash flow modelling

Description	Units	Total (USDm)
Net Revenue	(USD million)	5 377
Total Operating Expenditure	(USD million)	-2 895
Total Capital Expenditure	(USD million)	-810
Other Expenses (Environmental Bond Payments)	(USD million)	-117
Net pre-tax, pre-finance cashflow	(USD million)	1 555
Corporation Tax (24.5%)	(USD million)	-379
Interest & Fees (Fees on Environmental Bond)	(USD million)	-22
Net post-tax, pre-finance cashflow	(USD million)	1 155
Payback period	(years)	9
NPV 8% (post-tax, pre-finance)*	(USD million)	248
IRR	(%)	14.0%

**Northland HFS estimate an NPV 8% (post-tax, pre-finance) of USD 251 million*

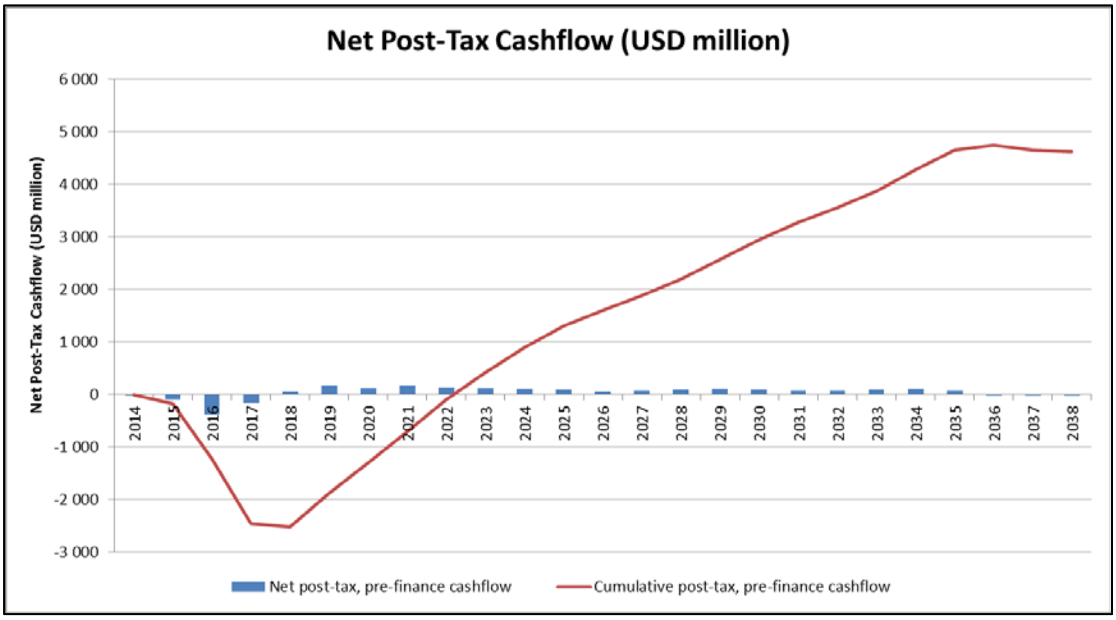
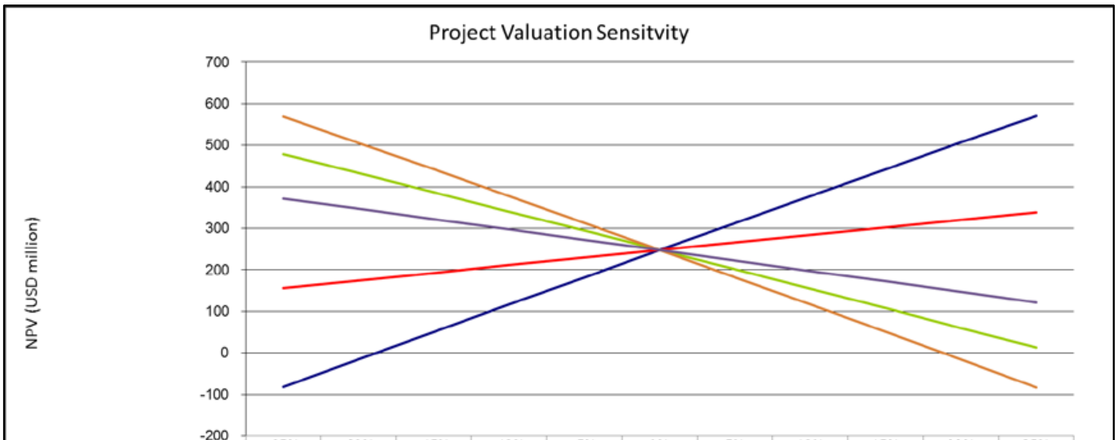


Figure ES 12: Annual net post-tax, pre-finance cashflow

For illustrative purposes the following analysis presents the sensitivity of the Project valuation (post-tax and pre-finance) for various capital costs, operating costs, commodity price, exchange rate and discount rate scenarios.

Figure ES 13 shows the Project valuation for varying single parameter sensitivities at an 8% discount rate for commodity price, operating costs, capital costs and EUR:USD exchange rate.



	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
Fe Price	(82)	(15)	51	117	182	248	313	377	442	506	571
Cu Price	156	175	193	211	229	248	266	284	302	320	338
Operating Expenditure	479	433	387	340	294	248	201	154	108	61	13
EUR:USD Exchange rate	569	506	442	377	313	248	182	117	51	(16)	(83)
Capital Expenditure	372	347	322	298	273	248	223	197	172	147	121

Figure ES 13: NPV (8%) single parameter sensitivities

Assuming all other assumptions remained unchanged, SRK notes that the Project would be roughly break-even should the Fe ore price fall by around 20%.

The Project appears less sensitive to variations in capital costs, with a 25% increase in capital expenditure resulting in the valuation falling by roughly half, from USD 248 million to USD 121 million.

At a fixed basecase discount rate of 8%, Table ES 12 shows the sensitivity of the Project valuation (USD million), to simultaneous changes in two parameters for; operating costs and Fe price, capital costs and Fe price, and operating costs and capital costs respectively. SRK notes that the Project is roughly break-even at:

- A decrease in Fe-ore price of 10% and simultaneous increase in operating costs of around 12%; and
- A decrease in Fe-ore price of 15% and simultaneous increase in capital costs of 10%.

At a variable discount rate, Table ES 13 shows the sensitivity of the Project valuation (USD million), to simultaneous changes in two parameters for; operating costs and Fe price, capital costs and Fe price, and operating costs and capital costs respectively.

Table ES 12: Twin Parameter Project Sensitivities in USD million - Fixed Discount Rate (8%)

		REVENUE V OPEX SENSITIVITY										
							Fe Price					
	247,6	**-25%**	**-20%**	**-15%**	**-10%**	**-5%**	**0%**	**5%**	**10%**	**15%**	**20%**	**25%**
Operating Expenditure	**-25%**	154	220	285	350	414	479	543	607	672	736	799
	-20%	108	173	238	303	368	433	497	562	626	690	754
	-15%	61	126	192	257	322	387	451	516	580	644	708
	-10%	13	79	145	210	276	340	405	470	534	598	662
	-5%	-34	32	98	164	229	294	359	424	488	552	617
	0%	-82	-15	51	117	182	248	313	377	442	506	571
	5%	-131	-63	4	70	136	201	266	331	396	460	525
	10%	-182	-111	-44	23	89	154	220	285	350	414	479
	15%	-234	-161	-92	-25	42	108	173	238	303	368	433
	20%	-290	-213	-141	-72	-6	61	126	192	257	322	386
	25%	-350	-266	-192	-121	-53	13	79	145	210	275	340
		REVENUE V CAPEX SENSITIVITY										
							Fe Price					
	247,6	**-25%**	**-20%**	**-15%**	**-10%**	**-5%**	**0%**	**5%**	**10%**	**15%**	**20%**	**25%**
Capital Expenditure	**-25%**	47	112	178	243	307	372	436	501	565	628	692
	-20%	21	87	153	218	283	347	412	476	540	604	668
	-15%	-4	62	127	193	258	322	387	451	516	580	644
	-10%	-30	36	102	167	233	298	362	427	491	555	620
	-5%	-56	11	77	142	208	273	337	402	467	531	595
	0%	-82	-15	51	117	182	248	313	377	442	506	571
	5%	-108	-41	26	92	157	223	288	352	417	482	546
	10%	-135	-67	0	66	132	197	263	328	392	457	522
	15%	-162	-93	-26	41	107	172	237	303	367	432	497
	20%	-188	-119	-52	15	81	147	212	278	343	407	472
	25%	-216	-146	-78	-11	56	121	187	252	318	382	447
		OPEX V CAPEX SENSITIVITY										
							Operating Expenditure					
	247,6	**-25%**	**-20%**	**-15%**	**-10%**	**-5%**	**0%**	**5%**	**10%**	**15%**	**20%**	**25%**
Capital Expenditure	**-25%**	601	555	510	464	418	372	326	280	233	187	140
	-20%	577	531	485	439	393	347	301	255	208	162	115
	-15%	552	507	461	415	369	322	276	230	183	137	90
	-10%	528	482	436	390	344	298	251	205	158	111	64
	-5%	503	457	411	365	319	273	226	180	133	86	39
	0%	479	433	387	340	294	248	201	154	108	61	13
	5%	454	408	362	315	269	223	176	129	82	35	-12
	10%	429	383	337	290	244	197	151	104	57	9	-38
	15%	405	358	312	265	219	172	125	78	31	-16	-64
	20%	380	333	287	240	194	147	100	53	5	-42	-90
	25%	355	308	262	215	168	121	74	27	-20	-68	-116

Table ES 13: Twin Parameter Project Sensitivities in USD million - Variable Discount Rate

DISCOUNT FACTORS V IRON PRICE

		Fe Price										
	1 154,5	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
NPV Discount Factors	0%	378,1	533,3	688,6	843,9	999,2	1 154,5	1 309,8	1 465,1	1 620,4	1 775,7	1 931,0
	2%	204,7	328,0	450,8	573,4	695,9	818,2	940,5	1 062,6	1 184,8	1 306,8	1 428,8
	4%	78,4	177,7	276,3	374,6	472,7	570,5	668,2	765,7	863,2	960,5	1 057,8
	6%	(14,1)	67,0	147,3	227,2	306,8	386,1	465,3	544,3	623,1	701,8	780,4
	8%	(82,2)	(15,1)	51,2	117,0	182,4	247,6	312,6	377,3	442,0	506,4	570,7
	10%	(132,3)	(76,1)	(20,8)	34,0	88,5	142,7	196,7	250,4	304,0	357,4	410,7
	12%	(169,0)	(121,6)	(74,9)	(28,7)	17,1	62,7	108,0	153,1	198,0	242,8	287,3
	14%	(195,8)	(155,4)	(115,6)	(76,3)	(37,4)	1,3	39,7	78,0	116,0	153,9	191,6
	16%	(215,2)	(180,3)	(146,2)	(112,4)	(79,0)	(45,9)	(13,0)	19,6	52,2	84,5	116,7
	18%	(228,8)	(198,6)	(169,0)	(139,8)	(111,0)	(82,3)	(54,0)	(25,8)	2,2	30,1	57,7
	20%	(238,1)	(211,7)	(185,9)	(160,5)	(135,3)	(110,4)	(85,8)	(61,3)	(37,0)	(12,8)	11,2

DISCOUNT FACTORS V COPPER PRICE

		Cu Price										
	1 154,5	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
NPV Discount Factors	0%	937,9	981,3	1 024,6	1 067,9	1 111,2	1 154,5	1 197,9	1 241,2	1 284,5	1 327,8	1 371,2
	2%	647,8	681,9	716,0	750,1	784,2	818,2	852,3	886,3	920,4	954,4	988,5
	4%	434,1	461,4	488,7	516,0	543,3	570,5	597,7	625,0	652,2	679,4	706,5
	6%	275,3	297,5	319,7	341,9	364,0	386,1	408,2	430,3	452,4	474,5	496,5
	8%	156,3	174,6	192,9	211,2	229,4	247,6	265,8	284,0	302,1	320,3	338,4
	10%	66,5	81,8	97,1	112,3	127,5	142,7	157,9	173,0	188,1	203,2	218,3
	12%	(1,6)	11,3	24,2	37,0	49,9	62,7	75,5	88,2	101,0	113,7	126,4
	14%	(53,5)	(42,5)	(31,5)	(20,6)	(9,6)	1,3	12,2	23,1	33,9	44,8	55,6
	16%	(93,1)	(83,6)	(74,2)	(64,7)	(55,3)	(45,9)	(36,6)	(27,2)	(17,9)	(8,6)	0,7
	18%	(123,3)	(115,0)	(106,8)	(98,7)	(90,5)	(82,3)	(74,2)	(66,1)	(58,0)	(50,0)	(41,9)
	20%	(146,2)	(139,0)	(131,8)	(124,7)	(117,5)	(110,4)	(103,4)	(96,3)	(89,2)	(82,2)	(75,2)

DISCOUNT FACTORS V OPERATING EXPENDITURE

		Operating Expenditure										
	1 154,5	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
NPV Discount Factors	0%	1 701,0	1 591,7	1 482,4	1 373,1	1 263,8	1 154,5	1 045,3	936,0	826,7	717,4	608,1
	2%	1 249,3	1 163,2	1 077,0	990,8	904,5	818,2	731,9	645,5	559,0	472,4	385,8
	4%	915,9	846,9	777,9	708,9	639,7	570,5	501,2	431,8	362,3	292,7	222,9
	6%	666,9	610,9	554,8	498,7	442,5	386,1	329,7	273,2	216,5	159,7	102,7
	8%	478,9	432,8	386,6	340,4	294,1	247,6	201,0	154,4	107,5	60,5	13,3
	10%	335,5	297,2	258,7	220,1	181,5	142,7	103,8	64,7	25,6	(13,8)	(53,4)
	12%	225,3	193,0	160,6	128,0	95,4	62,7	29,8	(3,2)	(36,4)	(69,7)	(103,3)
	14%	140,0	112,4	84,8	57,1	29,3	1,3	(26,8)	(55,0)	(83,3)	(111,8)	(140,6)
	16%	73,4	49,7	26,0	2,1	(21,8)	(45,9)	(70,1)	(94,4)	(118,9)	(143,5)	(168,4)
	18%	21,2	0,7	(19,9)	(40,6)	(61,4)	(82,3)	(103,4)	(124,5)	(145,8)	(167,3)	(188,9)
	20%	(19,8)	(37,8)	(55,8)	(73,9)	(92,1)	(110,4)	(128,9)	(147,4)	(166,1)	(184,9)	(203,9)

DISCOUNT FACTORS V CAPITAL EXPENDITURE

		Capital Expenditure										
	1 154,5	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
NPV Discount Factors	0%	1 308,1	1 277,4	1 246,7	1 216,0	1 185,3	1 154,5	1 123,8	1 093,1	1 062,4	1 031,7	1 001,0
	2%	963,0	934,1	905,2	876,2	847,2	818,2	789,2	760,1	731,0	701,9	672,7
	4%	707,6	680,3	652,9	625,5	598,0	570,5	542,9	515,3	487,6	459,9	432,1
	6%	516,5	490,6	464,6	438,5	412,3	386,1	359,9	333,5	307,1	280,7	254,1
	8%	372,0	347,2	322,5	297,6	272,6	247,6	222,5	197,4	172,1	146,8	121,5
	10%	261,5	237,9	214,2	190,5	166,6	142,7	118,7	94,6	70,4	46,2	21,9
	12%	176,4	153,9	131,2	108,4	85,6	62,7	39,7	16,6	(6,6)	(29,8)	(53,1)
	14%	110,4	88,8	67,0	45,2	23,3	1,3	(20,8)	(42,9)	(65,1)	(87,4)	(109,7)
	16%	58,8	38,0	17,2	(3,8)	(24,8)	(45,9)	(67,1)	(88,4)	(109,7)	(131,1)	(152,5)
	18%	18,2	(1,7)	(21,7)	(41,9)	(62,1)	(82,3)	(102,7)	(123,1)	(143,6)	(164,1)	(184,7)
	20%	(13,7)	(32,9)	(52,2)	(71,5)	(90,9)	(110,4)	(130,0)	(149,6)	(169,3)	(189,1)	(208,9)

DISCOUNT FACTORS V EUR:USD EXCHANGE RATE SENSITIVITY

		EUR:USD Exchange rate										
	1 154,5	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
NPV Discount Factors	0%	1 796,9	1 668,3	1 540,0	1 411,4	1 283,0	1 154,5	1 026,1	897,7	769,5	640,8	512,4
	2%	1 343,9	1 238,9	1 134,1	1 028,8	923,6	818,2	712,7	607,1	501,4	395,2	288,8
	4%	1 009,2	921,8	834,4	746,6	658,6	570,5	482,1	393,5	304,7	215,3	125,4
	6%	758,8	684,8	610,6	536,0	461,2	386,1	310,8	235,1	159,2	82,6	5,5
	8%	569,2	505,5	441,6	377,2	312,5	247,6	182,3	116,7	50,8	(15,8)	(83,0)
	10%	424,2	368,6	312,7	256,3	199,7	142,7	85,4	27,7	(30,3)	(89,0)	(148,3)
	12%	312,2	263,0	213,5	163,5	113,3	62,7	11,7	(39,6)	(91,3)	(143,6)	(196,5)
	14%	224,8	180,9	136,5	91,8	46,7	1,3	(44,5)	(90,6)	(137,0)	(184,1)	(231,7)
	16%	156,2	116,5	76,5	36,0	(4,8)	(45,9)	(87,4)	(129,2)	(171,4)	(214,0)	(257,3)
	18%	101,9	65,8	29,3	(7,6)	(44,8)	(82,3)	(120,2)	(158,4)	(196,9)	(236,0)	(275,5)
	20%	58,7	25,6	(7,9)	(41,8)	(76,0)	(110,4)	(145,3)	(180,4)	(215,8)	(251,7)	(288,0)

1.12 SRK Qualifications

The work undertaken by SRK in compiling this report has been managed by Mr Howard Baker, a Principal Mining Geologist with SRK. Mr Baker is a Qualified Person ("QP") as defined by the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") and outlined in National Instrument 43-101 of the Canadian Securities Administrators ("NI 43-101"). Mr Baker was also responsible for the Mineral Resource Estimates undertaken for the Project. As part of this work, SRK undertook a site visit and made first hand observations of the core collection and logging procedures employed and reviewed all data available for the Hannukainen deposit.

The Mineral Reserve and Mine Design aspects of the study were undertaken by Ms Colleen MacDougal, a Senior Mining Engineer with SRK. Ms MacDougall is a Qualified Person ("QP") as defined by the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") and outlined in National Instrument 43-101 of the Canadian Securities Administrators ("NI 43-101"). All work undertaken was reviewed by Mr Rick Skelton, a Corporate Mining Engineer with SRK. Mr Skelton is a Qualified Person ("QP") as defined by the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") and outlined in National Instrument 43-101 of the Canadian Securities Administrators ("NI 43-101").

The Metallurgical testwork and plant design review work in this report has been conducted by Dr David Pattinson, a Principal Process Engineer with SRK. Mr Pattinson is a Qualified Person ("QP") as defined by the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") and outlined in National Instrument 43-101 of the Canadian Securities Administrators ("NI 43-101").

Additional technical input provided by SRK to the HFS was carried out by SRK's Mr Dave Cooper and Mr Michel Noel who undertook the Waste Rock Design aspects of the study, SRK's Mr Andrew Barnes and Mr Mathew Dey who undertook the Acid Rock Drainage aspects of the study, SRK's Mr Phillip Mohr who undertook the Geotechnical aspects of the study and SRK's Mrs Sarah Johnson and Dr Tony Rex who undertook the Mine Water Management aspects of the study.

Review work of those aspects of the HFS not carried out by SRK was undertaken by SRK's Mr Michel Noel who managed the tailings aspects of the study being undertaken by Poyry, SRK's Mr Tim Fry who reviewed the infrastructure and concentrate logistics aspects of the study, SRK's Mr Richard Evans and Mrs Fiona Cessford who reviewed the environmental and permitting aspects of the study, and SRK's Mr Johan Bradley who reviewed the financial modelling aspects of the study.

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TECHNICAL REPORT ON THE HANNUKAINEN IRON-COPPER-GOLD PROJECT, KOLARI DISTRICT, FINLAND, JANUARY 2014

1 INTRODUCTION

This technical report has been prepared for Northland Mines OY ("Northland" or "the Company") by SRK Consulting ("UK") Ltd ("SRK") in connection with the publication by Northland of the Hannukainen Feasibility Study ("HFS"). Northland is a public limited liability company, domiciled in the Grand Duchy of Luxembourg, and governed by the Luxembourg law of August 10, 1915 on commercial companies, as amended. Northland Resources S.A. is the parent company, which in turn holds operating subsidiary companies in Sweden ("Northland Resources A.B") and Finland ("Northland Mines OY"). The common shares of the Company were listed on the Toronto Stock Exchange ("TSX") up to 15 March 2013, and are now primary listed on the Oslo Stock Exchange ("Oslo Børs").

Northland is principally an iron exploration and production company with properties in northern Sweden (The Kaunisvaara project comprising the Sahavaara and Tapuli deposits, currently in production) and Finland (Hannukainen comprising the Hannukainen and Kuervitikko deposits). This report describes the results of a review of the HFS which comprises the exploitation of the Hannukainen iron-copper-gold ("IOCG") project ("the Project") in the Kolari District, Finland ("the Mineral Asset").

The Technical Report has been prepared following the Guidelines of the CIM (Canadian Institute of Mining, Metallurgy and Petroleum) reporting code following the 2011 definitions and guidelines of National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101"), Form 43-101F1 - Technical Report (the "Technical Report"), and Companion Policy 43-101CP ("the "Companion Policy").

SRK has prepared the Mineral Resource and Mineral Reserve estimates presented in the HFS following the guidelines of the CIM code and SRK has provided direct geotechnical, hydrological, waste rock management and Acid Rock Drainage inputs to the Hannukainen mine design assumed by this. In addition, SRK has reviewed those aspects of the HFS completed by Northland and its other contractors and consultants to a sufficient level to enable SRK to present its own opinions on the project and an audited NPV for this.

The Project is an advanced exploration project which when constructed will comprise two conventional open pit mines and a magnetite processing operation producing an iron ("Fe") concentrate product. In addition, the Project will produce a combined copper ("Cu") / gold ("Au") concentrate.

The work undertaken by SRK in compiling this report has been managed by Mr Howard Baker, a Principal Mining Geologist with SRK. Mr Baker is a Qualified Person ("QP") as defined by the CIM. Mr Baker was also responsible for the Mineral Resource Estimates undertaken for the Project. As part of this work, SRK undertook a site visit and made first hand observations of the core collection and logging procedures employed and reviewed all data available for the Hannukainen deposit.

The Mineral Reserve and Mine Design aspects of the Project were undertaken by Ms Colleen MacDougal, a Senior Mining Engineer with SRK. Ms MacDougal is a Qualified Person ("QP") as defined by the CIM. All work undertaken was reviewed by Mr Rick Skelton, a Corporate Mining Engineer with SRK. Mr Skelton is a Qualified Person ("QP") as defined by the CIM.

The Metallurgical testwork and plant design review work in this report has been conducted by Dr David Pattinson, a Principal Process Engineer with SRK. Mr Pattinson is a Qualified Person ("QP") as defined by the CIM.

Additional technical input provided by SRK to the HFS was carried out by SRK's Mr Dave Cooper and Mr Michel Noel who undertook the Waste Rock Design aspects of the Project, SRK's Mr Andrew Barnes and Dr Mathew Dey who undertook the Acid Rock Drainage aspects of the Project, SRK's Mr Phillip Mohr who undertook the Geotechnical aspects of the project and SRK's Mrs Sarah Johnson and Dr Tony Rex who undertook the Mine Water Management aspects of the project.

Review work of those aspects of the HFS not carried out by SRK was undertaken by SRK's Mr Michel Noel who managed the tailings aspects of the project being undertaken by Poyry, SRK's Mr Tim Young who reviewed the infrastructure and concentrate logistics aspects of the project, SRK's Mr Richard Evans and Mrs Fiona Cessford who reviewed the environmental and permitting aspects of the Project, and SRK's Mr Johan Bradley who reviewed the financial modelling aspects of the Project. All of these team members visited the site and Northland offices throughout the technical studies and reviews being undertaken. Mr Howard Baker and Mr Johan Bradley attended multiple meetings with Northland personnel throughout the Project.

2 RELIANCE ON OTHER EXPERTS

SRK's opinion is based on information provided to SRK by Northland throughout the course of SRK's investigation, which in turn reflect various technical and economic conditions at the time of writing.

This report includes technical information, which requires subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK is not an insider, associate or affiliate of Northland, and neither SRK nor any affiliate has acted as advisor to Northland or its affiliates in connection with the Project. The results of the technical review by SRK is not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

SRK was heavily reliant upon information and data provided by Northland. However, SRK has, where possible, verified data provided independently, and has undertaken a site visit to review physical geological evidence for the deposit.

The Technical report contains extracts from Northland reports and communications. Other technical reports by external consultants and researchers used are referenced throughout the text.

The additional information reviewed in preparing this report has largely been provided directly by Northland and its associated consultants. SRK conducted face to face meetings with those consultants responsible for the sections of the HFS not undertaken by SRK. This included the process plant design undertaken by Jacobs UK Ltd ("Jacobs"), the infrastructure and utilities undertaken by Pöyry, the metallurgy undertaken by Pöyry, Pertti Lamberg of the Lulea University of Technology and Bo Arvidson Consulting LLC the tailings management undertaken by Pöyry (managed by SRK), the onsite and offsite infrastructure undertaken by Pöyry, the concentrate transport and logistics undertaken by Northland and Pöyry, the environmental and social impact assessment undertaken by Northland, ERM Pöyry and Ramboll, the iron ore market and price forecasting studies undertaken by Raw Materials Group and the economic evaluation undertaken by Northland.

Table 2-1: Third Party Consultants

Discipline	Responsible Company
Process Plant design	Jacobs
Infrastructure and Utilities	Pőyry
Onsite and Offsite Infrastructure	Pőyry
Metallurgy	Pőyry, Pertti Lamberg, Bo Arvidson Consulting LLC
Tailings Management	Pőyry (managed by SRK)
Concentrate Transport and Logistics	Northland, Pőyry
Environmental and Social Impact Assessment	Northland, ERM, Pőyry, Ramboll
Market Studies	Raw Materials Group

3 PROPERTY DESCRIPTION AND LOCATION

The Project comprises two iron-copper-gold deposits: Hannukainen and Kuervitikko. It is part of two elongated clusters of magnetite deposits occurring in the Pajala and Kolari Ore Districts, located in Sweden and Finland respectively (Figure 3-1). The two districts are located on either side of the Muoniojoki River, marking the international boundary between the two countries. To date, as many as 30 magnetite deposits have been identified within an area of some 1,600 km^2 (40 by 40 km), all located within Northland's exploration claim areas.

The Project area is located 25 km northeast of the municipal centre of Kolari in Finnish Lapland. The closest major city is Rovaniemi, Finland; some 170 km southeast of Kolari with 40,000 inhabitants (see Figure 3-1). The Project area can be located on general sheet line system (Yleislehtijako) sheets 271410 and 271411, 1:20,000 series.

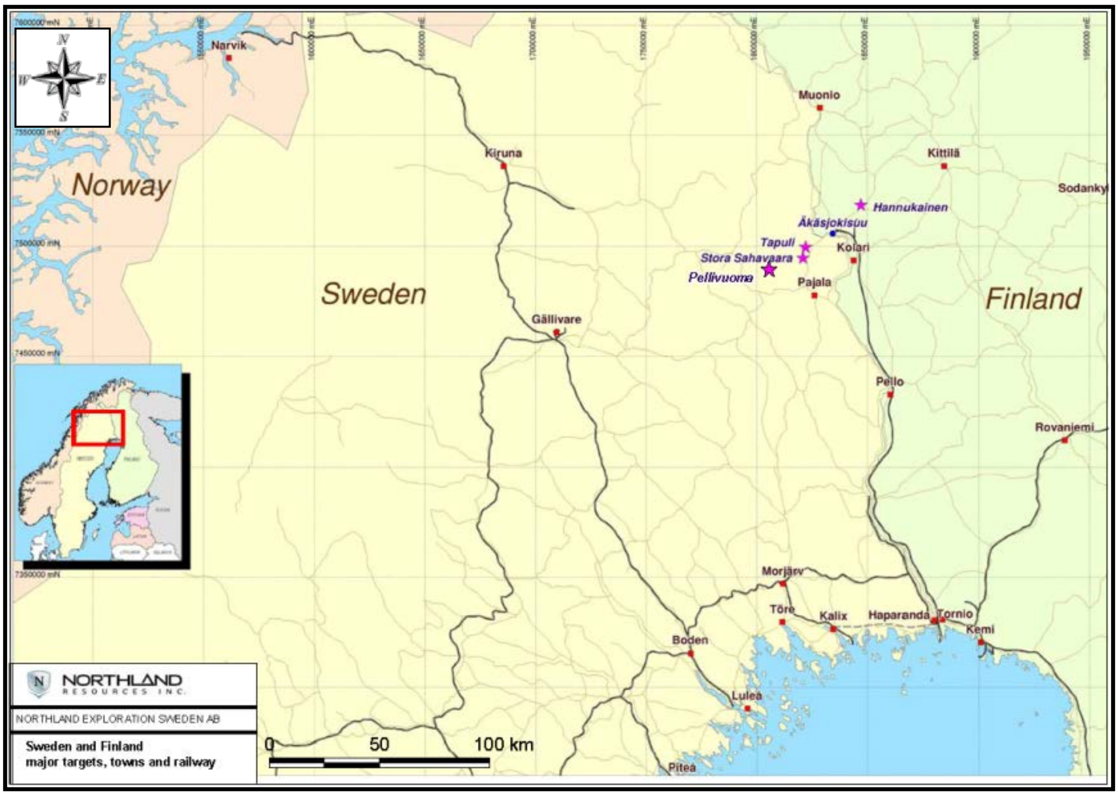


Figure 3-1: Project location (Source: Northland October 2010)

3.1 Property Description and Ownership

The Project is located within the Northland's exploration claims and the applied Mining permit (concession) as shown in Figure 3-2. These exploration claims and applied mining permit are 100% owned by Northland Mines Oy, a wholly-owned subsidiary of Northland Resources S.A.

Northland's applied Mining permit comprises following granted exploration claims:

- At the Hannukainen site:
  - 8126/1-18 and 25-29 (Claim names Hannukainen 1-18 and 25-29)
  - 8426/1-3 and 7-9 (HAN 44-46 and 50-52)
  - 8680/1, 2, 4-8 (HAN 57, 58, and 60-64).

- At the Rautuvaara site:
  - o Claims 8124/4 and 6-18 (Kolari 18 and 20-32)
  - o Applied ore prospecting permit ML2012:0201-01H, which is extension for granted claim 8373/1 according the new Mining Act (claims expired 2013-01-12).

Finish Safety and Chemical Agency, Tukes has been the surveillance and permit consideration authority in Finland as of 1 July 2011. Tukes decides on the applications filed for permits and rights, and runs the mining register in Finland. The matters pending with the Ministry of Employment and the Economy prior to 1 July 2011 subject to the old Mining Act (503/1965) shall be handled in compliance with the requirements of the old Act, unless otherwise laid down in the transitional provisions of the new Mining Act (1 July 2011).

Part of Northland's current 'granted claims', for example in Hannukainen area, has now expired (Table 3-1). Extensions have been filed for all expired claims covering the project areas and will be handled in compliance with the requirements of the old Mining Act. However, the waiting time for processing the claims has increased substantially in recent years. This is due to a large backlog of applications at the Finnish Mining authority, Tukes.

A claim is a mineral right that gives the holder (holding a prospective licence) the exclusive right to carry out exploration of extractable minerals and to exploit them after meeting certain conditions. If it can be proved that the land contains extractable minerals, then the holder may have the land appropriated for mining and has the right to exploit the minerals. The claim is valid for between two and five years from the issue date and the Mining authority can grant an extension of three years. A claim does not bestow any surface rights and the project includes no surface rights. The surface rights are owned partly by the Finnish government, partly by private companies and partly by private individuals.

Northland has filed a mining concession application on the 22 December 2010 and updated the application on the 23 June 2011 and on the 28 March 2013. The mining concession application allows Northland to continue exploration for the duration of the concession application, as described below from an extract from the Tukes website: "If the mining concession application has been filed before the claim has lapsed, then the rights to the claim will remain in force until the mining concession has been approved. This means that exploration may continue in the same manner as described in the previous claim restrictions, as long as the annual landowner compensation (claim compensation) payment is apportioned. This situation will continue until the mining concession decision is made."

Northland does not yet own the surface rights to the Project area. Land acquisition will commence once a positive HFS has been completed and company go-ahead received. The properties in the area affected (to be purchased) have all been evaluated, and negotiations/discussions with the affected landowners will begin in the fall. Part of access to land is the land planning process, which has been initiated.

Northland also has a number of other granted licences and licenses pending in the surrounding area. The list of Northland's permits within the Mining permit is shown in Table 3-1. It shows that many of the granted claims have expired; however, Northland is in the process of updating its claims and mining concessions with the Mining authority.

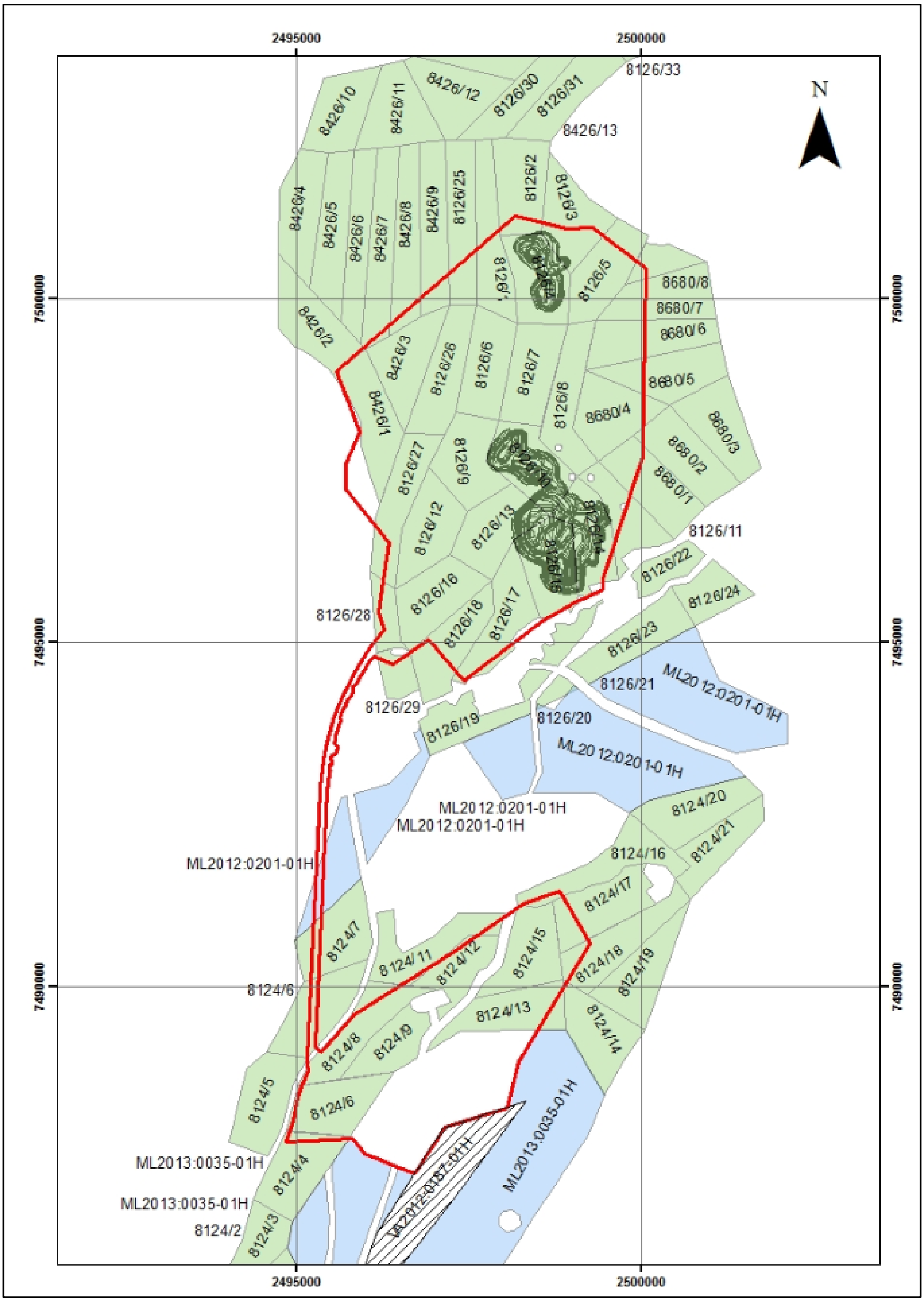


Figure 3-2: Northland exploration NEF-Granted claims and mining concession (red) in the Project area (Source: SRK Oct 2012)

Table 3-1: Northland claims statuses (expired in red) (Source: Northland December 2013)

Applied mining permit (concession)			
Register No	**Name**		**Applied**
K8126	Hannukainen and Rautuvaara		2010-12-22
At Hannukainen Site			
Claim No.	**Claim Name**	**Date of Grant**	**Expires**
8126/1	Hannukainen 1	2006-03-20	2011-03-20
8126/2	Hannukainen 2	2006-03-20	2011-03-20
8126/3	Hannukainen 3	2006-03-20	2011-03-20
8126/4	Hannukainen 4	2006-03-20	2011-03-20
8126/5	Hannukainen 5	2006-03-20	2011-03-20
8126/6	Hannukainen 6	2006-03-20	2011-03-20
8126/7	Hannukainen 7	2006-03-20	2011-03-20
8126/8	Hannukainen 8	2006-03-20	2011-03-20
8126/9	Hannukainen 9	2006-03-20	2011-03-20
8126/10	Hannukainen 10	2006-03-20	2011-03-20
8126/11	Hannukainen 11	2006-03-20	2011-03-20
8126/12	Hannukainen 12	2006-03-20	2011-03-20
8126/13	Hannukainen 13	2006-03-20	2011-03-20
8126/14	Hannukainen 14	2006-03-20	2011-03-20
8126/15	Hannukainen 15	2006-03-20	2011-03-20
8126/16	Hannukainen 16	2006-03-20	2011-03-20
8126/17	Hannukainen 17	2006-03-20	2011-03-20
8126/18	Hannukainen 18	2006-03-20	2011-03-20
8126/25	Hannukainen 25	2006-03-20	2011-03-20
8126/26	Hannukainen 26	2006-03-20	2011-03-20
8126/27	Hannukainen 27	2006-03-20	2011-03-20
8126/28	Hannukainen 28	2006-03-20	2011-03-20
8126/29	Hannukainen 29	2006-03-20	2011-03-20
8426/1	HAN 44	2009-02-11	2014-02-11
8426/2	HAN 45	2009-02-11	2014-02-11
8426/3	HAN 46	2009-02-11	2014-02-11
8426/7	HAN 50	2009-02-11	2014-02-11
8426/8	HAN 51	2009-02-11	2014-02-11
8426/9	HAN 52	2009-02-11	2014-02-11
8680/1	HAN 57	2010-10-05	2015-10-05
8680/2	HAN 58	2010-10-05	2015-10-05
8680/4	HAN 60	2010-10-05	2015-10-05
8680/5	HAN 61	2010-10-05	2015-10-05
8680/6	HAN 62	2010-10-05	2015-10-05
8680/7	HAN 63	2010-10-05	2015-10-05
8680/8	HAN 64	2010-10-05	2015-10-05
At Rautuvaara Site			
Claim No.	**Claim Name**	**Date of Grant**	**Expires**
8124/4	Kolari 18	2006-03-13	2011-03-13
8124/6	Kolari 20	2006-03-13	2011-03-13
8124/7	Kolari 21	2006-03-13	2011-03-13
8124/8	Kolari 22	2006-03-13	2011-03-13
8124/9	Kolari 23	2006-03-13	2011-03-13
8124/10	Kolari 24	2006-03-13	2011-03-13
8124/11	Kolari 25	2006-03-13	2011-03-13
8124/12	Kolari 26	2006-03-13	2011-03-13
8124/13	Kolari 27	2006-03-13	2011-03-13
8124/14	Kolari 28	2006-03-13	2011-03-13
8124/15	Kolari 29	2006-03-13	2011-03-13
8124/16	Kolari 30	2006-03-13	2011-03-13
8124/17	Kolari 31	2006-03-13	2011-03-13
8124/18	Kolari 32	2006-03-13	2011-03-13
ML2012:0201-01H	Rautu 1-9	Applied 2012-11-14	

3.2 Required Labour

Northland has the human resources presently to plan and design the mine. Should the HFS be positive, a company decision to proceed be made and relevant permits be obtained, Northland would commence a recruitment process to man the future operation.

3.3 Additional Permits and Payments

The mineral rights to the Project are entirely owned by Northland.

Although Northland has the right to cut down trees, build access roads, and carry out bulk sampling on the land throughout the claim, it is responsible for damage to the forests, land and existing infrastructure and is required to compensate the land owner for any damages.

A water permit is required for drainage purposes, and a permit for bulk sampling is also required. Additional environmental permits will be required before commencement of operations.

4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

4.1 Accessibility

Rovaniemi and Kittilä airports are both located within reasonable driving distance from Kolari. There are several scheduled daily flights between Helsinki and both airports. The Project is located on the northern side of secondary highway 940. Local access roads are in good condition since the deposit has been in production relatively recently.

4.2 Climate

Finland belongs wholly to the temperate coniferous-mixed forest zone with cold, wet winters, where the mean temperature of the warmest month is no lower than 10°C and that of the coldest month no higher than -3°C, and where the rainfall is moderate in all seasons. The monthly averages are +15ºC in July and -13ºC in January.

The mean temperature in Finland is several degrees higher than of other areas in these latitudes, such as Siberia and south Greenland. The temperature is raised by the Baltic Sea, inland waters and, above all, by airflows from the Atlantic, which are warmed by the Gulf Stream.

4.3 Local resources

Rovaniemi, the administrative centre of Finnish Lapland, is located 200 km southeast of Hannukainen (Figure 3-1). Northland has located its administrative office to Rovaniemi because of its proximity, size and availability of services which include a regional technical centre of the Geological Survey of Finland ("GTK") and its analytical laboratory. Northland also maintains field offices in Pajala and Kolari.

The towns of Pajala in Sweden and Kolari in Finland provide most of the support services for the Project. They have basic administrative offices for small towns as well as healthcare centres, schools, food outlets, etc. Kolari has historically been a mining community serving the old iron mines at Hannukainen and Rautuvaara as well as a limestone quarry and cement factory at Äkäsjokisuu, where Northland's core processing facility is located. The regional industrial base is now dominated by small businesses involved in forestry, agriculture and manufacturing. Several hotels, shops and restaurants provide accommodation for the influx of tourists visiting Lapland.

4.4 Infrastructure

Infrastructure in the area is good, in some places excellent. This is due the previous mining and other industrial operations at the area. All the major and secondary highways at the area are paved roads. There is also a good network of private gravel roads and gravel sealed forestry roads. These roads are easy to use for accessing drilling sites with heavy equipment. The Project site is connected to the national rail system with still manoeuvrable side track that has been used to transport iron ore products and cement to the south of Finland. The railhead located at the old Rautuvaara mining site is 10 km to the south of Hannukainen. Along the railroad system, there is an access to all major ports in Finland. Still operational, high voltage power lines are running to Rautuvaara mine site a few kilometres southwest of Hannukainen.

4.5 Physiography

The landscape of the Kolari area has been sculpted by the most recent glaciations spanning a period between three and ten thousand years ago. Smoothed by erosion and quaternary deposits, the area is composed of undulating glacial moraine and sand hills typically covered with pine and spruce forests. Large mires usually cover the lowlands between the dry hills. Small creeks, ponds and lakes are common. The Äkäsjoki River runs just south of the Hannukainen area into the Muonio River at the border of Finland and Sweden. Outcrops are sparse covering roughly 2% or less of the Kolari area.

5 HISTORY

5.1 Early History and Rautaruukki

5.1.1 Hannukainen

The Kolari region has been known for its iron ore resources for centuries. First historical records indicate that in the late 17th century, the Juvakaisenmaa Fe deposit provided ore feed to the Köngäs ironworks in Pajala. During the Second World War, Vuoksenniska Oy carried out Fe exploration in the area, which was continued by Suomen Malmi Oy from 1956–1960. Patents and claims were transferred to Otanmäki Oy in 1960. Otanmäki continued the feasibility studies of the nearby Rautuvaara deposit until 1967 when Otanmäki Oy was amalgamated with Rautaruukki. From late 1969 to early 1970, Rautaruukki re-evaluated the early plans to exploit the Rautuvaara Iron deposit and a formal decision to open the Rautuvaara mine was made in May 1970; mining commenced in 1975. Meanwhile, Rautaruukki continued exploration in the surrounding area and in 1974 exploration was focused in the Hannukainen area.

Ground magnetic surveys led to the discovery of several ore bodies, named as Kuervaara, Laurinoja, Lauku, and Vuopio. The overburden thickness and stripping ratio were most favourable at the Kuervaara ore body, and open pit mining was started in May 1978. Ore was hauled 10 km to the Rautuvaara mine site to provide additional feed for the Rautuvaara plant. The most valuable ore body by its in situ value was, however, at Laurinoja. The decision to exploit Laurinoja was made by the Rautaruukki in June 1981, with a flotation plant built at Rautuvaara, and commenced the production of Cu-concentrate from Hannukainen.

5.1.2 Kuervitikko

The Kuervitikko deposit was also first found with ground magnetic surveys conducted by Rautaruukki in 1974. Two separate magnetic anomalies were located, which were investigated with 7 drillholes between 1974 and 1977. Rautaruukki returned to the site between 1986 and 1987 when the Rautuvaara copper plant was operating; and drilled 31 new holes to investigate the possibilities for open-pit mining. GTK re-sampled and re-assayed 32 of these holes between 1992 and 1993.

5.1.3 Mining

Between 1978 and 1990, Rautaruukki mined approximately 4.5 Mt of iron ore from two open pits from the Laurinoja and Kuervaara Zones on the Property. After Rautaruukki left the area, GTK continued its investigations.

Between 1990 and 1995, the Rautuvaara mill and Hannukainen Deposit were leased to Outokumpu Mining OY, who mined some additional 0.45 Mt of magnetite ore from Laurinoja open pit, and processed ore from its other deposits (such as Saattopora and Juomasuo) in the Rautuvaara plant.

5.2 Polar Mining

From 2003 to 2005, the deposit area was held by Polar Mining - Northland acquired the claims in the area in 2005 by map staking. SRK was not provided with any additional information on Polar Mining.

5.3 Northland

Northland became active in the area in February 2005. The operation started with data collection and compilation, followed by re-logging and re-assaying of historical core in 2005-2006. In summer 2006, the Company performed a geological mapping programme covering the Hannukainen area. An extensive drilling programme covering all the five targets of the deposit (Kuervaara, Vuopio, Laurinoja, Lauku and Kivivuopio) began in May 2006. Since then, 553 holes have been drilled throughout the area with the total length of drill core summing up to 75,500 m. Ground magnetic and gravity measurements in six profiles (16,130 m) were completed by Suomen Malmi Oy upon request from Northland across the whole Hannukainen target area in 2007.

Bench-scale metallurgical test work was carried out in 2007 at SGS facilities in Lakefield, Ontario, Canada (“SGS Lakefield”) under the supervision of Corus Consulting. A total of 70 m of drill core weighing approximately 700 kg was collected from two diamond drillholes located next to drillhole HAN06009 at the centre of the Laurinoja mineralisation. The results confirmed that both a high-grade iron product and Cu+Au concentrate could be produced.

Further metallurgical drilling was conducted in 2008, 2009, 2010, and 2011. Several metallurgical testwork reports have been produced using this data, including a study in 2011 by Bo Arvidson Consulting LLC.

Additional metallurgical testwork has been completed during 2012 by the Lulea University of Technology with the work programme being managed by Dr Pertti Lamberg (Professor in Geometallurgy).

5.4 Previous Mineral Resource Estimates

SRK has not undertaken a review of the historical Mineral Resource Estimates and can therefore not comment on the validity of the work undertaken.

5.4.1 Micon 2007 MRE

An NI43-101 compliant Mineral Resource Estimate for the Project was prepared by Micon International Limited (“Micon”) in 2007. The Estimate was included in the 2007 WGM report entitled “Mineral Resource Estimate for the Hannukainen Deposits, Kolari area, Finland”. The MRE was based on 340 drillholes completed before 2007.

Micon used cut-off grades of 10-25% Fe for low grade Fe units, >25% Fe for high grade Fe units, and a Cu cut-off of 0.1% Cu for two Cu domains in Laurinoja. Kuervitikko was not estimated.

Micon created a sub-blocked model with parent blocks 10 x 10 x 2 m and utilised true inverse distance cubed as a means for grade interpolation.

Measured Mineral Resources were defined as those portions of the mineralised area which is drilled at an interval of 50 m by 50 m with a minimum of three samples. Each block has a restriction to use only one sample per drillhole and estimated with a minimum of two octants. Indicated Mineral Resources were defined as those portions of the mineralised area which is generally drilled on a grid of 100 m by 100 m. The blocks were estimated with a minimum of three samples, utilizing a maximum of one sample from a drillhole. Inferred Mineral Resources were defined as those portions of the mineralised area which is drilled at a spacing of 200 m by 200 m.

In total, for the Hannukainen deposit, Micon report 84.6 Mt of Measured and Indicated Mineral Resources averaging 34.6% Fe, 0.2% Cu and 93 ppb Au (0.093 g/t). Micon also report 81.8 Mt of Inferred Mineral Resources. Table 5-1 shows the 2007 Micon Mineral Resource statement.

Table 5-1: Micon 2007 Mineral Resource Statement

Zone		Measured				Indicated				Total				Inferred			
		Tonnes ('000)	Grade			Tonnes ('000)	Grade			Tonnes ('000)	Grade			Tonnes ('000)	Grade		
			Fe %	Cu %	Au (PPb)		Fe %	Cu %	Au (PPb)		Fe %	Cu %	Au (PPb)		Fe %	Cu %	Au (PPb)
Laurinoja	Main	32,094	39.4	0.32	178	3,936	36.8	0.23	124	36,029	39.2	0.31	172	3,316	36.4	0.26	170
	Cu-Lower	1,304	23.2	0.29	91	59	22.7	0.20	55	1,363	23.1	0.29	90	67	39.6	0.13	21
	Cu-Upper	2,005	17.8	0.28	103	484	17.4	0.25	147	2,489	17.7	0.27	111	63	16.8	0.15	114
	Low Grade	2,371	17.7	0.07	25	94	16.9	0.07	24	2,464	17.7	0.07	25				
Kivivuopio	Fe-ore													37,807	36.3	0.14	36
Vuopio	Fe-Ore					8,368	39.6	0.11	58	8,368	39.6	0.11	58	30,571	37.8	0.12	27
	Low Fe-Ore					2,695	23.0	0.05	22	2,695	23.0	0.05	22	5,407	17.7	0.05	9
Kuevaara	Fe-Ore	7,992	32.1	0.12	33	9,587	32.5	0.10	19	17,579	32.3	0.11	25				
	Low Fe-Ore	1,145	18.8	0.07	18	3,779	20.8	0.07	15	4,924	20.3	0.07	16				
Lauku	North	3,993	36.6	0.11	14	1,430	33.3	0.10	9	5,423	35.7	0.11	13	4,349	37.3	0.12	27
	South	2,235	41.4	0.12	30	1,028	44.7	0.10	9	3,263	42.4	0.11	23	46	45.7	0.12	18
	Total	**53,138**	**35.6**	**0.25**	**122**	**31,458**	**32.9**	**0.11**	**43**	**84,597**	**34.6**	**0.20**	**93**	**81,625**	**35.7**	**0.13**	**36**

Note: NI43-101 Compliant Mineral Resource, using a 15% Fe cut-off grade.

5.4.2 WGM 2010 MRE

A Mineral Resource Estimate for the Project was prepared by Watts, Griffis and McOuat ("WGM") and reviewed by George H Wahl & Associates Consulting ("G Wahl") in 2010. The Estimate was included in the 2010 WGM report entitled "Technical Report on the Mineral Resource Estimates and Preliminary Assessment of the Project, Finland, for Northland S.A.".

The final Mineral Resource Estimate follows NI 43-101 guidelines and definitions with the Mineral Resources classified as Measured, Indicated and Inferred Mineral Resources.

G Wahl developed the solids, which were then utilised by WGM (adding minor adjustments) for the MRE. G Wahl used a 12% Fe_calc formula or 2500 ppm Cu cut-off to define the domain boundaries, where Fe_calc = %Fe Total – (1.5*S). This resulted in separate solids being created for Kuervaara/Vuopio, Lauku/Laurinoja, Kivivuopio and Kuervitikko.

WGM created a block model with 15 x 15 x 5 m block dimensions, and utilised true inverse distance squared as a means for grade interpolation.

WGM utilised estimation criteria in order to classify the model, based on two stages. In stage one, blocks populated in the first and second estimation pass (with a search ellipse of <100 x 100 x 50 m) were classified as Indicated, and blocks populated in the third pass were classified as inferred. The second stage of classification upgraded blocks within 50 m of the Northland drilling from Indicated to Measured Mineral Resources.

In total, for Hannukainen and Kuervitikko, WGM reports 136 Mt of Measured and Indicated Mineral Resources averaging 32% Fe Total, 0.17% Cu and 0.085 g/t Au. WGM also reports 20 Mt of Inferred Mineral Resources. This was based on a Whittle optimisation study for Hannukainen and a cut-off grade of 15% Fe Total for Kuervitikko. In addition to the Mineral Resources stated above, WGM also reported 88 Mt of Inferred Mineral Resources grading 31.7% Fe Total, 0.13% Cu and 0.041 g/t Au below the Whittle shell at Hannukainen. This was not restrained by a cut-off grade or any economic parameters; however, WGM stated that these Mineral Resources may become economic by deepening an open pit or by underground extraction.

Table 5-2 shows the 2010 WGM Mineral Resource statement.

Table 5-2: WGM 2010 Mineral Resource Statement

HANNUKAINEN				
CLASSIFICATION	MT	FE(%)	CU(%)	AU(g/t)
MEASURED	101	33.80	0.17	0.067
INDICATED	9	35.00	0.13	0.023
MEAS+IND	**110**	**33.90**	**0.17**	**0.064**
INFERRED (above Whittle shell)	1	31.30	0.09	0.020
INFERRED (below Whittle shell)	88	31.70	0.13	0.041
KUERVITIKKO				
CLASSIFICATION	MT	FE(%)	CU(%)	AU(g/t)
MEASURED	-	-	-	-
INDICATED	26	23.80	0.17	0.175
MEAS+IND	**26**	**23.80**	**0.17**	**0.175**
INFERRED	19	21.70	0.15	0.165

Note: Whittle shell applied to Hannukainen, and a 15% Fe Total cut-off grade applied to Kuervitikko.

6 GEOLOGICAL SETTING AND MINERALIZATION

6.1 Regional Geology

The Fennoscandian shield is the largest exposed Precambrian crustal domain in Europe, covering most of Finland and Sweden as well as significant parts of north western Russia. The bedrock of the northern part of the Fennoscandian shield consists of Archean basement, Palaeoproterozoic greenstone and schist belts (Central Lapland greenstone belt, CLGB), ca. 1.9 Ga granulite belt and Svecofennian 1.93-1.77 Ga granitoids (Figure 6-1). The CLGB was formed during prolonged stages of rifting of the Archaean craton, with sedimentation and magmatism in intracratonic and cratonic margin rift settings between 2.5 and 1.9 Ga, and was subjected to multiphase deformation and metamorphism during orogenic events between 1.92 and 1.77 Ga.

Several crustal-scale structures are known in the area, one of which is the NNE-SSW striking Pajala Shear Zone (“PSZ”) which straddles the border between Finland and Sweden (Figure 6-2). The PSZ is up to 50 km wide and at least a 150 km long crustal-scale shear zone system that outline the boundary between the Norrbotten craton in the west and Karelian craton in the east. The structural lineaments that comprise the PSZ were initially formed during the continent-continent collision of the Norrbotten and Karelian cratons in 1.89-1.86 Ga and were subsequently reactivated during later orogenic events between 1.83 and 1.79 Ga.

The peak metamorphic conditions in the northern Fennoscandian shield vary between greenschist and granulite facies conditions. Three ductile deformation stages and subsequent brittle stages have been distinguished from the area.

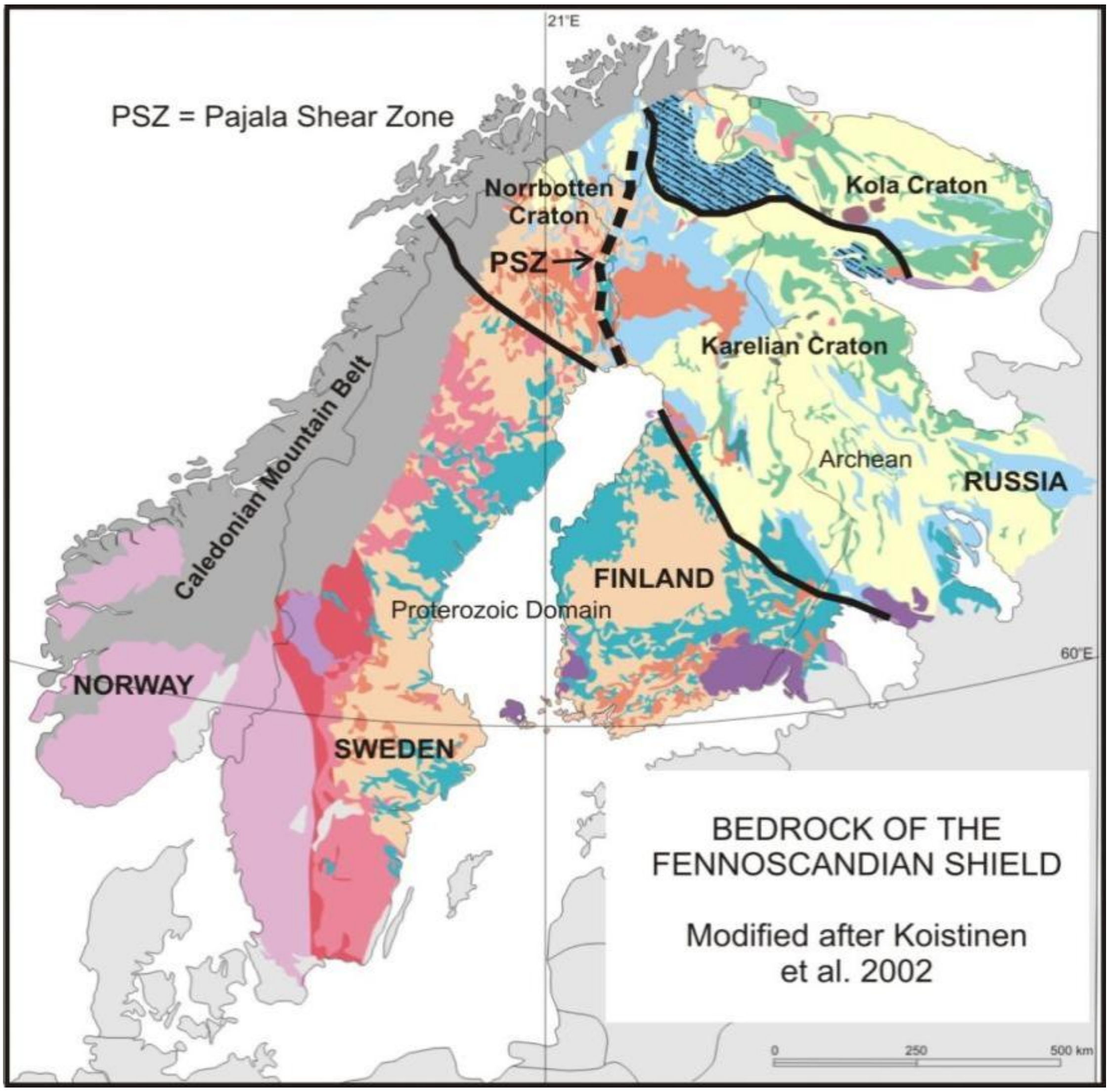


Figure 6-1: Fennoscandian Shield (Source: Northland October 2010)

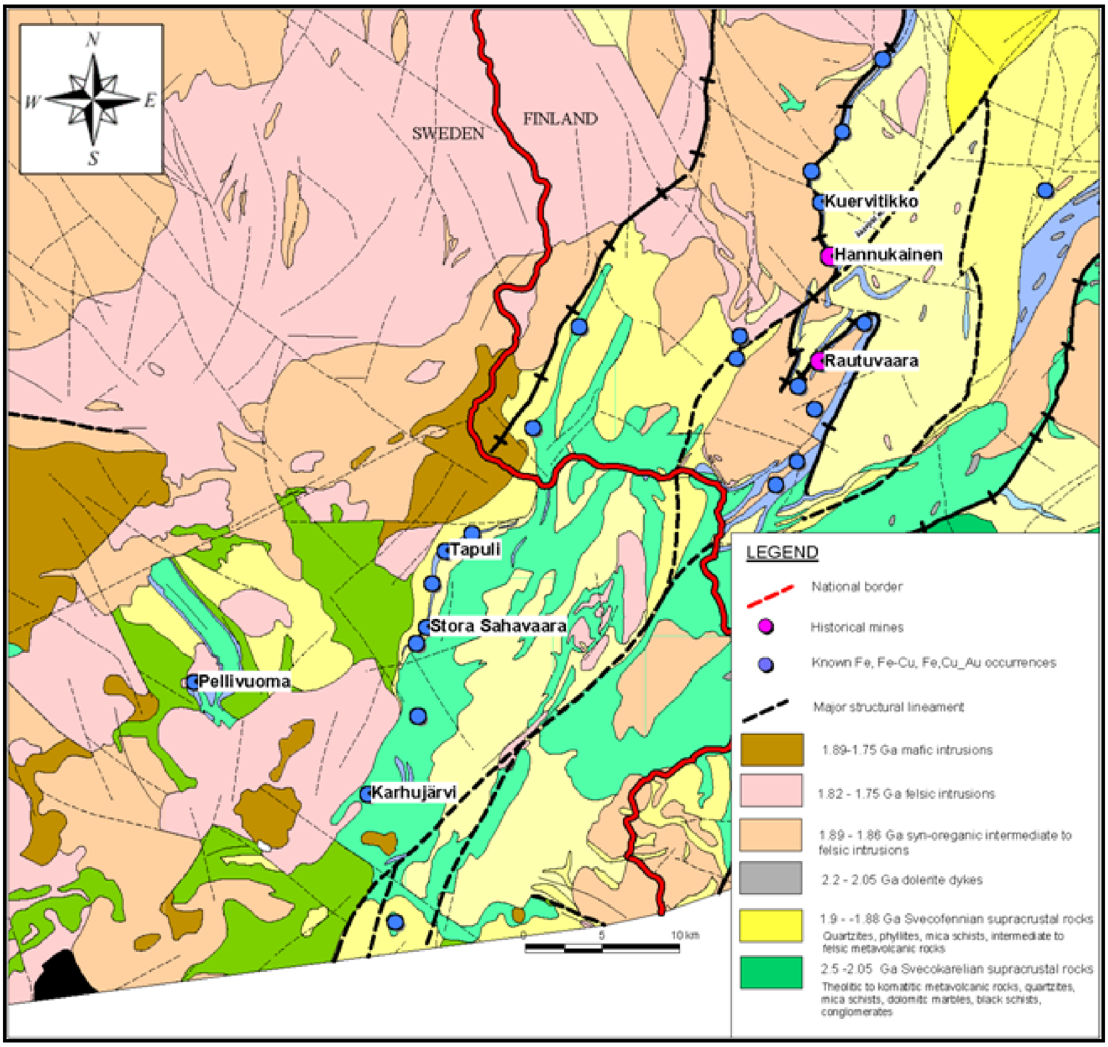


Figure 6-2: Regional Geology of the Kolari-Pajala area (Source: Northland October 2010)

6.2 Local Geology

The Hannukainen deposit displays several features typical of IOCG deposits, which have recently been widely recognized as a global class of ore deposits. The location of the mineralization is strongly structurally controlled; with Fe-Cu-Au ore lenses located on a bend of a mineralized thrust zone. To the south of the deposit, the Äkäsjoki strike-slip shear zone cross-cuts the thrust zone. The reverse thrust at Hannukainen dips 20-30° to the west, where strong lineations are developed as a result of the thrusting. At Hannukainen, the lineation plunges about 30° towards the southwest, whereas the foliation follows the direction and dip of the thrust structure.

The hanging wall host rocks consist of Svecofennian Haparanda suite monzonite and diorite, whereas the footwall package comprises mainly supracrustal units, mainly mafic metavolcanic rocks (amphibolites). In association with the amphibolite units, there are thin beds of quartz-feldspar schist, which is commonly pyrrhotite-bearing and contains variable amounts of graphite. A sequence of mica gneiss occurs below the amphibolite unit which overlies quartzite, which is located at the base of many of the drillholes. Pegmatite, aplite and granite dykes with varying thicknesses commonly cross-cut the whole sequence.

In general, all the rocks in the sequence, excluding the youngest granites, are intensely altered. Alteration at Hannukainen displays a deposit-scale zoned pattern where three different alteration zones can be distinguished around the ore bodies in both the hanging wall and footwall host rocks. The alteration assemblages vary somewhat depending on the primary lithology, but the general pattern is that albite ±scapolite are the dominant alteration minerals in distal zones, biotite and K-feldspar dominate the assemblage in intermediate zones and the proximal alteration zone is characterized by additional clinopyroxenes, with varying amounts of magnetite, amphibole and calcite. The iron ore itself displays similar alteration assemblages to the proximal alteration zone. Most of the sulphides occur in the iron ore and proximal alteration zones, but both iron and copper sulphides are locally abundant in all altered rocks independent of the alteration assemblages. The zoned alteration pattern may repeat itself at different scales throughout the sequence. Some zones, particularly the intermediate zone, may be poorly developed or even missing. Similarly, proper magnetite skarn core is locally missing or it is very thin and pinches and swells.

The Kuervitikko deposit alteration assemblages define a similar zoning as described for Hannukainen, above. The most significant differences are that at Kuervitikko, K-feldspar and biotite are less common in hangingwall distal alteration zone, and there is an albitite (±quartz) unit between the diorite and clinopyroxene-amphibole skarn. In addition, amphibole is slightly more abundant as a proximal alteration mineral than at Hannukainen. Also, sulphides appear to be more abundant in the distal alteration zones, especially in the albitite.

Table 6-1 gives an overview of the physical dimensions of the mineralized areas at Hannukainen.

Table 6-1: Physical dimensions of the mineralized areas at Hannukainen

Mineralized Lens	Length (m)	Down-dip width (m)	Max Thickness	Ave Thickness (m)
Kuervaara / Vuopio	1900	800	50	10-30
Laurinoja / Lauku / Kivivuopio	2100	2000	50	10-30
Kuervitikko	1200	500	50	10-20

Figure 6-3 shows an interpreted geological cross-section from Northland geologists. The lithological units and mineralization horizons are depicted.

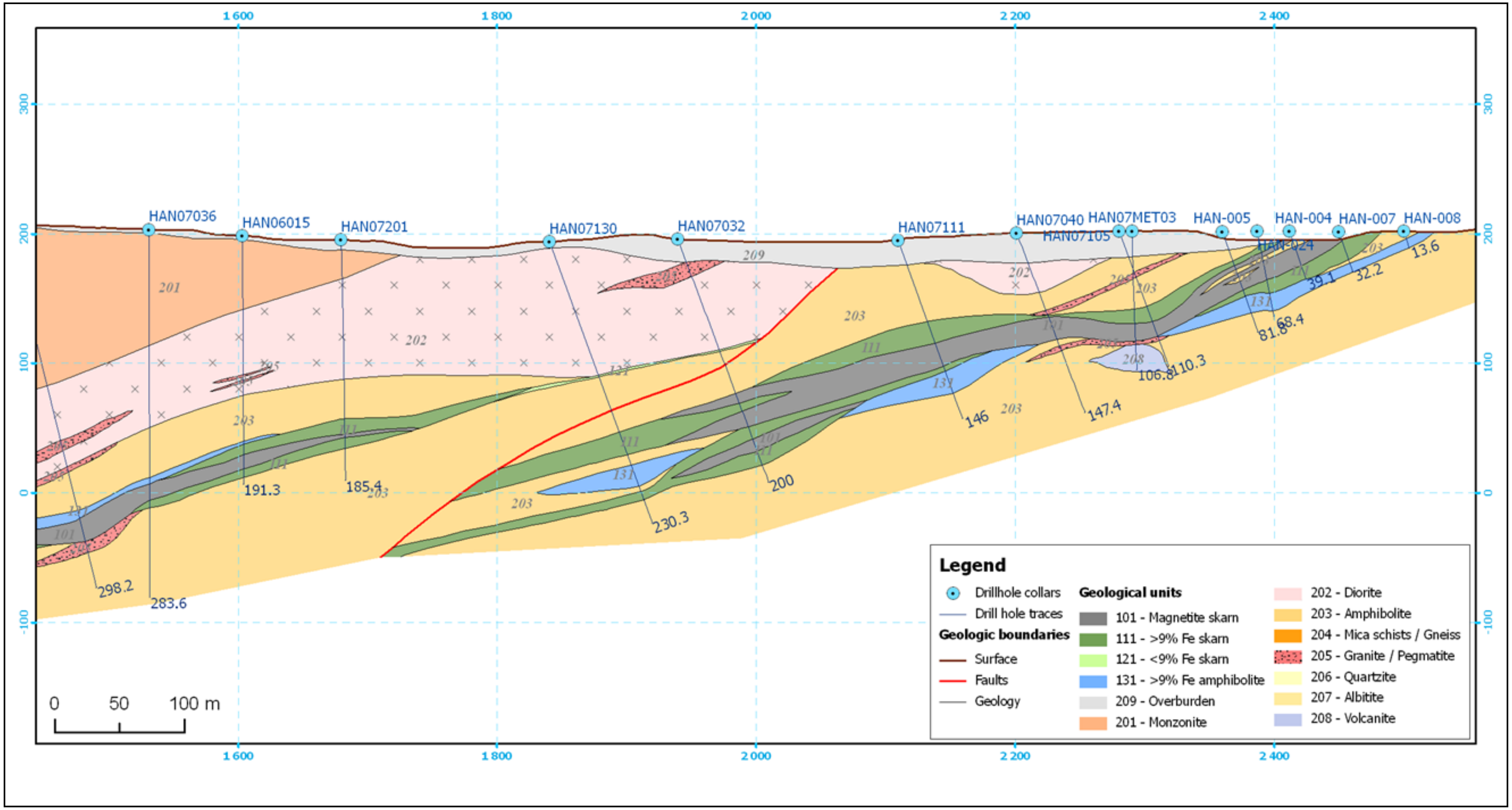


Figure 6-3: **Cross-section XS6300 showing the geological units and mineralization (Source: Northland October 2010)**

6.3 Mineralization

The mineralized lenses are hosted by skarns formed through metasomatic alteration near to the contact zone between the supracrustal metavolcanic rocks and monzonite-diorite intrusions. The dominant host rock for the Fe-mineralisation is clinopyroxene skarn with varying amounts of amphibole, magnetite, biotite, albite, scapolite, garnet, calcite and sulphides. Main oxide mineral is magnetite and pyrite, pyrrhotite and chalcopyrite are the dominant sulphide phases; minor amounts of molybdenite occur in association with chalcopyrite in places. Native gold has been detected in silicate gangue, chalcopyrite and magnetite. Locally, amphibole-, scapolite-, or garnet-dominated horizons occur within the skarn and the magnetite-rich lenses referred as magnetite skarns or iron ore. Cu-Au-mineralization is usually hosted by magnetite skarns and in lesser amount by surrounding clinopyroxene skarns.

Three different textures found at the Project are shown below. Figure 6-4 shows a piece of drill core from the Project, showing massive magnetite adjacent to massive chalcopyrite.

Figure 6-4: Drill core showing massive magnetite and chalcopyrite (Source: SRK February 2011)

Figure 6-5 shows a more disseminated magnetite distribution and a veined chalcopyrite/pyrite texture within a skarn host rock. Figure 6-6 shows a patchy and disseminated texture within a skarn host rock.

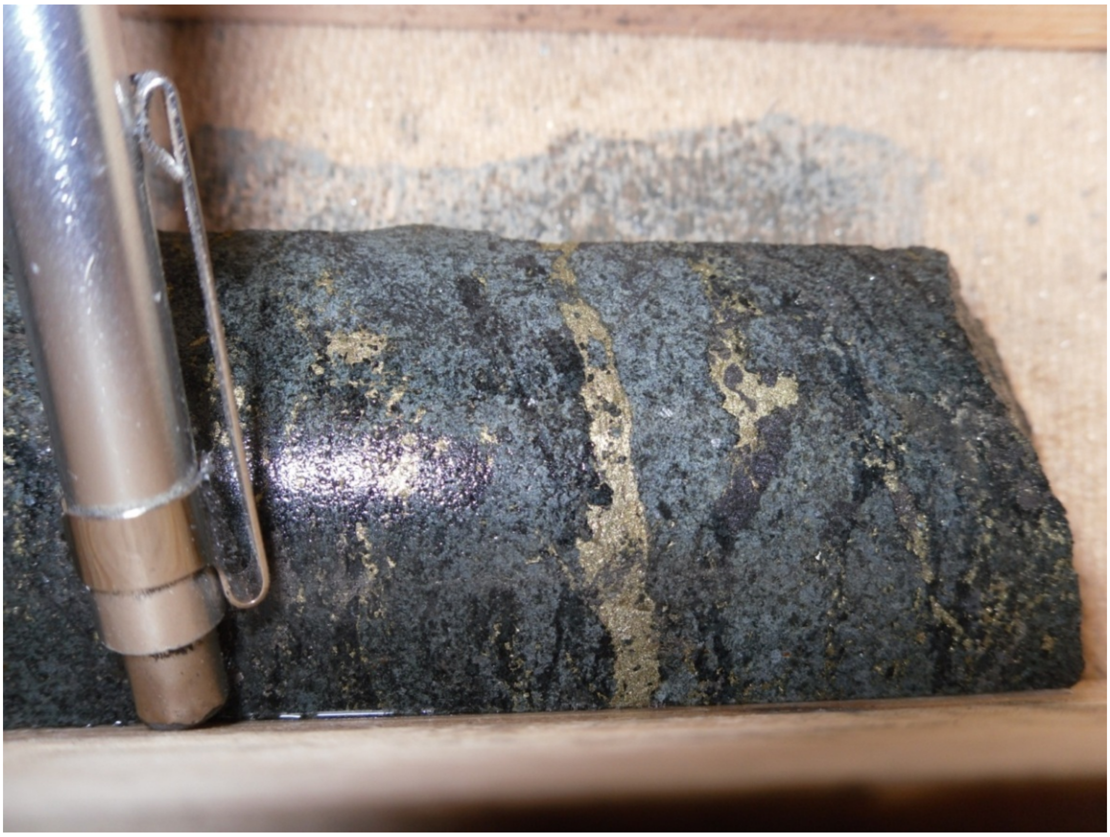


Figure 6-5: Drill core showing veins of magnetite and chalcopyrite/pyrite in skarn (Source: SRK February 2011)

Figure 6-6: Drill core showing patchy and disseminated magnetite and chalcopyrite/pyrite in skarn (Source: SRK February 2011)

7 DEPOSIT TYPES

7.1 Hannukainen Deposit Type

Historically, the deposits of the Pajala-Kolari Iron Ore field have been interpreted as examples of a broad group of magnetite-dominated, Ca-Mg- and Mg-silicate skarn-hosted deposits, which occur throughout Northern Sweden. These deposits are located within a Karelian volcano-sedimentary domain and are generally referred to as “skarn iron ores” and as such may be metamorphosed banded iron formations or some other similar type of syngenetic iron formation (for example, Frietsch, 1997). The deposits are comprised of several bands, or lenses of magnetite skarn, concordant with the surrounding skarn and sedimentary host rocks with associated copper and gold mineralisation. The Project displays several features typical of IOCG deposits:

- the brecciated textures commonly observed in the mineralised zones indicate that the magnetite is epigenetic in origin;
- spatial correlation to the shear zone suggests the possibility that the mineralisation is structurally controlled; this theory is also put forward by Johnson (2010);
- although syngenetic iron formations are known in the Karelian sequence further west of Pajala area, no examples are known in the Pajala-Kolari district; and
- available isotopic dating of the mineralisation from the Project suggests that mineralisation formed relatively "late", these ages thus being more consistent with epigenetic than syngenetic origin.

In the surrounding area, the Tapuli, Sahavaara and Pellivuoma deposits (also owned by Northland) are devoid of any significant Cu or Au mineralisation; however, they display some other features of an IOCG-type of system. One typical feature for the known IOCG-belts (for example, Cloncurry in Australia) is that numerous epigenetic magnetite and/or hematite deposits occur amongst the Fe-Cu-Au deposits.

7.2 Regional Deposits

7.2.1 Significant Fe and Cu deposits in Northern Fennoscandia

The economically most important iron deposits in Northern Fennoscandia are the Kiruna and Malmberget magnetite-apatite deposits, both currently operated by Luossavaara-Kiirunavaara AB (“LKAB”).

The Kiruna deposit, located 150 km northwest from the Pajala area, was discovered in 1696 and has been mined on a regular basis since 1900. The ore is currently mined underground.

The Kiruna iron deposit consists of a 5 km long, up to 100 m thick, steeply dipping mineralisation with the sole mineral of economic interest being magnetite. The mineralisation is hosted by felsic volcanic rocks within a Svecofennian supracrustal sequence known as the Kiruna Porphyry Group (Offerberg *et. al.,* 1967). A recent genetic model of the Kiruna deposit suggests that the magnetite was crystallized directly from an iron-phosphorus rich magma (Nyström & Henriques, 1994). It has been proposed that the Kiruna-type magnetite-apatite deposits would have been formed as an end-member of the IOCG class of iron ores (Hintzman *et. al.,* 1992).

Another currently operating iron mine is the Malmberget magnetite-apatite deposit located at Gällivare in Sweden, about 100 km northwest from Pajala. The Malmberget deposit consists of some 20 ore bodies over an underground area about 5 x 2.5 km. The mineralisation type is the same as found at Kiruna. The Malmberget deposit is hosted by highly metamorphosed felsic volcanic rocks which are considered to be metamorphosed equivalents of the Kiruna Porphyry Group rocks that host the Kiruna deposit. Seven of the known orebodies are currently being mined by LKAB.

The largest copper deposit currently in production in Europe, Aitik, is located some 18 km away from Malmberget. Aitik was discovered in 1930 and has been in production since the 1960s.

The bulk of the mineralisation at Aitik is hosted in rocks of the Svecofennian Porphyry Group and specifically altered intermediate to felsic metavolcanic and subvolcanic intrusions next to a quartz-monzodiorite of which the latter is also weakly mineralised. The deposit has been interpreted as a metamorphosed porphyry deposit (for example, Wanhainen *et. al.,* 2003).

8 EXPLORATION

SRK has reviewed a proportion of the Northland drill core located in the Northland core shed in Kolari. SRK has also visited the drill rigs used for the 2011 metallurgical drilling programme. SRK is confident that the drilling sampling methodology and quality used for the Northland drilling campaigns are appropriate for use in this study.

8.1 Drilling

Exploration drilling was carried out on the Hannukainen deposit by Northland from 2006-2012, totalling 408 exploration drillholes for 63,170 m. Northland have also drilled 8 geotechnical holes in 2011 for 1,888 m, 4 hydrological holes in 2011, and 153 metallurgical holes between 2007-2012 for 17,000 m. It was also drilled extensively by Rautaruukki between 1974 and 1986 for exploration and production purposes, totalling 293 drillholes for 33,200 m. The description of the drilling programmes is given below in section 9. Significant intervals of Fe-Cu-Au mineralisation were intercepted by this historical drilling campaign. Significant intercepts include: 34.75 m grading 45.6 % Fe Total and 1.1% Cu in drillhole LAU-170; 24.95 m grading 45.6% Fe Total and 0.6% Cu in drillhole LAU-171.

Northland's exploration programme started in 2005 immediately after property acquisition. The 2005 programme included the re-logging and re-assaying of historic drill core drilled by Rautaruukki on both the Hannukainen and Kuervitikko deposits. This core is stored at GTK's core storage library at Loppi in southern Finland. In 2005, Northland re-logged 175 of these Hannukainen drillholes and re-sampled 100 holes. For the Kuervitikko Deposit, Northland re-logged and re-sampled 28 drillholes totalling 2,247 m. Northland reports (Holma et al., 2008) that a considerable amount of this historic drill core was destroyed prior to Northland's involvement due to earlier sampling by Rautaruukki and GTK.

All Northland drilling was undertaken with WL-76 drill rods, producing 57.5 mm diameter drill core. The drilling was shared between Lapin Asbestos Oy (later Northdrill), Oy Kati AB and SMOY.

Significant intercepts from the Northland drilling programmes include: 30.2 m grading 40.6% Fe Total and 0.85% Cu in hole HAN06005; 25.7 m grading 44.1% Fe Total and 0.3% Cu; 28 m grading 39.6% Fe Total and 0.3% Cu in hole HAN08MET27.

8.2 Geophysical Surveys

8.3 Pre-Northland

Magnetic surveys were conducted in the Hannukainen area by Rautaruukki in the 1970s, which produced several anomalies, which were named Kuervaara, Vuopio, Laurinoja and Lauku. Later, more extensive magnetic surveys discovered Kuervitikko. SRK was not provided with additional data relating to these surveys and cannot comment on the quality of the data.

8.4 Northland

In May 2008, ground geophysical surveys were carried out in Kuervitikko by GeoVista. Two profiles covering 6 line-km of gravity lines and a total of 37 profiles covering 51.8 line-km of magnetic total field were measured. The evaluation of the gravity and magnetic data was performed using anomaly surface maps and 2D forward modelling. The aim of the modelling was to estimate the geometry, strike and dip of the bodies that cause the anomaly in the gravitational and/or magnetic field. The forward modelling technique used for the evaluation of both gravity and magnetic data has many degrees of freedom and the interpreted dips might reflect near surface geometry verified by drilling, even though the models extend to depths of several hundreds of metres.

The gravity data indicate a fairly smooth positive anomaly in the central part of both measured profile lines. There is a distinct regional trend in the data indicating high density rocks in the west and lower density rocks in the east. The dip from the horizontal plane of the magnetite breccia rocks is approximately 20-30° to the west and the thickness estimated at 40-80 m. The right hand rule strike of the model bodies is estimated at about N195°E, estimated from the orientation of the positive anomaly shown in magnetic total field anomaly map.

The magnetic total field data indicate a distinct positive anomaly that coincides with the positive gravity anomaly. The anomaly is elongated in the NNE-SSW direction with the length of approximately 900-1000 m and half width in the east-west direction is approximately 200-300 m. To the east of the magnetic maximum there is a distinct magnetic low and to the west the magnetic field decreases slowly with distance, which indicates that the source body/bodies dip to the west. Small density contrasts between the magnetite-breccia and the amphibolites makes the modelling results of gravity measurements uncertain. The magnetic modelling of the anomaly shows a strong correlation with the results achieved from the gravity modelling.

Northland also completed at the high-resolution deep reflection seismic experiment (“HIRE”) programme carried out by GTK. The seismic reflection survey comprised six vibroseis lines (total length of 71.7 km) and two explosion seismic lines (total length 8.7 km) and was carried out in the Hannukainen-Rautuvaara Fe-Cu-Au exploration area in March to April 2008. The results showed that the study area is characterized by strong, high-amplitude reflectivity in the uppermost 5 km. The most prominent regional structure being an extensive system of three major reflective layers which range in thickness from less than 200 m to about 1 km. The layers were correlated laterally over distances of 10 to 20 km and the reflectors form an open asymmetric antiform which has one limb dipping about 20º SW under the monzonite intrusions in the west, and the other limb in a subhorizontal position under quartzite and other metasediments in the eastern part of the survey area. The large-scale structure can be interpreted as thrusting from SW to NE and be related to a major bedrock structure in western Lapland, the Kolari Shear Zone. Thrusting has taken place in the same direction as the well-developed lineation of the area. The SW-NE Äkäsjoki shear zone is interpreted as a subvertical strike-slip fault developed in the direction of thrusting.

Reflectors in the Hannukainen area correlate with known Fe-Cu-Au deposits, and the results suggest that the strong reflectivity is due to iron bearing lithologies, sulphides, skarn and amphibolite within monzonite, diorite and metasediments. Reflectors also correlate with magnetic and electrically conductive layers.

8.5 Geochemical surveys

8.5.1 Northland

In 2008, a representative set of 2007 drill core was analysed for a more extensive suite of elements than included in the regular research package. The purpose of the task was to achieve a detailed understanding of the geological characteristics of the highly altered deposit. Another aim was to gather background information of the distribution of U, Th and Cs in the Kuervitikko deposit. Based on the evidence given by the study of high field strength elements ("HFSE"), such as Al, Ti, Zr and REE, the host rocks for mineralisation at Kuervitikko consists of both the syn-orogenic Haparanda-type intrusions and Savukoski Group supracrustal rocks, that is, hanging wall diorites and footwall amphibolites.

Locally there is evidence that mineralisation overprints the oldest generation of the 1.82-1.78 Ga pegmatite dykes. REE patterns also suggest that there is a common source for hangingwall monzonites and diorites; that the hangingwall quartz-albite rocks are in fact silicified shear zones (thrust zone); and that the footwall amphibolites represent two distinct metavolcanic types with a different source as well as different temporal and depositional environments. The ultimate source of the fluids and/or metals in the deposit is interpreted to be metasomatized mantle; indicating a possible link to carbonatites. This model could be applied to other Fe-deposits in the Kolari area as well, as they too are located at the cratonic margin and it is likely that the mantle below the cratonic boundary was metasomatized during subduction related to collision of the Norrbotten and Karelian cratons during the Svecofennian orogenic events.

9 DRILLING

9.1 Historical

Rautaruukki completed exploration drilling on the Project prior to Northland's ownership, between 1974 and 1986. Earlier drilling was reportedly carried out by SMOY and Otanmäki Oy in the 1950s and 1960s; however, no information was available for SRK to review.

The Rautaruukki drilling comprises 293 drillholes with a total meterage of 33,202 m. No core recovery data was available for this drilling period. A portion of the remaining core is held at the GTK archive in Loppi, southern Finland.

9.2 Northland 2006-2007

Northland's 2006 exploration programme consisted of a geological mapping and diamond drilling at Hannukainen. There is a lack out outcrop in the area, due to a thick cover of glacial sediments, restricting the value of the bedrock mapping and surface sampling. The drill programme consisted of 45 vertical drillholes totalling 8,745 m of core, testing the Laurinoja Zone of the Hannukainen Deposit and in-filling between historic Rautaruukki drillholes.

In 2007, Northland continued exploration at the Hannukainen deposit and also initiated exploration of the Kuervitikko deposit. At Hannukainen, more drilling was completed, ground geophysical surveys were conducted and bench scale metallurgical testwork was started at SGS Lakefield using drill core samples. Ground magnetic and gravity measurements along six profiles aggregating 16.13 line-km covering the entire Hannukainen area were completed. The Hannukainen drill programme consisted of 206 holes totalling 29,981 m. This total included three holes drilled to secure more sample material for metallurgical testwork.

Northland's 2007 programme at Kuervitikko consisted of the drilling of 38 holes totalling 4,311 m.

9.3 Northland 2008

The drilling operation conducted during 2006 and 2007 was continued with 28 drillholes totaling 4,056 m in 2008. Drilling was mostly designed to infill gaps in the existing drilling grid.

As the Company engineers were considering the option to use the existing open pit for test quarrying, four drillholes were completed at the edges of the open pit to explore the dimensions of the mineralisation as close to the pit as possible. Test quarrying operations were however later postponed.

The poorly explored south western continuum of the mineralisation was tested with few holes, in order to test the structural model where the two mineralised horizons were tectonically stacked on top of each other.

The northern extremities of the magnetic anomaly (Pöllölä target), which follows the Hannukainen thrust zone, was also tested with 9 drillholes.

In winter 2007-2008, metallurgical drilling was undertaken to produce a bulk metallurgical sample for testing at SGS. Test holes were distributed over the whole Hannukainen mineralisation, excluding the Kivivuopio deep seated ore lens. The metallurgical programme included 36 drillholes, which commenced in December 2007 and totaling 4,666 m, of which some 1,126 m of mineralised material was shipped to the SGS laboratories in Lakeland, Canada, in October 2008.

Condemnation drilling was conducted on the eastern side of the Hannukainen-Kuervitikko area, comprising 30 holes totaling 4,490 m.

All Northland drilling was undertaken with WL-76 drill rods, producing 57.5 mm diameter drill core. The drilling was shared between Lapin Asbestos Oy (later Northdrill), Oy Kati AB and SMOY.

9.4 2011 Metallurgical drillholes

Northland completed a metallurgical drillhole programme in May 2011. This comprised 117 holes totaling 12,620 m of drilling. The locations of the drillhole collars are shown in Figure 9-1. This sampling programme was designed to collect samples for metallurgical testwork and this data has been incorporated into the Mineral Resource Estimate.

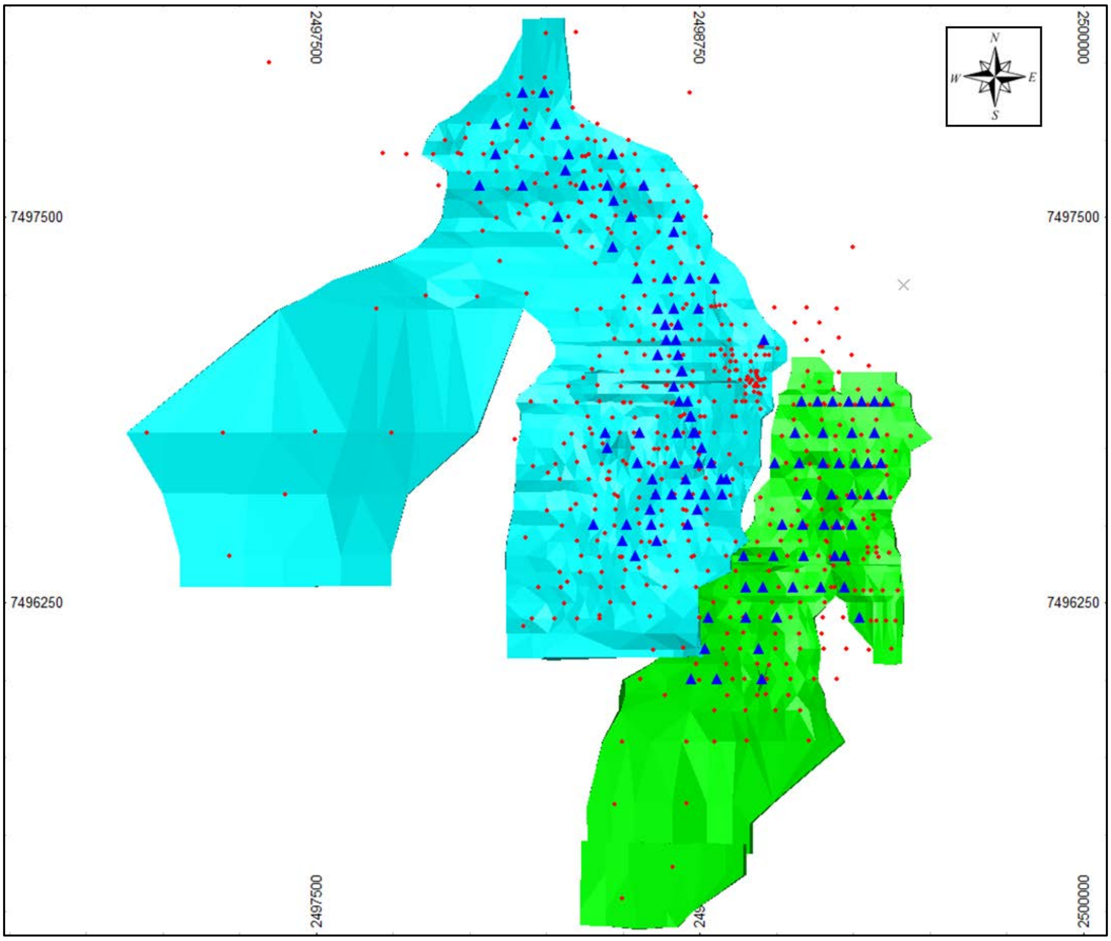


Figure 9-1: 2011 Metallurgical drillholes (blue triangles) with previous drillhole collars (red dots) and SRK Hannukainen mineralisation wireframes (Source: SRK Oct 2012)

9.5 2011-2012 Northland Deep Infill Drilling

In 2011, Northland created an exploration plan with the aim of infilling the current drilling grid at depth to test for continuity. In total, 15 deep infill holes were drilled for 5,800 m. the holes increased the confidence in the model towards the southern end of Hannukainen and in between Kivivuopio and Laurinoja.

9.6 Drilling Summary

From 1974-1986, Rautaruukki drilled 293 holes totalling 33,202 m of core. From 2006-2008, Northland drilled a total of 429 holes totalling 62,018 m of core. In addition, 30 condemnation holes were completed by Northland during the 2006-2008 campaign.

Table 9-1 provides a summary of the drilling carried out at the Project to date (excluding the recently completed 2011 metallurgical holes). The entire exploration drillhole collar dataset is shown in Figure 9-2.

Table 9-1: Drilling Summary (including geotechnical and metallurgical holes)

Period	Deposit	Holes Drilled	% of Total	Meterage
Rautaruukki	Hannukainen	244	28%	28,432
	Kuervitikko	49	6%	4,770
	Total	**293**	**34%**	**33,202**
Northland 2006	Hannukainen	42	5%	8,745
	Total	**42**	**5%**	**8,745**
Northland 2007	Hannukainen	204	24%	29,476
	Kuervitikko	38	4%	4,311
	Total	**242**	**28%**	**33,787**
Northland 2008	Hannukainen	61	7%	8,292
	Kuervitikko	84	10%	11,194
	Total	**145**	**17%**	**19,486**
Northland 2011	Hannukainen	138	16%	19,494
	Total	**138**	**16%**	**19,494**
Northland 2012	Hannukainen	2	0.2%	643
	Total	**2**	**0.2%**	**643**
	Grand Total	**862**	**100%**	**115,356**

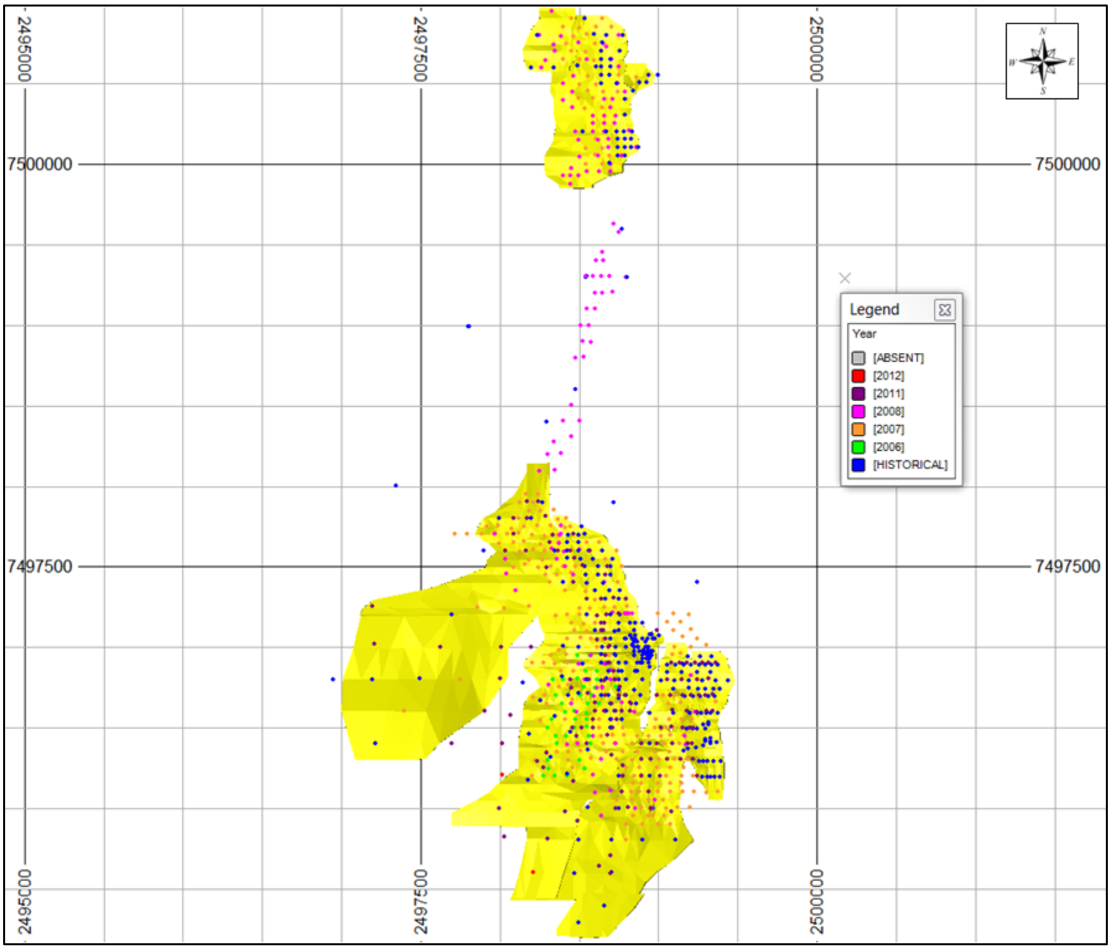


Figure 9-2: Drillhole collar locations (coloured by year drilled) with SRK mineralisation wireframes (Source: SRK Oct 2012)

9.7 Summary of Drilling Results

A total of 28,685 assays have been analysed from the 862 drillholes in the Project area which amounts to >40,000 m of samples.

As a result of the sampling and logging of the drillholes, 11 mineralisation wireframes in three areas were digitized using Datamine software. Hannukainen and Kuervitikko are two separate deposits, with a gap of >1.6 km between them containing sparse low grade mineralisation. Hannukainen is split into two main areas: the eastern area and the western area. The western area, comprising of a high grade Fe core, a low grade Fe halo, a high grade Cu area and internal waste lenses, is the largest and contains the most drillhole intercepts. This zone covers the areas formerly known as Laurinoja, Lauku and Kivivuopio. It measures 2.1 km along strike (across Laurinoja and Lauku together), 2 km across strike (across Laurinoja and Kivivuopio) and is <50 m thick. The mineralisation has a general strike of 160º and dip of between 10-30º towards the west, with the Laurinoja and Kivivuopio lenses plunge at ~30º towards the southwest. The low grade halo contains approximately 50 Mm^3 of material.

The eastern area comprises a high grade Fe core, low grade Fe halo and internal waste lenses with these wireframes covering the Kuervaara and Vuopio areas. It measures 1.9 km along strike, 800 m across strike (at its widest point) and is <50 m thick. The mineralisation has a general strike of 160º and dip of between 0-20º towards the west, plunging at approximately 40º towards the southwest. The low grade halo contains approximately 30 Mm^3 of material.

Kuervitikko comprises Fe-rich, Cu-rich and Fe-poor, Cu-rich areas, along with internal waste lenses. It measures 1.2 km along strike, 500 m across strike (at its widest point) and is <50 m thick. The mineralisation has a general strike of 165º and dip of between 5-20º towards the west, plunging at approximately 20º towards the southwest. The combined Fe and Cu wireframes contains approximately 14 Mm^3 of material.

9.8 Interpretations of Results

As a result of the drilling and sampling programmes, SRK digitized mineralisation wireframes at both the Hannukainen and Kuervitikko deposits. For the Kuervaara/Vuopio area, high grade Fe and low grade Fe wireframes were created, along with several internal waste lenses. For the Laurinoja/Lauku/Kivivuopio areas, high grade Fe, low grade Fe, and high grade Cu wireframes were created along with several internal waste lenses. For the Kuervitikko deposit, Fe-rich and Fe-poor, Cu-rich wireframes were created along with several internal waste lenses. The area in between Hannukainen and Kivivuopio contains patchy mineralisation, which was not wireframed.

10 SAMPLING PREPERATION, ANALYSIS, AND SECURITY

The sample preparation and analysis was partly conducted by the issuer, and partly by an independent company to the issuer. The structure of the preparation is highlighted below.

10.1 Pre-Northland

There is limited data on the sampling procedures conducted by Rautaruukki. No information is currently available on the specific sampling procedures used for the historical drilling campaigns. It is understood by Northland employees that major element analysis was conducted using XRF, and Au was determined using fire assay.

Northland has performed verification procedures using the historical data to quantify a bias.

10.2 Northland

Only the mineralized interval (+ buffers) of the core is included in sampling, although random samples are occasionally taken from sulphide rich parts of the hanging wall. Sample intervals start and stop a few metres before and after the visual limits of the ore and selected sample intervals are marked on the core boxes with the respective sample number and the sample interval to be cut. The maximum sample interval length is 1.40 m for the silicate rocks and 1 m for iron ore. Core losses less than 50 cm are included in sampling intervals and when the length of core loss exceeds 50 cm, the sampling continuum is broken and the gap is registered in the batch list. During the logging stage the core recovery is measured and the core is measured marking down each 1 m interval to the box. Core losses are marked as CL, together with the length of the core loss. When geological and geotechnical logging and sampling has been completed, the boxes are photographed both wet and dry.

Metallurgical drillholes are sampled by marking on the core boxes the beginning and the end of the mineralization, that is, the skarn interval, to produce a composite sample from each of the drillholes. The core within the marked boundaries is immediately wrapped into plastic bags and stored to the cold storage of the Äkäsjokisuu core shed.

10.3 Northland Chain of Custody, Sample Preparation, and Analyses

Northland's technicians and/or geologists picked up cores at the drill site on a daily basis when drills were operating. From there, the core was transported to Northland's logging and core handling facility at Kolari by pick-up trucks and/or trailers.

After marking the sample intervals and photographing, the core is cut in two halves by Northland's technical crew using a diamond saw. Each sample is bagged in a PVC bag with a sample tag containing the sample ID number. Bags are then closed with nylon cable ties and shipped to laboratory for assays by either Northland's personnel or a shipping company. The analytical procedures used by Northland are as follows:

High temperature drying of samples.

1. Fine crushing of the samples to >70% passing 2 mm size.
2. Riffle splitting the sample to two splits of which another is stored as 2 kg coarse reject in Northland's storage at Äkäsjokisuu. The other half is pulverized in a flying disc or ring and puck-style grinding mills into >85% passing 75 micron (200 mesh) grain size.

3. 100 g of the pulverized sample is stored as QC spare and another 100 g as duplicate sample. The QC spare samples taken from the pulp reject is sent to a secondary laboratory for assays.
4. Scoop sampling of ampoule (100 g) size samples directly from pulverizing mill for analytical purposes.
5. Of each pulverized sample, Northland stores one ampoule (about 100 g), QC spares are split and 1.8 kg is pulp reject.
6. Sent for assaying at external laboratory.

Two laboratories were mainly used for the Hannukainen assaying: GTK (later renamed Labtium; based in Rovaniemi) and ALS Chemex (Vancouver). SGS Lakefield was used for the 2008 metallurgical testwork assaying only. All three laboratories are accredited.

GTK/Labtium used two methods for assaying:

1. Code 704 (P): Au by lead fire assay (inductively coupled plasma optical emission spectrometry "ICP-OES") on 25 g sub-sample, with 10 ppb Au detection limit; The detection limits for each analyte is shown in Table 10-1.and
2. Code 720 (P): Multi-element analysis using inductively coupled plasma atomic emission spectrometry ("ICP-AES") following sodium peroxide fusion on a 0.2 g sub-sample.

Table 10-1: GTK/Labitum ICP-OES Analytes and Detection Limits

Element	Symbol	Detection limit [%]
Aluminium	Al	0.01
Antimony	Sb	0.01
Arsenic	As	0.01
Calcium	Ca	0.001
Cobalt	Co	0.003
Chromium	Cr	0.002
Copper	Cu	0.01
Gold	Au	10 ppb
Iron	Fe	0.05
Potassium	K	0.02
Magnesium	Mg	0.001
Manganese	Mn	0.005
Molybdenum	Mo	0.005
Nickel	Ni	0.05
Phosphorus	P	0.01
Lead	Pb	0.02
Sulphur	S	0.01
Titanium	Ti	0.01
Vanadium	V	0.005
Zinc	Zn	0.005

ALS Chemex used several analysis types; however, the majority were analysed using AU-ICP21 for Au and ME-ICP81 for all other analytes. The ppb detection limits for each analyte is shown in Table 10-2.

Table 10-2: ALS Chemex ICP-OES Analytes and Detection Limits

Element	Symbol	Lower Detection Limit (ppb)	Upper Detection Limit (ppb)
Aluminium	Al	0.01	50
Arsenic	As	0.01	10
Calcium	Ca	0.05	50
Cobalt	Co	0.002	30
Chromium	Cr	0.01	30
Copper	Cu	0.005	30
Golg	Au	0.001	10
Iron	Fe	0.05	100
Potassium	K	0.1	30
Magnesium	Mg	0.01	30
Manganese	Mn	0.01	50
Nickel	Ni	0.005	30
Lead	Pb	0.01	30
Sulphur	S	0.01	60
Silica	Si	0.01	50
Titanium	Ti	0.01	30
Zinc	Zn	0.01	30

10.4 Core Storage

Northland's core storage facility in Kolari is manned or locked 24 hours a day. The core facilities are shown in Figure 10-1.

Figure 10-1: Northland's core storage facility in Kolari (Source: SRK February 2011)

10.5 Density Measurements

Northland used the Archimedes' method of weighing the specimens dry and immersed in water, also taking into account the porosity, to determine the density (reported as specific gravity, SG). The samples vary in weight from approx 750 g to a maximum of 1,500 g. The scale is connected to the computer for automatic registration of weights to avoid later errors in data registration. The scale is considered to be accurate to 0.01 g.

Northland performed density determinations on a regular basis. For Hannukainen, 4,662 density determinations were available, mostly from what is considered to be within the mineralisation envelopes. The iron content was determined as described in the assay section and the density was determined using the following method:

Wet specimens are placed in a basket and then submerged in a vessel containing water. The mass of each specimen in water is then recorded. Samples are wiped dry and the mass of each specimen is recorded. A balance is used for mass determination, the accuracy of which is not known. Based on the recorded readings it is, however, believed to be accurate to 0.1 g and is especially equipped with a hook and basket for weighing in water.

The density of the samples is calculated using the following formula:

$$\text{Density} = \text{mass (air)} / (\text{mass (air)} - \text{mass (water)})$$

A review of the density data estimated into the SRK model can be found in section 13.4.

10.6 Fusion Database

Prior to 2011, a Fusion database system was employed by Northland. The following processes were employed for use with Fusion.

When the drillhole logging process is completed, the hole is "Checked In" into Fusion Remote using the Fusion Client tool. When assays become available from the laboratory (ALS Chemex) they are imported directly into Fusion remote by the database administrator. The laboratory (ALS Chemex) will typically return the CSV import file using the required laboratory import file format.

The laboratory import function within Century Systems conducts validation on the assays prior to and during the import process; in particular it checks:

- that the sample numbers have been entered into the database;
- that the structure of the file supplied by the laboratory is correct;
- that all the necessary columns are present in the file; and
- that the field and laboratory standards are assessed against the defined tolerances.

10.7 Northland Exploration Database

The Fusion database software used by Northland prior to 2011 caused major issues with intervals tables entered into the database, so was replaced by an internally built database called Northland Exploration Database ("NED"). The new databasing system ensures that the data entered by the employee is not altered by the database software once it is entered into the system. Figure 10-2 shows the complete Northland databasing software. Northland Exploration Software Suite ("NESSIE") is the software used by the employees to record every day activities, such as drilling meterage, logs, sampling, which then all feeds into the final NED database.

SRK believes this software is more user-friendly, and more transparent, allowing for fewer databasing errors.

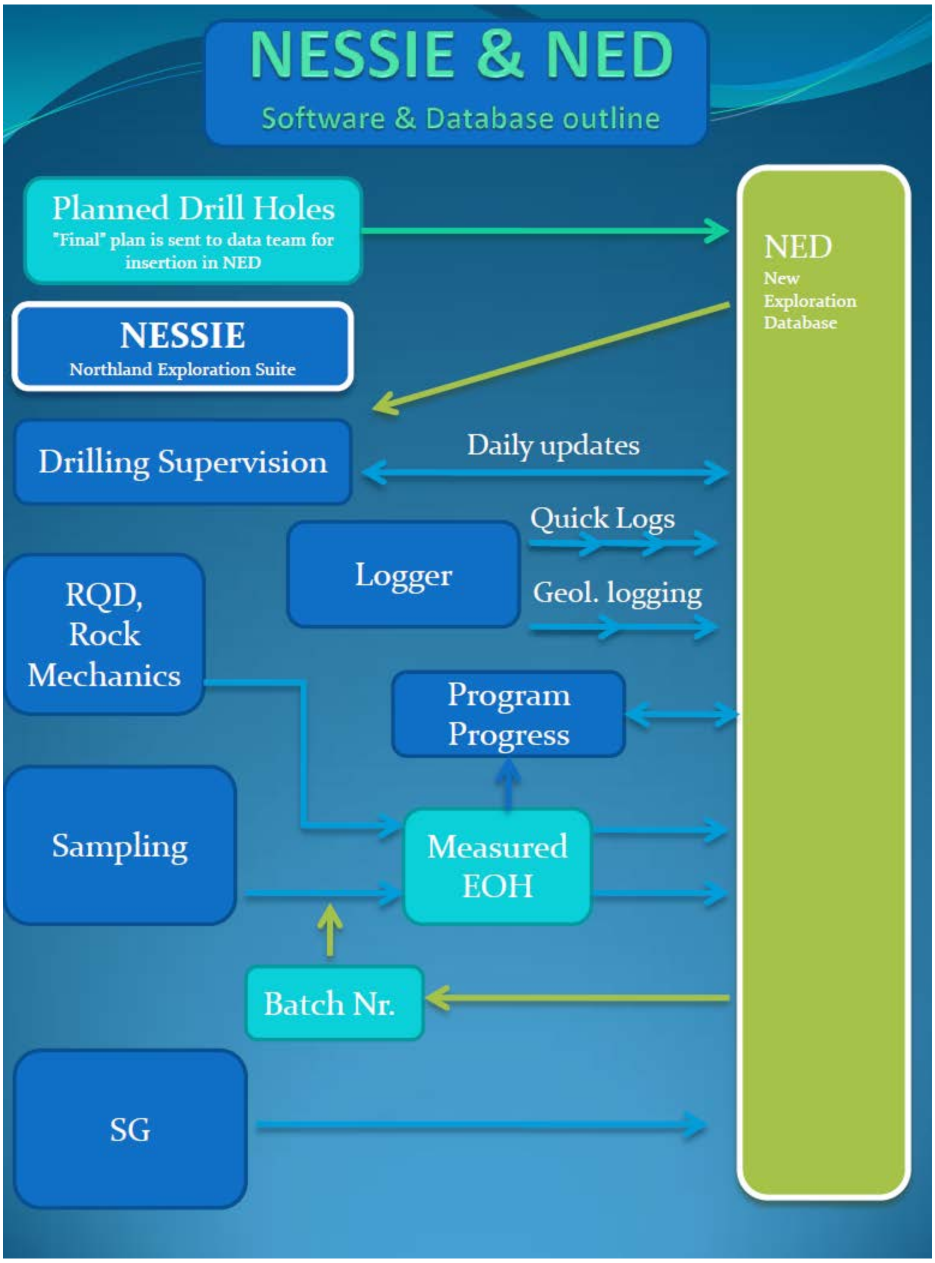


Figure 10-2: NESSIE and NED Northland's internal databasing software (Source: Northland)

10.8 QAQC Procedures

Northland prepared a rigorous QAQC programme prior to the 2006-2008 drilling sampling with the same protocols being used at all their current projects. This comprised duplicates, blanks and standards, as described below.

The pre-Northland drilling performed by Rautaruukki did not contain any QAQC data. Northland performed re-assaying and twinned drilling to validate the historical data.

10.8.1 Duplicates

Inter-laboratory comparison

Laboratory duplicates are made when the laboratory makes two separate pulps independently from the same sample. Northland has requested that the primary laboratory sends the reject pulp (QC spare) to be analysed in another laboratory using the same methods as the original. Comparison of the two analyses gives an indication of validity of laboratory preparation procedures and sample variability. According to Northland’s current protocol, three QC spare samples out of 84 assays submitted should be analysed in the secondary laboratory Northland staff, project geologists and the database manager periodically compare the QC spare analyses to ensure that preparation procedures are adequately homogenizing the sample and replication of analysis is acceptable.

Laboratory Duplicates

Sample pulps produced during the laboratory sample preparation procedure are normally analysed and then stored for future reference. These pulps are re-analysed on a random basis at the primary laboratory to determine reproducibility. Pulp homogenization should result in geochemical results with little variability when compared to the original. According to the current procedure, Northland re-analysis should constitute approximately 10% of the samples. When receiving the assays from the laboratories, it is reported that Northland staff monitor the inter-laboratory variations to ensure compliance to acceptable limits of variation.

10.8.2 Blanks

Blanks or samples with no contained mineralisation are being submitted with each batch of samples sent to the laboratory. The Company´s blank material has been collected from a location known to be devoid of any Fe, Cu and Au mineralisation. Results from these samples indicate if there is any contamination introduced during the sample preparation or analytical procedures. Approximately two blanks in every 84 samples should be submitted to the primary laboratory. If any significant contamination is noted, the analytical laboratory should be notified and corrective measures taken to resolve the potential problem. In addition to Company blanks, the primary laboratory constantly analyses in-house zero blanks. These results enable company staff to further verify the data validity.

10.8.3 Certified Standards

The evaluation of standard geochemical results is the most effective way to confirm laboratory accuracy. Certified standards are sample pulps prepared, packaged, and certified to contain known values of certain elements. The Northland standards NEF-1, NEF-2, NEF-3, NES-1, NES-2, and NES-3 have been prepared by SGS, Lakefield (who used four verification laboratories: GTK, Finland; ALS Chemex, Vancouver; ACME, Vancouver; OMAC, Ireland) that has certified their content. The standards NAG53, NAG55, NAU77 and NAU79, were created by CDN and originally named CDN-CS-3, CDN-CS-5, CDN-CS-7, CDN-CS-9. These contain varying amounts of various ore forming elements within the compositional range expected to be found in the studied deposits. These are being constantly submitted together with unknown samples to the primary laboratory. Northland protocol states that, on average, three Northland standards per 84 samples should be submitted for analysis. In addition to Company standards, the primary laboratory constantly analyses in-house standards - these results enable company staff to further verify the data validity.

10.8.4 Northland QAQC Data Verification

The data verification process is ongoing for the duration of sampling. Data verification includes a statistical analysis of duplicates, standards, blanks, and other types of data obtained during the exploration phase and will determine if analytical procedures, sample preparation, or sampling procedures need to be modified to obtain accurate and verifiable results. Comparisons between the primary and secondary laboratories are undertaken in order to validate laboratory results.

10.9 SRK QAQC Analysis

SRK undertook an analysis of the QAQC data provided by Northland from the 2006-2012 drilling programmes. This includes blanks, standards and duplicates as described above. Only the two main laboratories, ALS Chemex and GTK/Labtium analysed samples that form the QAQC database.

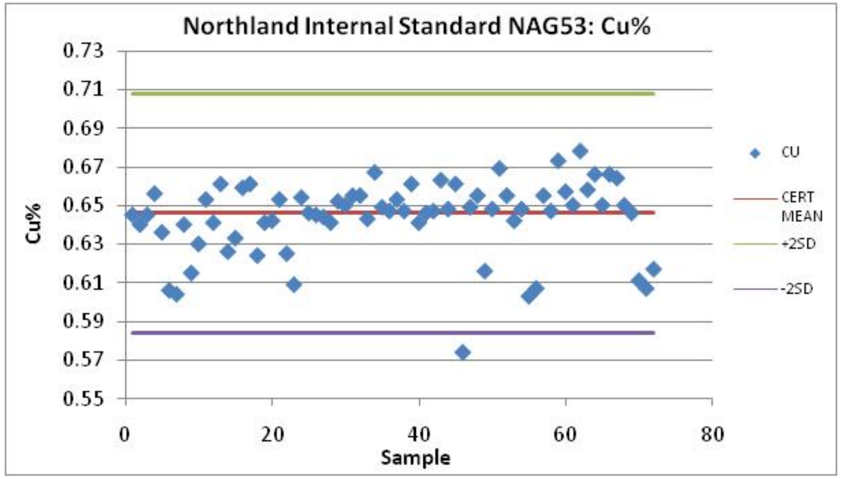
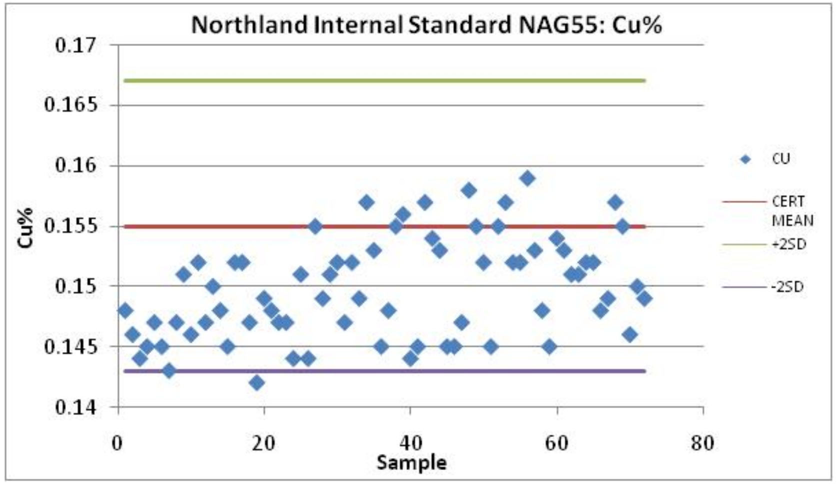
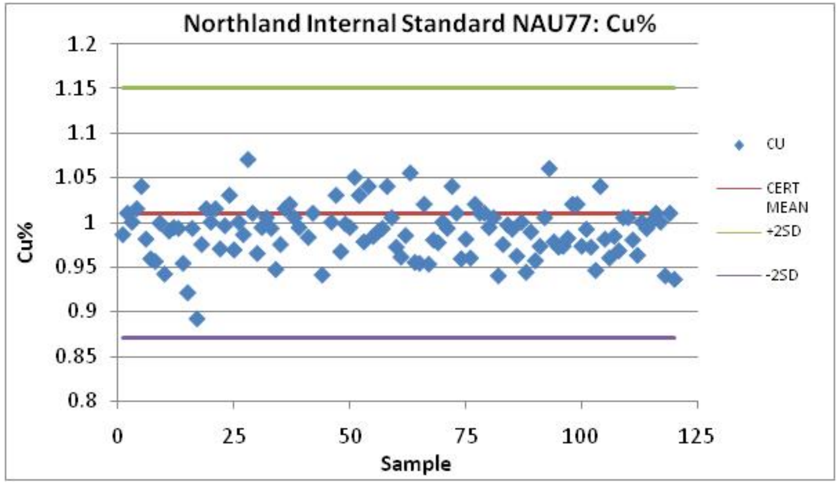
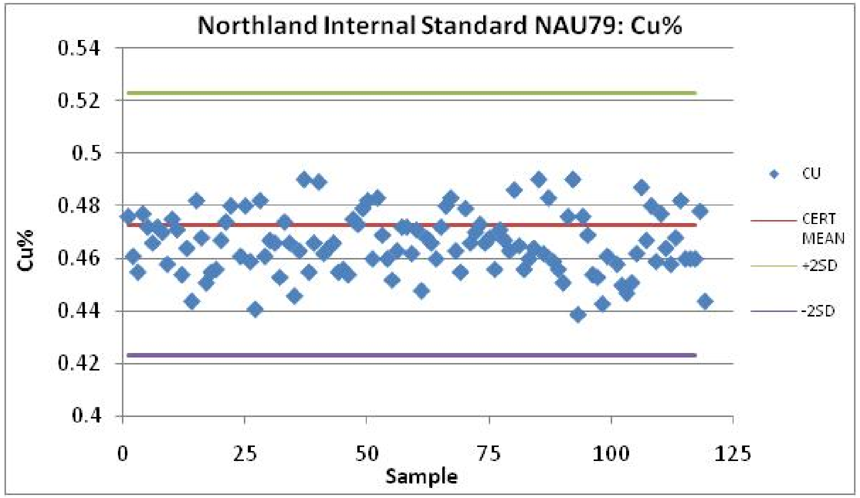
10.9.1 Northland Standards

In total, 8 different internal standards were used during the Hannukainen sampling programme. ALS Chemex analysed 572 Fe and Cu assays, equating to approximately a 4% insertion rate; GTK/Labtium analysed 332 Fe assays and 340 Cu assays, also equating to approximately a 4% insertion rate. The standards were created to try to represent the different ore types present at Hannukainen. Standards NAG53, NAG55, NAU77 and NAU79 were chosen to represent the Cu and Au mineralisation, whereas NEF-1, NEF-2 and NEF-3 were chosen for to represent the Fe mineralisation.

Figure 10-3 shows the internal standard assays compared to the certified mean and two standard deviations from the certified mean. They are numbered by the date the assay result was received by Northland, and so should give an indication of any bias with time.

Of the Cu standards, NAG-53 shows excellent results, with a small spread about the mean and >99% of data inside two standard deviations ("SD"). NAG-55 shows an initial negative bias that improves slightly with time. The results for both NAU-77 and NAU-79 showed good accuracy, with all data inside two SD. Fe standard NEF-1 shows a large scatter with the mean of the data being 2 Fe% different to the certified mean. The Cu results are also shown for NEF-1 with a more accurate assay being evident. Northland state that this standard was based on limited results, and so the 'performance gates' (±2SD) need to be improved. NEF-2 and NEF-3 both show better results than NEF-1, however, both show a negative bias and a large scatter. The 2011 assays show similar results to pre-2011 results.

In general, SRK is confident with the quality of the standard results and believe they are of acceptable quality for use in this MRE. That said, some of the standards do show a slight negative bias, indicating a potential underestimate in grade.

NAG53: %Cu	NAG55: %Cu
	
NAU77: %Cu	**NAU79: %Cu**
	

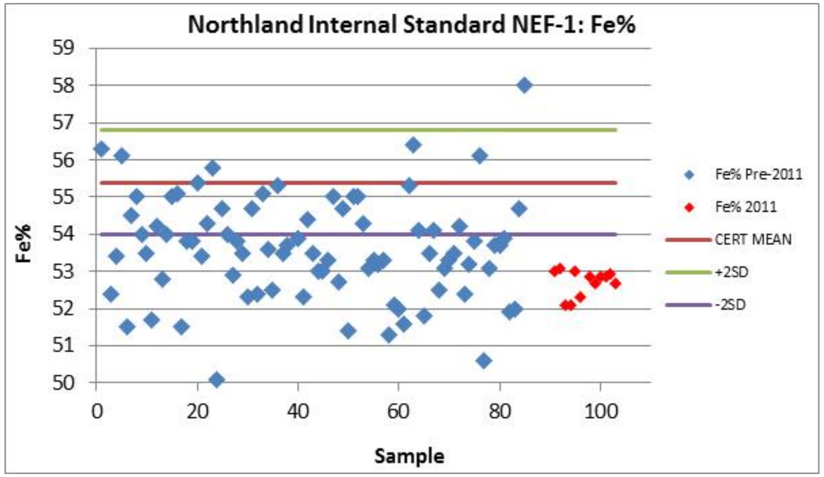
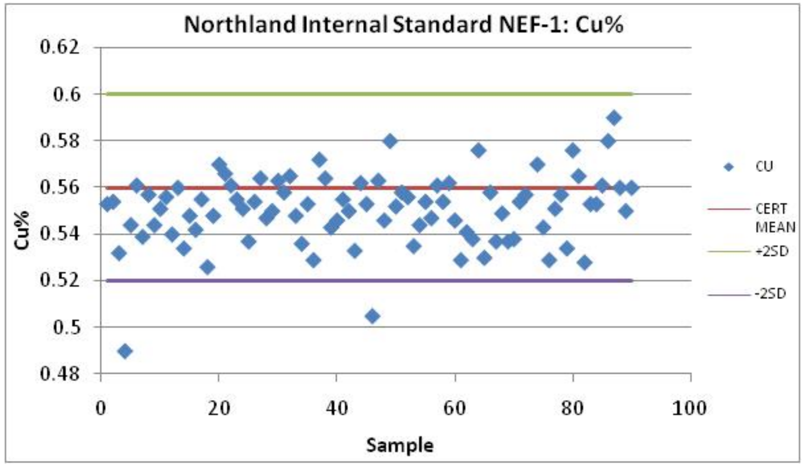
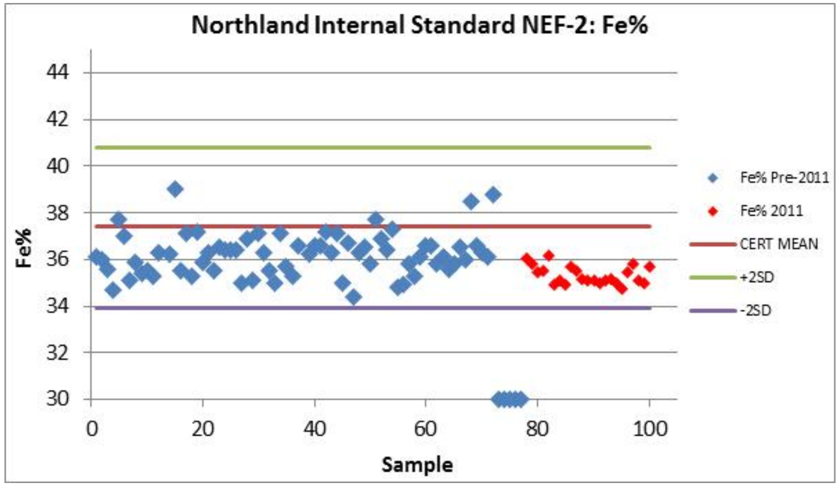
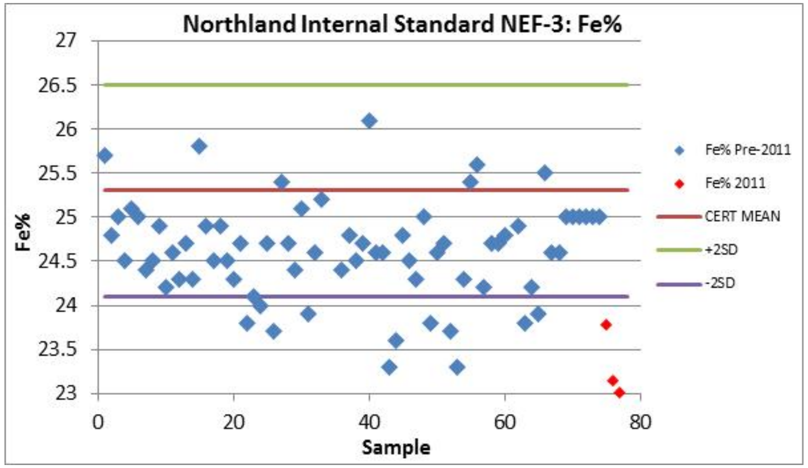
NEF-1: Fe Total	NEF-1: %Cu
	
NEF-2: %Fe Total	**NEF-3: %Fe Total**
	

Figure 10-3: Northland Internal Standards

10.9.2 Northland Blanks

In total, Northland submitted 303 blanks for analysis; 167 at ALS Chemex (approximately 1% insertion rate) and 136 at GTK/Labtium (approximately 1% insertion rate). The results are shown in Figure 10-4 to Figure 10-6, arranged in order of dates the assays were received. With the exception of a few outliers, Figure 10-4 shows a consistent assay range for Cu, being close to the quoted detection limit of 0.005%. Figure 10-5 shows a wide spread of data, with an average grade of 10% Fe Total for pre-2011 samples, then an average of <1% in 2011. The change is due to the change of material used as blanks. Prior to 2011, a diabase blank material was used, which was inappropriate for the Fe mineralisation due to the aproximately 10% inherent Fe grade. The low-grade Fe mineralisation contains sporadic magnetite, but can have a mean grade of approximately 15%, which SRK considered too close to the blank material in the previous MRE report. In 2011 report, Northland changed the blank material to quartzite from Nilsiä, Eastern Finland, which has an inherently low Fe grade.

SRK is confident that the Northland blank results are of acceptable quality for use in this MRE. The new blank utilised is more appropriate for the Fe-Cu-Au mineralisation than the previously used diabase.

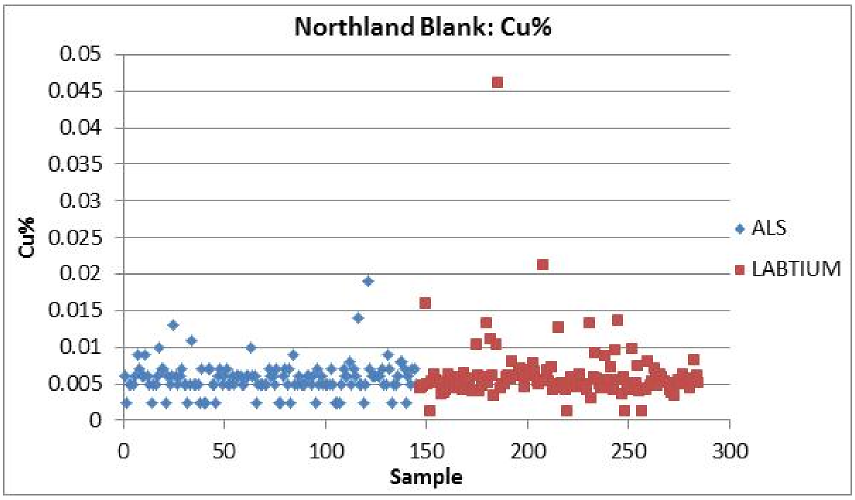


Figure 10-4: Northland blank material: %Cu

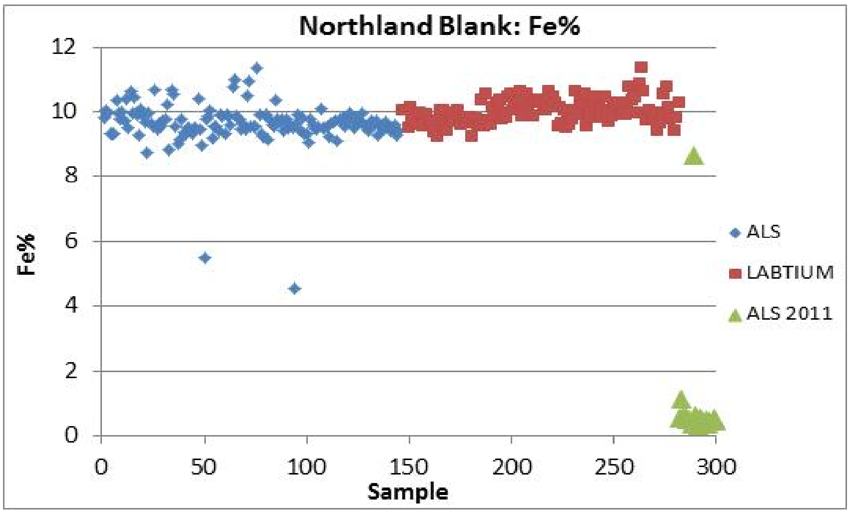


Figure 10-5: Northland blank material: %Fe Total

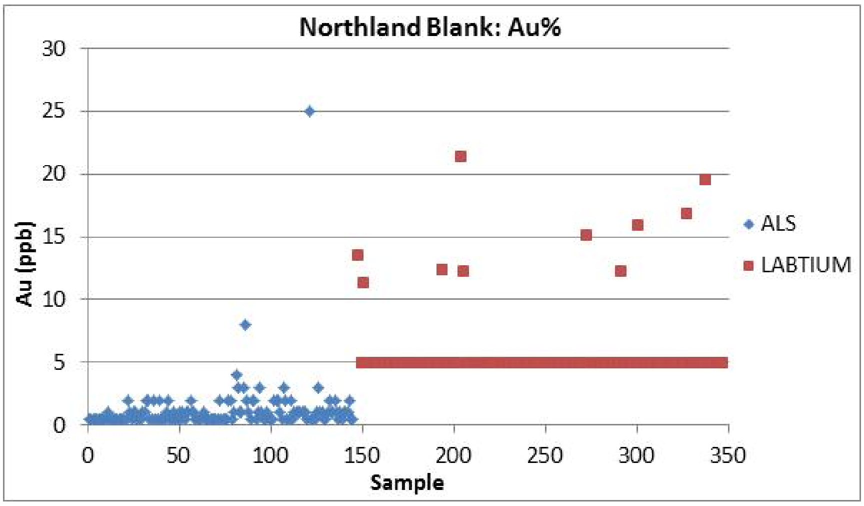


Figure 10-6: Northland blank material: Au (ppb)

10.9.3 Laboratory Standards/Blanks

In total, ALS Chemex used 46 different in-house standards/blanks, with varying Fe, Cu and Au values; comprising 1,526 Cu assays across 14 different standards and 1,438 Fe assays across 10 different standards. This equates to an insertion rate of approximately 10%. The GTK/Labtium laboratories used 13 different in-house standards/blanks, with varying Fe, Cu and Au values; comprising 867 Cu assays across 13 different standards and 845 Fe assays across 13 different standards. This equates to an insertion rate of approximately 10%.

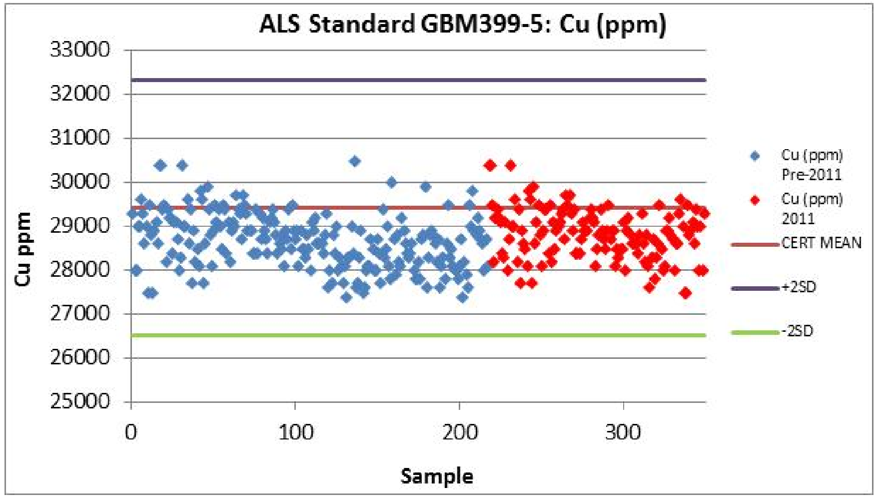
Figure 10-7 shows a sample of the in-house standards and blanks used by SGS and GTK/Labtium between 2005 and 2012. There are many other standards which contain low numbers of samples, which are not displayed here. The samples are arranged in order of the date received from the laboratory and should show potential bias with time.

In general, the SGS standards show no major biases, with a good spread of data about the certified mean, and >95% of the data falling within two SD. There are slight negative and positive biases seen in individual graphs, but nothing to suggest the quality of the assay database has been compromised.

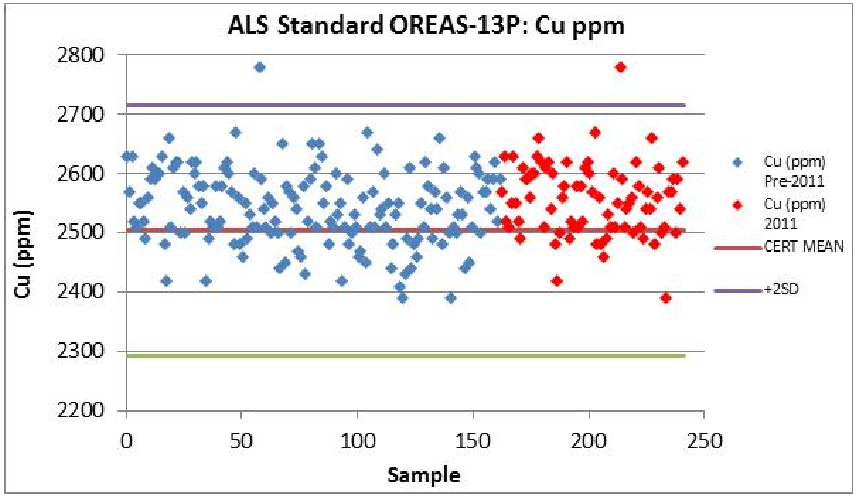
The GTK/Labtium standards are fewer in number and the certified mean and standard deviation data were difficult to acquire. They also show poor results, especially QCSU-1a, which has certified data, and shows the majority of the data outside of two SD and a definite negative bias.

In general, SRK is confident that the quality of the in-house laboratory standard and blank results is of acceptable quality for use in this MRE. That said, some of the standards indicate a slight negative bias, which may lead to a slight underestimation in grade. There are also some concerns over the quality of some of the standard material used by GTK/Labtium.

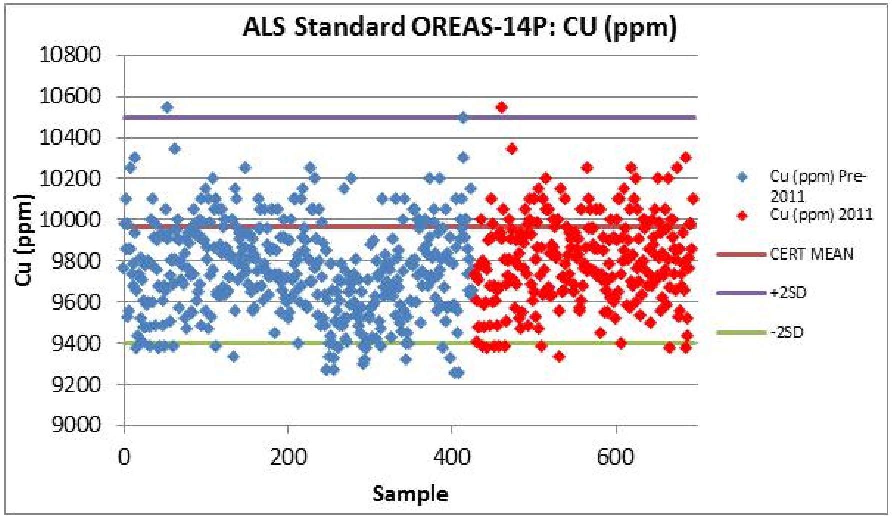
GBM399-5: Cu (ppm)



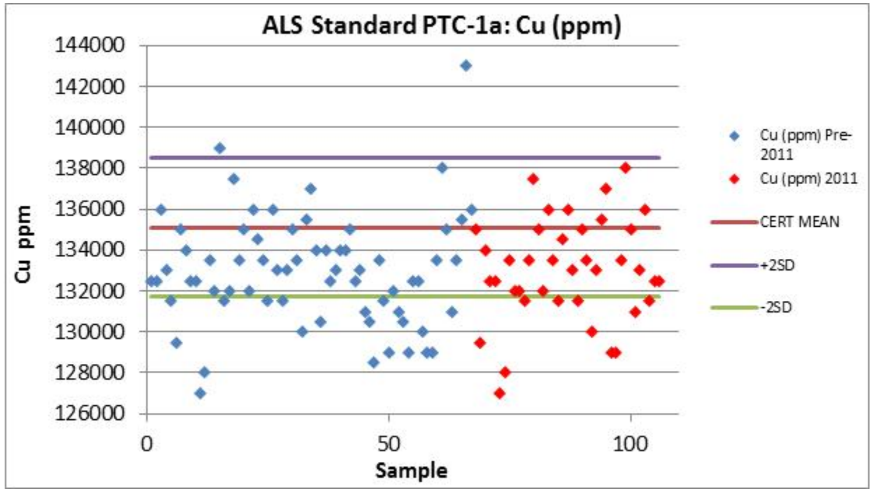
OREAS-13P: Cu (ppm)

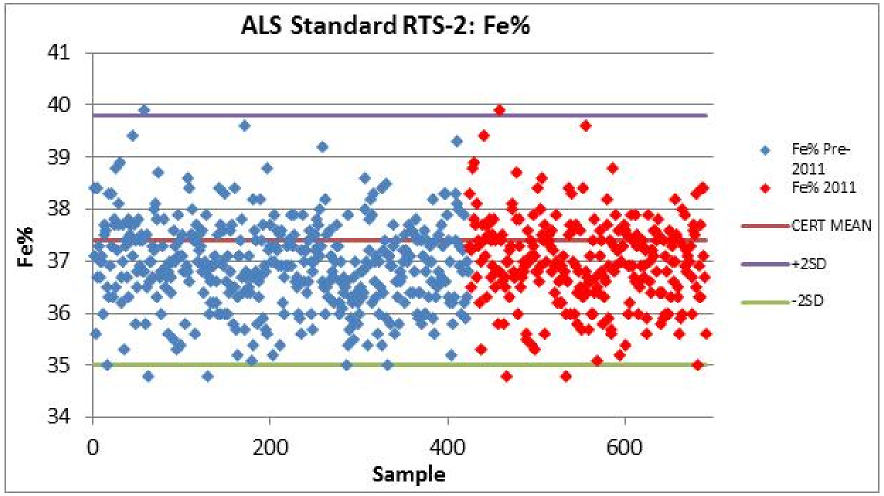
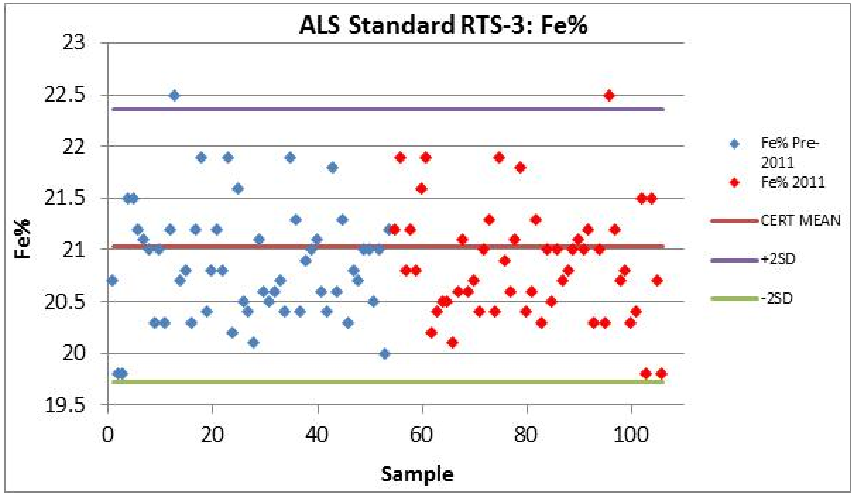
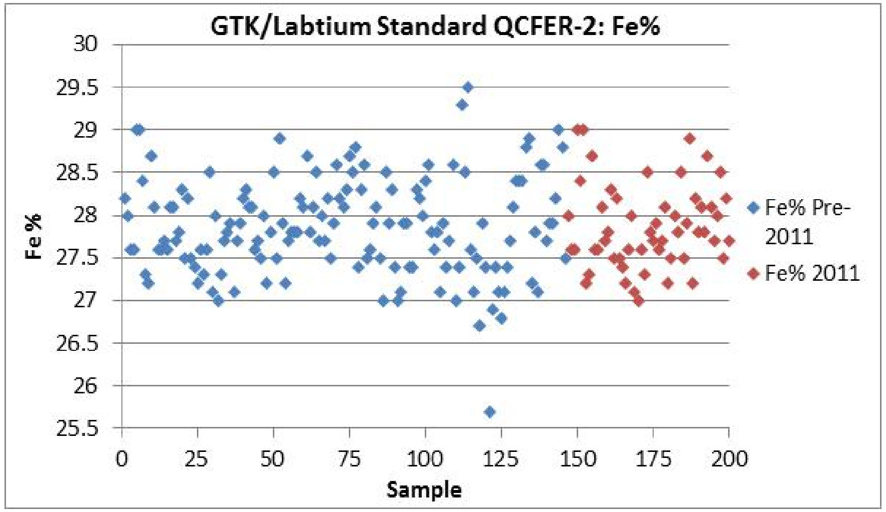
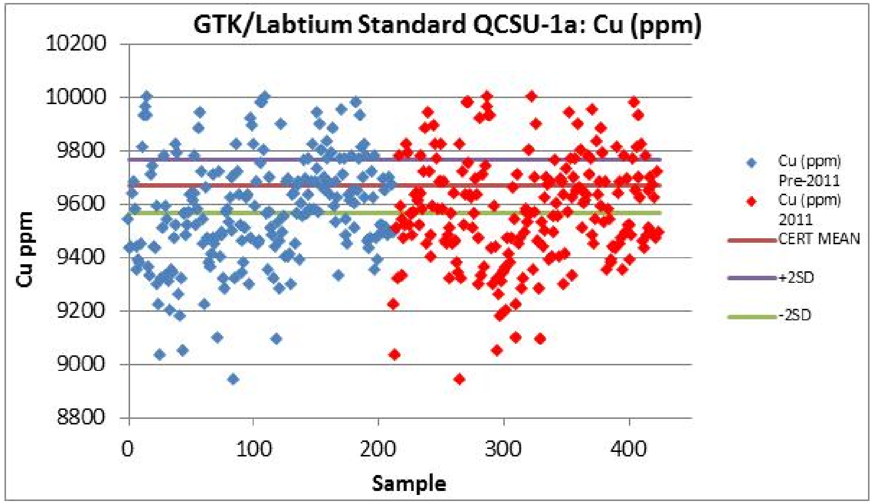


OREAS-14: Cu (ppm)



PTC-1a: Cu (ppm)



RTS-2: %Fe Total	RTS-3: %Fe Total
	
QCFER-2: %Fe Total	**QCSU-1a: Cu (ppm)**
	

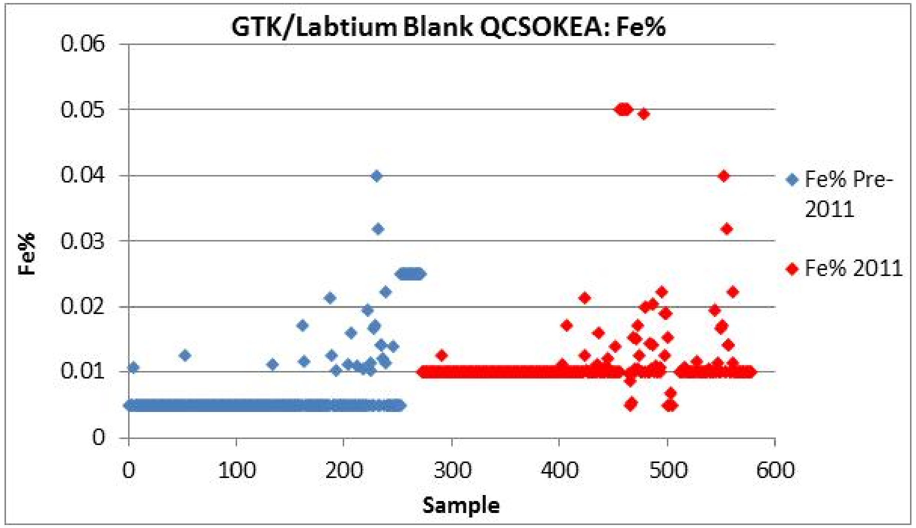
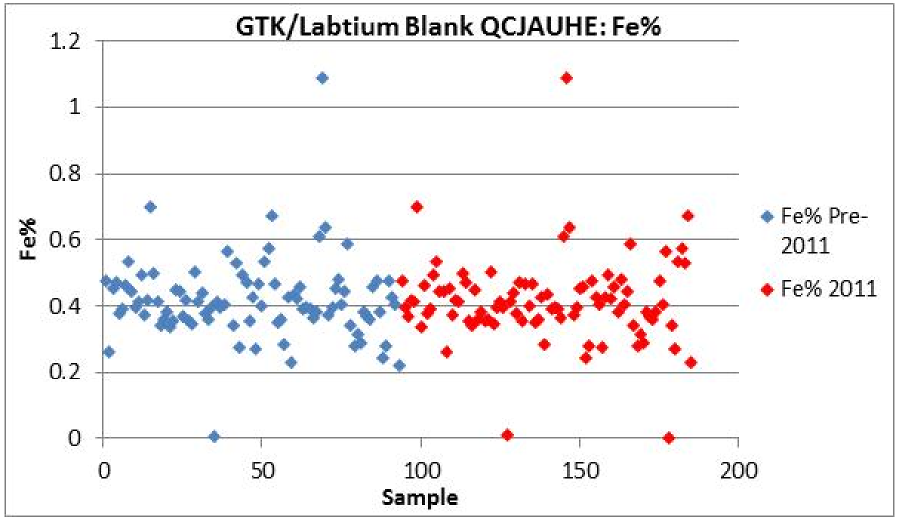
QCSOKEA	QCJAUHE
	

Figure 10-7: In-house laboratory standards and blanks

10.9.4 Duplicates

The laboratory duplicate assays are compared to the original assays to check the consistency and precision of the laboratories. In total, there are 885 ALS Chemex duplicates (insertion rate of approximately 3.5%). and 384 Labtium duplicates (insertion rate of approximately 5%).

The laboratory duplicates show excellent correlation, with a correlation coefficient of close to 1 for both the ALS Chemex and Labtium samples, as shown for %Fe Total and %Cu in Figure 10-8 and Figure 10-9.

SRK is therefore confident in the repeatability of the sample preparation and analysis of these samples. That said, and as described above, some of the standards show a slight negative bias at both laboratories, so it may be possible that both laboratories have slightly underestimated grade. SRK does not however consider this to be material to the Project.

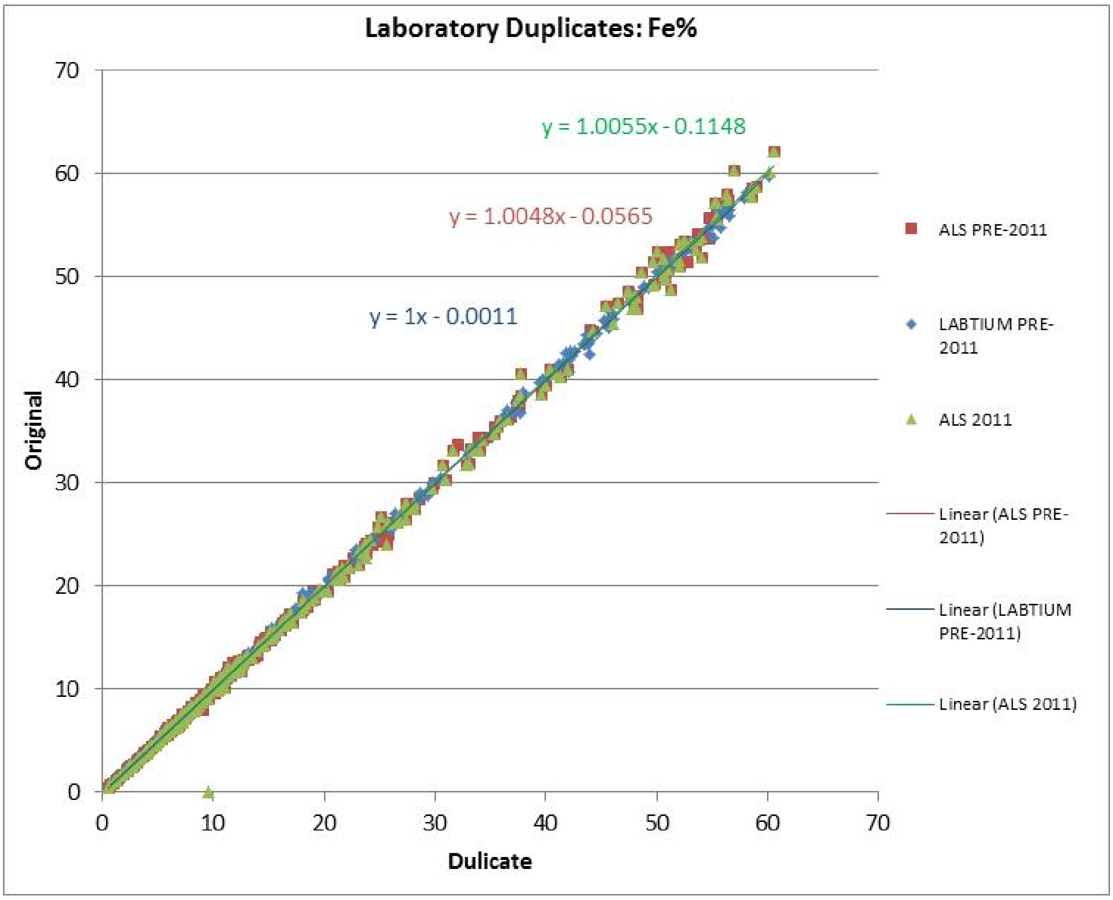


Figure 10-8: Laboratory duplicates: %Fe Total

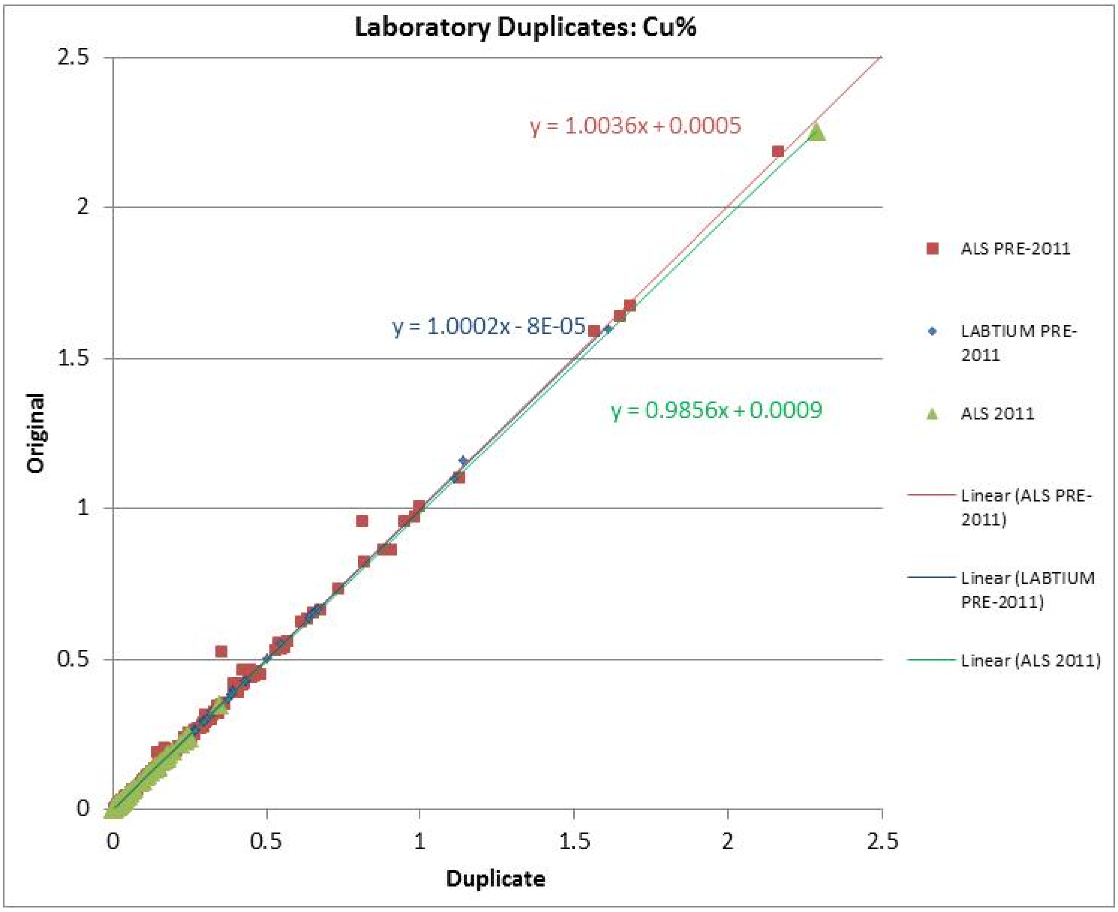


Figure 10-9: Laboratory duplicates: %Cu

10.9.5 Inter-laboratory Duplicates

In order to ascertain the accuracy of the assaying laboratories, duplicates are assayed by a secondary (umpire) laboratory. In total, ALS Chemex had 307 assays duplicated at GTK/Labtium (insertion rate of approximately 2%) and GTK had 195 assays duplicated at ALS Chemex (insertion rate of approximately 2.5%).

The duplicate data is plotted against the original assay data for Fe and Cu in Figure 10-10 and Figure 10-11. The duplicate assays show excellent correlation to the original assays, with a correlation coefficient close to 1. SRK can deduce that there are no biases in the laboratories, and is confident that are performing consistently and accurately.

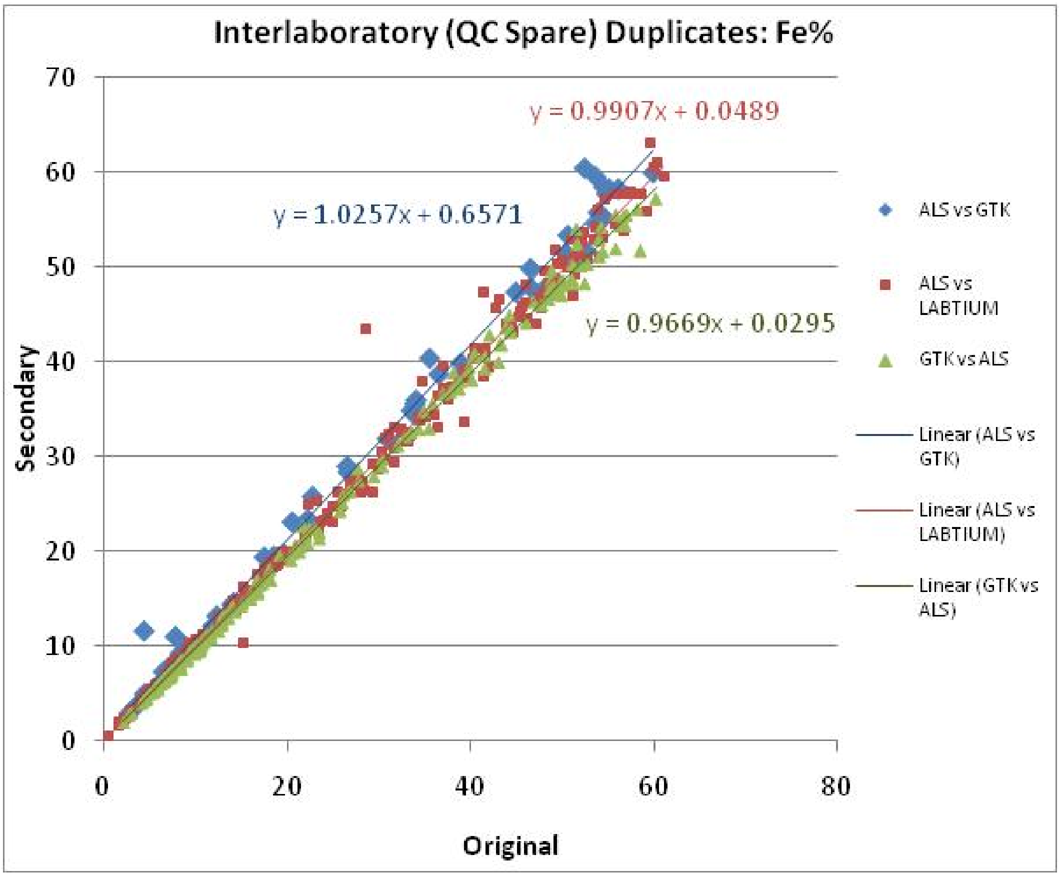


Figure 10-10: Umpire laboratory duplicates: %Fe Total

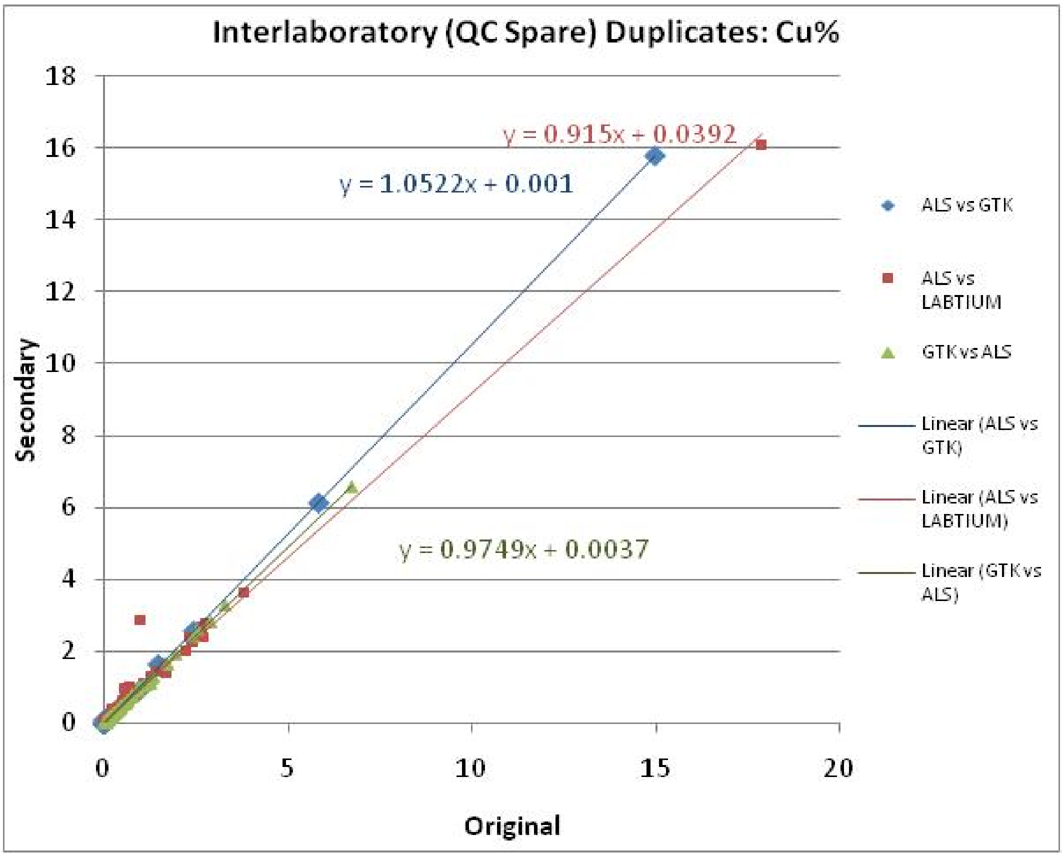


Figure 10-11: Umpire laboratory duplicates: %Cu

10.9.6 QAQC Summary

Northland used a rigorous QAQC programme throughout the sampling programme at Hannukainen. This comprised sampling blanks, duplicates and standard reference material, both inserted by Northland and by the laboratory. The two main assaying laboratories, ALS Chemex and GTK/Labtium, produced good quality data, with no material biases between the laboratories or over time. However, a slight negative bias can be seen in the standard assays, resulting in a possible under estimation in grade for the Project. The GTK/Labtium standards showed poor results and certified values were difficult to acquire. The recommended insertion rates were not matched for the inter-laboratory duplicates or Northland blank material, however, they were for the Northland standards, and also the laboratory standards were inserted at a high rate. Overall, the QAQC material was inserted at a rate of approximately 10% by Northland, and an additional 10% by the laboratories. SRK considers this to be appropriate.

The blank material submitted by Northland prior to 2011 was deemed inappropriate for use as an Fe blank due to the inherent approximately 10% nature of the diabase used. It is appropriate for the Cu and Au mineralisation, however, and shows a lack of contamination at the laboratories. Northland changed this blank material for a quartzite with inherently low Fe, Cu and Au grades. SRK deems this appropriate and recommends continuing to utilise this new blank material.

SRK is confident that the quality of the data across the main two laboratories is adequate for a Mineral Resource Estimate. The historical data does not incorporate any QAQC check samples, but has been verified by re-assaying the majority of the core to confirm the quality of the data. The slightly negative bias seen in the standard assays results in a potential underestimation in grade for the Project, however, this is not deemed to be material overall.

10.10 Core Recovery Analysis

The drill core observed during the site visit to the Pajala core shed showed very good recovery and a good quality core, with no obvious impact on the quality of the Mineral Resource Estimate. Appendix A shows the core recovery analysis for Hannukainen drill data. Generally the core loss is very low, with the vast majority of holes having >99% recovery, and the vast majority with an RQD >80%. There is no core recovery data for the historical drillholes.

11 DATA VERIFICATION

Qualified Person Howard Baker (MAusIMM(CP)) has verified that the data provided by Northland seems to be correct and viable for use in a Mineral Resource Estimate.

11.1 Data Received

SRK was provided with the following list of documents and files to assist with the Mineral Resource Estimate:

- Drillhole Data:
  - Drillhole Database, including collar coordinates, down-hole survey measurements, elemental assay data, lithological logging data, structural measurement data, sulphide and oxide logging data, magnetic susceptibility readings, Satmagan readings and Davis Tube recovery testwork ("DTT") data;
  - QAQC data to accompany the assay data; and
  - documents outlining external and internal database validation and structure.
- Geological and mineralisation cross-sections:
  - geological and mineralisation interpretations by Northland along all drill lines in Hannukainen and Kuervitikko.
- Topographic survey:
  - high resolution fly-over topographic survey in DXF format; and
  - former pit plans, digitised by Northland to show actual pit surface, taking account of the lake surface which forms the surface of the flyover data topographic surface.

11.2 Database Validation

Before commencing the 2010 WGM Mineral Resource Estimate, WGM and G Wahl completed a database validation exercise on the entire Hannukainen dataset. Errors were found and corrected in the database, which was then used for their estimate. SRK also carried out checks to validate the database and ensure the quality of data is adequate for a CIM compliant Mineral Resource Estimate.

11.2.1 SRK database validation

In February 2011, SRK's Howard Baker and Ben Lepley visited the core storage facilities in Kolari to review some of the drill core, and validate it against the assay data and lithological logging. The primary goal was to observe and confirm the mineralised horizons, which are generally demarcated by the geological logging code MGTS (magnetite skarn) along with Fe Total grades in excess of 15%. SRK is satisfied that the logging and assaying data correlate well with each other and the core.

SRK imported the drillhole data in Datamine Studio 3 software to validate the files provided by Northland. All interval files and collar files were clean and valid drillholes were created. The elevations of the collars match closely with the topography due to the high accuracy of the topographic survey, along with the high accuracy of the differential GPS survey points of the collars. No adjustments were made to the collar locations.

11.2.2 Historical assay data

Northland has used three different accredited laboratories since they acquired the Project: ALS Chemex, GTK/Labtium and SGS. In addition, the previous operators Rautaruukki undertook assaying during the 1970s and 1980s.

In total, 34% of the drillholes in the Hannukainen database were drilled between 1974 and 1986 by Rautaruukki and the additional 66% between 2005 and 2012 by Northland. The historical assays do not contain any QAQC data and most of the historical core is stored in storage facilities in Helsinki. It was not possible for SRK to inspect this core in the time period of the Mineral Resource Estimate. That said, Northland re-assayed 73% of the historical data leaving only 6% of the entire assay database as historical invalidated data. Less than 2% of the assay database has been mined-out during the mining activities of Rautaruukki in two separate pits in Kuervaara and Laurinoja. This data was included for the Mineral Resource Estimate, with the material modelled above the pits being excluded from the block model.

In order to validate the original historical assays, Northland conducted an analysis comparing the original assays to the re-assays, as shown in Figure 11-1 and Figure 11-2. The re-assays are length-weighted as the sample lengths were different to the original sample intervals. The re-assays show a strong correlation to the historical data for Fe. There are some major outliers, but generally re-assays show a high level of precision. The Cu results show a larger scatter, although this is partly to be expected due to the more erratic and nuggety nature of the Cu mineralisation within the deposit. The data shows a tighter cluster at lower Cu values.

Overall, SRK is confident that the quality of the historical data is sufficient for a CIM compliant Mineral Resource Estimate to be completed and as such the entire database has been utilised.

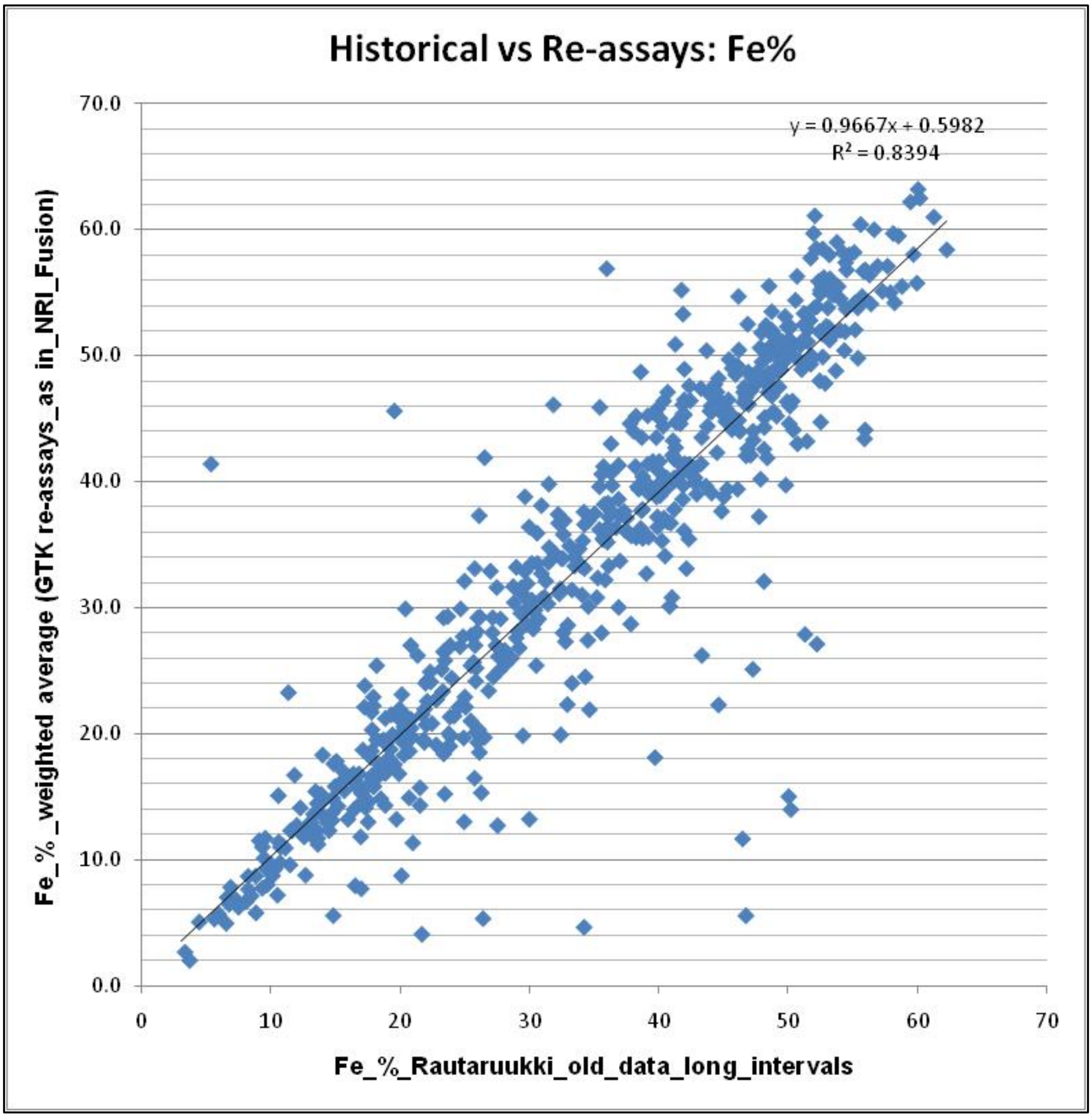


Figure 11-1: Historical vs Re-assayed results for %Fe Total

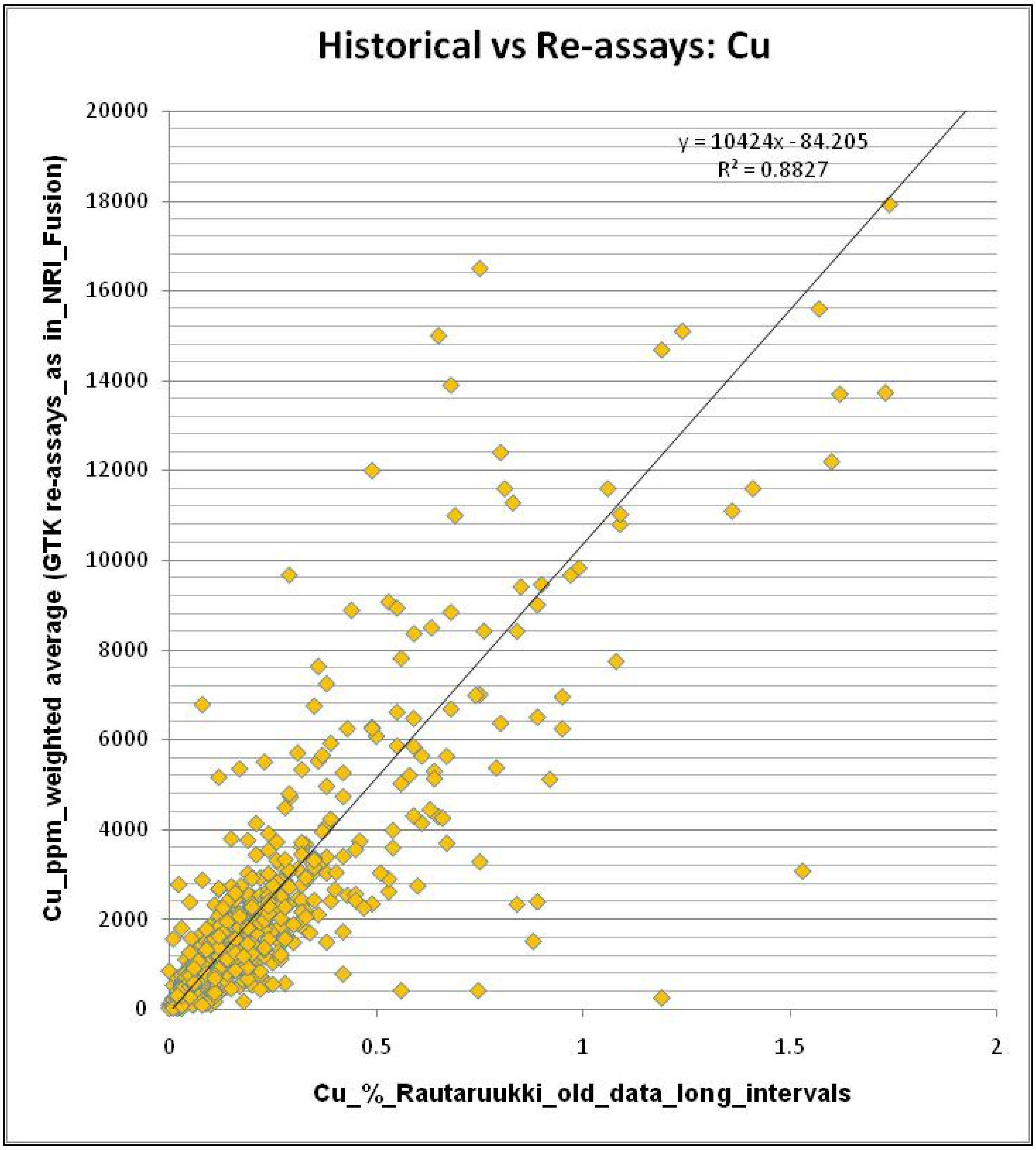


Figure 11-2: Historical vs Re-assayed results for Cu ppm

11.2.3 Duplicate drillholes

To check for short-scale variability between drillholes, down-hole comparisons were made between sets of drillholes within 5 m of one another. Figure 11-3 and Figure 11-4 show two of the closest drillhole comparisons. They both show a strong correlation to one another, indicating that the short scale variability of the Fe grades is relatively low.

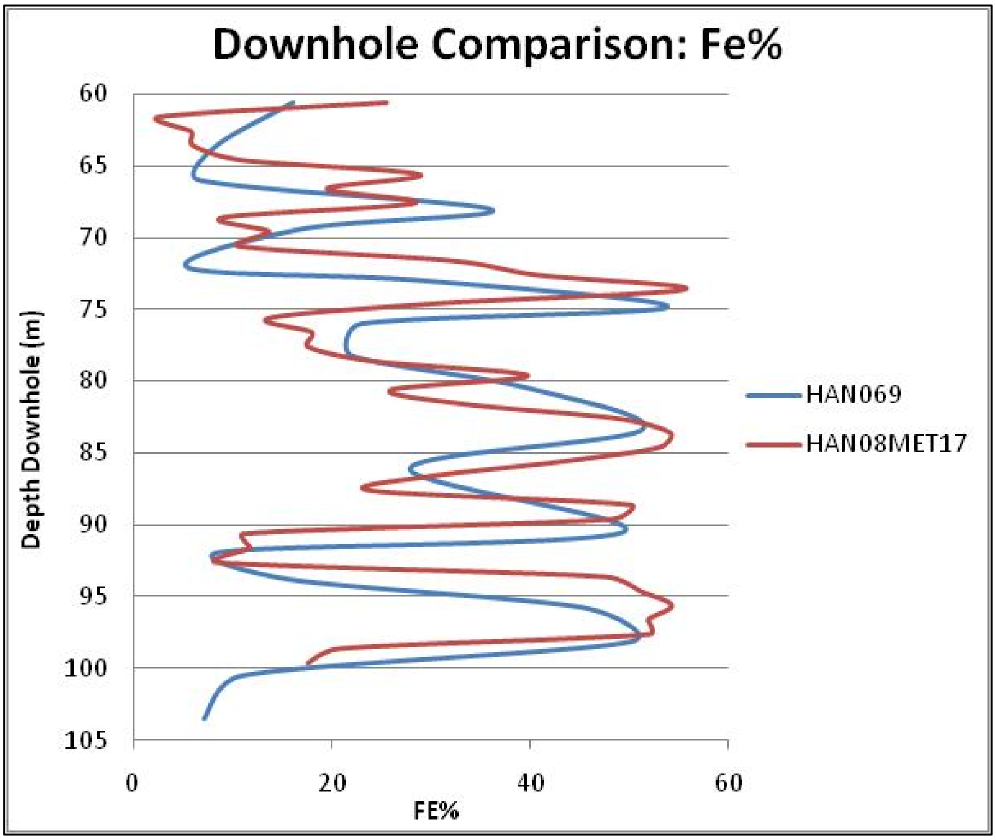


Figure 11-3: Down-hole comparison for %Fe Total - HAN069 vs HAN08MET17

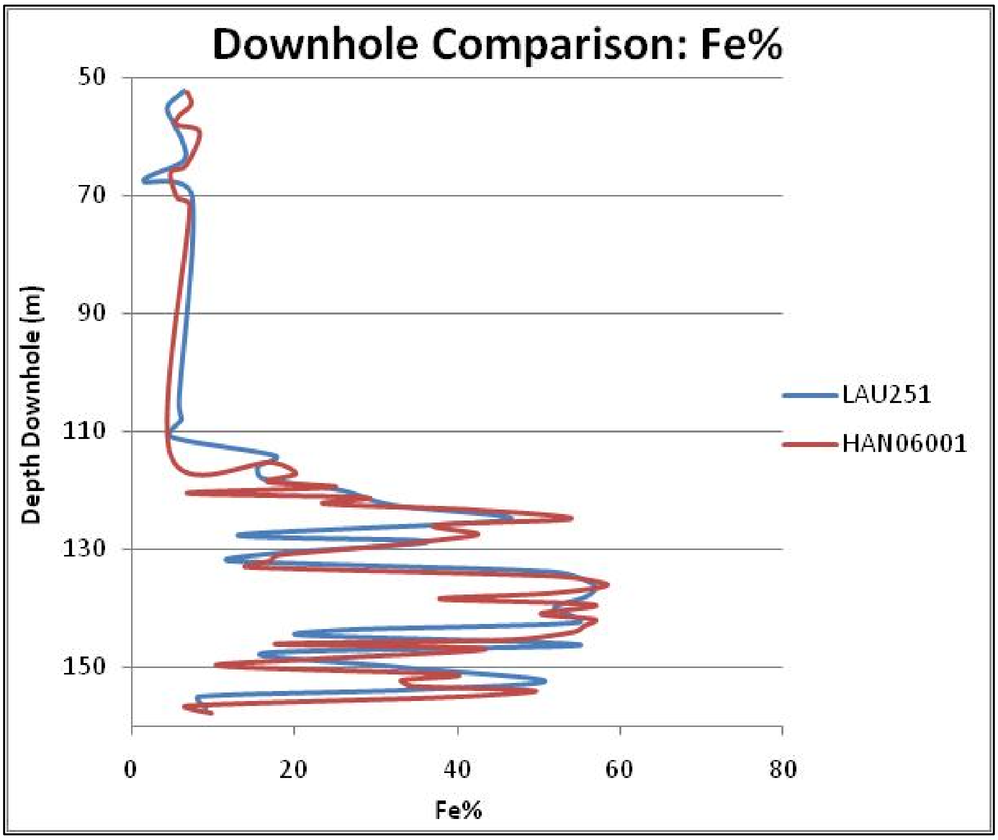


Figure 11-4: Down-hole comparison for %Fe Total - LAU251 vs HAN06001

11.3 QAQC

The quality control measures that Northland has put in place are discussed in the previous section. It is SRK's opinion that the procedures adopted have led to a reliable database, although SRK does recognise that a slight underestimate in Fe grade may be possible as highlighted by the standard assay results. The overall insertion rate of approximately 10% of the assays is considered appropriate.

Prior to 2011, blanks were inserted with an inherent Fe value but with negligible Cu and Au. Northland changed this blank material prior to the 2011 assaying to a quartzite with low Fe, Cu and Au values. SRK believes this blank material is more appropriate and should continue to be used.

SRK is confident that the quality of the data is sufficient for a CIM compliant MRE.

11.4 Topographic Survey

A topographic survey was conducted to produce an accurate topographic surface (digital terrain model), which was used use to cut mineralisation wireframes to report volume and tonnes accurately. The terrain model supplied by Northland was produced as part of the "Tapuli Hydrological Study", from laser scanning data compiled on 24 May 2008. The aerial images were collected simultaneously with laser scanning and orthomosaic's were generated. The terrain model was delivered as ground points in triangulated irregular network ("TIN") format and contour lines in the coordinate system of the Project. The accuracy of terrain model was estimated better than 10 cm on hard surfaces and better than 15 cm on dense forest areas. In addition, and using historical cross sections, a wireframe surface was generated to represent the base of the existing open pits, currently below the pit lake. This surface was utilised to remove the mined out areas from the block model.

SRK has verified that this data is viable for use in a Mineral Resource Estimate.

11.5 SRK Comment on Data Quality

SRK is confident that the quality of the data provided by Northland is fit for purpose. The collar, down-hole survey, interval files all validated and were verified in 2011 by three individual sources: Northland, G Wahl and SRK. SRK, in previous commissions, suggested that the Fusion database system was replaced, due to the numerous errors it created. This has been achieved by Northland, who has replaced Fusion with an internally built database, which appears to be more transparent and has not caused any internal errors. Comparisons of historical and modern assays show strong correlations, and therefore SRK has deemed them compatible for use together in the Mineral Resource Estimate. Twin drillholes show that the short-scale variability is low, giving SRK additional confidence in the quality of the historical data. The recent (2011) metallurgical and infill (2011-2012) drilling sampling has also been shown to be compatible with the previous data.

The topographic survey shows excellent correlation to the surveyed drillhole collars, and has a high resolution, allowing for accurate block model definition. As such, no edits were made to the collar file or topography surface during the Mineral Resource Estimation process.

12 MINERAL PROCESSING AND METALLURGICAL TESTING

Northland has conducted several metallurgical testwork programmes since taking ownership of the Project in 2005. The data has been analysed by external consultants and Northland employees. Section 12.1, below, is a memo by G Wahl which is a review of all data accumulated before January 2010. Section 12.2, below, is a report by Bo Arvidson Consulting LLC (B.Arvidson), which outlines the processing recovery functions used for this Mineral Resource Estimate.

Additional metallurgical testwork has been completed during 2012 by the Lulea University of Technology with the work programme being managed by Dr Pertti Lamberg (Professor in Geometallurgy).

12.1 Hannukainen and Kuervitikko Fe Recovery

The following is taken from a memo by G Wahl entitled “Hannukainen and Kuervitikko Fe Recovery”. The memo highlights the metallurgical testwork conducted to date by Northland and adds some context for selecting data and testwork appropriate for a grade based formula with which to populate the recoverable Fe attribute in the block model.

The following describes previous Davis Tube and Satmagan Testwork, which is predominantly focused on Hannukainen samples. Currently no testwork is available for Kuervitikko other than some historic Satmagan (SRK Note: since this memo was written, additional testwork has been completed on 40 samples from Kuervitikko: SGS, 2010b; and additional testwork in Hannukainen: SGS 2010a).

The following comprises a list of raw data/reports which pertain to Satmagan and DDT results.

- 18 DDT and 18 Satmagan Samples on HAN07009B – SGS (2008). Proj File 11610-001 DT Variability Tests.xls.;
- 11 DDT Samples – GTK rerun of SGS samples – CL Consulting Report Aug, 2008;
- 103 DDT and 103 Satmagan Cluster Samples – SGS (2009). Proj File 12032-004 DT Variability Tests.xls;
- Final #3 Flowsheet Recoveries based on 103 Cluster Samples – 12032-007 Global Cluster Flowsheet 3.xls;
- 585 Satmagan Results – 1202-006 Head Assays – Rev 3.xls;
- Compilation of Satmagan Results – HAN-KUE_Satmagan_Compil_PTP_141209. Xlxs;
- Proposed DDT Samples for Kuervitikko – Kuervitikko DDT Samples.xls;
- Hannukainen DDT Tests Hooey Memo Nov 3, 2008; and
- CL Consulting Mineralogical Study on Hannukainen DDT-Test Samples by K Heinanen and J Hognas.

12.1.1 Flow Sheet #3

Flow sheet #3, proposed for the Preliminary Economic Assessment ("PEA") (Hatch, 2010), is based on crushing and grinding, a chalcopyrite rougher, followed by Low Intensity magnetic Separation ("LIMS") and then a pyrrhotite rougher to create a Fe concentrate with a majority of the sulphides removed. In flow sheet #3 testwork, all three clusters (high grade, medium grade and low grade grouped samples) including the 14.8% head Fe of cluster #3 produced a Fe concentrate grade above 65%Fe with very low S. (See 12032-007 Global Cluster Flowsheet3.xls).

12.1.2 Mineralogy and Recoverable Fe

Three mineralogical studies highlight the occurrence of Fe within Fe oxides, sulphides or silicates at Hannukainen. This work is described in a mineralogical study conducted by GU Ivanyuk et al in October 2008 on 23 specimens over a mineralised interval extending from 119.45-178.7 m in drillhole HAN-06003. The second study by CL Consulting is comprised of 10 samples from Hannukainen. The third study conducted by SGS in 2008 (Ref No 11610-001) was based on drillhole HAN7009B.

Ivanyuk et al (2008) identified non-magnetite iron as being contained primarily in silicates and sulphides. Diopside and amphiboles as well as pyrite, pyrrhotite and chalcopyrite were identified as being dominant within the magnetite skarn. Ivanyuk found that magnetite, sulphides, diopside and amphiboles are concentrated within ore bands interlayered with plagioclase or quartz rich bands.

The silicate bands contained rare magnetite dispersed within silicate grains or occupying silicate grain boundaries. The ore rich magnetite bands contain massive aggregate magnetite grains or silicate grains cemented by magnetite.

Ivanyuk indicates a mean diopside content in the magnetite skarn of 23.0 wt%, while amphiboles account for 13% by weight. Diopside was observed to contain micron sized inclusions of magnetite. Mean FeO content in the Ivanyuk diopside samples was 11.1 wt%. Amphiboles were found to contain an average of 17.8 wt%FeO.

Work by CL Consulting indicates that the magnetite content of the Hannukainen ores is very pure. The MgO content is mainly below 0.2%. Work by Ivanyuk et al (2008) indicates that magnetite grains are generally homogeneous but can contain micro-inclusions of chalcopyrite, ilmenite and uraninite. The main impurities identified in the magnetite were identified as manganese and aluminium which replace Fe. Based on 19 specimens, Ivanyuk indicates a mean weight % chemical composition for magnetite of 93.37%FeO, 0.13%Al_2O_3, 0.12%MnO, 0.07%MgO, and 0.09% SiO_2. Work by SGS (See 11610-001 in Sept 2008) indicates that microprobe testwork on samples from HAN7009B also indicated that the Hannukainen ores were relatively clean of impurities. Mean Fe content from 29 magnetite grains was 70.4%. Al_2O_3 was the most prominent impurity at 0.28% and traces of MgO, MnO and SiO_2 were also noted on most magnetite grains.

Ivanyuk found that pyrite is the second most abundant mineral phase after magnetite. Skarn pyrite is usually later than magnetite and pyrite metacrystals often include magnetite relics. Pyrite is often found to replace pyrrhotite along fissures. Pyrite is a common oxidation product of pyrrhotite. The mean pyrite composition by weight percent is 53.11% S and 47.17% Fe. Microprobe work on 10 grains by SGS (SGS Rep No 11610-011) indicates that pyrite was comprised of an average of 52.9wt% S and 46.2wt% Fe.

Ivanyuk suggests that pyrrhotite-chalcopyrite is associated with the deeper portion of the deposit, while the upper portion of the deposit is dominated by pyrite-chalcopyrite. Ivanyuk suggests that initially pyrrhotite was widespread throughout the orebody and that the upper portion of the orebody was oxidized to pyrite. This observation needs to be verified as to whether this trend occurs throughout the deposit or is a localized feature in this particular drillhole. It is quite likely that oxidation is directly related to proximity to structure. As the Hannukainen deposit is hosted within a regional structural domain, the depth relationship to oxidation may vary significantly due to the depth extent and the variable intensity of structures.

Monoclinic pyrrhotite is ferromagnetic while hexagonal pyrrhotite is a non-magnetic form of pyrrhotite. The Fe content between these two end members also differs. Monoclinic pyrrhotite contains 60.5% Fe, while hexagonal pyrrhotite contains 63.5%Fe.

Ivanyuk observes that pyrrhotite occurs with chalcopyrite between grains of magnetite and silicates. Both monoclinic and hexagonal pyrrhotite was identified. The mean chemical composition of pyrrhotite was found by Ivanyuk to average 39.18% S and 60.70% Fe. Work by CL Consulting (Heinanen Aug, 2008) also evaluated the Fe and S proportions in pyrrhotite. Microprobe work by CL Consulting indicates that the Fe content returned an average of 60.3% Fe. Microprobe work by SGS (SGS Rep No 11610-001) on 11 grains indicates that pyrrhotite was comprised of an average of 39.5wt% S, and 59.03wt% Fe. This results in a formula of $Fe_{0.83}S$. This work also found that pyrrhotite contained appreciable amounts of cobalt (0.10%).

Ivanyuk observed that chalcopyrite occurs as intergrowths with pyrite and pyrrhotite and in between magnetite and silicates, as veinlets with pyrite and as lens like segregations. The chemical composition is equal to the theoretical formula. The same conclusion of no major impurities in the chalcopyrite was reached by SGS in their study on grains from drillhole HAN7009B (SGS Rep No 11610-001).

CL Consulting's work on 10 samples indicated that two gangue minerals, diopside and hornblende represent the two dominant Fe containing silicates. Microprobe work indicated that the Fe content in these two minerals is constant. CL Consulting indicates that the main gangue mineral diopside contains 12% FeO and that the main part of Fe losses in silicates is due to this mineral. This corresponds to the 11% FeO identified by Ivanyuk. Microprobe work by SGS (SGS Rep No 11610-011) indicates that pyroxene was comprised of an average of 15.4wt% FeO.

12.1.3 Davis Tube Testwork

Hooey (2008) describes Davis Tube testwork (“DTT”) on 206 assay rejects which were 5 m composites from Laurinoja (84), Lauku (63) and Vuopio (59). Recovery of Fe was calculated by removing the FeS component of the DDT magnetic concentrate using the ratio of 61.1%Fe/41.1%S derived from the CL Consulting work. A large number of samples did not show good separation of iron. A second procedure was tried by changing the magnetic flux density setting and tube speed frequency setting however recoveries decreased in the second run.

A set of 11 composites from the above intervals were sent to SGS to compare against the GTK/Labtium results. Overall recoveries were found to be nearly identical.

A report by CL Consulting was then commissioned to investigate the poor recoveries. The conclusion of the study was that the lower recoveries are not a result of complex ore structures or agglomerates but technical. The authors suggest issues with the DTT procedures including grain size, quantity of sample used, amount of water used, separation time, magnitude of the magnetic field and tube position. Quantitative mineralogical analysis (“MLA”) showed that the magnetite content in the magnetite concentrate was low ranging from 50-90%.

Discussion with Johan Hognas of Northland indicates that GTK/Labtium, which completed the DTT, had not used the Davis Tube equipment for many years and did not have an operator who was experienced with this equipment.

As there is doubt whether these results were completed by a QP, it is not recommended that the earlier 10 or 206 DTT samples be relied upon to generate recovery formulas.

A second phase of DTT was completed on a very limited population of 18 variability samples (SGS Rep No 11610-001) from drillhole HAN07009B.

A third phase of DTT was based on the cluster studies completed by SGS (GTurner-Saad, 2008) (SGS Rep No 12032-001&004). The clusters represent three grade ranges: high, medium and low grade. Weighted average Fe grades for the Cluster 1, 2 and 3 samples were 46.75, 29.34 and 13.99% Fe. These samples were pulverized to 100 microns and a 2 x 20 g of each sample was submitted for DTT with settings of intensity at 1.5 Amps, frequency at 100 stroke/minute, wash water at 1.0L/minute and wash water time of 4 minutes.

Individual Davis tube concentrates and tailings were submitted for whole rock analysis (“WRA”), S, Satmagan and BMA (ball mill accelerator) to establish recoveries. Assays from concentrates and tailings and mineralogical results were used to establish the head grade.

The 103 cluster study DTT results provide the most representative suite of samples in terms of grade distribution and distribution throughout the Hannukainen deposit. The link between recoveries generated between the individual 103 samples and the three metallurgical cluster composites is somewhat compromised as the third cluster composite contains grades which are below the modelled cut off and result in an average 13.8% Fe head grade while the mean head of the above 15% Fe cluster #3 samples used to generate recovery curves in SGS report No 12032-001&004 have a mean head grade of 17.7% Fe.

12.1.4 Satmagan

Three sets of Satmagan data are available. A historic Rautaruukki Satmagan dataset contains 190 Kuervitikko and 1,720 Hannukainen Satmagan results. As part of the cluster study, Satmagan data was collected for 103 individual samples. As part of the 2009 metallurgical sampling programme, an additional 585 Satmagans were taken. All Satmagan results indicate a good correlation with head Fe. Both the 2009 sets of Satmagan produce a much cleaner correlation than the historic Rautaruukki data. Also of interest is that the Kuervitikko Satmagan-Fe regression shows a concentration of Fe grades around the 17% range which reflects the lower grade of the overall Kuervitikko Mineral Resource.

12.1.5 Qemscan Results

Qemscan was performed by SGS on the 18 HAN7009B samples. When compared against assay values, the Qemscan indicated a fairly good agreement (Fig 7 of SGS Report No 11610-001) with a correlation of 0.9848. In addition, nearly 100% of the Cu was found to be associated with chalcopyrite.

Qemscan was run on six separate size fractions ranging from -20 micron to -600 micron. All size fractions were dominated by magnetite (48.4%) followed by pyroxenes (21.7%), amphiboles (9.19%) and feldspars (6.93%). Pyrite and pyrrhotite were represented by 2.29% and 3.22% respectively. The remainder consisted of non-opaque minerals with significant traces of chalcopyrite (0.77%).

Approximately 90% of the available Fe was recoverable as magnetite and was fairly consistent between all size fractions. The remaining Fe was largely contained within pyrrhotite (5.23%), chalcopyrite (0.60%) and pyrite (3.10%). Only 1.44% Fe was distributed within the remaining non-opaque minerals.

The sulphur was distributed amongst pyrrhotite (44.9%), pyrite (46.5%) and chalcopyrite (8.19%).

12.1.6 Weight Recovery

SGS results in Rep No 11610-001 generated a very strong correlation of 0.98 between Satmagan in the head and weight recovery (“Wt Rec”) for 18 samples with a Wt Rec in mags = 1.04*Satmagan. Weight recovery versus Qemscan FeOx in head indicated a perfect correlation of R2=1.00 with a Wt Rec in mags of 1.03*FeOx. The correlation between weight recovery and direct head Fe assay was also strong at R2=0.96 and a regression formula of Wt Rec in mags=1.66*Fe-18.10.

For the 103 cluster samples tested in SGS Report No 12032-001&004 issued in July of 2009, SGS generated correlation plots in order to assess recovery relationships. Head assays below 15% Fe were not included in these regressions. The relationship between Satmagan in the head and weight recovery was very strong on R2=0.97. SGS notes that the relationship between Qemscan and weight recovery was also very good at R2=0.96. The relationship between WRA Fe assay and the weight recovery indicated the best correlation of R2=0.99. A weight recovery in mags regression was generated where Wt Rec in mags = Fe in Head - 1.59-13.86.

12.1.7 Fe Recovery and Recoverable Fe

The SGS Report No 116110-001 regressions of Fe recovery % versus Fe Head by WRA % indicate a weaker correlation. Correlations between recoverable Fe by DTT were very good at R^2=0.97 and correlation between recoverable Fe by DTT and Satmagan was R2=1.00. Correlation between S Head Grade and S concentrate grade indicated a fair amount of scatter with a R^2= 0.66. Correlation between S head grade and magnetic concentrate grade indicated a scattered trend with no relationship with recovery. A slight relationship occurred between pyrrhotite head and S grade and recovery at R^2= to 0.52 and 0.55 respectively.

The SGS Report 12032-001&004 were based on the 103 Global Cluster samples. The testwork indicated that the relationships for weight, recoverable Fe and S content are very good, while the SiO_2 model was slightly more scattered. SGS concluded that the weight recovery and recoverable Fe models only required Fe head assays. A global model which predicts overall Fe recovery after pyrrhotite flotation was developed which showed that the Fe losses in the sulphides were directly proportional to the S grade in the LIMS concentrate. Head versus Fe recovery indicated a correlation of R^2=0.78. The relationship between Fe in the head and Satmagan and Davis tube both indicated strong correlations at R2=0.99 and R^2=0.99 respectively.

It was recommended that Fe recovery for the Hannukainen PEA could either be reasonably based on the 2009 Satmagan results or the >15% Fe Davis Tube cluster study sample results contained in the SGS Rep No 12032-001&004. For the purposes of the PEA it was recommended that the recovery function for Hannukainen also be assumed for Kuervitikko until the initial set of Kuervitikko DTT results become available.

12.2 Preliminary Hannukainen metallurgical recovery functions

In 2011, Recovery functions were developed for the Hannukainen deposit by Bo Arvidson of Bo Arvidson Consulting LLC as part of on-going metallurgical testwork for the HFS.

Extracts from a memo by Bo Arvidson entitled “Preliminary Hannukainen Metallurgical Recovery Functions based on limited data”, dated March 2011 are quoted here to describe the generation of the recovery functions that have been utilised by SRK in generating the Mineral Resource Statement.

12.2.1 Discussion of SGS data

The previous studies undertaken by SGS and described above shows the development of linear magnetic product and Fe recovery functions based on QEMSCAN (“QS”) Fe oxide determinations, Satmagan data and elemental head assays of Fe by X-ray fluorescence (“XRF”) in WRA. Due to the relatively fine grinding that is required, problems can occur in the separation process with small particles being carried into the wrong product stream. This was studied by SGS and explains the surprisingly modest magnetite recovery described. This is one of several aspects that will be further investigated in the on-going HFS testwork programmes.

The previous testwork shows that a strong relationship between the Davis Tube magnetic weight recovery and the head Fe grade by WRA, as shown in Figure 12-1, has determined that there is no Fe recovery at the head grade of less than 10.9% Fe.

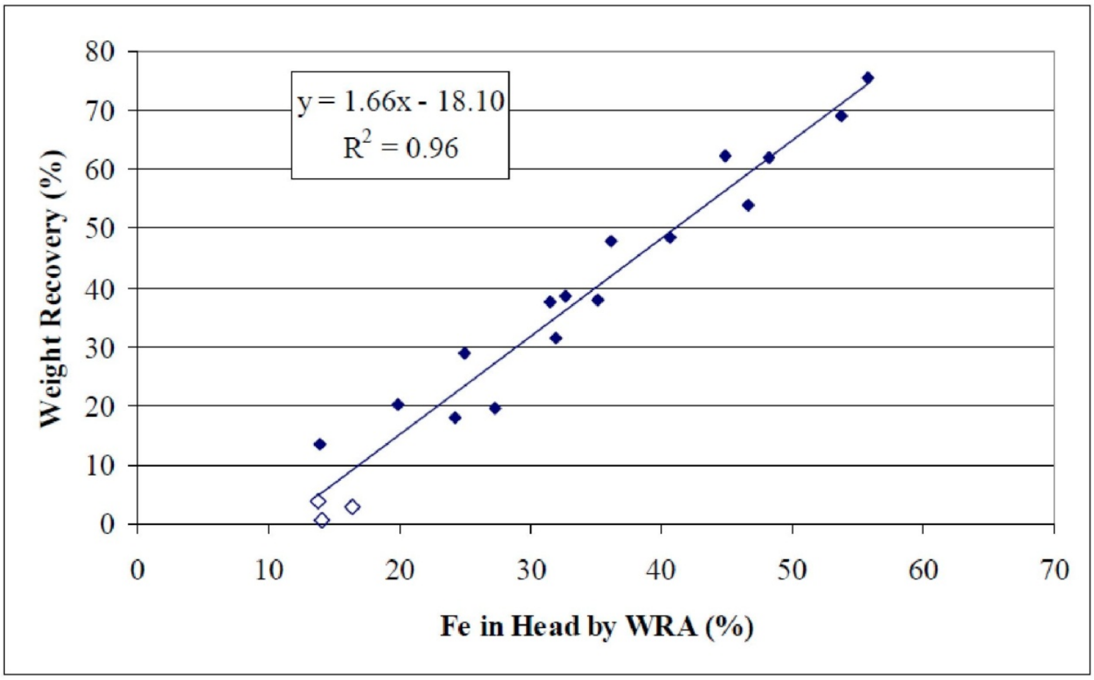


Figure 12-1: Correlation between Direct Fe and Weight Recovery

The previous testwork by SGS did not attempt to develop a recovery functions for Cu or Au, with the main focus of the testwork to establish and understand the effect of sulphide (pyrite and pyrrhotite) content on S grade and recovery in the magnetic concentrate. No meaningful relationship was subsequently generated as illustrated in Figure 12-2.

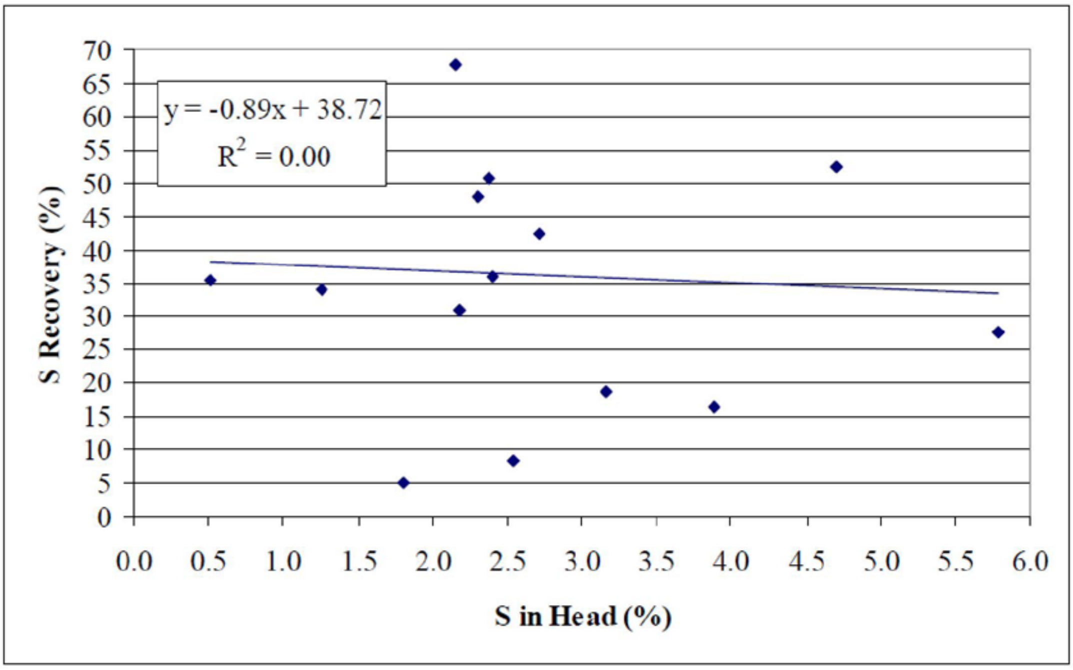


Figure 63: Correlation between S Head Grade and S Recovery

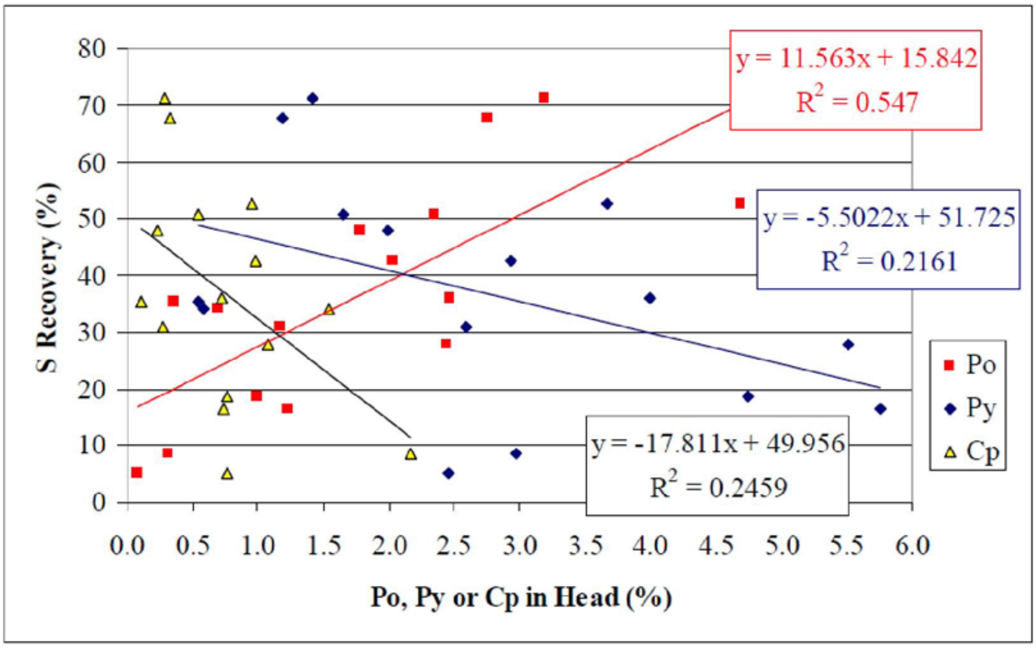


Figure 65: Correlation between Po, Py and Cp Head Grade and S Recovery

Figure 12-2: Correlations between S head Grade and S recovery (Figure 63) and correlation between Po, Py, and Cp Head Grade and S recovery (Figure 65)

12.2.2 Development of preliminary Fe, Cu and Au recovery functions

The recovery function developed and highlighted below utilise samples from a single drillhole and it is clear that the samples utilised are not sufficient enough in quantity to represent the various metallurgical and geological domains identified at Hannukainen.

The proposed recovery function for $Fe_{magnetite}$ made use of all the bench test metallurgical data, including the zero recovery point as determined by the Davis Tube magnetic recovery vs head Fe grade. A high power polynomial function provided the best fit as shown in Figure 12-3 to Figure 12-5 for Fe, Cu and Au respectively.

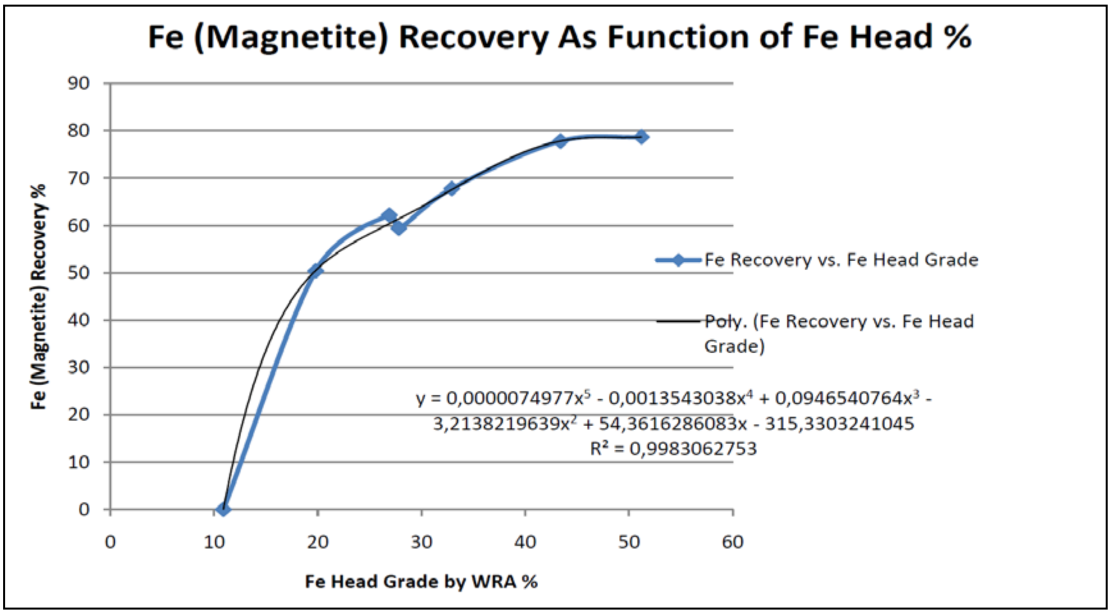


Figure 12-3: Fe processing recovery function

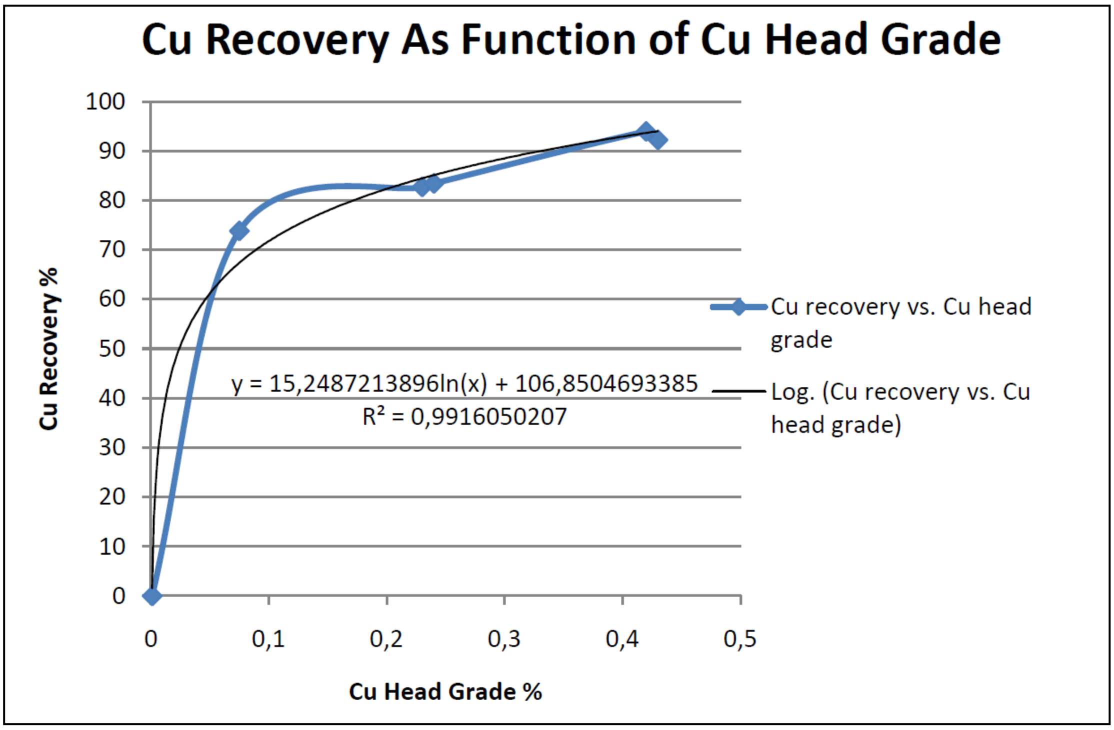


Figure 12-4: Cu processing recovery function

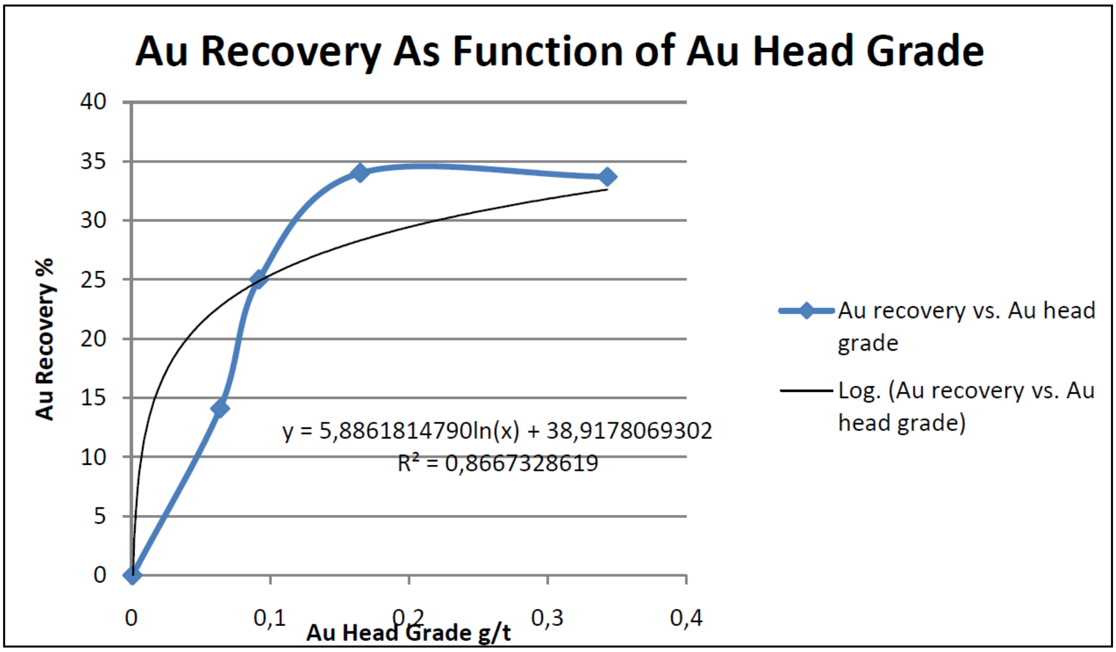


Figure 12-5: Au processing recovery function

12.2.3 2012 Testwork Update

Additional metallurgical testwork has been completed during 2012 by the Lulea University of Technology with the work programme being managed by Dr Pertti Lamberg (Professor in Geometallurgy). This is discussed in section 16.

13 MINERAL RESOURCE ESTIMATE

13.1 Introduction

A maiden Mineral Resource Estimate has been produced and classified using the guidelines approved by the CIM. SRK's Howard Baker managed the Mineral Resource Estimation.

13.2 Statistical Analysis – Raw Data

A statistical study of the available Hannukainen data was undertaken to determine suitable geological domains to be used in the Mineral Resource Estimation. Using the Northland geological logging, all rock codes relating to the skarn body were analysed. Table 13-1 shows the minor lithologies within the skarn/magnetite, skarn/clinopyroxene and skarn major lithology, their formulae, and their theoretical iron content. Of these lithologies, magnetite skarn ("MGTS") represents 45% of the logged skarn intervals, clinopyroxene skarn ("CPXS") represents 45%, with the others representing <1% of the total skarn intervals. It is clear from the %Fe Total from each lithology that it could be possible to have elevated Fe grades from non-Fe oxide minerals, which must be considered for domaining purposes.

Table 13-1: Formulae and iron content of all skarn minor lithologies

Minor Lithology	Formula of Main Mineral	%Fe Total content of Main Mineral
ACTS: actinolite skarn	$Ca_2(Mg,Fe)5Si_8O_{22}(OH)_2$	Variable: 0 - 29%
CHLS: chlorite skarn	$(Mg,Fe)_3(Si,Al)_4O_{10}(OH)_2{\cdot}(Mg,Fe)_3(OH)_6$	Variable: <10%
CPAS: clinopyroxene-actinolite skarn	$XY(Si,Al)_2O_6$ (XY could be Na, Ca, Mg, Fe, Li) and $Ca_2(Mg,Fe)5Si_8O_{22}(OH)_2$	Variable (depends on pyroxene: augite = 12%, aegirine = 24%, and actinolite ratio): 0- 24%
CPXS: clinopyroxene skarn	$XY(Si,Al)_2O_6$ (XY could be Na, Ca, Mg, Fe, Li)	Variable (depends on pyroxene: augite = 12%, aegirine = 24%): 0-24%
HBLS: hornblende skarn	$Ca(Mg,Fe)_4Al(Si_7Al)O_{22}(OH,F)_2$	Variable (depends on iron-content: Hornblende = 1.7%; ferrohornblende = 25%): 1.7 - 25%
MGTS: magnetite skarn	Fe_3O_4	72.4%
SCAS: scapolite skarn	Na4[Cl(AlSi3O8)3]-Na4[CO3(Al2Si2O8)3]	0%
SRPS: serpentine skarn	$(Mg, Fe)_3Si_2O_5(OH)_4$	Variable (depends on serpentine type: clinocrysotile = 0%; antigorite = 14%; lizardite = 0%): 0 - 14%

Northland geologists report that when logging, for an interval to be considered MGTS, a 30% visual magnetite cut-off is used to differentiate between all the other skarn types. Consequently, although the majority of the high grade (and with it, the magnetite) exists within the magnetite skarn, due to the logging method adopted by Northland, it is inevitable that a relatively high magnetite content could also exist in all the other lithologies, and cannot be used exclusively to determine magnetite bearing domains.

Assay Histograms

Figure 13-1 shows the %Fe Total histogram for all major lithology intervals logged as skarn. It shows two clear populations with a natural break at around 30% Fe Total. Figure 13-2 shows Fe% for all skarn intervals logged as MGTS as the minor lithology. It clearly shows a single population, which accounts for the high grade spike in Figure 13-1. Figure 13-3 shows %Fe Total for all skarn intervals logged as CPXS as the minor lithology. It clearly shows a single, population which accounts for the low grade spike in Figure 13-1

SRK can deduce from this that the logging by Northland geologists appears to be highly accurate, which allows for a high degree of confidence in the logged data. However, due to the presence of very high grades (>40% Fe Total) within the CPXS, along with some very low grades in the MGTS, minor lithology cannot be used as the sole domaining tool.

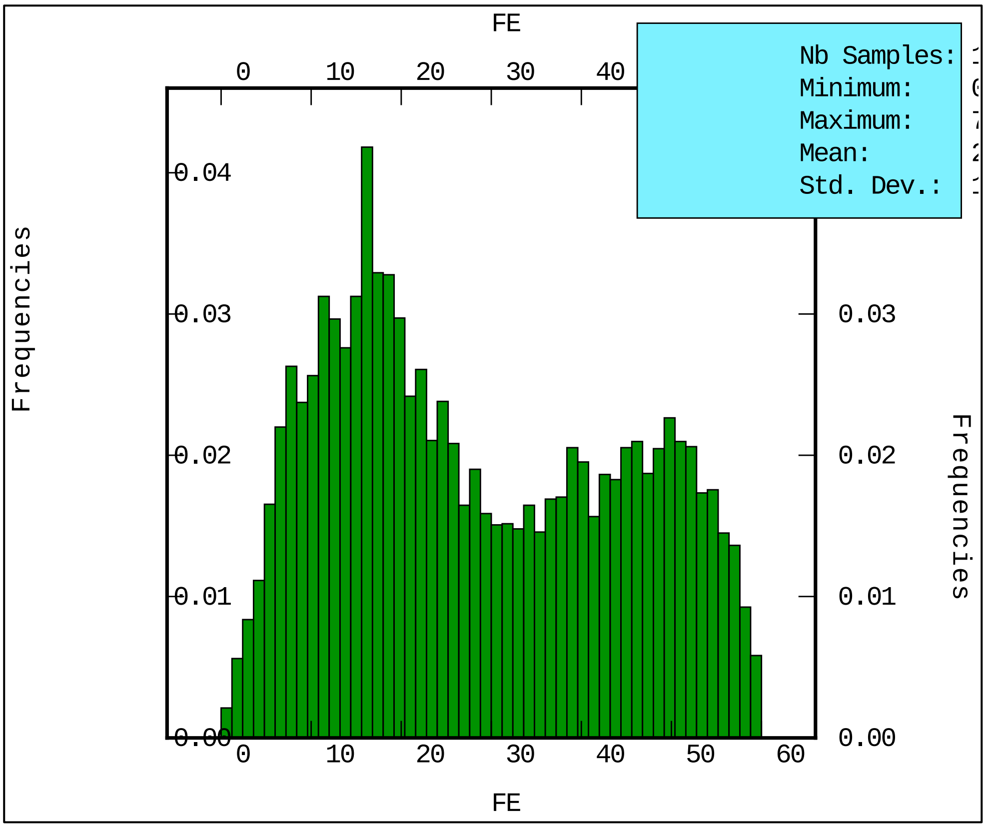


Figure 13-1: Histogram of %Fe Total within skarn major lithology intervals

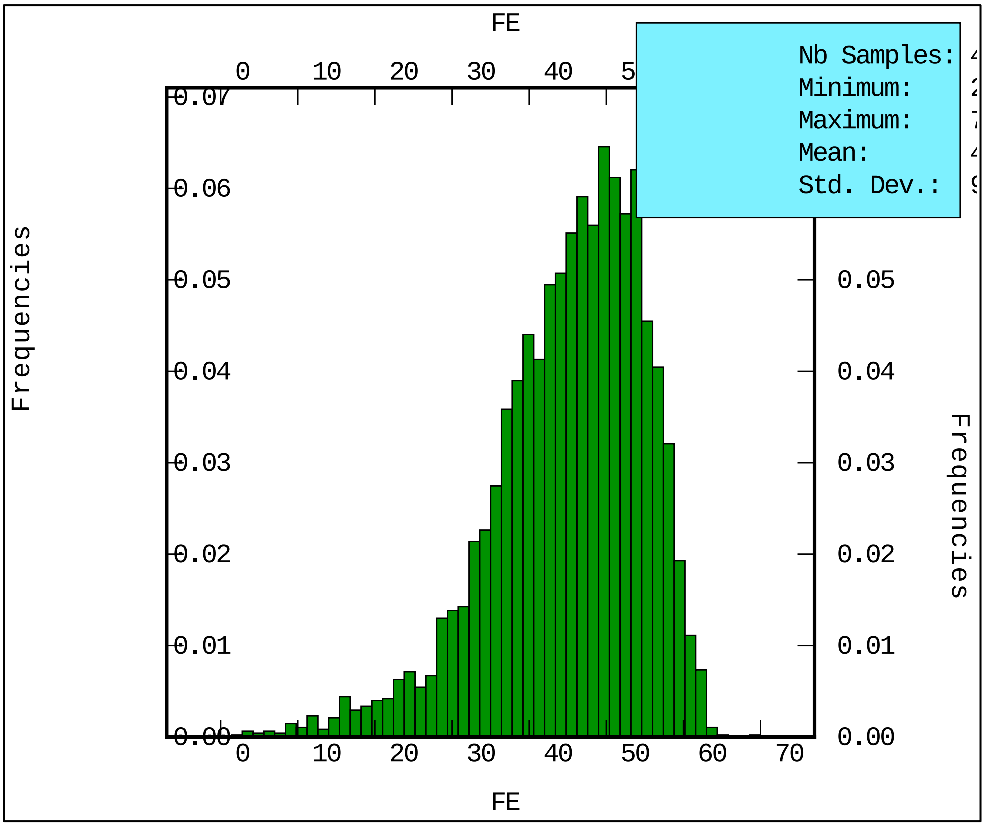


Figure 13-2: Histogram of %Fe Total within skarn major lithology intervals and MGTS minor lithology intervals

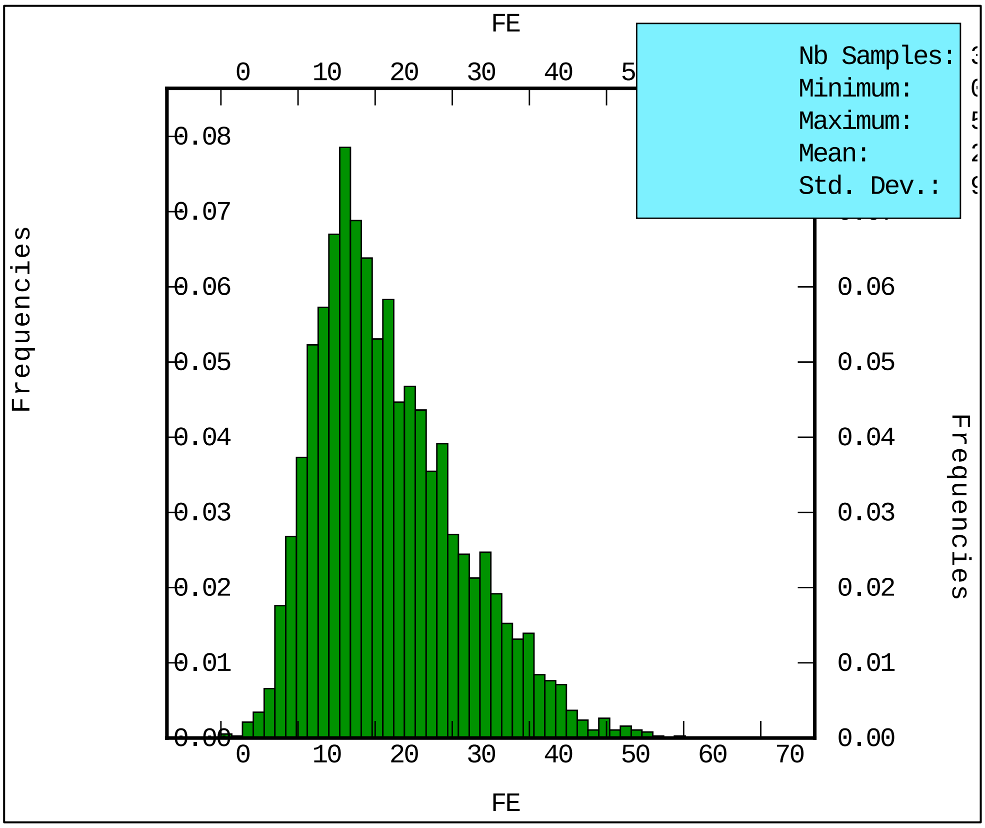


Figure 13-3: Histogram of %Fe Total within skarn major lithology intervals and CPXS Minor Lithology Intervals

Assay Probability Plots

To define theoretical domaining tools, further statistical analysis was undertaken. Figure 13-4 shows the %Fe Total probability plot for all skarn intervals. It shows population breaks at around 15% and 50%. The MGTS/CPXS divide is also shown. Figure 13-5 shows the %S probability plot, with breaks around 3%, 6% and 12%. The 6% break shows the largest inflection; however, the plot is reasonably smooth and is not clear.

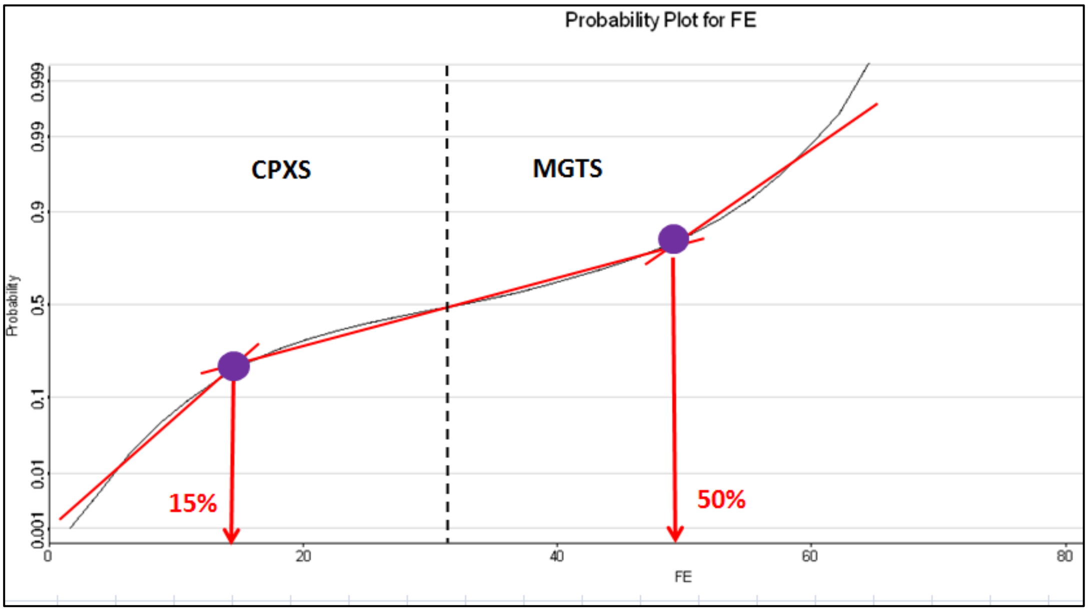


Figure 13-4: Probability plot for all skarn intervals: %Fe Total

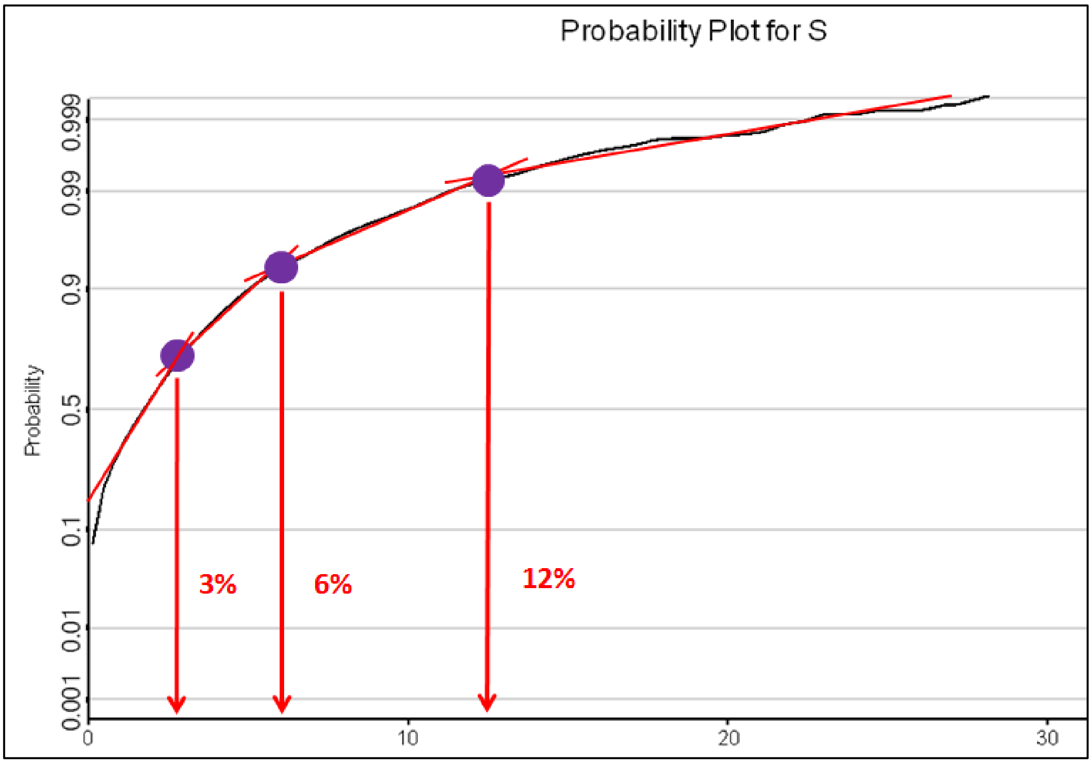


Figure 13-5: Probability plot for all skarn intervals: %S

Figure 13-6 shows the probability plot for Cu (ppm), with a major break at 1000 ppm and a minor break around 2500 ppm.

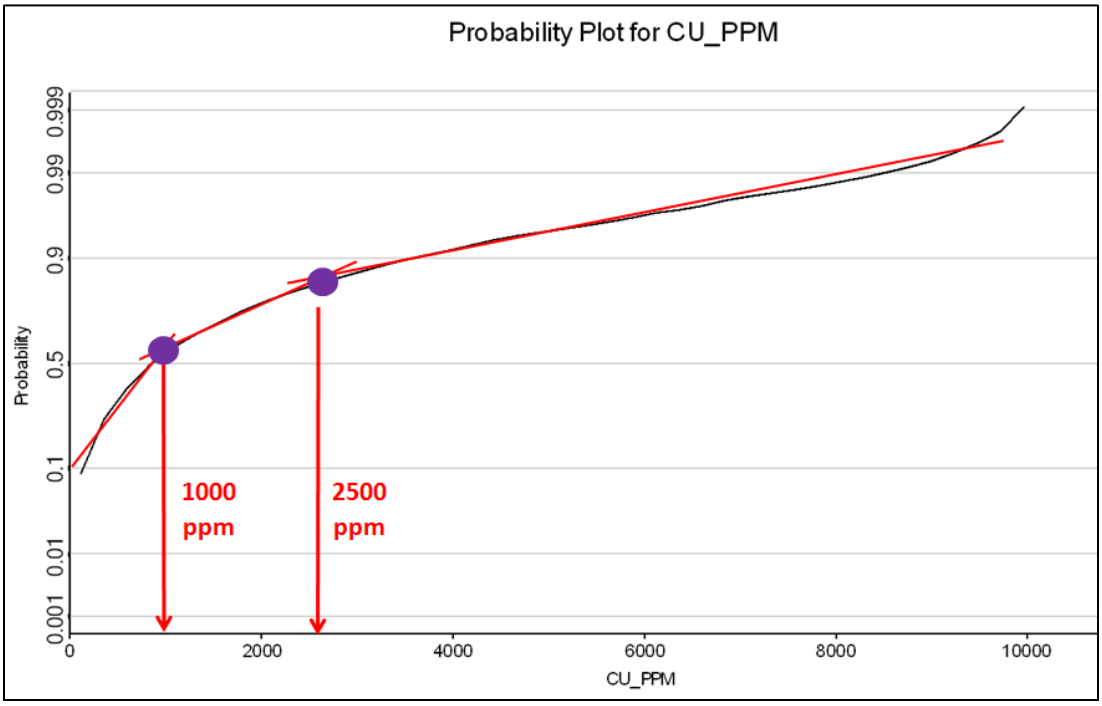


Figure 13-6: Probability plot for all skarn intervals: Cu (ppm)

Assay Scatterplots

The probability plots above provided additional evidence for domaining, but cannot be the only domaining tool due to the ambiguous nature of some of the population breaks.

Figure 13-7 shows a scatterplot of %Fe Total against %S for all skarn intervals. It is clear that the material logged as MGTS has a generally tight cluster of >30% Fe Total, <6% S, with some scattered high S values. The CPXS data shows a larger spread, with the majority of data <4% S and >10% Fe Total. The majority of the "other" data, including intervals logged as skarn, amphibolite, and schist contains <20% Fe Total, and <2% S. Again, the populations are blurred and domain boundaries are difficult to define precisely. There is no clear Fe sulphide population where Fe and S increase steadily together (shown by the blue arrow).

Figure 13-8 shows a scatterplot of Cu (ppm) against %S for all skarn intervals. It also shows blurred populations, with the MGTS, CPXS and other lithologies all containing high and low Cu, related to S grades of generally <6%. The strongest trend shows an increase in Cu with a relatively stable S grade. This likely represents the chalcopyrite population, which is the key Cu-bearing mineral in Hannukainen. The 1000 ppm population identified in Figure 13-6 is not as clear, but still shows the vast majority of data <6% S. Figure 13-8 also shows a second trend of data, with increasing S not related to increasing Cu. This is related to the non-Cu bearing sulphides present, such as pyrite and pyrrhotite.

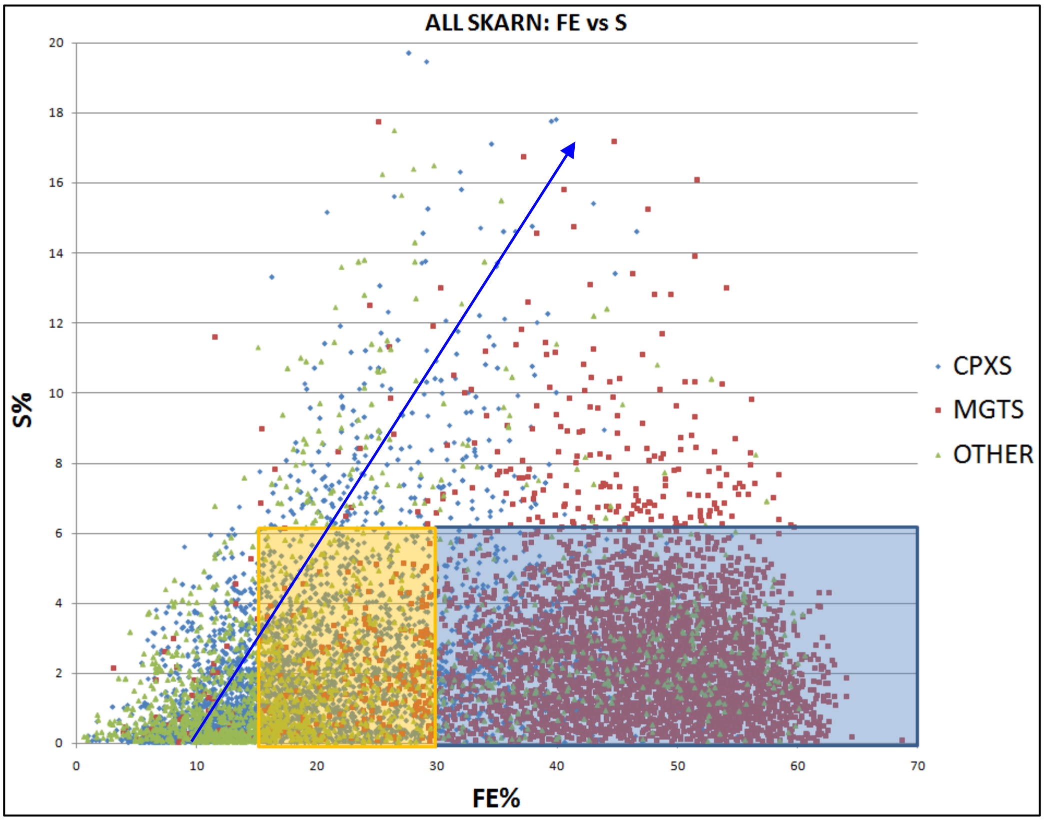


Figure 13-7: %Fe Total vs %S for all skarn intervals (blue box = high-grade Fe; yellow box = low grade Fe)

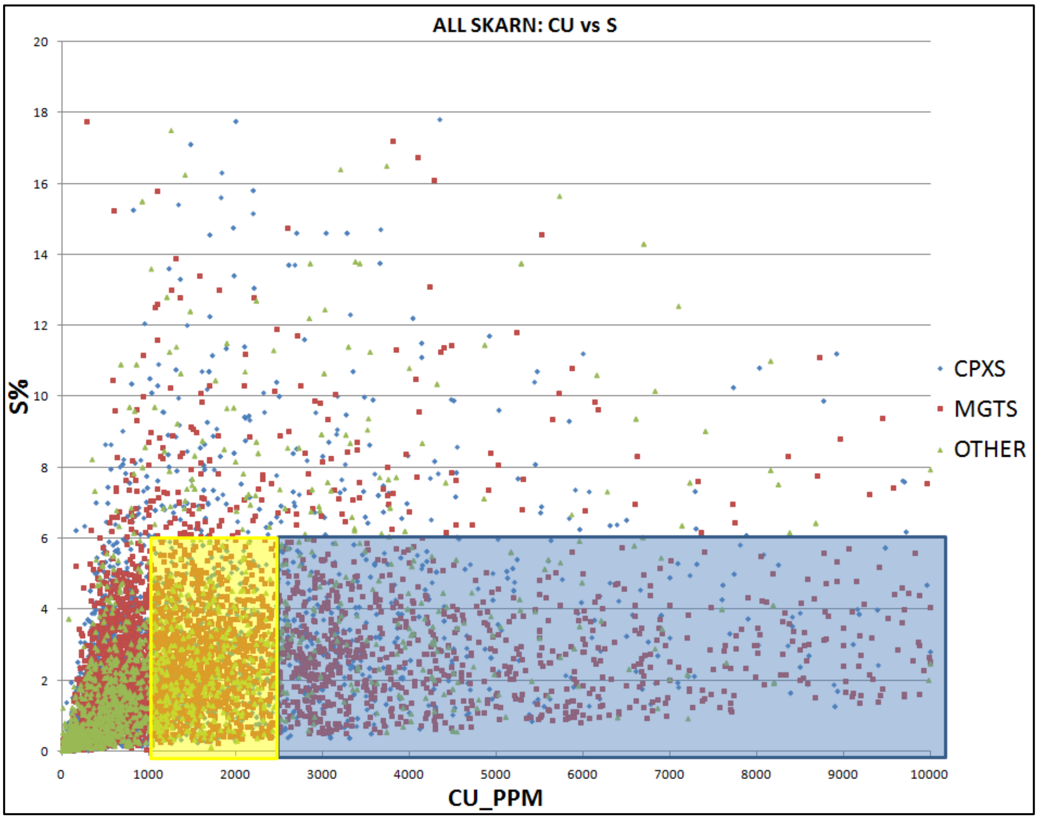


Figure 13-8: Cu (ppm) vs %S for all skarn intervals (blue box = high grade Cu; yellow box = low grade Cu)

Magnetic Susceptibility

Over 41,000 magnetic susceptibility readings were taken at Hannukainen, spanning the historical and modern drilling campaigns. One reading was taken per metre of core, in one position along the metre length. Whilst this is not considered to be representative of the entire core length, clear trends are observed in the data. Figure 13-9 shows the magnetic susceptibility readings against %Fe Total for MGTS and CPXS logged intervals. There is a large scatter of data, however, the majority of the CPXS data falls below 0.5. The MGTS data also contains readings below 0.5, and so this cannot be used as a hard boundary.

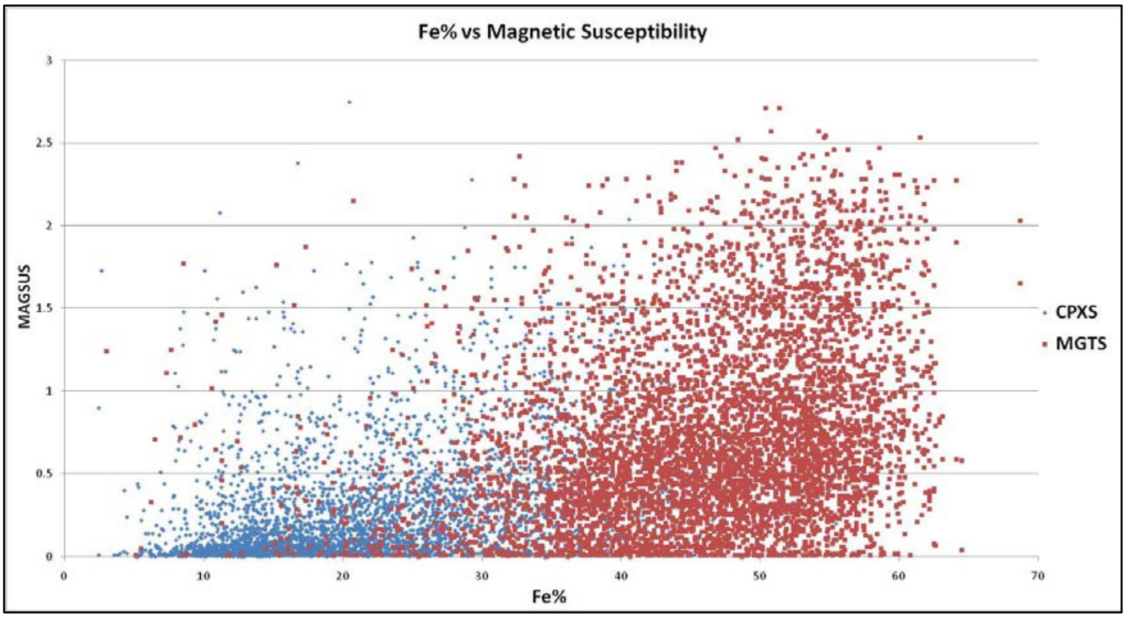


Figure 13-9: %Fe Total vs Magnetic susceptibility

Satmagan

Only 580 Satmagan analyses were taken at Hannukainen, due to a large number of Davis Test Tube recovery testwork analyses being undertaken. The lack of data means they could not be used as a domaining criterion, but where present were used to confirm the magnetic mineralisation.

13.2.1 Theoretical Domaining

The theoretical domaining criteria used by SRK to outline mineralisation zones was mainly based on the assay statistics. A cut-off of >15% Fe Total was proposed in conjunction with a ceiling S cut-off of 6% for the mineralisation envelope. A cut-off of 30% Fe Total was used to define an internal and continuous high grade domain. It was not possible to develop an additional and continuous three dimensional domain with Fe values in excess of 50% as identified in the statistical study and shown in Figure 13-4. In addition, Cu-domains are proposed where the Fe mineralisation envelope does not enclose the Cu mineralisation. Domain boundaries of 1000 ppm for low grade, and 2500 ppm for high grade, are proposed. In addition, magnetic susceptibility and Satmagan readings can be used to increase accuracy around mineralisation boundaries with low Fe or Cu, or elevated S, values.

Geological cross-sectional interpretations by Northland also contain useful information about the nature of the geometry and dip and strike of the mineralisation lenses.

At the Project, Fe, Cu, Au and cobalt ("Co") are all of potential economic interest and may all be pivotal in controlling the economic viability of the Project. For this reason, they all must be considered when domaining and estimating. The Fe mineralisation is by far the most prevalent and has previously been the most important commodity driving the economic potential. SRK has prioritised the domaining of the Fe due to this reason. The high grade Cu mineralisation is more localised, and mainly contained within the Fe mineralisation. The Au and Co mineralisation is entirely contained within the Fe and Cu.

13.2.2 Mineralogical Problems

The presence of Cu-bearing sulphides, mainly in the form of chalcopyrite, has led to elevated S values across the deposit. There is also elevated S associated with other Fe-sulphides, such as pyrite and pyrrhotite. These non-Cu sulphide minerals are difficult to domain-out, as they are pervasive and are often associated with the chalcopyrite. Using a sulphur ceiling, in combination with magnetic susceptibility/Satmagan readings, the majority of pyrite should be excluded from wireframes, where it is not associated with chalcopyrite. The pyrrhotite has generally been found to be of the magnetic variety, and so could cause additional complications when distinguishing between magnetite, especially if the pyrrhotite contains low S grades.

In general, the Fe sulphides are spotty and comprise a small percentage of the mineral assemblage, and so should not have a large adverse effect on the Project.

13.2.3 Actual Mineralisation Domaining and Modelling

The deposit modelling was conducted in Datamine Studio 3 and Surpac software and comprised the following:

1. Importing the collar, survey, assay, geology, Satmagan and magnetic susceptibility data into Datamine to create a de-surveyed drillhole file.
2. Importing the topography data file (former pit profiles had to be digitised into the topographic wireframe due to the topographic data using the pit lakes as the surface; this surface was provided by Northland to SRK).
3. The creation of mineralisation wireframes in Datamine based on the domain guidelines set out above.
4. The creation of waste lithology wireframes in Datamine, based on lithology codes generated by Northland geologists.
5. The creation of an empty block model in Surpac coded by ZONE and Copper Zone ("CUZONE") to distinguish the different geological domains, including mineralisation and waste lithologies.
6. Importing the block model into Datamine to undertake the interpolation process.

In practice, the domaining criteria worked effectively and separate domains were created using the criteria described above. Datamine Studio 3 was used to create strings around the suggested domains, which were in turn used to create 3D-wireframes. In total, 7 Fe ore domains, four Cu ore domains, and three internal waste domains were modelled.

In general, wireframes were extended one drillhole distance down-dip and up-dip, and half a drill spacing along strike in both directions. Where the domains were seen to pinch-out, the wireframes were shortened, which is often the case with the high grade lenses.

A low grade (15-30% Fe Total) Fe mineralisation shell was created for the Kuervaara and Vuopio areas combined. These areas showed good geological continuity and so it was decided not to separate them based on their historical names. An internal high grade core was created within the low grade halo; this was based on a >30% Fe Total, <6% S cut-off. An internal waste domain was created comprising several lenses of <15% Fe Total, generally being logged amphibolite material. No additional Cu domains were created, as the majority of the mineralisation contained low to medium (approximately 1000 ppm) Cu grades. Small areas of higher grade Cu were observed, but were not deemed continuous enough to domain separately. The Kuervaara/Vuopio wireframes are shown in Figure 13-10 and Figure 13-11.

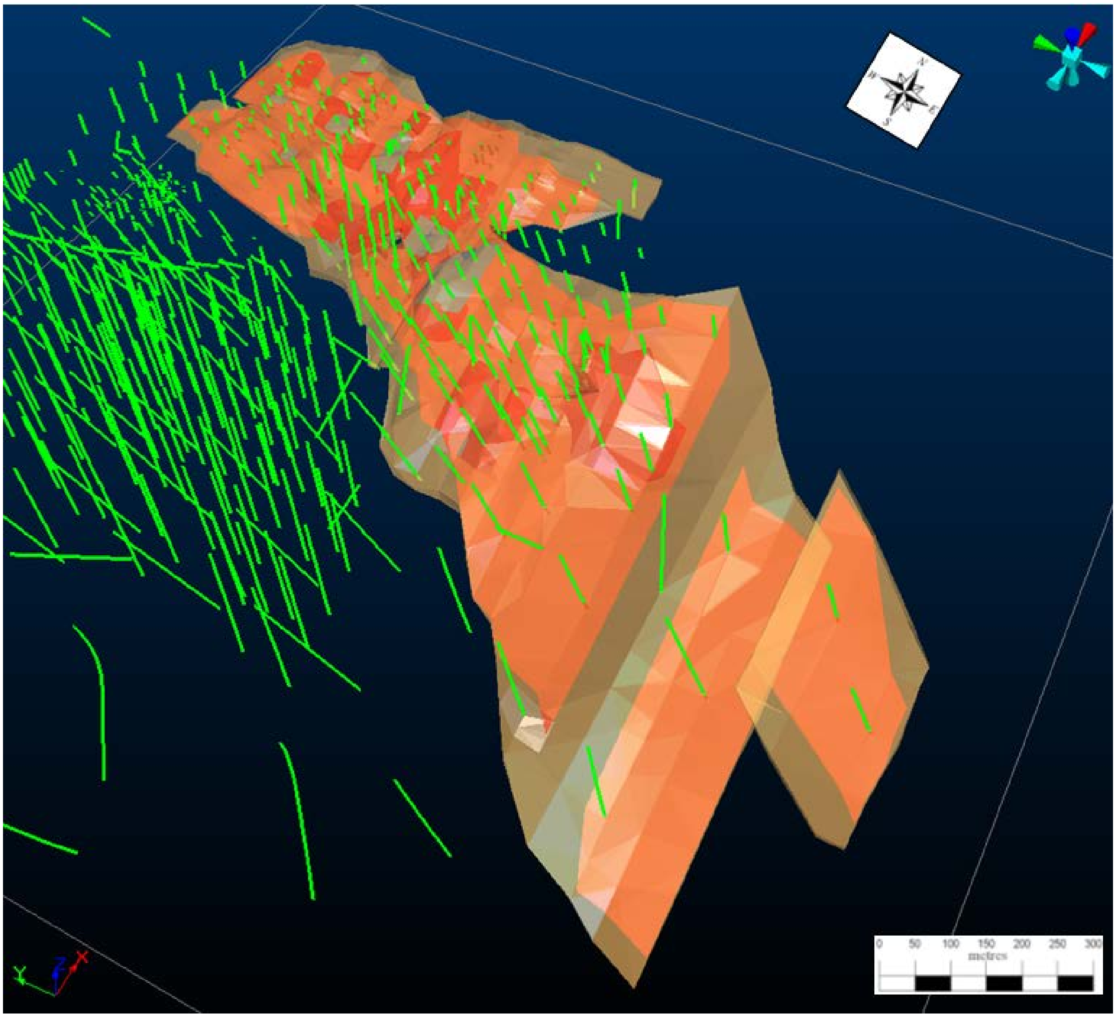


Figure 13-10: Kuervaara/Vuopio low grade Fe (Zone 110, pale orange), high grade Fe (Zone 111, red) and internal waste wireframes (Zone 119, grey) (Source: SRK Oct 2012)

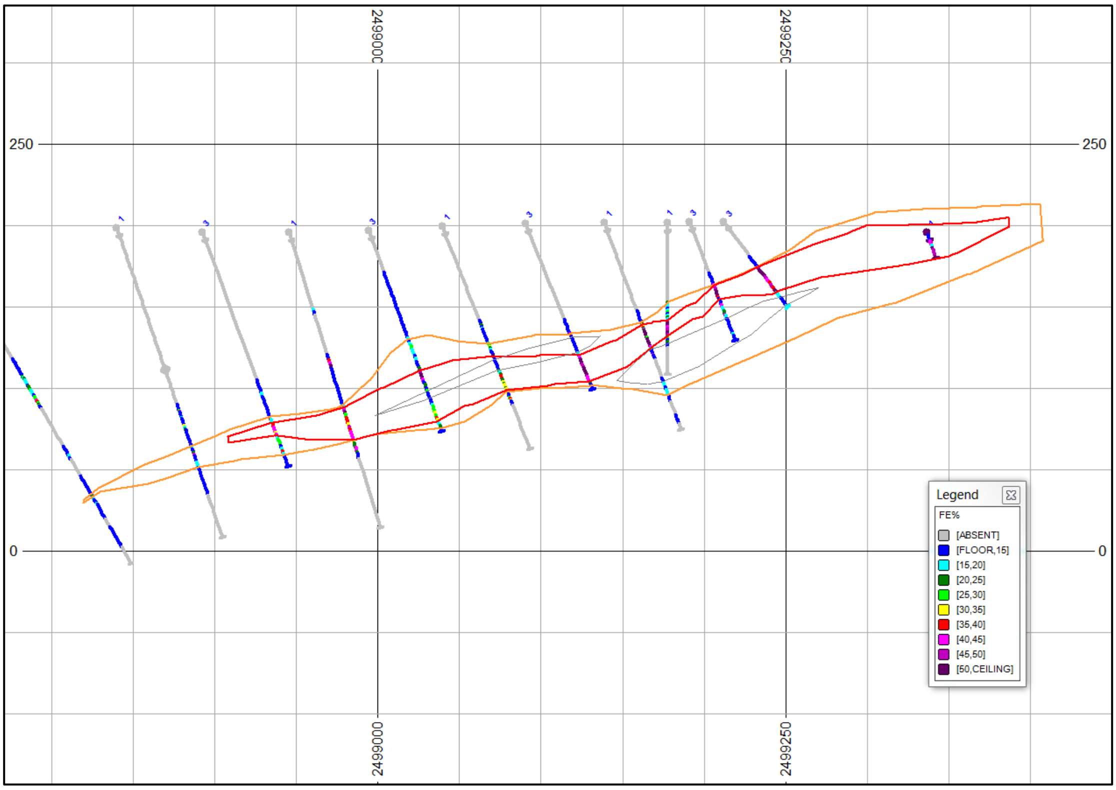


Figure 13-11: XS6400 Cross-section through Kuervaara, showing low grade, high grade and internal waste wireframes. Drillhole code: 1 = pre-2011 holes; 2 = 2011-2012 infill holes; 3 = 2011 metallurgical holes; (Source: SRK Oct 2012)

The Laurinoja, Lauku and Kivivuopio lenses were also domained together as one mineralisation unit due to the high continuity between the areas. Again, a low grade Fe halo and an internal high grade Fe domain was created, both showing excellent sectional continuity. Several internal waste lenses were also created. A clear high grade Cu domain exists, contained within the Laurinoja area, which was separated from the Fe mineralisation using a 2500 ppm cut-off (but no S cut-off). This area was highlighted in a study by Johnson (2010), which indicated prospective Cu-Au zones at the Hannukainen deposit. It sits at a slightly oblique angle to the Fe-mineralisation, and is thought to represent a secondary structure which probably post-dated the magnetite deposition. The majority of this domain falls within the high grade Fe domain, but can also occur in the low grade halo and outside of the Fe mineralisation.

The Laurinoja/Lauku/Kivivuopio wireframes are shown in Figure 13-12 to Figure 13-17.

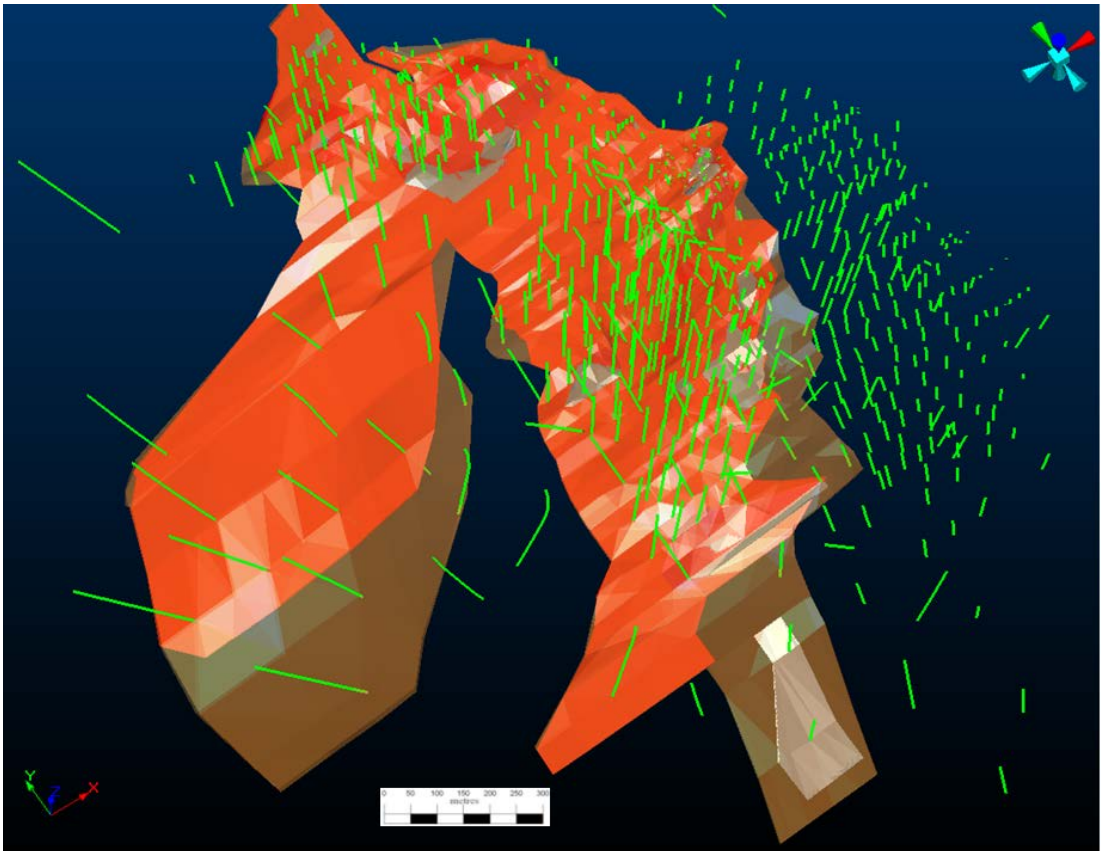


Figure 13-12: Laurinoja/Lauku/Kivivuopio low grade Fe (pale orange), high grade Fe (red) and internal waste wireframes (grey) (Source: SRK Oct 2012)

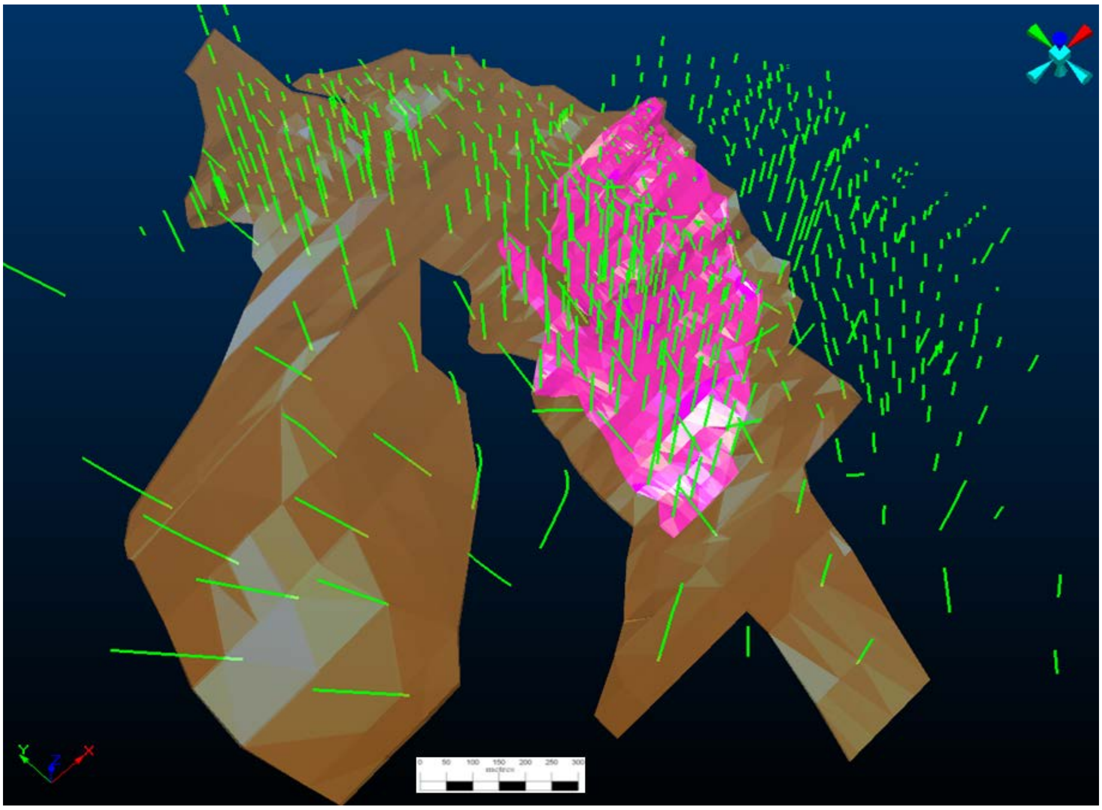


Figure 13-13: Laurinoja high grade Cu wireframe (CUZONE 3, pink) (Source: SRK Oct 2012)

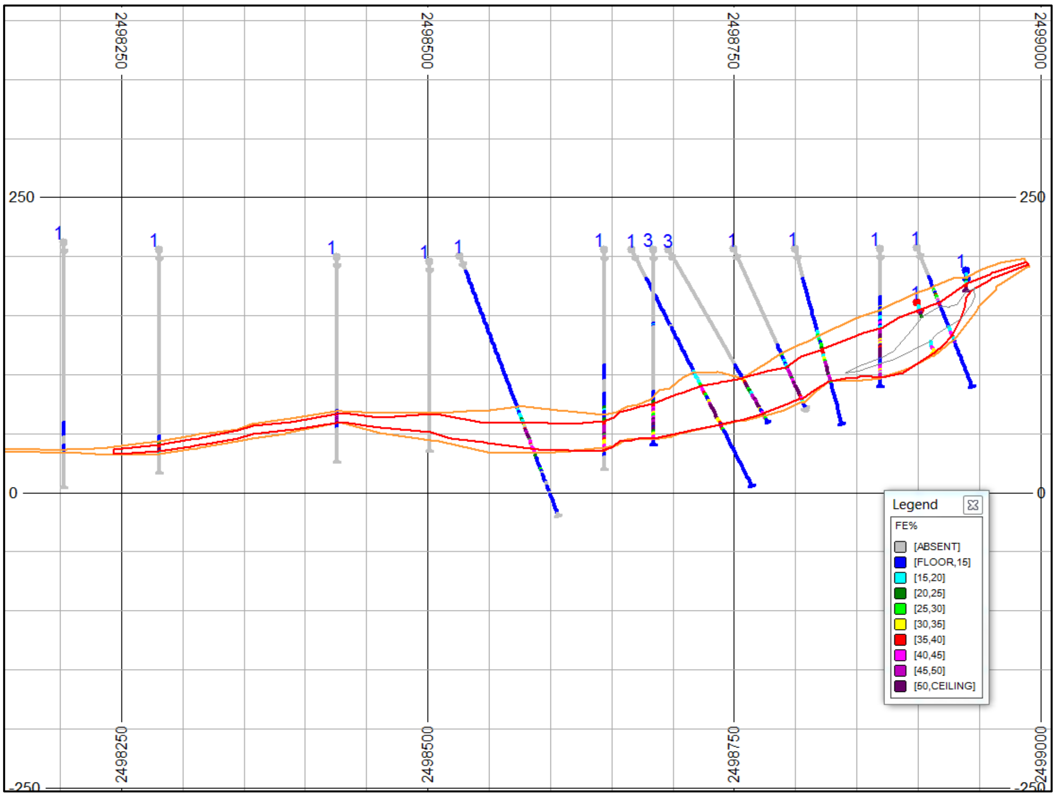


Figure 13-14: XS6900 Cross-section through Laurinoja, showing low grade Fe (orange), high grade Fe (red) and internal waste (grey) wireframes (Source: SRK Oct 2012)

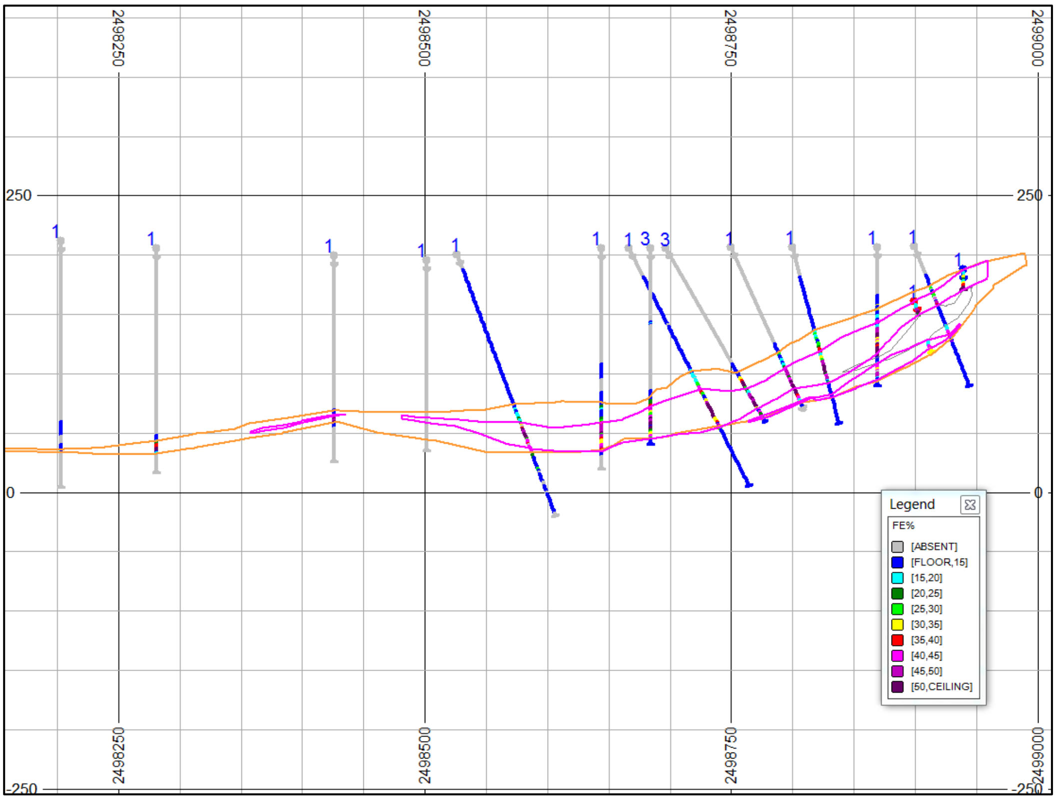


Figure 13-15: XS6900 Cross-section through Laurinoja, showing low grade Fe (orange) and high grade Cu (pink) wireframes (Source: SRK Oct 2012)

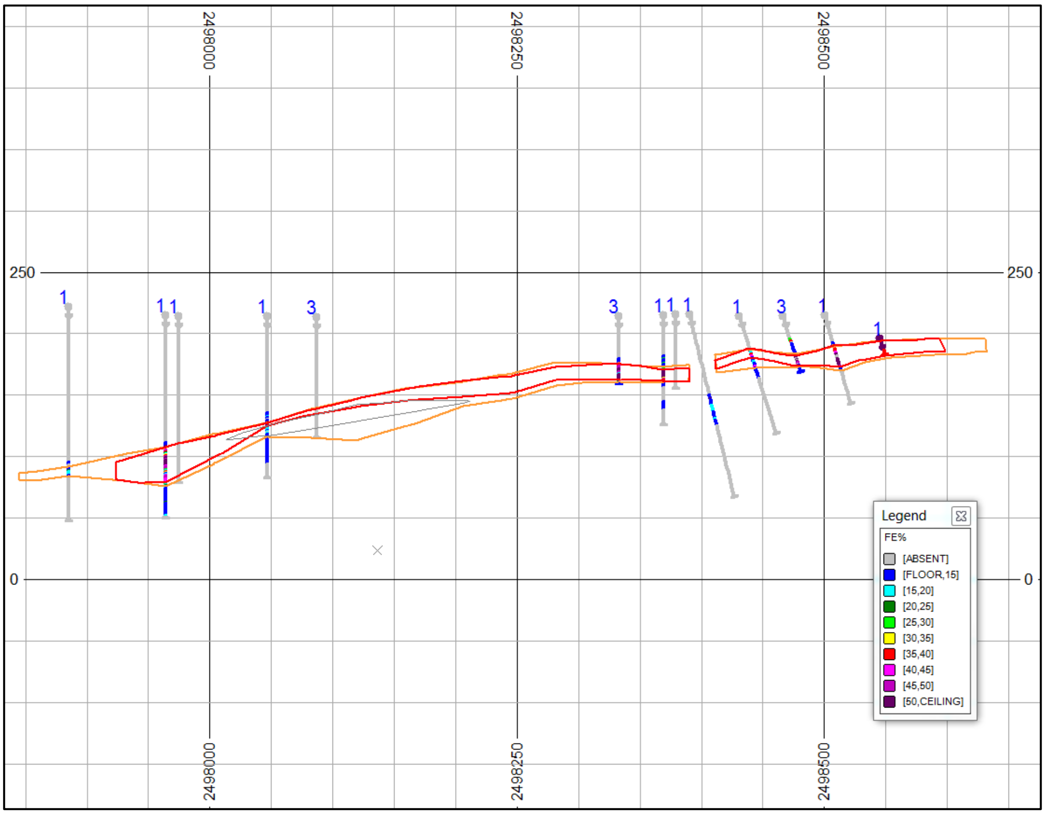


Figure 13-16: XS7700 Cross-section through Lauku, showing low grade Fe (orange), high grade Fe (red) and internal waste (grey) wireframes (Source: SRK Oct 2012)

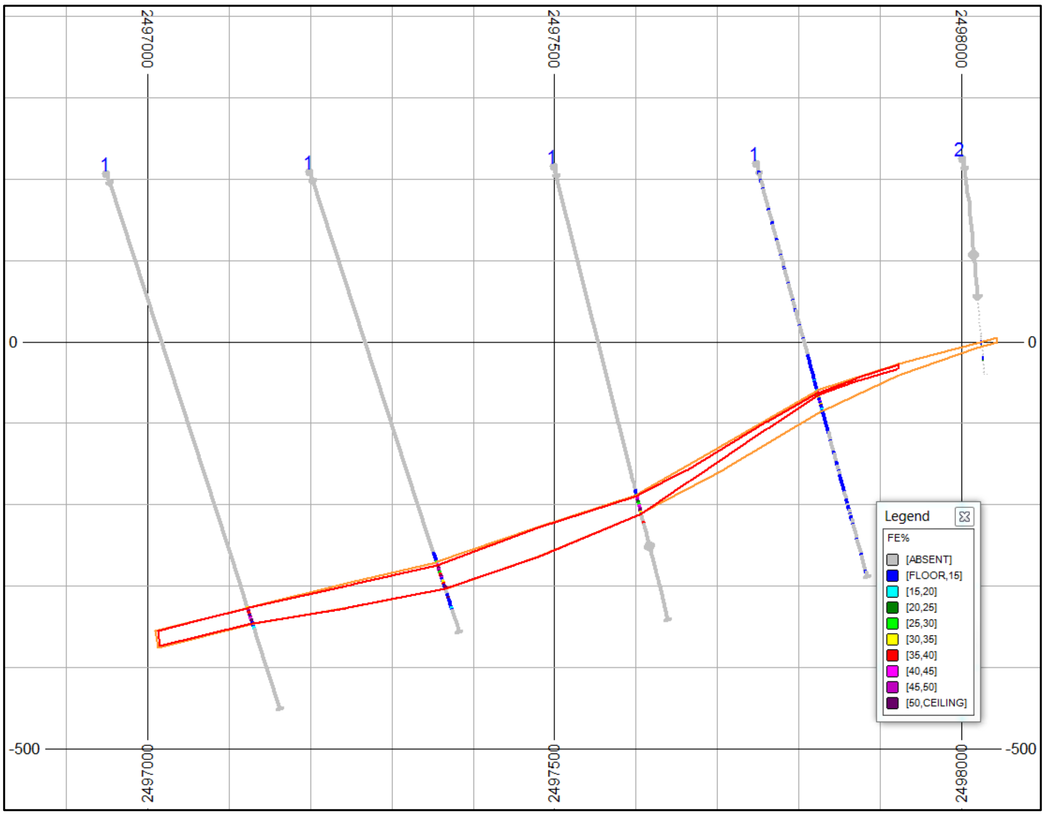


Figure 13-17: XS6800 Cross-section through Kivivuopio, showing low grade Fe (orange) and high grade Fe (red) wireframes (Source: SRK Oct 2012)

Kuervitikko contains lower grades than that of the main Hannukainen zones, with the absence of a high grade core in most sections. It was apparent on some sections, but not deemed continuous enough to allow for separate domaining. A general Fe domain of material >15 % Fe Total, <6% S was therefore created. Kuervitikko also contains elevated Cu grades, but it more homogenous than that of Laurinoja. There are several areas where low grade Fe and high grade Cu exist, lying outside of the principal Fe domain so it was considered appropriate to create a separate domain in order to encapsulate these Cu grades. A grade of >1000 ppm Cu was used as an approximate cut-off, with no S ceiling.

The wireframes for Kuervitikko are shown from Figure 13-18 to Figure 13-20.

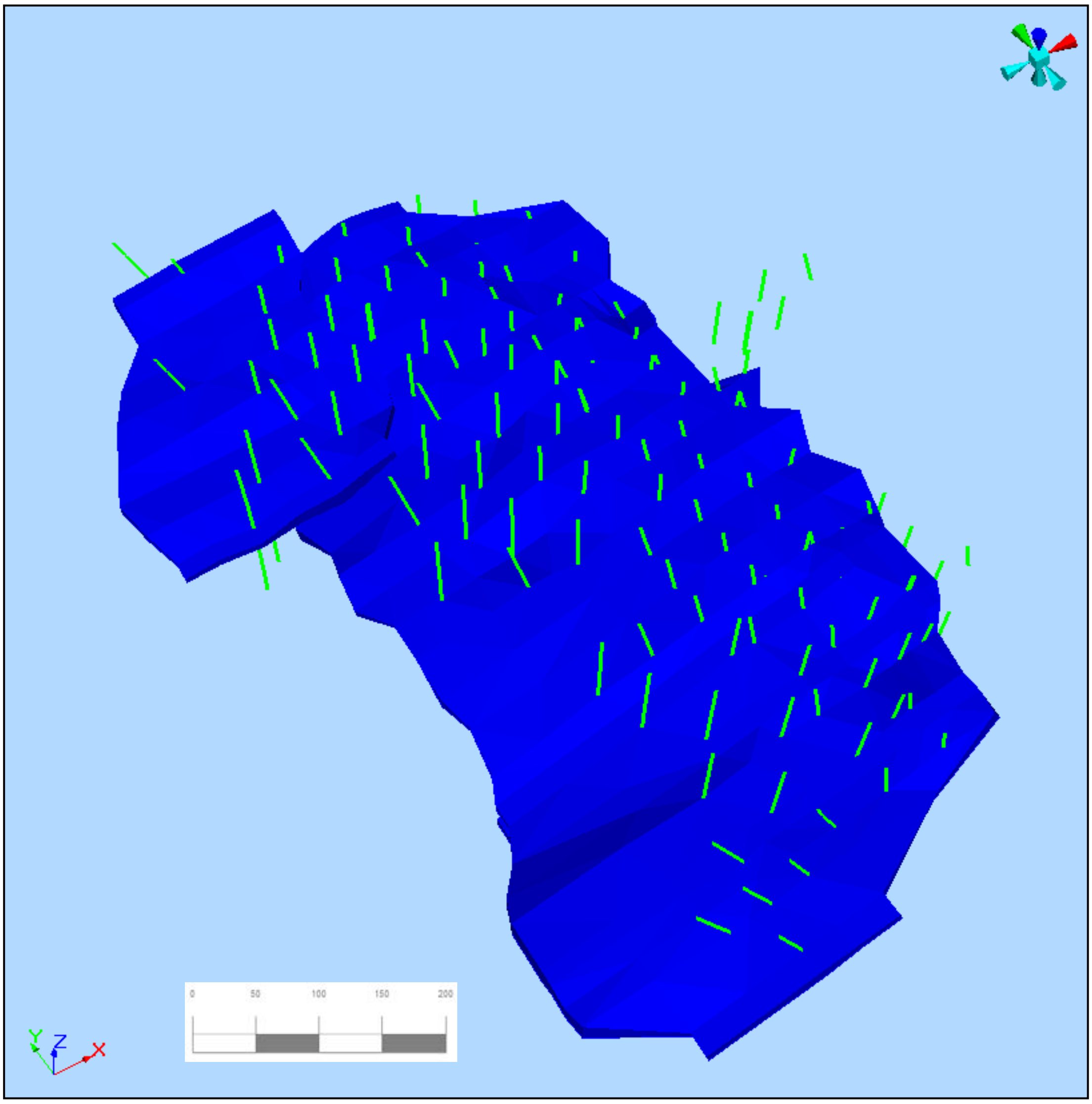


Figure 13-18: Kuervitikko Fe (Zone 130) wireframe (Source: SRK Oct 2012)

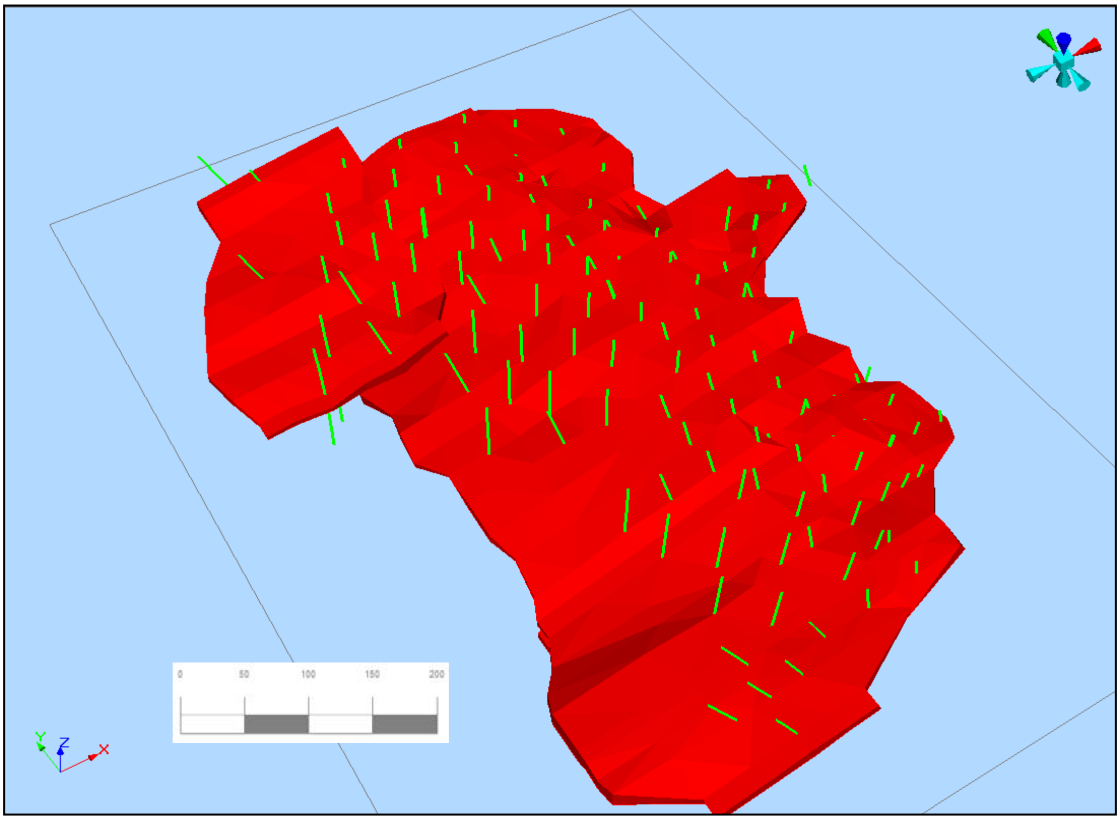


Figure 13-19: Kuervitikko Fe-poor, Cu-rich (Zone 132) wireframe (Source: SRK Oct 2012)

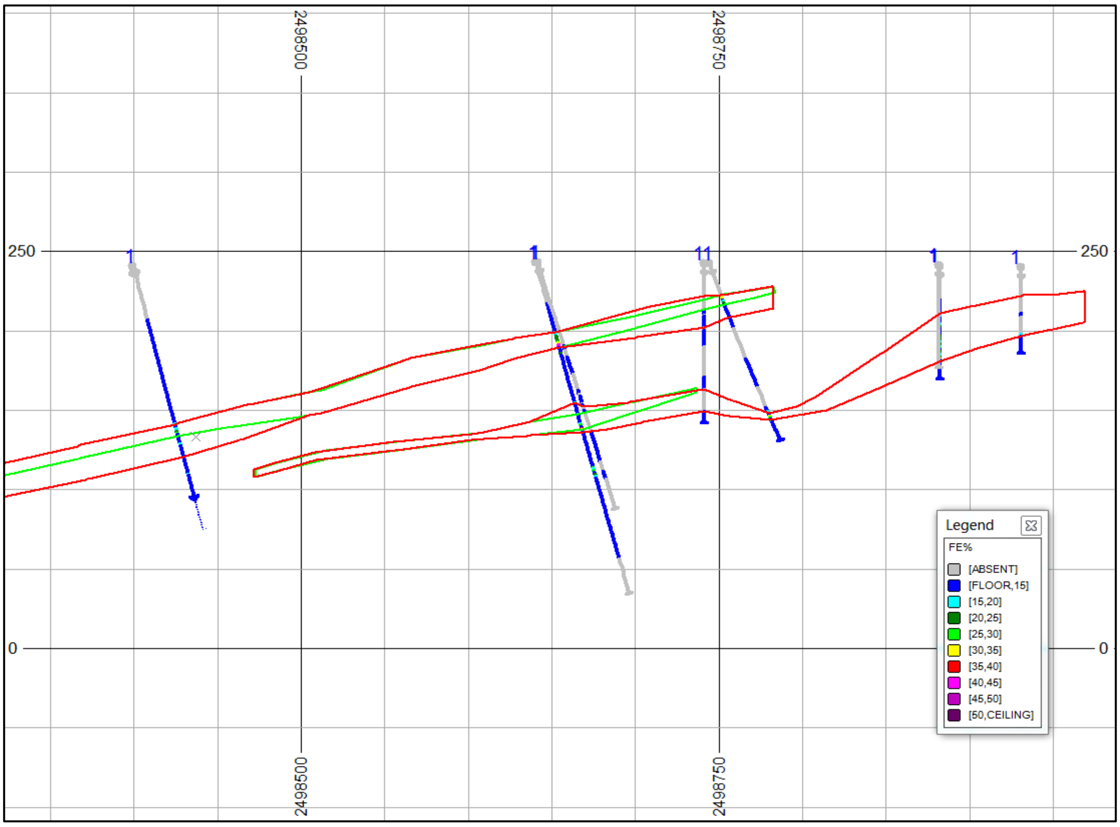


Figure 13-20: XS-10500 Cross-section through Kuervitikko, showing Fe (green) and Cu (red) wireframes (Source: SRK Oct 2012)

13.2.4 Lithological Modelling

In addition to the mineralisation wireframes, an overburden surface and several lithological units were digitised, as shown in the block model in Figure 13-21. The overburden surface, which mainly comprises glacial till material with some overlying peat, was created using lithological logging codes by Northland geologists. This created an accurate surface in areas of drilling, and an average depth of 15 m was used to extend the wireframes to the edge of the model extents.

Lithological codes provided by Northland were also utilised to create to create five lithological surfaces in order to code the model. Additional lithological units are present in the database, however there were insufficient entries to create additional units. The quartzite unit extends to the base of the model, and the monzonite to the surface, due to lack of lithological information at depth. This may not be representative of the actual lithology.

In general, the lithological package is reasonably uniform over the entire strike length, with an apparent stacked succession from monzonite and diorite (intrusives), through amphibolite (metavolcanic rock), mica schist/gneiss and quartzite (metasediments). The mineralised skarn unit lies around the upper contact between the metavolcanic and intrusive units, generally within the amphibolite. This contact represents the major thrust fault in the area, which is thought to be the conduit for the mineralising fluids.

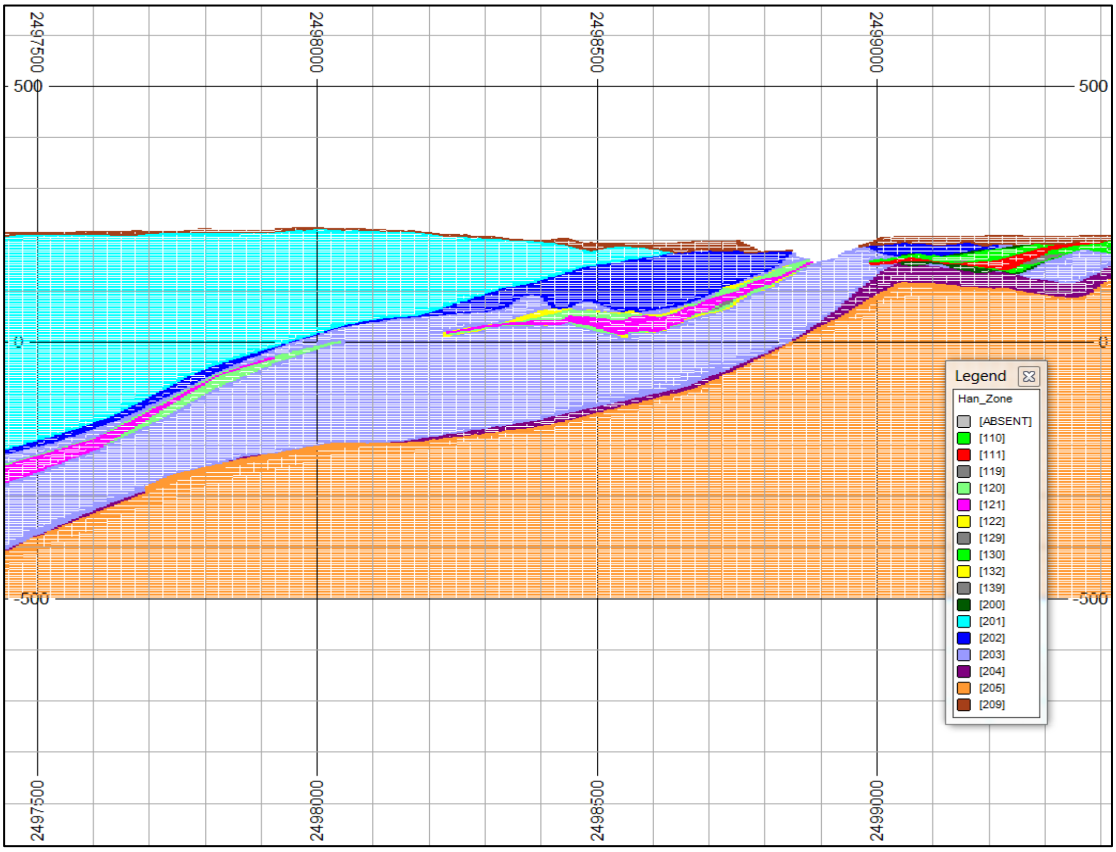


Figure 13-21: Block model coloured by Zone code showing lithology, from top: 209 = overburden; 201 = monzonite; 202 = diorite; 203 = amphibolite; 200 = skarn (which includes all mineralised zones coded 1); 204 = mica schist/gneiss; 205 = quartzite (Source: SRK October 2012)**

13.2.5 MRE Domain Codes

Table 13-2 and Table 13-3 show the domain codes generated for the Hannukainen model and subsequently used to control the interpolation process.

As a result of the secondary Cu-Au-Co-S phase of mineralisation identified at Hannukainen, two separate estimates are required to account for the difference in mineralisation styles. The drillholes and model were given a “ZONE” code for the Fe (and all oxides) estimation, and “CUZONE” code for the Cu-Au-Co-S estimation.

Figure 13-22 shows the empty block model coded by zone, and Figure 13-23 shows the block model coloured by CUZONE.

Table 13-2: Model Fe domain codes used for the MRE

AREA	FE DOMAIN CODE (ZONE)	EXPLANATION
Kuervaara/ Vuopio	110	Low grade Fe
	111	High grade Fe
	119	Internal waste
Laurinoja/ Lauku/ Kivivuopio	120	Low grade Fe
	121	High grade Fe
	122	High grade Cu, waste Fe
	129	Internal waste
Kuervitikko	130	Fe
	132	Cu
	139	Internal waste

Table 13-3: Model Cu domain codes used for the MRE

AREA	CU DOMAIN CODE (CUZONE)	EXPLANATION
Kuervaara/ Vuopio	1	Zones 110 & 111 combined
Laurinoja/ Lauku/ Kivivuopio	2	Low grade Cu (zones 120, 121 combined)
	3	High grade Cu (zone 122)
Kuervitikko	4	Zones 130 & 132 combined
Internal Waste	5	Zones 119, 129 & 139 combined

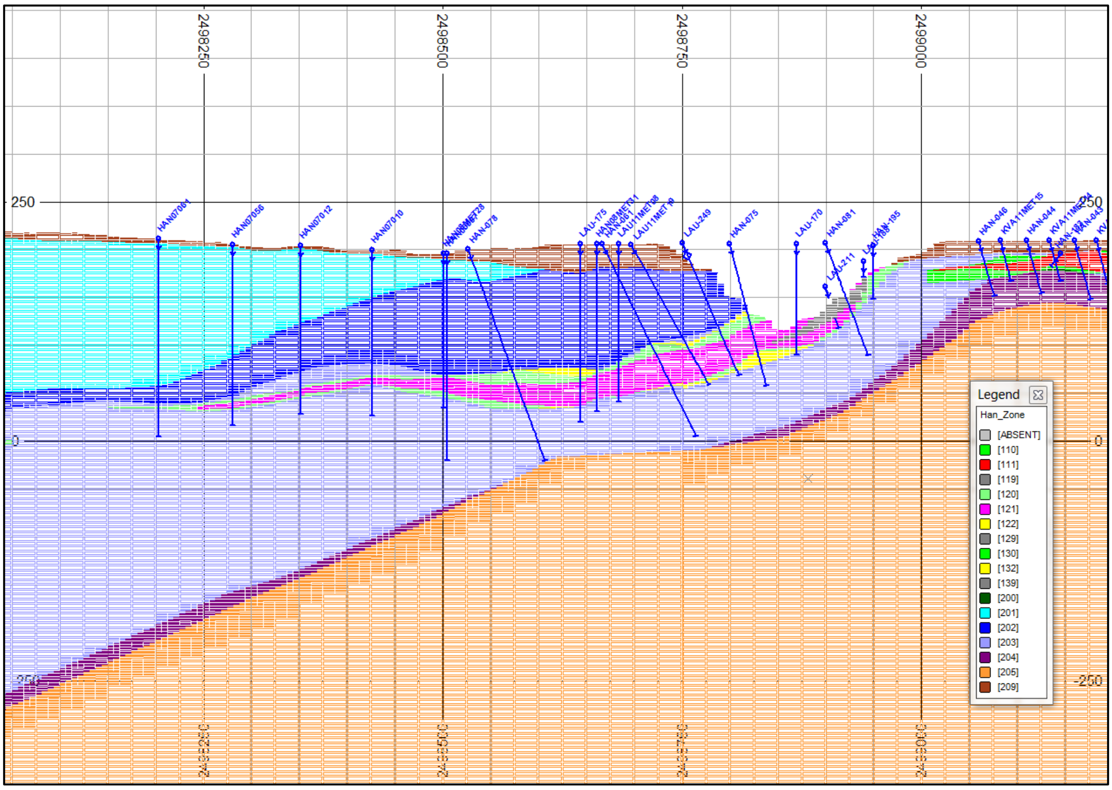


Figure 13-22: Block model coded by ZONE (Source: SRK Oct 2012)

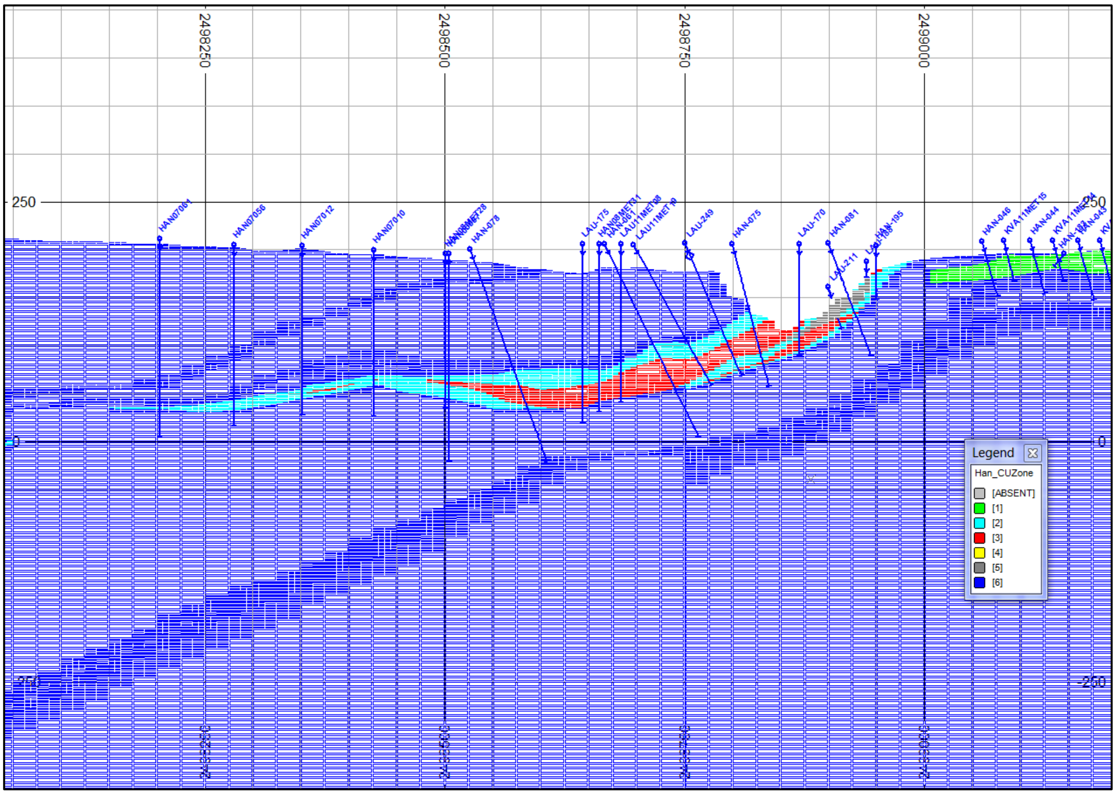


Figure 13-23: Block model coded by CUZONE (Source: SRK Oct 2012)

13.3 Statistical Analysis – Domained Data

Prior to undertaking the interpolation, a statistical study was undertaken on the geological domains to determine their suitability for purpose and to confirm that the appropriate estimation domains have been generated.

13.3.1 Compositing

Data compositing is undertaken to reduce the inherent variability that exists within the population and to generate samples more appropriate to the scale of the mining operation envisaged. It is also necessary for the estimation process, as all samples are assumed to be of equal weighting, and should therefore be of equal length.

The Hannukainen samples have been composited from the collar downwards. Compositing is conducted down-hole, with the composite process being controlled by the wireframe surfaces relating to the new geological domain. Two composite files were created – one based on ZONE codes, for the Fe and related elements and oxides estimate and the other based on CUZONE codes, for the Cu, Au, S and Co estimation.

The Hannukainen deposit comprises large areas of thin mineralisation. With the addition of a high grade internal domain, each domain was made thinner still. As a result, a short composite length is required to show the small scale variability and, as such, SRK chose a 2 m composite length. In the 2010 WGM estimate, a 5 m composite file was used, and so to test the effect of compositing to 5 m composites, a 2 m and 5 m composite length analysis for the Fe estimation file was undertaken.

13.3.2 Composite Length Analysis

The estimation process assumes an equivalent weighting per composite. It is therefore necessary to discard or ignore remnant composite intervals that are generated in the down-hole compositing process to avoid a bias in the estimation. Three composite length analysis ("CLA") runs were undertaken on the 2 m composite drillhole file that compared the effect of discarding remnant samples of 0.5 m, 1 m and 1.5 m lengths. A statistical study of the CLA runs is then undertaken with a comparison against the weighted composite statistics of all data within the 2 m composite file. This was run on both Fe and Cu estimation drillhole files.

This process was also conducted on a 5 m composite file for comparison purposes, with four runs, removing 1, 2, 3 and 4 m samples from the file. The CLA showed that all samples <2 m should be discarded, which comprised a total of 18% of samples. This large percentage is due to the thin nature of the domains. As a result of this analysis, it was decided to use 2 m composites.

From the 2 m Fe composite file, discarding lengths of <1 m had the least effect on the statistical mean of Fe Total. It was therefore decided to remove all samples from the 2 m composite file with a length <1 m. This equated in a total of 836 out of 8,661 samples (9%) being removed from the composite drillhole file, with an average Fe Total grade of 26.8%.

From the 2 m Cu composite file, discarding lengths of <1.5 m had the least effect on the Cu mean. However, this resulted in discarding >10% of samples, and so discarding lengths of <1 m was chosen instead. This equated in a total of 596 out of 8,940 samples (6%) removed at an average grade of 0.2% Cu.

13.3.3 Grade Capping

High grade capping is often used in nuggety deposits, such as with Au and Cu mineralisation. High values can be smeared over large distances and can affect the mean grade, leading to overestimation. The high grade caps were investigated using log probability plots for Au, Cu and S. Capping the composite files made negligible differences to the mean grades, and the lack of extreme values made capping meaningless. SRK decided not to cap the composite files.

13.3.4 Domain Statistics

Table 13-4 shows the declustered composite statistics per Fe estimation domain (ZONE) for the Hannukainen deposit and Table 13-5 shows the declustered statistics per Cu estimation domain (CUZONE).

Zone 110 (Kuervaara/Vuopio low grade Fe domain) shows mean grades of 21.52% Fe Total; Zone 111 (Kuervaara/Vuopio high grade Fe domain) shows a mean grade of 40.24% Fe Total; Zone 120 (Laurinoja/Lauku/Kivivuopio low grade Fe domain) shows a mean grade of 22.01% Fe total; Zone 121 (Laurinoja/Lauku/Kivivuopio high grade Fe domain) shows a mean grade of 43.25% Fe total; Zone 122 (Laurinoja/Lauku/Kivivuopio high grade Cu, high S domain) shows a mean grade of 18.39% Fe Total; Zone 130 (Kuervitikko Fe domain) shows a mean grade of 24.07% Fe Total and Zone 132 (Kuervitikko Cu domain) shows a mean grade of 13.90% Fe Total.

CUZONE 1 (Kuervaara/Vuopio high and low grade Fe domains combined) shows mean grade of 931 ppm Cu and 22 ppb Au: CUZONE 2 (Laurinoja/Lauku/Kivivuopio high and low grade Fe domains combined, excluding zone 122) shows mean grades of 1,198 ppm Cu and 37 ppb Au; CUZONE 3 (Laurinoja/Lauku/Kivivuopio high grade Cu domain inside Zone 122) shows mean grades of 5,185 ppm Cu and 320 ppb Au; CUZONE 4 (Kuervitikko) shows mean grades of 1,981 ppm Cu and 204 ppb Au; CUZONE 5 (waste zones 119, 129, 139) shows mean grades of 582 ppm Cu and 22 ppb Au.

The Coefficient of Variation (“CoV”) can be used to describe the shape of the distribution and is defined as the ratio of the standard deviation to the mean. A CoV greater than one indicates the presence of some erratic high values that may have an impact on the final estimation within the mineralisation zones.

Table 13-4 shows that CoV values greater than 1 are restricted to minor elements along with Au and Cu. This is due to the spotted high grade nature of these elements. Grade capping would help to lower the CoV, but has been proven to be inappropriate for this deposit.

Table 13-4: Hannukainen Declustered Composite Statistics by ZONE

ZONE	FIELD	NREC	NSAMP	MIN	MAX	RANGE	MEAN	VAR	STAND DEV	COV
110	FE	9352	1314	2.27	57.81	55.54	21.42	64.78	8.05	0.38
110	SIO2	9352	707	12.60	69.53	56.93	38.29	60.15	7.76	0.20
110	MGO	9352	1269	0.51	12.00	11.49	5.74	2.95	1.72	0.30
110	AL2O3	9352	1269	0.74	17.04	16.30	6.76	7.83	2.80	0.41
110	CAO	9352	1269	0.52	26.72	26.20	12.48	14.24	3.77	0.30
110	P	9352	1269	0.00	0.61	0.61	0.07	0.00	0.07	0.97
110	MN	9352	1296	0.02	1.09	1.06	0.17	0.01	0.11	0.68
110	TIO2	9352	1269	0.03	1.38	1.35	0.38	0.03	0.17	0.45
110	K2O	9352	1269	0.01	5.68	5.67	0.57	0.37	0.61	1.07
111	FE	9352	1356	6.83	63.05	56.22	40.31	118.39	10.88	0.27
111	SIO2	9352	797	4.81	52.90	48.09	22.49	81.01	9.00	0.40
111	MGO	9352	1256	0.51	10.70	10.19	4.38	2.04	1.43	0.33
111	AL2O3	9352	1256	0.48	18.32	17.84	3.54	6.28	2.51	0.71
111	CAO	9352	1256	1.40	25.78	24.38	8.03	10.88	3.30	0.41
111	P	9352	1256	0.00	0.35	0.35	0.06	0.00	0.04	0.73
111	MN	9352	1307	0.04	2.11	2.07	0.14	0.01	0.11	0.80
111	TIO2	9352	1256	0.03	0.95	0.92	0.22	0.02	0.14	0.62
111	K2O	9352	1256	0.01	4.71	4.70	0.28	0.10	0.32	1.14
119	FE	9352	324	1.47	31.28	29.81	10.06	15.42	3.93	0.39
119	SIO2	9352	198	27.60	71.30	43.70	47.09	44.93	6.70	0.14
119	MGO	9352	324	0.28	9.85	9.57	5.07	2.83	1.68	0.33
119	AL2O3	9352	324	1.85	18.27	16.42	11.69	8.07	2.84	0.24
119	CAO	9352	324	0.74	28.75	28.01	13.34	25.10	5.01	0.38
119	P	9352	324	0.01	0.59	0.59	0.11	0.00	0.07	0.62
119	MN	9352	324	0.01	0.47	0.46	0.14	0.01	0.09	0.64
119	TIO2	9352	324	0.07	1.12	1.05	0.62	0.05	0.21	0.35
119	K2O	9352	324	0.03	7.11	7.08	1.01	1.49	1.22	1.21
120	FE	9352	1340	2.86	58.20	55.34	22.26	71.79	8.47	0.38
120	SIO2	9352	569	11.20	75.00	63.80	40.65	85.28	9.23	0.23
120	MGO	9352	1186	0.08	26.74	26.66	5.08	7.57	2.75	0.54
120	AL2O3	9352	1186	0.37	18.30	17.93	5.90	12.19	3.49	0.59
120	CAO	9352	1186	0.19	27.67	27.47	11.08	28.25	5.32	0.48
120	P	9352	1186	0.00	0.53	0.53	0.07	0.00	0.07	1.02
120	MN	9352	1231	0.02	2.35	2.33	0.20	0.01	0.12	0.60
120	TIO2	9352	1186	0.01	1.60	1.59	0.27	0.04	0.21	0.76
120	K2O	9352	1186	0.00	5.90	5.90	0.78	0.68	0.83	1.06
121	FE	9352	2349	1.86	63.20	61.34	42.83	102.22	10.11	0.24
121	SIO2	9352	1024	4.71	87.07	82.37	22.63	106.58	10.32	0.46
121	MGO	9352	1935	0.29	19.32	19.03	4.22	2.03	1.43	0.34
121	AL2O3	9352	1935	0.23	17.00	16.77	3.14	5.66	2.38	0.76
121	CAO	9352	1935	0.05	20.27	20.22	6.02	9.94	3.15	0.52
121	P	9352	1935	0.01	0.50	0.49	0.05	0.00	0.05	1.00
121	MN	9352	2129	0.02	2.65	2.63	0.22	0.06	0.25	1.15
121	TIO2	9352	1935	0.01	1.11	1.10	0.16	0.02	0.12	0.77
121	K2O	9352	1935	0.01	4.94	4.93	0.49	0.32	0.56	1.15
122	FE	9352	516	1.00	45.20	44.20	12.96	66.45	8.15	0.63
122	SIO2	9352	258	16.40	96.78	80.39	51.17	115.01	10.72	0.21
122	MGO	9352	499	0.04	16.40	16.36	3.93	5.85	2.42	0.62
122	AL2O3	9352	499	0.02	20.20	20.18	11.15	22.22	4.71	0.42
122	CAO	9352	499	0.03	23.48	23.45	7.65	16.52	4.06	0.53
122	P	9352	499	0.00	0.64	0.64	0.08	0.00	0.06	0.73
122	MN	9352	503	0.01	0.42	0.41	0.11	0.01	0.08	0.68
122	TIO2	9352	499	0.01	2.04	2.03	0.42	0.07	0.27	0.64
122	K2O	9352	499	0.03	6.95	6.92	1.39	1.09	1.04	0.75
129	FE	9352	298	0.38	56.37	55.99	8.91	34.63	5.88	0.66
129	SIO2	9352	146	11.94	78.70	66.76	53.98	78.09	8.84	0.16
129	MGO	9352	272	0.05	12.35	12.30	4.06	5.20	2.28	0.56
129	AL2O3	9352	272	1.19	21.90	20.71	12.15	16.96	4.12	0.34

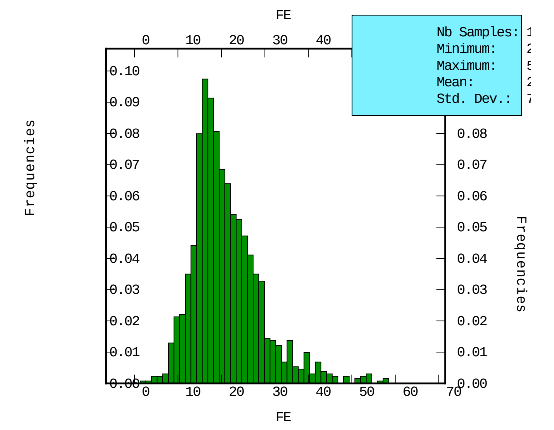
ZONE	FIELD	NREC	NSAMP	MIN	MAX	RANGE	MEAN	VAR	STAND	COV
129	CAO	9352	272	0.38	27.62	27.24	9.37	27.11	5.21	0.56
129	P	9352	272	0.01	0.45	0.45	0.10	0.00	0.06	0.58
129	MN	9352	273	0.01	0.49	0.48	0.13	0.01	0.10	0.77
129	TIO2	9352	272	0.01	1.50	1.49	0.52	0.09	0.30	0.58
129	K2O	9352	272	0.03	8.79	8.76	1.49	2.49	1.58	1.06
130	FE	9352	1039	1.41	59.19	57.78	23.50	98.42	9.92	0.42
130	SIO2	9352	932	6.13	86.88	80.74	42.70	132.34	11.50	0.27
130	MGO	9352	959	0.01	21.30	21.29	3.37	10.22	3.20	0.95
130	AL2O3	9352	959	0.14	17.37	17.23	8.02	14.60	3.82	0.48
130	CAO	9352	959	0.10	37.22	37.12	4.80	44.16	6.65	1.38
130	P	9352	959	0.01	0.21	0.21	0.04	0.00	0.02	0.57
130	MN	9352	1137	0.01	0.92	0.91	0.13	0.03	0.17	1.32
130	TIO2	9352	959	0.01	1.62	1.61	0.48	0.08	0.28	0.58
130	K2O	9352	959	0.06	11.37	11.31	0.74	0.59	0.77	1.03
132	FE	9352	204	2.71	42.74	40.03	13.95	36.73	6.06	0.43
132	SIO2	9352	167	11.71	83.74	72.03	46.74	187.82	13.70	0.29
132	MGO	9352	185	0.51	17.03	16.52	4.82	5.76	2.40	0.50
132	AL2O3	9352	185	0.38	18.99	18.62	8.66	15.71	3.96	0.46
132	CAO	9352	185	0.05	39.58	39.53	8.38	62.16	7.88	0.94
132	P	9352	185	0.01	0.19	0.19	0.05	0.00	0.03	0.53
132	MN	9352	262	0.00	0.98	0.97	0.15	0.04	0.19	1.27
132	TIO2	9352	185	0.02	2.17	2.15	0.51	0.13	0.37	0.71
132	K2O	9352	185	0.06	4.57	4.51	1.03	0.76	0.87	0.85
139	FE	9352	47	4.33	24.13	19.80	10.06	12.02	3.47	0.34
139	SIO2	9352	47	8.77	67.66	58.89	36.80	274.97	16.58	0.45
139	MGO	9352	47	1.24	10.49	9.24	4.64	4.42	2.10	0.45
139	AL2O3	9352	47	1.68	17.57	15.89	7.87	19.62	4.43	0.56
139	CAO	9352	47	0.65	41.43	40.78	19.32	221.65	14.89	0.77
139	P	9352	47	0.01	0.14	0.13	0.05	0.00	0.03	0.56
139	MN	9352	47	0.02	0.91	0.89	0.36	0.08	0.28	0.76
139	TIO2	9352	47	0.12	0.91	0.79	0.41	0.05	0.22	0.53
139	K2O	9352	47	0.16	3.89	3.73	1.27	0.58	0.76	0.60

Table 13-5: Hannukainen Declustered Composite Statistics by CUZONE

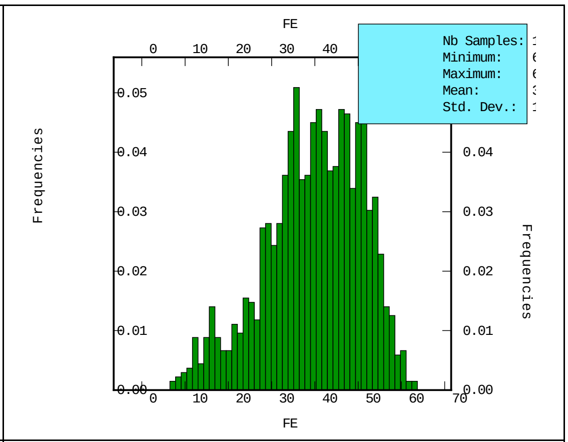
CUZONE	FIELD	NREC	NSAMP	MIN	MAX	RANGE	MEAN	VAR	STAND DEV	COV
1	CU_PPM	9303	2663	10	62,754	62,744	1,046	2,716,628	1,648	1.58
1	AU_PPB	9303	2508	0.5	1,303	1,302	25	3,456	59	2.35
1	S	9303	2664	0.01	22.3	22.2	2.9	4.9	2.2	0.77
1	CO_PPM	9303	1944	5	656	651	144	7,222	85	0.59
2	CU_PPM	9303	2779	15	97,422	97,407	1,401	4,719,113	2,172	1.55
2	AU_PPB	9303	2614	0.5	18,951	18,950	45	103,901	322	7.09
2	S	9303	2721	0.01	20.9	20.9	2.0	2.6	1.6	0.81
2	CO_PPM	9303	2065	5	1,448	1,443	142	11,943	109	0.77
3	CU_PPM	9303	1401	94	36,000	35,906	5,320	18,657,088	4,319	0.81
3	AU_PPB	9303	1296	5.0	14,383	14,378	336	445,672	668	1.99
3	S	9303	1327	0.11	11.6	11.5	2.7	2.3	1.5	0.57
3	CO_PPM	9303	897	12	684	672	181	10,131	101	0.56
4	CU_PPM	9303	1479	25	23,332	23,307	1,985	3,315,271	1,821	0.92
4	AU_PPB	9303	1475	0.5	8,963	8,962	203	114,813	339	1.67
4	S	9303	1189	0.03	14.5	14.4	2.5	4.9	2.2	0.88
4	CO_PPM	9303	1439	5	724	719	115	6,870	83	0.72
5	CU_PPM	9303	671	10	12,657	12,647	566	1,091,185	1,045	1.85
5	AU_PPB	9303	664	0.5	1,270	1,270	22	3,162	56	2.60
5	S	9303	666	0.01	12.8	12.8	0.9	1.4	1.2	1.26
5	CO_PPM	9303	530	5	478	473	57	2,219	47	0.83

Figure 13-24 to Figure 13-26 show the %Fe Total, %S, Cu (ppm) and Au (ppb) composite histograms for the individual domains. The data shows mostly normal populations for %Fe Total, with slightly negatively skewed data for the high grade domains and slightly positively skewed data for the low grade domains. The Cu and Au populations show generally positively skewed distributions, with the exception of CUZONE 3, which represents the high-Cu-Au population within Laurinoja, and shows a slightly more normal population as a result of grouping the high Cu-Au values.

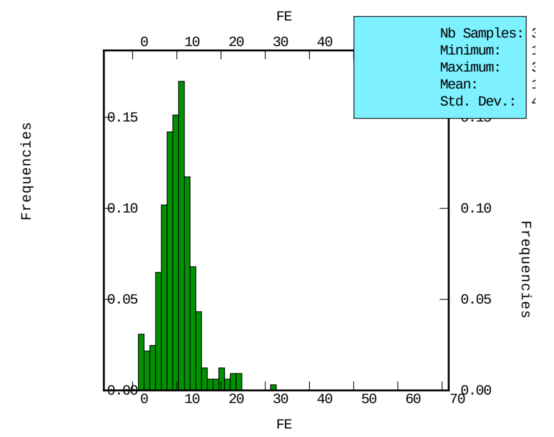
The skewed populations correspond with the poor CoV highlighted above, caused by the nuggety nature of the Cu and Au distribution in the deposit.



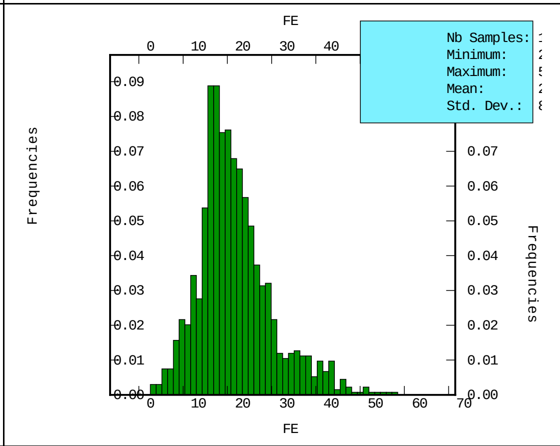
Zone 110: %Fe Total



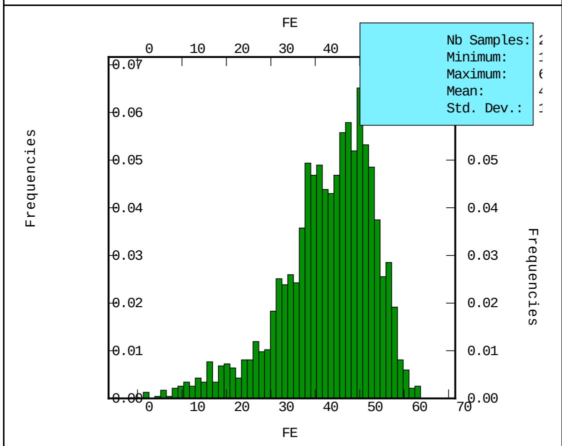
Zone 111: %Fe Total



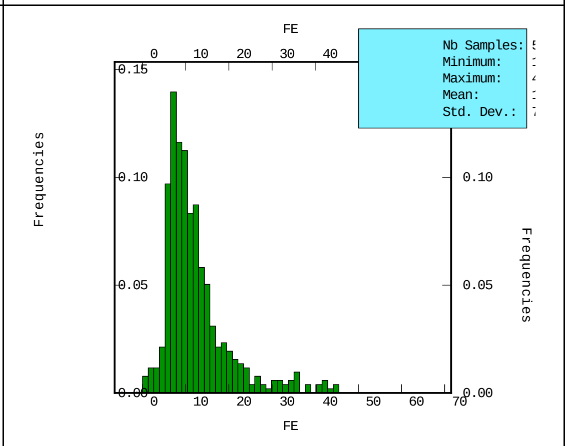
Zone 119: %Fe Total



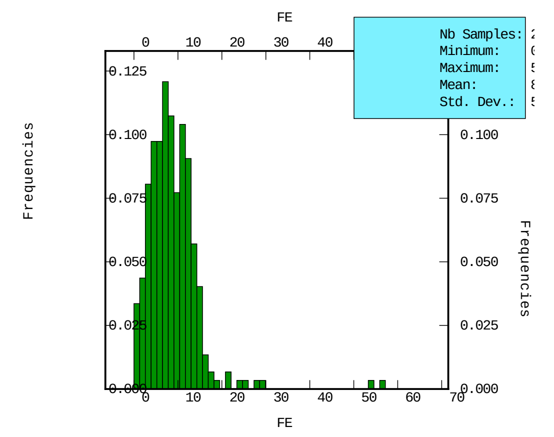
Zone 120: %Fe Total



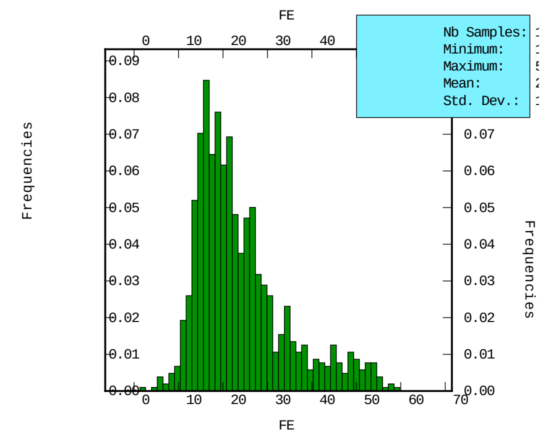
Zone 121: %Fe Total



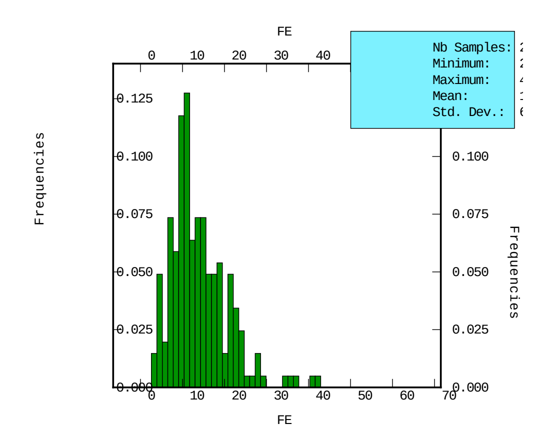
Zone 122: %Fe Total



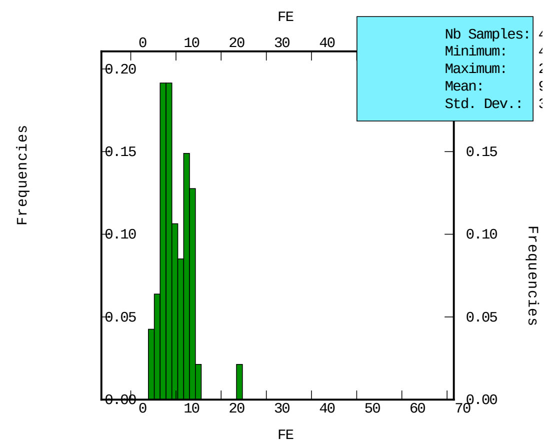
Zone 129: %Fe Total



Zone 130: %Fe Total

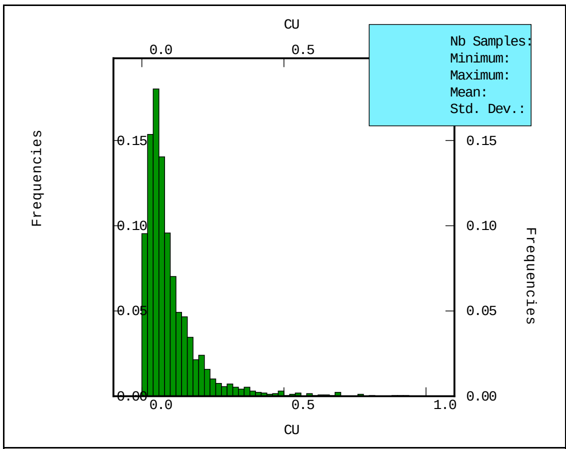


Zone 132: %Fe Total

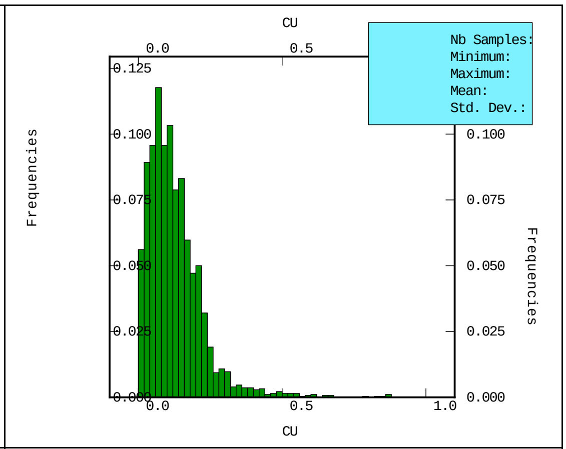


Zone 139: %Fe Total

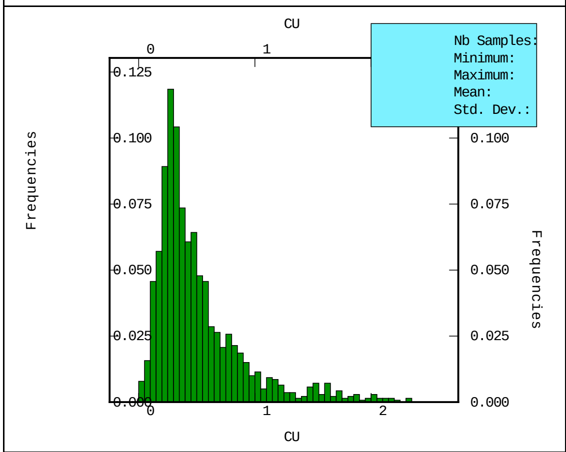
Figure 13-24: %Fe total histograms by Zone



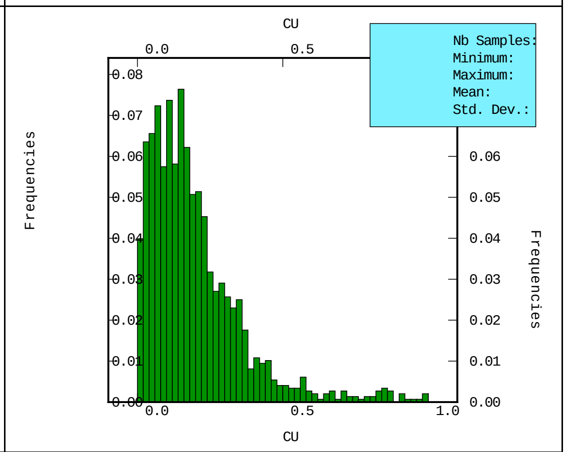
CUZONE 1: CU



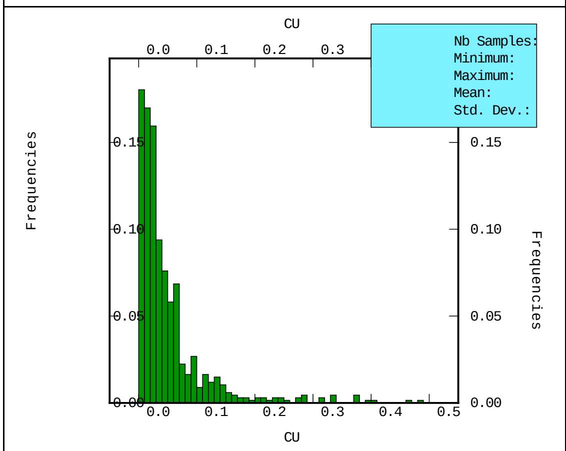
CUZONE 2: CU



CUZONE 3: CU



CUZONE 4: CU



CUZONE 5: CU

Figure 13-25: %Cu Histograms by CUZONE

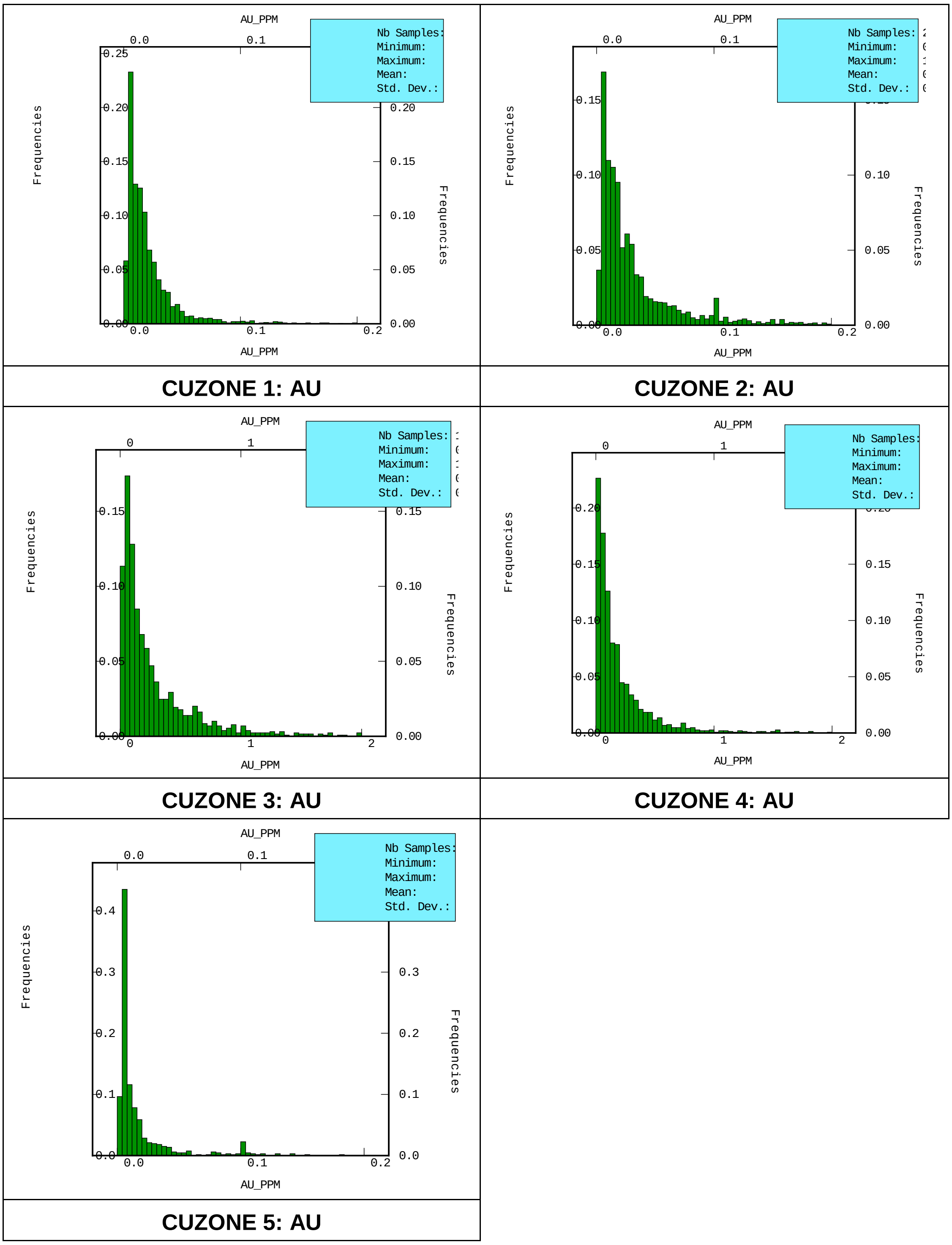
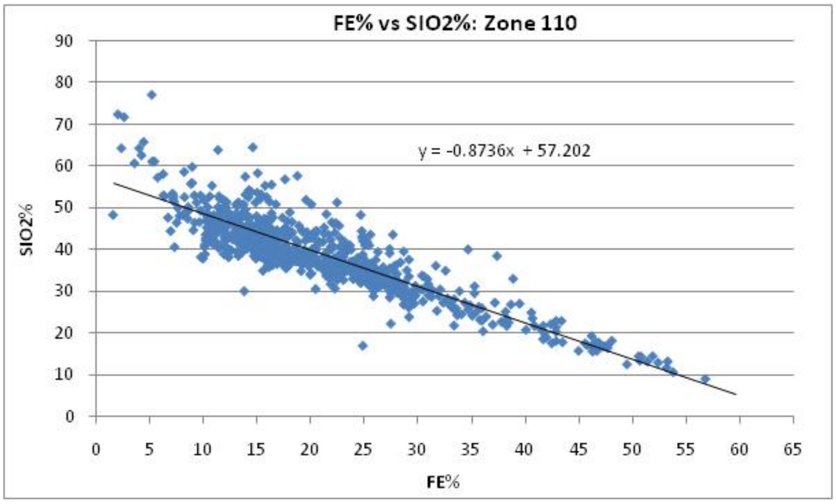


Figure 13-26: Au (ppm) Histograms by CUZONE

Historical SiO2 Assays

All major elements and oxides used in this estimate, except for SiO_2, were assayed for during the historical drilling campaign in the 1970s and 1980s. The lack of SiO_2 data has been accounted for in the estimate by using a regression formula with Fe, per zone (except Zone 122, which was joined with Zone 120 due to the lack of samples) to populate the absent SiO_2 data.

The regression plots and the formula calculated to populate the missing SiO_2 values are shown in Figure 13-27.

<table>
<tr><td>ZONE 110</td><td>
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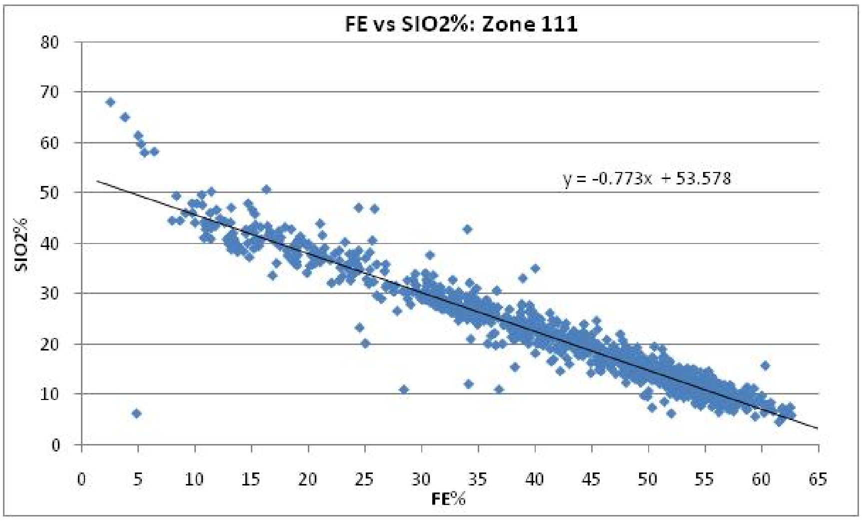
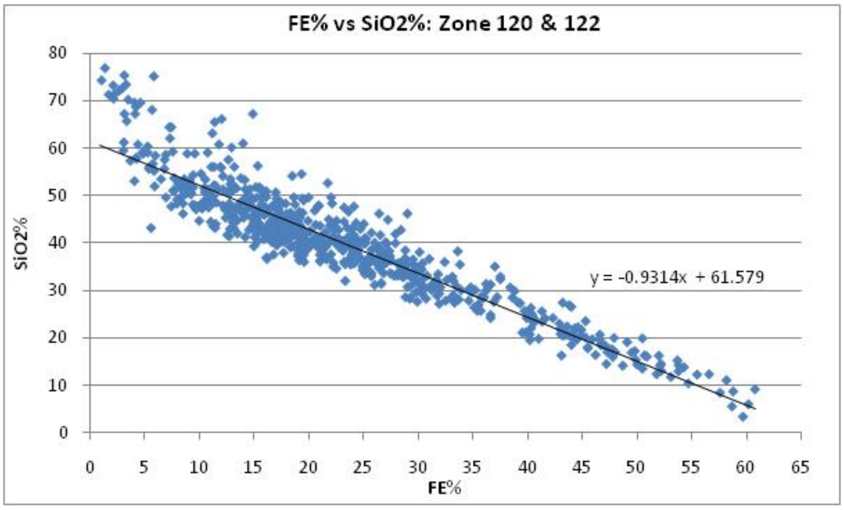
<tr><td>ZONE 111</td><td>
</td></tr>
<tr><td>ZONE 120 (also including zone 122)</td><td>
</td></tr>
</table>

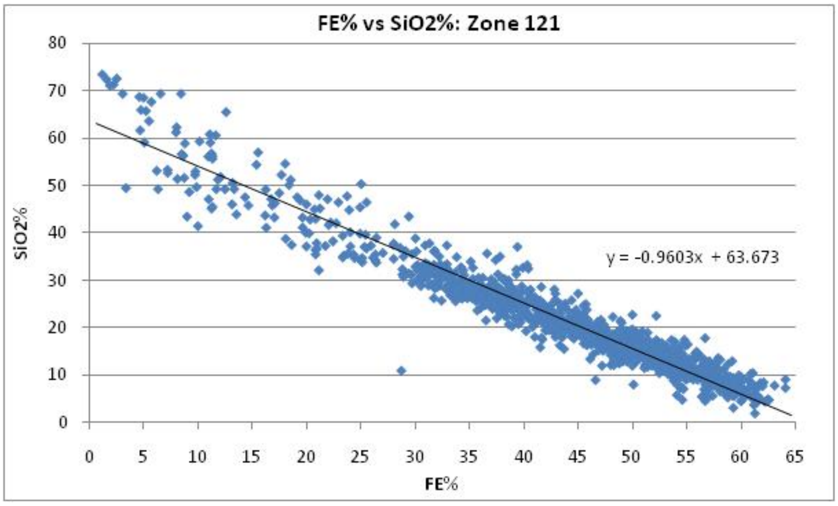
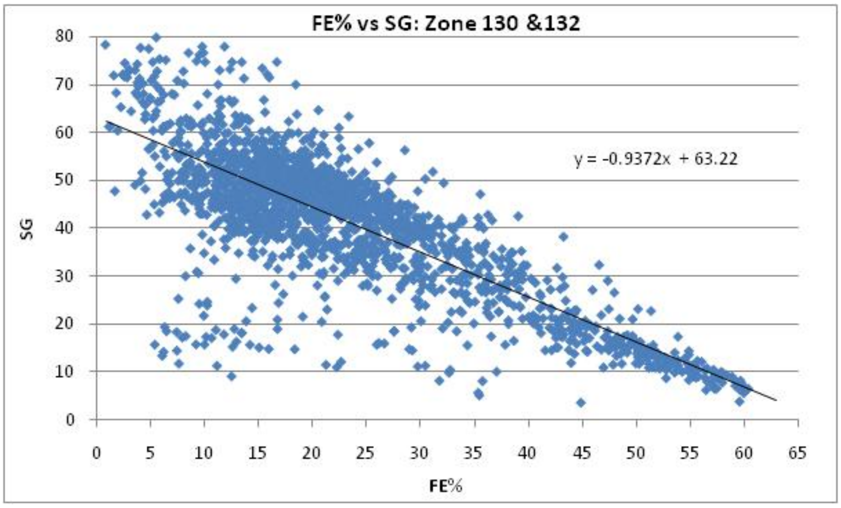
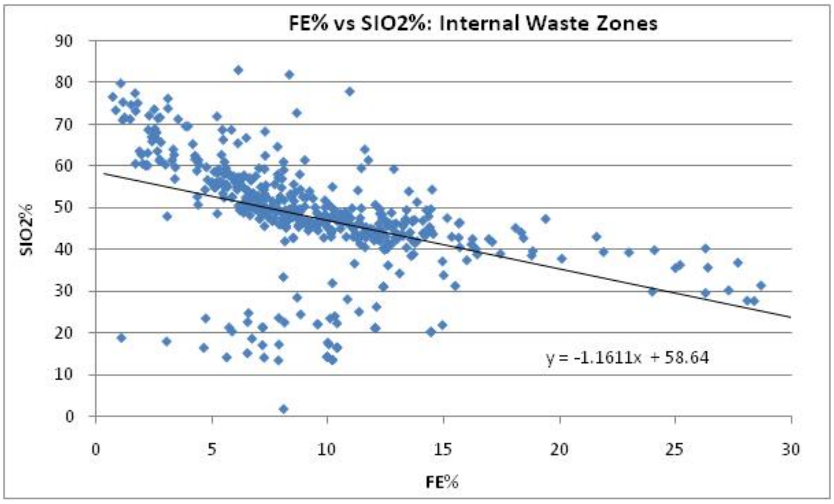
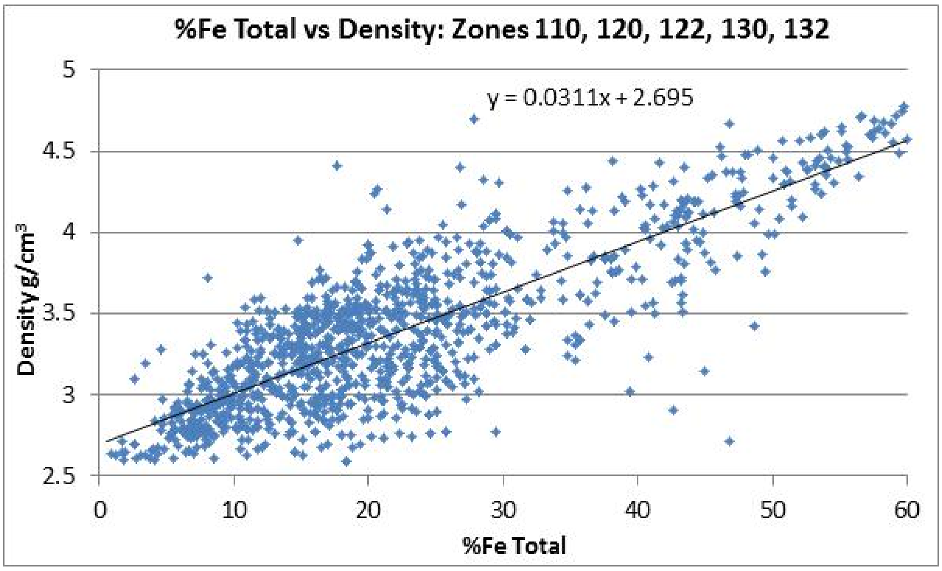
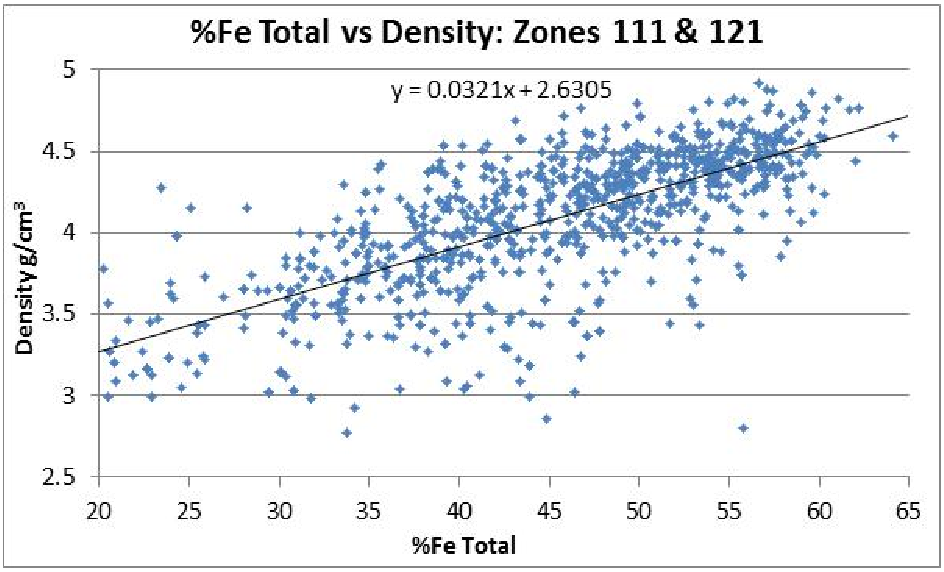
ZONE 121	
ZONE 130 and 132	
WASTE ZONES	

Figure 13-27: Fe-SiO2 Regression Plots

13.4 Density Analysis

The density (SG) values provided by Northland were compared to the %Fe Total values per domain via scatterplots, with a strong correlation being exhibited. Based on the interpolated %Fe total, the density was calculated using the regression formulas determined per domain. Similar domains were joined together to improve the quality of the regression.

The regression trends and the corresponding formula used to assign density to the block model are shown in Figure 13-28.

Low Grade Fe (Zones 110, 120, 122, 130 and 132)	
High Grade Fe (Zones 111, 121)	

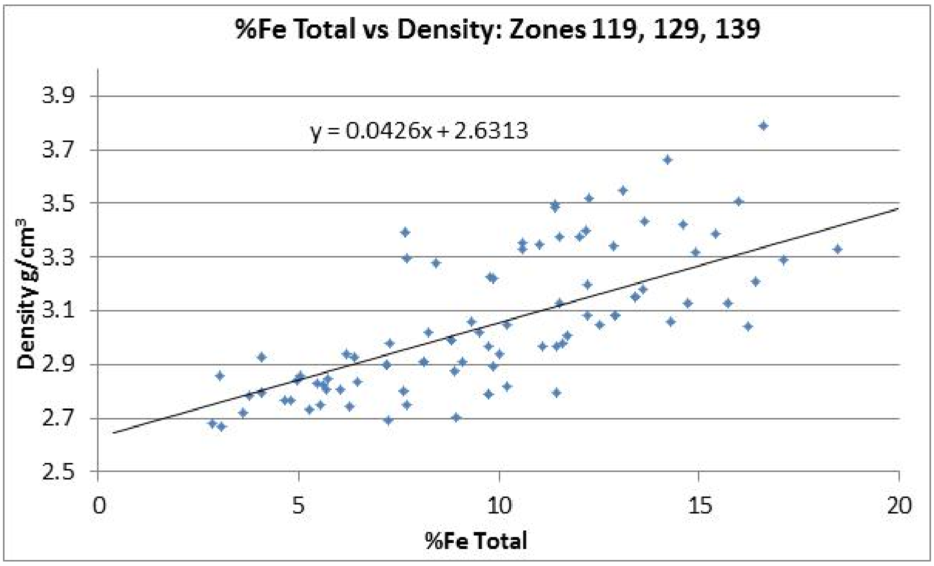
Internal Waste (Zones 119, 129, 139)	

Figure 13-28: %Fe Total vs SG per mineralisation domain

13.4.1 Waste Rock Density

For the waste rock density values, averages per modelled lithology were utilised. A value of 2.2 g/cm^3 was used for the overburden glacial till material, which was used by WGM in the 2010 estimate and is deemed appropriate by SRK.

The density values assigned to each lithology are shown in Table 13-6.

Table 13-6: Waste Lithology Densities

ZONE	LITHOLOGY	DENSITY (g/cm^3)
200	Skarn	3.0
201	Monzonite	2.8
202	Diorite	2.9
203	Amphibolite	3.0
204	Mica Schist/ Gneiss	3.0
205	Quarzite	2.7
209	Overburden	2.2

13.5 Block Model Framework and Coding

A regular sub-celled block model was created and coded using the wireframes in Surpac mining software before being imported into Datamine Studio 3 software for the estimation process. The block model framework is shown in Table 13-7. A block size of 25 m in the X and Y directions represents half the average drillhole spacing, and a 5 m block height (Z) was chosen to due to the thin nature of the mineralisation.

Table 13-7: Block model framework

	ORIGIN	PARENT BLOCK SIZE	NO.BLOCKS	SUB-BLOCK MINIMUM
X	2496500	25	140	6.25
Y	7495000	25	248	6.25
Z	-500	5	164	1.25

13.6 Geostatistical Study

13.6.1 Variography

The composited drillhole database, coded by the modelled domains, was imported into ISATIS software for the geostatistical analysis. Variography was attempted on the two separate estimation sample files – the Fe and related elements and oxides file (coded by ZONE), and the Cu, Au, S, Co file (coded by CUZONE). The Fe sample file variography was based on three groups of domains: low grade, high grade and waste, separately. These domains were considered statistically similar and could be combined together to create more robust variograms. The Cu sample file variography was attempted per zone, due to the varying populations.

Directional experimental semi-variograms were produced for Fe Total, Cu, Au, Al_2O_3, CaO, K_2O, MgO, Mn, P, S, SiO_2, TiO_2 and Co. The semi-variograms were produced using a 2 m (composite length) lag in the down-hole direction allowing the short-scale structures and nugget variance to be determined. Along strike (160º strike, 30º plunge) and down-dip (20º to the west) variograms were then produced with the nugget fixed from the down-hole variogram, and using a lag spacing of 25 m with a 50% tolerance being applied to the lag spacing. This spacing was chosen to mirror the average drillhole spacing within the Project. The plunge was applied to account for the apparent plunging nature of the mineralisation. The overall strike of all zones would produce a plane of 160º strike and 15-20º dip to the west-southwest direction. However, the majority of the mineralisation appears to have a stronger trend (plunge) towards 190º. The variogram uses a tolerance of 45º on the angle, which should account for the small differences in strike and dip of the individual domains. Variogram maps were attempted to fine tune the direction, but were unclear and did not show a dominant direction to be used. The variography produced good quality variograms for most elements for the high grade, low grade and waste groups.

The variogram plane used to define the along strike, down-dip and down-hole variograms is shown in Figure 13-29.

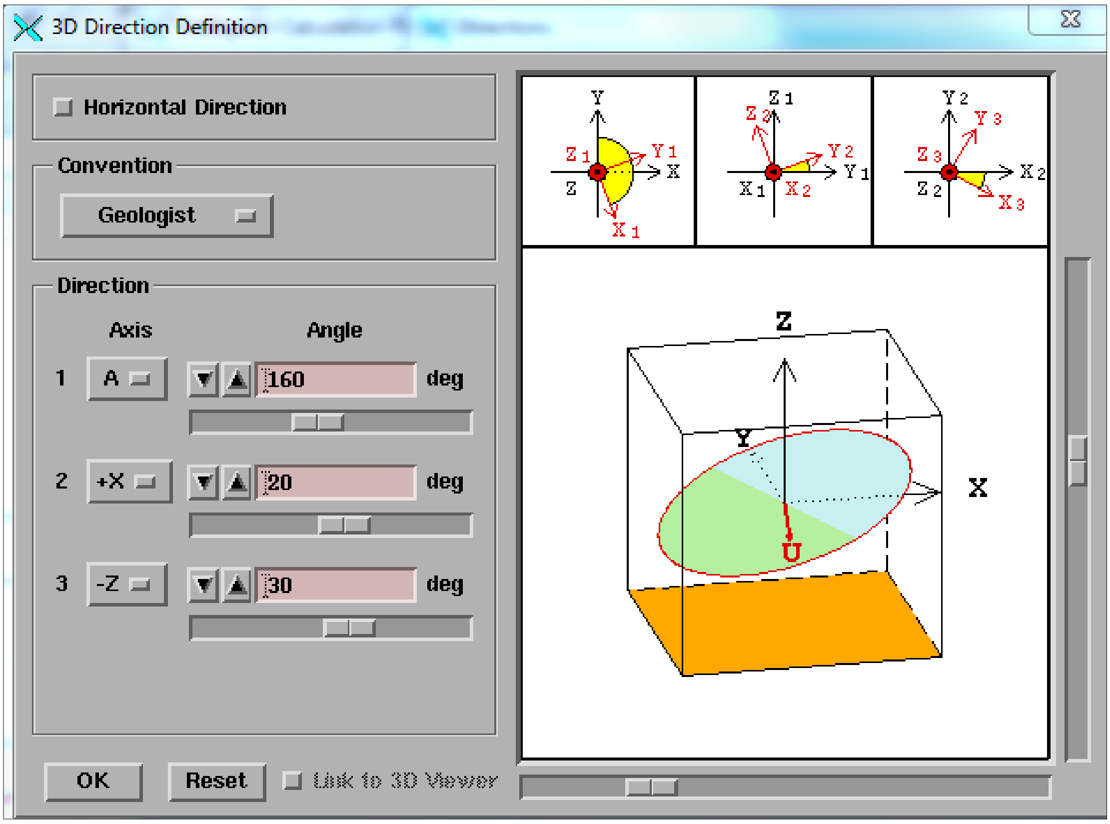


Figure 13-29: Directional variogram plane

The Fe variograms for the high grade and low grade groups are shown Figure 13-30. Robust variograms can be seen in all three directions: along strike, down-dip and across-strike (equivalent to down-hole).

The Cu and Au variograms for each mineralised CUZONE are shown in Figure 13-31. The Cu and Au variograms do not show the high quality of the Fe variograms, but are still adequate to assign distance weightings in the Ordinary Kriging ("OK") process.

The results of the variography are shown in Table 13-8.

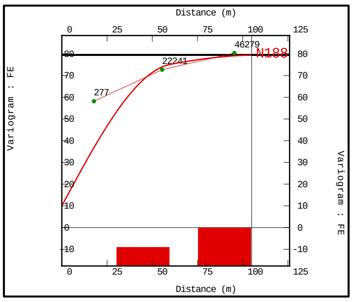
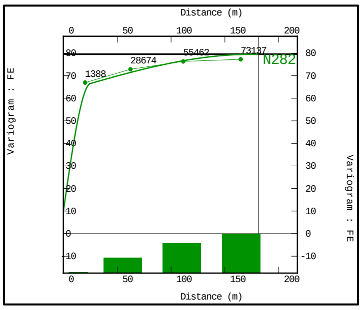
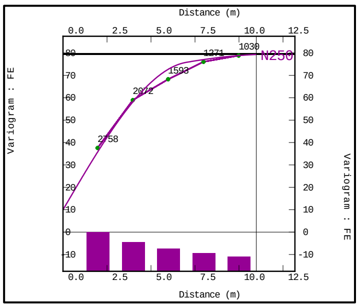
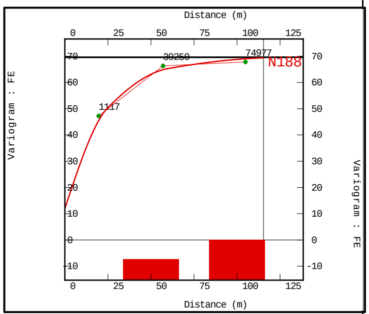
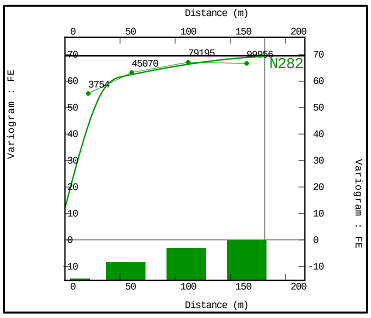
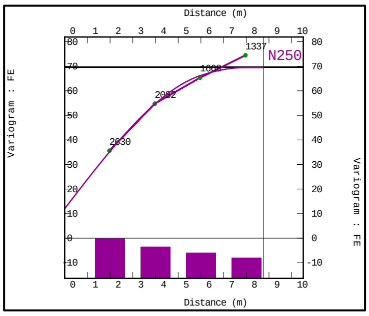
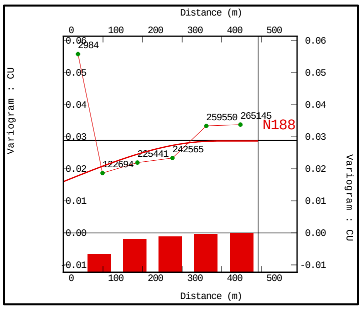
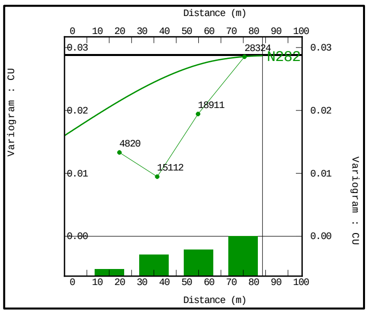
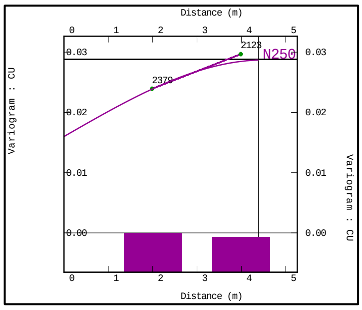
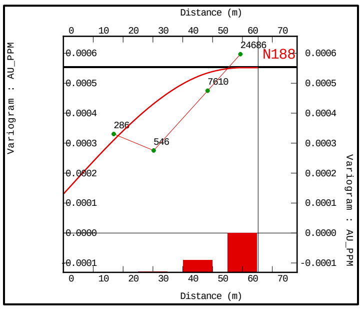
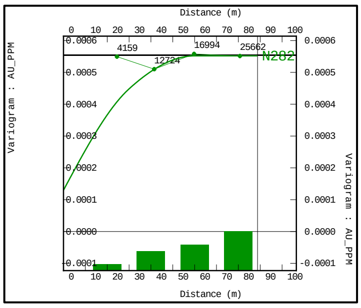
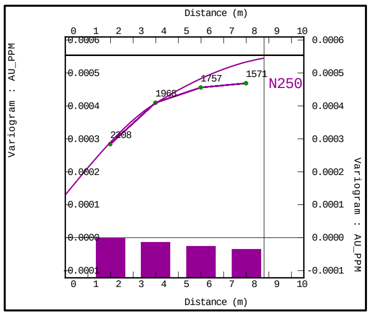
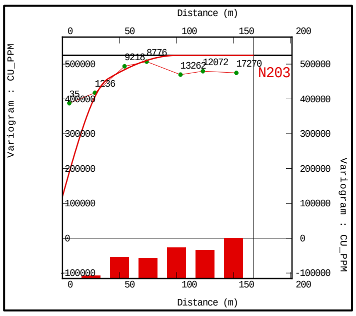
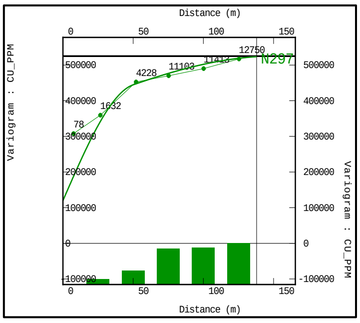
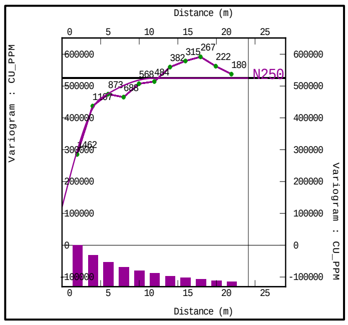
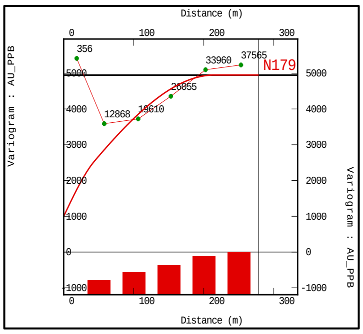
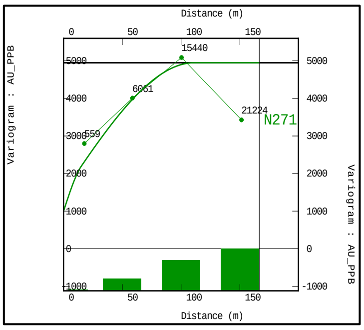
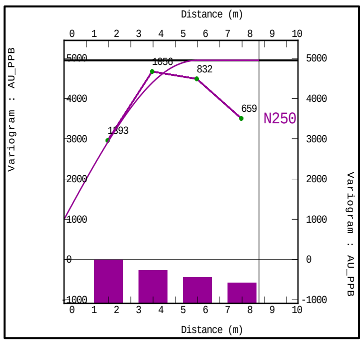
LOW GRADE			
	Along Strike	**Down-dip**	**Down-hole/Across strike**
FE			
HIGH GRADE			
	Along Strike	**Down-dip**	**Down-hole/Across strike**
FE			

Figure 13-30: Directional Fe variograms for low grade and high grade groups

<table>
<tr><th colspan="4">CUZONE 1</th></tr>
<tr><th></th><th>Along Strike</th><th>Down-dip</th><th>Down-hole/Across strike</th></tr>
<tr><td>CU</td>
<td></td>
<td></td>
<td></td></tr>
<tr><td>AU</td>
<td></td>
<td></td>
<td></td></tr>
<tr><th colspan="4">CUZONE 2</th></tr>
<tr><td>CU</td>
<td></td>
<td></td>
<td></td></tr>
<tr><td>AU</td>
<td></td>
<td></td>
<td></td></tr>
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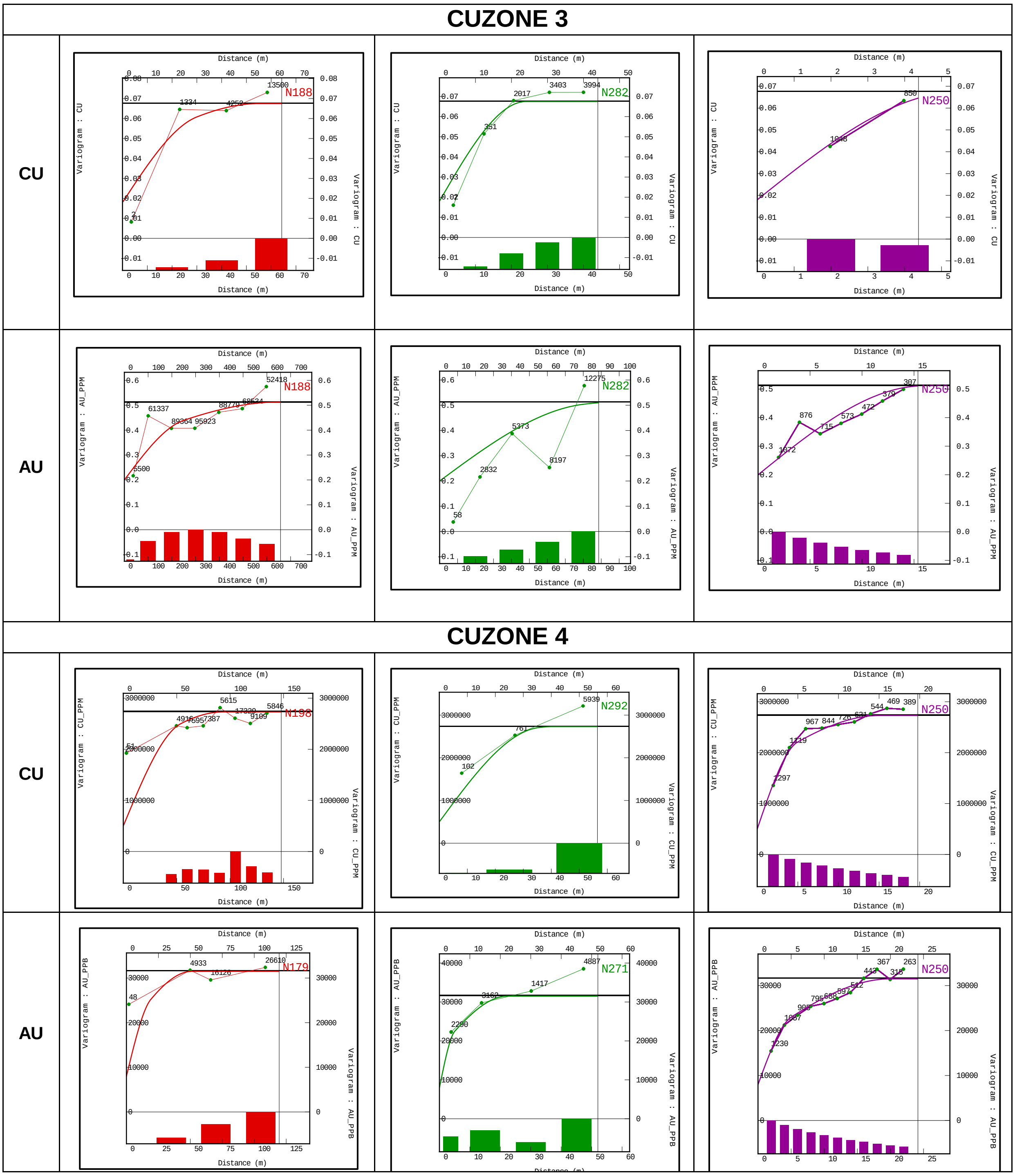


Figure 13-31: Directional Cu and Au variograms per CUZONE

Table 13-8: Fe Zone Variography Results

Group	Assay	Nugget	Along Strike Range	Down-dip Range	Down-hole Range	Sill 1	Along Strike Range	Down-dip Range	Down-hole Range	Sill 2	Along Strike Range	Down-dip Range	Down-hole Range	Sill 3	Total Sill	Nugget%
LOW GRADE	FE	10.000	60	25	7	53.000	110	175	12	16.500					79.5	13
	AL2O3	2.000	25	25	5	2.000	100	125	17	4.500	800	800	45	3.450	11.95	17
	SIO2	10.000	50	75	5	20.000	75	125	15	45.000	550	350	18	28.000	103	10
	MN	0.001	100	110	20	0.010	800	650	22	0.010					0.020	3
	CAO	2.000	75	100	10	15.000	1200	1000	85	22.000					39	5
	MGO	0.500	50	25	5	1.500	100	150	35	2.400	900	800	35	2.800	7.2	7
	TIO2	0.005	110	125	12	0.029	850	850	45	0.020					0.054	9
	P	0.0003	50	50	7	0.001	150	75	25	0.001	500	300	25	0.001	0.003	10
	K2O	0.100	225	50	6	0.2300	850	1000	20	0.220					0.55	18
HIGH GRADE	FE	12.000	25	40	6	20.000	60	50	8	25.000	125	200	8	12.500	69.5	17
	AL2O3	1.000	100	20	5	3.000	125	75	10	0.800	650	300	20	1.200	6.0	17
	SIO2	10.000	25	55	5	50.000	110	140	10	39.500					99.5	10
	MN	0.001	65	75	8	0.035	110	125	9	0.013					0.049	2
	CAO	0.500	75	60	9	9.500	650	400	9	1.800					11.8	4
	MGO	0.150	20	15	6	1.000	75	70	13	0.820					1.97	8
	TIO2	0.002	50	50	5	0.005	100	110	15	0.0060	600	400	17	0.005	0.018	11
	P	0.0003	15	25	6	0.001	20	100	18	0.001	300	400	20	0.001	0.002	12
	K2O	0.060	75	125	6	0.110	1100	450	22	0.056					0.226	27

Table 13-9: Cu Zone Variography Results

CUZONE	Assay	Nugget	Along Strike Range	Down-dip Range	Down-hole Range	Sill 1	Along Strike Range	Down-dip Range	Down-hole Range	Sill 2	Along Strike Range	Down-dip Range	Down-hole Range	Sill 3	Total Sill	Nugget%
1	CU ppm	0.016	350	70	5	0.004	400	90	5	0.009					0.029	56
	AU ppb	0.0001	50	30	4	0.0001	60	60	10	0.0003					0.001	24
	S	1.0	50	25	6	1.500	75	25	14	1.300	250	150	20	1.05	4.85	21
	CO ppm	1000	100	25	6	3000	200	70	15	3100					7100	14
2	CU ppm	12.0	40	50	5	25	100	150	12	15.500					52.5	23
	AU ppb	1.000	50	15	5	0.500	225	110	6	3.450					4.950	20
	S	0.800	25	25	5	0.400	65	60	6	0.800	200	250	8	0.530	2.53	32
	CO ppm	2000	80	70	6	4000	85	80	7	3500	85	150	15	2400	11900	17
3	CU ppm	0.018	30	24	6	0.024	55	24	6	0.026					0.068	27
	AU ppb	80	35	50	5	125	75	175	8	145					350	23
	S	0.5	60	65	8	1.400	225	275	12	0.370					2.270	22
	CO ppm	1200	65	100	7	7000	350	375	12	1550					9750	12
4	CU ppm	50	55	35	5	123	100	40	15	100					273	18
	AU ppb	8	25	5	5	10	55	22	20	14					31.5	25
	S	0.800	75	75	7	2	150	75	25	1.25	155	250	25	0.70	4.75	17
	CO ppm	1000	50	150	8	4000	275	225	17	1000					6000	17

13.6.2 Variography Summary

The composited data allowed for the generation of robust variograms in the down-hole, along-strike (160º, 30º plunge) and down-dip (20º to the west) directions for all element/oxide fields.

The results of the variography are used in the interpolation to assign the appropriate weighting to the sample pairs being utilised to calculate the block model grade. The total ranges modelled have also been used to help define the optimum search parameters and the search ellipse dimensions used in the interpolation. Ideally, sample pairs that fall within the range of the variogram where a strong covariance exists between the sample pairs should be utilised if the data allows. Applying a 2/3 rule to the total range of the variograms in the search ellipse dimensions forces the interpolation to use samples where the covariance between samples exists. In this case, the Fe variography produced shorter ranges than many of the other elements. To ensure that the first pass estimation uses only the data exhibiting covariance, it was decided to use the 2/3 range of the Fe variogram for the first pass search ellipse. Table 13-10 shows the chosen search ellipse radii. The across-strike radii were enlarged to ensure adequate samples were chosen in this direction.

The same principal was applied to the Cu estimation (Cu, Au, S and Co), in general using the 2/3 average ranges of Cu as the search radii, with alterations to ensure the search ellipse was large enough to ensure adequate samples were selected. The chosen radii are shown in Table 13-11.

As a result of the variography, OK was deemed the most appropriate interpolation technique.

Table 13-10: Variogram ranges and Search Ellipse radii for Fe estimation

ZONE	Variogram Ranges	Along Strike	Down Dip	Across Strike
LOW GRADE	All Elements: 2/3 Average Rounded	485	440	25
	Fe	110	175	10
	2/3 Rounded Fe	75	115	10
	Search Ellipse Radius Chosen	75	115	20
HIGH GRADE	All Elements: 2/3 Average Rounded	275	185	10
	Fe	125	200	10
	2/3 Rounded Fe	85	135	5
	Search Ellipse Radius Chosen	85	135	10

Table 13-11: Variogram ranges and Search Ellipse radii for Cu estimation

CUZONE	Variogram Ranges	Along Strike	Down Dip	Across Strike
1	All Elements: 2/3 Average Rounded	150	60	10
	2/3 Rounded Cu	265	60	5
	Search Ellipse Radius Chosen	**200**	**60**	**20**
2	All Elements: 2/3 Average Rounded	100	110	10
	2/3 Rounded Cu	65	100	10
	Search Ellipse Radius Chosen	**65**	**100**	**20**
3	All Elements: 2/3 Average Rounded	120	140	10
	2/3 Rounded Cu	35	15	5
	Search Ellipse Radius Chosen	**50**	**50**	**10**
4	All Elements: 2/3 Average Rounded	100	90	15
	2/3 Rounded Cu	65	25	10
	Search Ellipse Radius Chosen	**65**	**50**	**20**

13.7 Removed Drillholes

For the estimation, where two drillholes were in close proximity (<5 m), one hole in each pair was removed. This was undertaken to avoid negative Kriging weights from being used in the Mineral Resource Estimate.

Table 13-12 shows the holes removed from the estimation drillhole file. Where both twinned holes contained assays, the hole with the fewest assays was removed.

Table 13-12: Table of removed drillholes from the estimation file

HOLE-ID	ASSAYS
HAN07MET01	NO
HAN07MET02	NO
HAN08MET04	NO
HAN08MET06	NO
HAN08MET12	NO
HAN08MET13	NO
HAN08MET14	NO
HAN08MET15	NO
HAN08MET19	NO
HAN08MET21	NO
HAN08MET22	YES
HAN08MET23	NO
HAN08MET28	NO
HAN08MET26	NO
HAN08MET32	NO
HAN-069	YES
LAU-180	YES
LAU-200	YES
LAU-194	YES
LAU-205	YES
LAU-193	YES
LAU-208	YES
LAU-251	YES
LAU-261	YES
LAU-262	YES
LAU-176	YES
LAU-233	NO
LAU-245	NO
HAN-117	YES
HAN-110	YES
HAN-183	YES
KUE-132	YES
KUE-307	YES
KUE-313	YES
KUE-319	YES
KUE-184	YES
KUE-127	YES
KUE07017	YES
KUE-232	YES

13.8 Grade Interpolation

Grade has been estimated into the block model with properties as described in Table 13-7. The variography results allowed for grade estimates for each of the modelled domains to be calculated using OK, applying hard boundaries for the different estimation domains.

Kriging was carried out in three passes for each domain, the first search uses the chosen variogram range (generally 2/3 Fe variogram, or 2/3 Cu variogram), the second search is double the first, and the third search is ten times the size of the first. These multiple searches ensure all blocks within the modelled mineralised domains are interpolated a grade value, however, the confidence in the estimate decreases with each search used. The discretisation grid was set at 5 x 5 x 2 m in all cases.

13.8.1 Dynamic Anisotropy

Dynamic anisotropy uses true dip and dip direction values to assign dip and dip direction to the search ellipse. Each block has a dip and dip direction value, which was estimated into the model prior to grade estimation. The mineralisation wireframes were cut into sections perpendicular to the strike, plunge and dip of the wireframes, to created section strings. These strings contain directional information which is estimated into each block. Figure 13-32 shows a section through the block model coloured by dip. It shows the dip varying through the section, with horizontal sections with a dip between 0-5º, and the steepest sections >25º.

Figure 13-33 shows the result of the dynamic anisotropy with individual search ellipses depicted, which represent the changing dip and dip direction of each block.

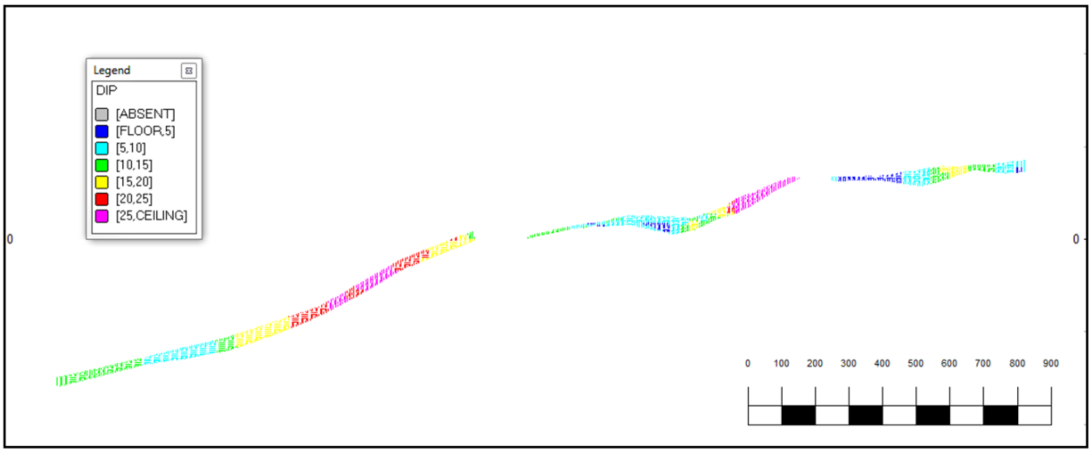


Figure 13-32: Block Model coloured by Dip (Source: SRK Oct 2012)

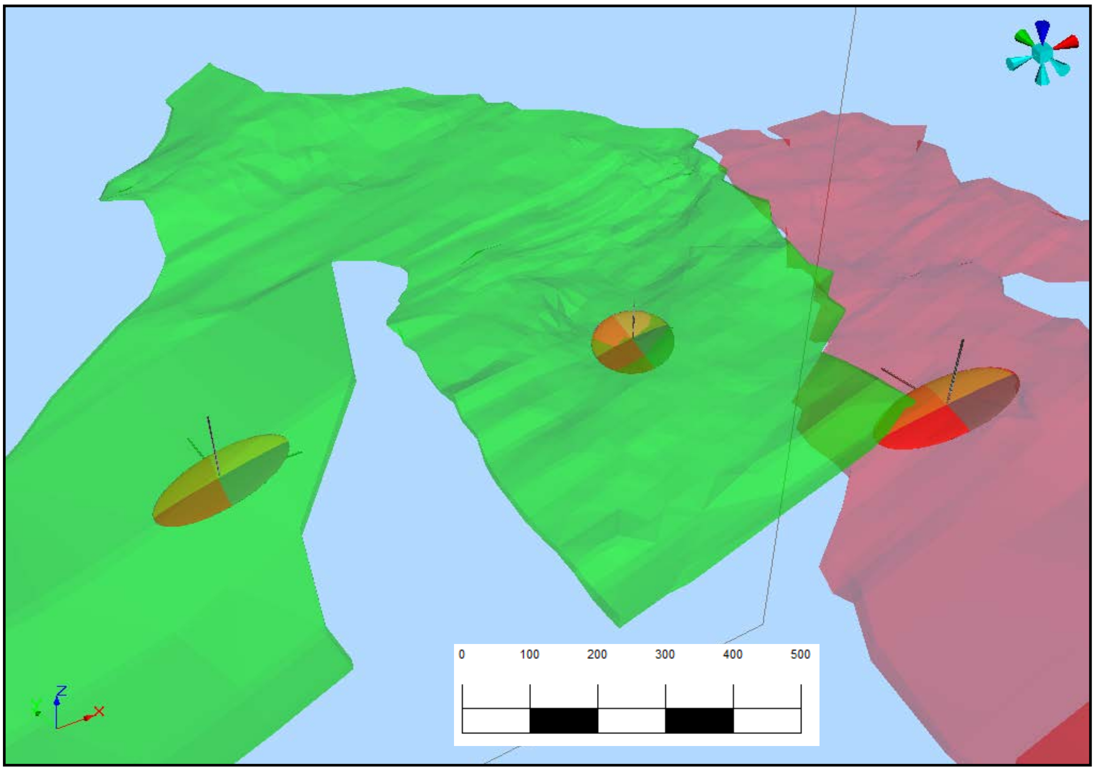


Figure 13-33: Example of search ellipses (Source: SRK Oct 2012)

13.8.2 Quantitative Kriging Neighbourhood Analysis

To better define the ideal search parameters used in the interpolation, Quantitative Kriging Neighbourhood Analysis (“QKNA”) was also undertaken on the data.

QKNA, as presented by Vann *et al,* (2003), is used to refine the search parameters in the interpolation process to help ensure ‘conditional unbiasedness’ in the resulting estimates. ‘Conditional unbiasedness’ is defined by David (1977) as “…on average, all blocks Z which are estimated to have a grade equal to Zo will have that grade”. The criteria considered when evaluating a search area through QKNA, in order of priority, are (Vann *et a.,* 2003):

- the slope of regression of the ‘true’ block grade on the ‘estimated’ block grade;
- the weight of the mean for a simple kriging;
- the distribution of kriging weights, and proportion of negative weights; and
- the kriging variance.

Under the assumption that the variogram is valid, and the regression is linear, the regression between the ‘true’ and ‘estimated’ blocks can be calculated. The actual scatter plot can never be demonstrated, as the ‘true’ grades are never known, but the covariance between ‘true’ and ‘estimated’ blocks can be calculated. The slope of regression should be as close to one as possible, implying conditional unbiasedness. If the slope of regression equals one, the estimated block grade will approximately equate to the unknown ‘true’ block grades (Vann *et al,* 2003).

During OK, the sum of the kriging weights is equal to one. When Simple Kriging (“SK”) is used, the sum of kriging weights is not constrained to add up to one, with the remaining kriging weight being allocated to the mean grade of the input data. Therefore, not only the data within the search area is used to krige the block grade, but the mean grade of the input data also influences the final block grade. The kriging weight assigned to the input data mean grade is termed “the weight of the mean”. The weight of the mean of a SK is a good indication of the search area as it shows the influence of the Screen Effect. A sample is ‘screened’ if another sample lies between it and the point being estimated, causing the weight of the screened sample to be reduced. The Screen Effect is stronger when there are high levels of continuity denoted by the variogram. A high nugget effect (low continuity) will allow weights to be spread far from a block to reduce bias (Vann *et al,* 2003). The weight of the mean for a SK demonstrates the strength of the Screen Effect the larger the weight of the mean, the weaker the Screen Effect will be. The general rule is that the weight of the mean should be as close to zero as possible. QKNA is a balancing act between maximising the slope of regression, and minimising the weight of the mean for a SK (Vann *e. al,* 2003). The margins of an optimised search will contain samples with very small or slightly negative weights. Visual checks of the search area should be made to verify this. The proportion of negative weights in the search area should be less than 5% (Vann *et al,* 2003).

QKNA provides a useful technique that uses mathematically sound tools to optimise a search area. It is an invaluable step in determining the correct search area for any estimation or simulation exercise.

13.8.3 Hannukainen QKNA

The search ellipse radii were based on the variogram ranges, as described above. SRK utilised dynamic anisotropy to guide the search ellipse directions.

For this MRE, 24 different QKNA runs were analysed for Fe and Cu estimates, separately, for the largest domains – Zone 121 and Cuzone 3. The parameters changed were:

- the minimum number of samples used to estimate each block;
- the maximum number of samples used to estimate each block; and
- the maximum number of samples used per drillhole.

In order to choose the most effective parameters, three criteria were analysed (along with ensuring the block and sample means were sensible):

- blocks filled in the first pass search ellipse (search volume 1);
- slope of regression ; and
- kriging variance.

The results are shown in Figure 13-34 to Figure 13-36 for the Fe estimate, and Figure 13-37 to Figure 13-39 for the Cu estimate. The different QKNA run numbers relate to a change in the parameter, for example, run 1 for minimum samples used for estimating each block utilised 3 samples minimum, run 2 utilised 5 samples minimum, run 3 utilised 10 samples etc.

For the Fe estimate, the chosen minimum number of samples used was five (QKNA run 2). This was mainly chosen due to the drop in blocks filled in search volume 1, along with an unremarkable fall in slope of regression and increase in kriging variance with a further increase in minimum number. The chosen maximum number of samples used was 50 (QKNA run 2). Increasing the maximum further made little difference to the estimate, and so the previously used 50 was maintained. The maximum number of samples per drillhole was chosen as 5 (QKNA run 4), which was considered optimal for all three criteria.

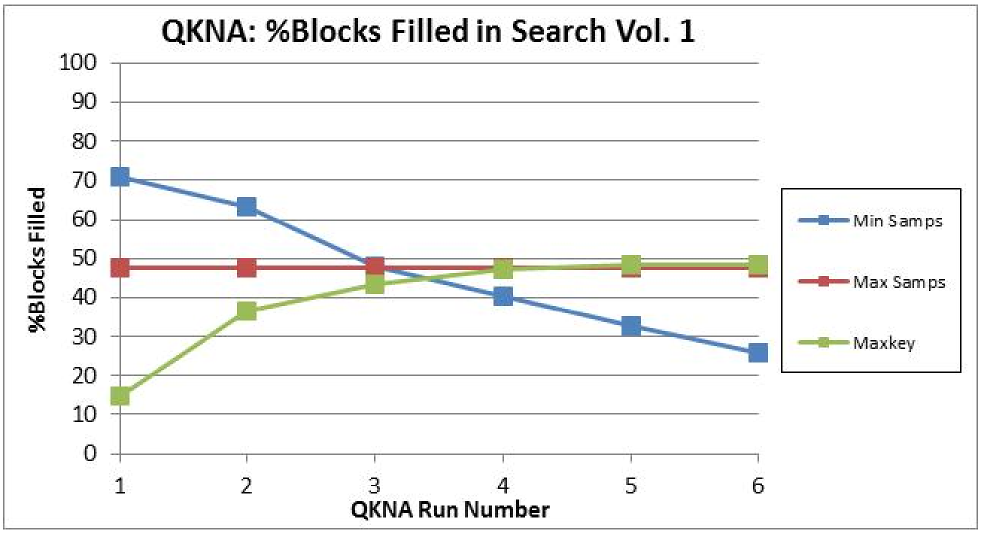


Figure 13-34: QKNA Blocks filled in Fe Search Volume 1

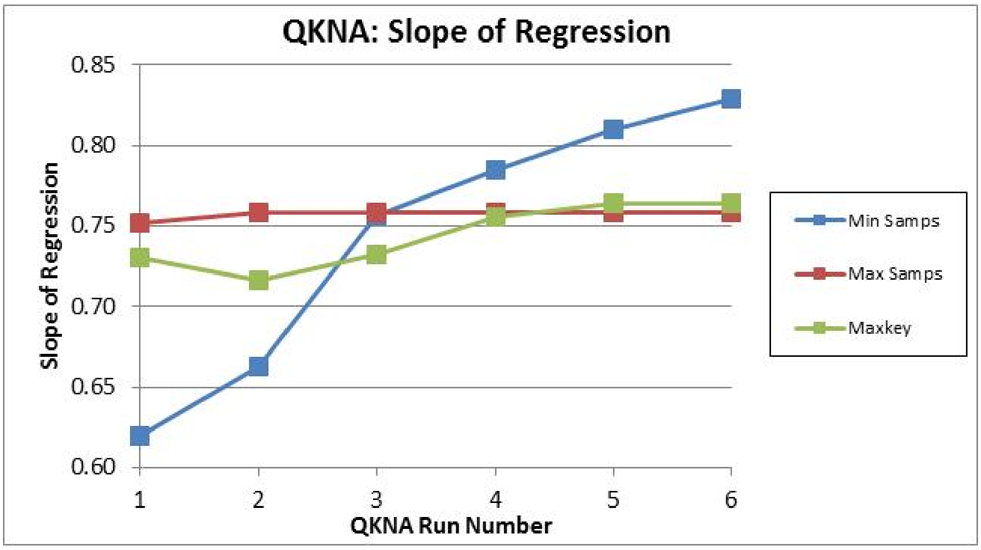


Figure 13-35: QKNA average Fe Slope of Regression

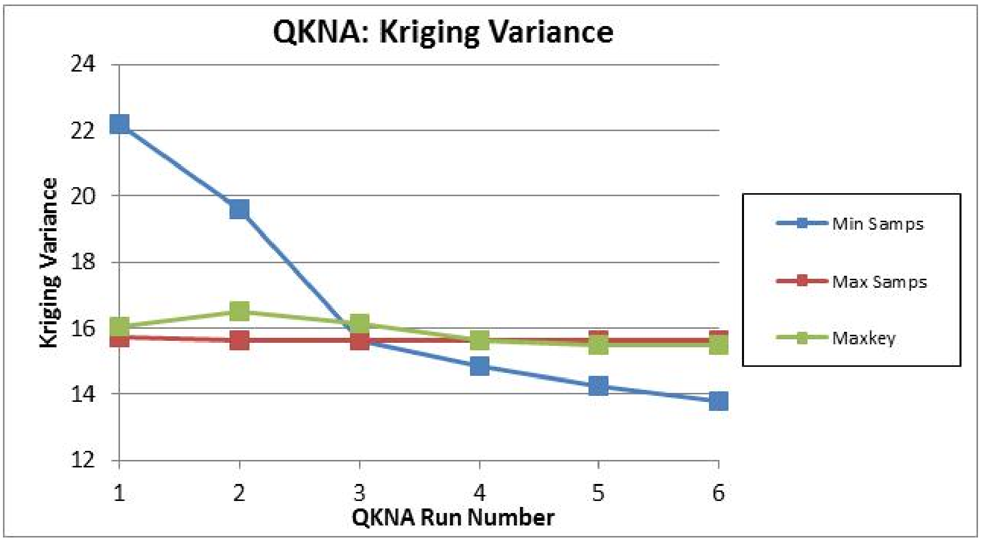


Figure 13-36: QKNA average Fe Kriging Variance

For the Cu estimate, the chosen minimum number of samples used was 10 (QKNA run 3). This was mainly chosen due to the high number of blocks filled, improved slope of regression values and drop in kriging variance. The chosen maximum number of samples used was 50 (QKNA run 2). Increasing the maximum further made little difference to the estimate, and so the previously used 50 was maintained. The maximum number of samples per drillhole was chosen as 5 (QKNA run 4), which was also considered optimal for all three criteria.

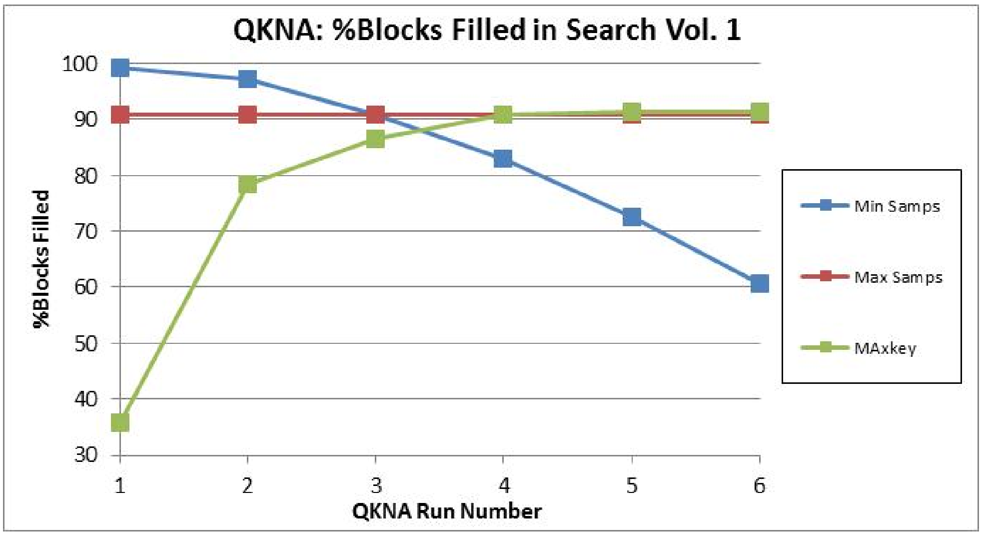


Figure 13-37: QKNA Blocks filled in Cu Search Volume 1

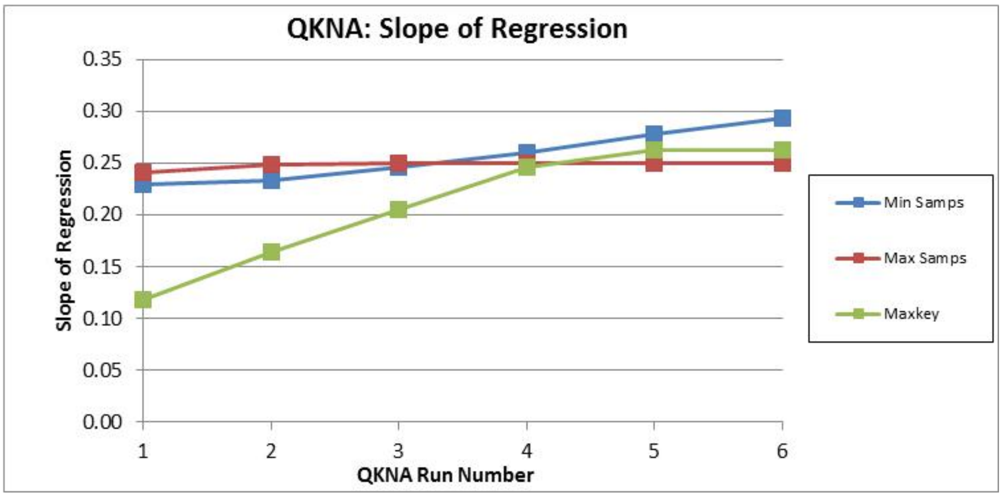


Figure 13-38: QKNA average Cu Slope of Regression

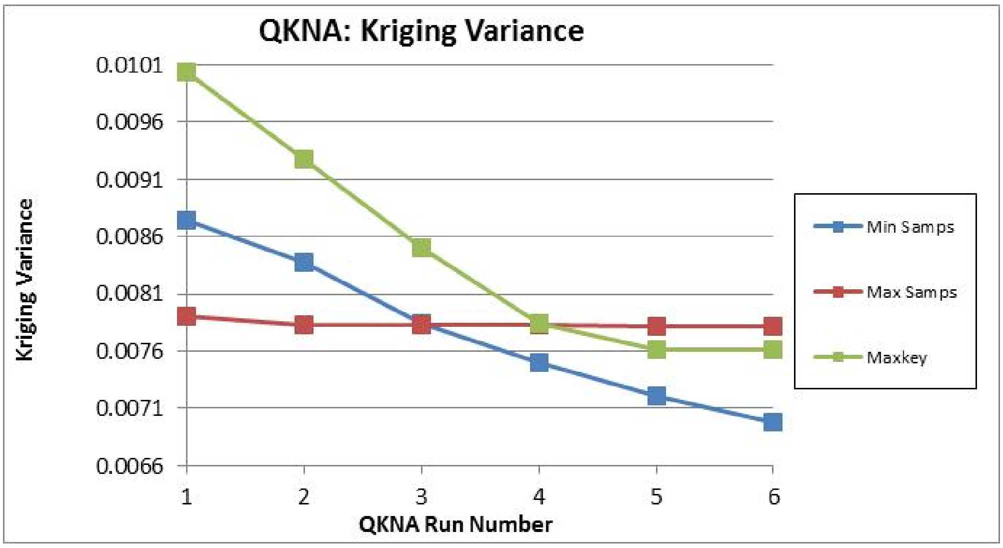


Figure 13-39: QKNA average Cu Kriging Variance

13.8.4 Block Size Sensitivity Analysis

A block size sensitivity test was carried out on the Hannukainen model to evaluate the effective selectivity of the grade tonnage curve.

The range of block sizes used for the sensitivity analysis is given in Table 13-13. The chosen parent block size in the Hannukainen Mineral Resource model is 25 x 25 x 5 m. The Mineral Resource model was estimated using the parent block size, and sub-blocks estimated within the parent block. Throughout this study, the parent cell has been taken as the block size and sub-blocking has not been applied.

The parent block size is approximately half the average drill spacing, with the Z dimension controlled by an estimated, open pit mining bench height. The short range structure of the variogram is controlled by downhole variography, so the behaviour of the variogram along strike, at short distances is not known. This may have a significant impact on the sensitivity analyses for the smaller blocks. The Selective Mining Unit ("SMU") size has been estimated by the Company to be approximately 6 x 6 x 5 m, being 180 m^3. The impact of changing to 10 m high benches was also investigated.

A range of smaller block sizes was investigated, as well as one size larger than the Mineral Resource OK model. The smaller blocks are useful for evaluating the potential impact of selective mining on the grade tonnage curve, whereas the larger blocks help to determine the potential selectivity for the parent blocks.

Table 13-13: Block sizes for sensitivity analysis

Model	X	Y	Z	Volume	% of Parent Block
SMU	6	6	5	180	6%
2	10	10	5	500	16%
3	15	15	5	1125	36%
4	20	20	5	2000	64%
5	**25**	**25**	**5**	**3125**	**100%**
6	50	50	10	25000	800%
7	25	25	10	6250	200%
8	20	20	10	4000	128%
9	15	15	10	2250	72%
10	6	6	10	360	12%

The grade tonnage curves for the various block sizes tested are shown in Figure 13-40. The results reported relate to all material that has been interpolated in runs one, two and three of the estimation process. For the sensitivity analysis, all of the mineralised zones are combined for re-estimation. The tonnages are presented as a percentage, with 100% being equivalent to the total tonnes above a zero cut-off grade

In addition to the grade tonnage curves, the grade and tonnage differences from the OK model were tabulated for a zero cut-off grade. The grade tonnage comparisons are given in Table 13-14.

Table 13-14: Grade/Tonnage comparisons

OPTION	BLOCKSIZE	TONNAGE	GRADE	TONNAGE DIFF TO OK MODEL	GRADE DIFF TO OK MODEL
OKMODEL	25,25,5	226,015,606	29.3	0.0%	0.0%
SMU	6,6,5	231,302,011	29.5	2.3%	0.7%
2	10,10,5	231,326,351	29.4	2.3%	0.6%
3	15,15,5	231,297,927	29.5	2.3%	0.8%
4	20,20,5	230,646,174	29.5	2.0%	0.8%
5	25,25,5	230,694,102	29.5	2.1%	0.7%
6	50,50,10	236,064,283	29.6	4.4%	1.0%
7	25,25,10	233,524,342	29.4	3.3%	0.5%
8	20,20,10	231,666,775	29.5	2.5%	0.7%
9	15,15,10	231,006,140	29.5	2.2%	0.7%
10	6,6,10	231,428,410	29.5	2.4%	0.7%

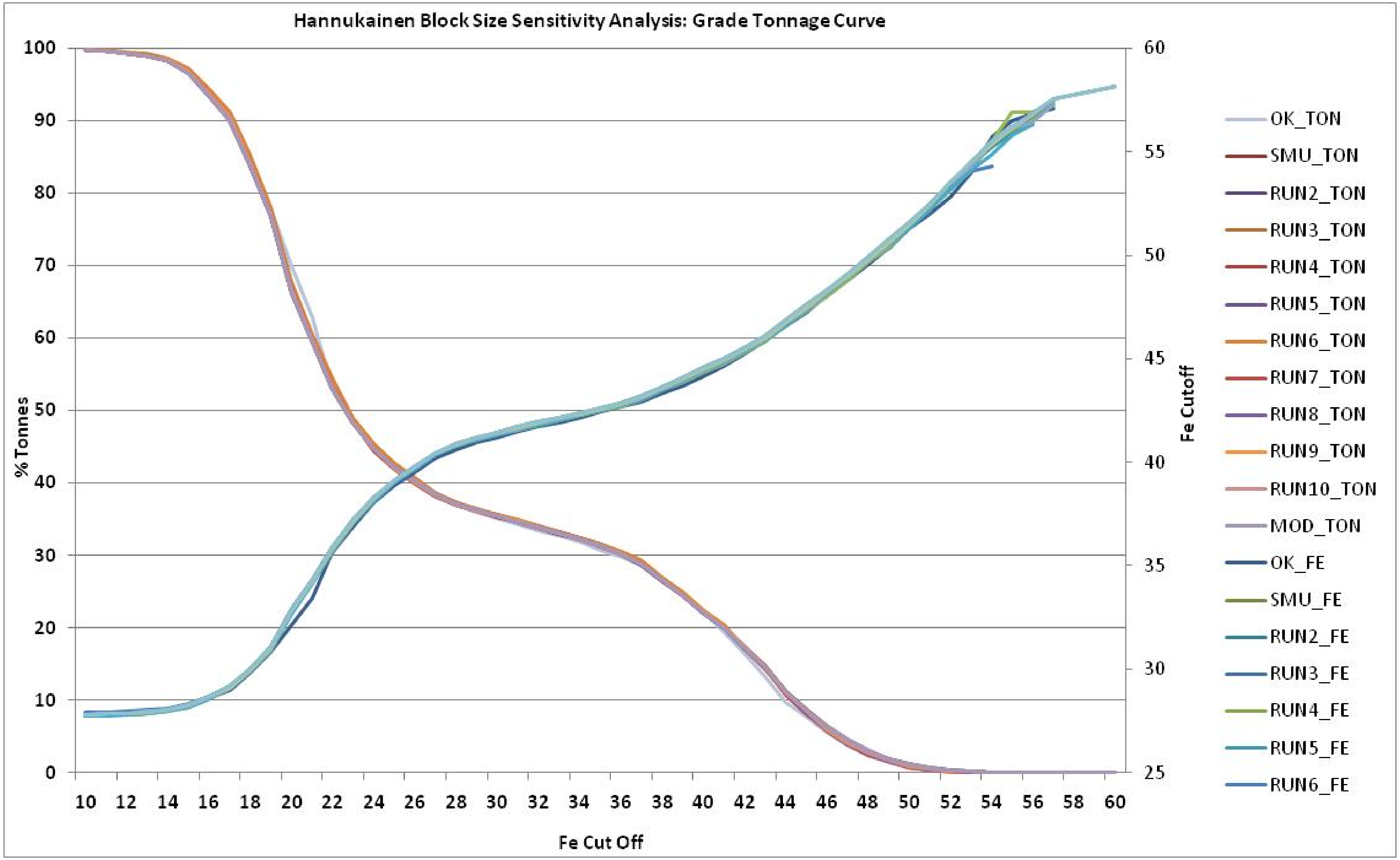


Figure 13-40: Global grade tonnage curves for Hannukainen compared with the OK block model grade tonnage curves

Comparing the grade tonnage curves over the full range of block sizes gives an indication of the sensitivity of the Mineral Resource model to different SMU sizes. As Table 13-14 and Figure 13-40 both illustrate, the Mineral Resource model is not sensitive to the changing block size. This indicates that no bias has been introduced by selecting the Mineral Resource model OK block size that relates to half of the dominant drillhole spacing. Only minor differences are observed relating to both tonnes and grade when all tested block sizes are compared to the Mineral Resource model. The small differences between the OK model and the sensitivity runs can be attributed to the improved accuracy of a sub-celled model. The shapes of the both the grade and tonnage curves are also very similar indicating similar distributions and degrees of smoothing at any given cut-off grade.

Furthermore, SRK considers that the SMU of 6.25 x 6.25 x 5 m, as chosen by the Company, appears to be appropriate and over the life of the Project should yield comparable tonnes and metal content.

13.8.5 Block Model Validation

In order to check that the estimation has worked correctly, the model has been validated through a visual and statistical comparison and through the generation of validation slices through the model.

Visual Validation

The block model was inspected by SRK on sections to compare the composite grades to the block model grades. The model and composite %Fe Total and %Cu grades show excellent correlation, as shown in Figure 13-41 to Figure 13-50.

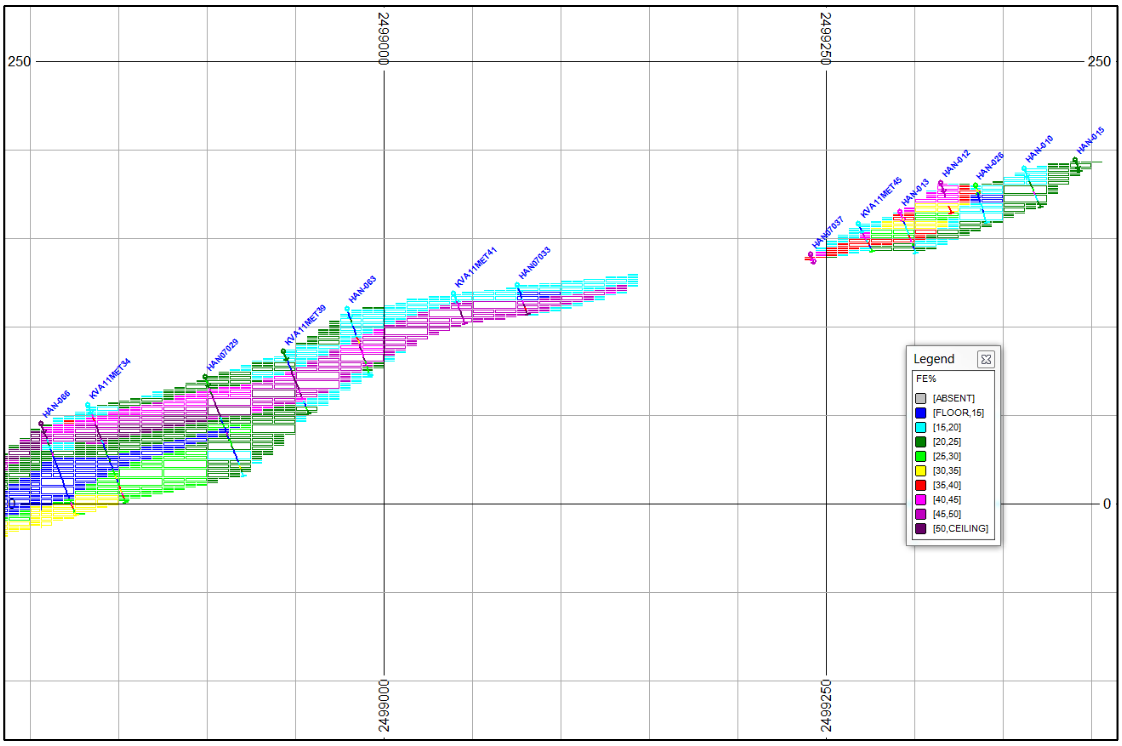


Figure 13-41: XS6200 cross-section through Kuervaara showing block model and composite file coloured by %Fe Total (Source: SRK Oct 2012)

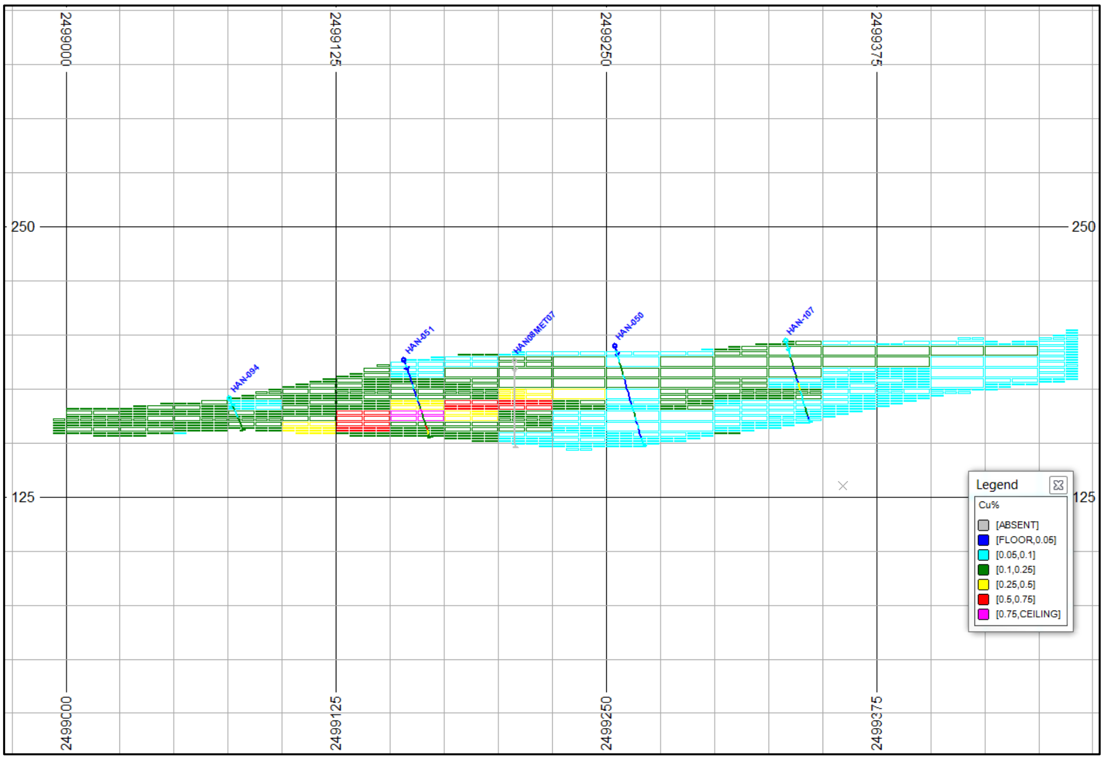


Figure 13-42: XS6750 cross-section through Kuervaara showing block model and composite file coloured by %Cu (Source: SRK Oct 2012)

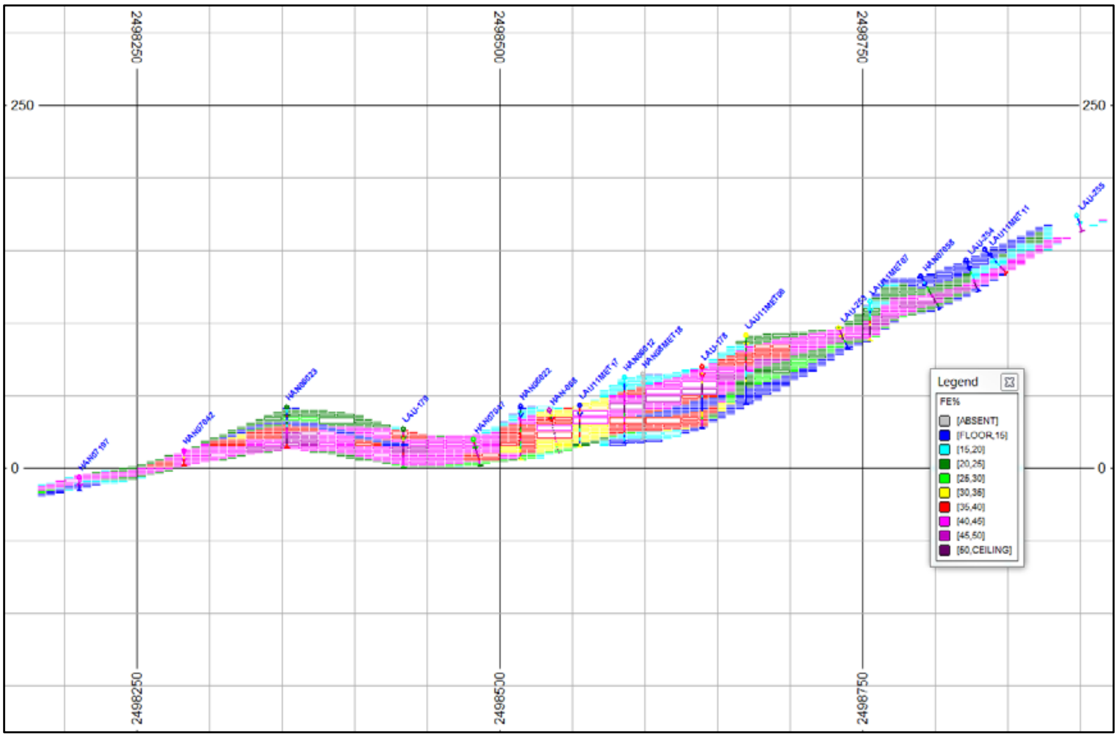


Figure 13-43: XS6700 cross-section through Laurinoja showing block model and composite file coloured by %Fe Total (Source: SRK Oct 2012)

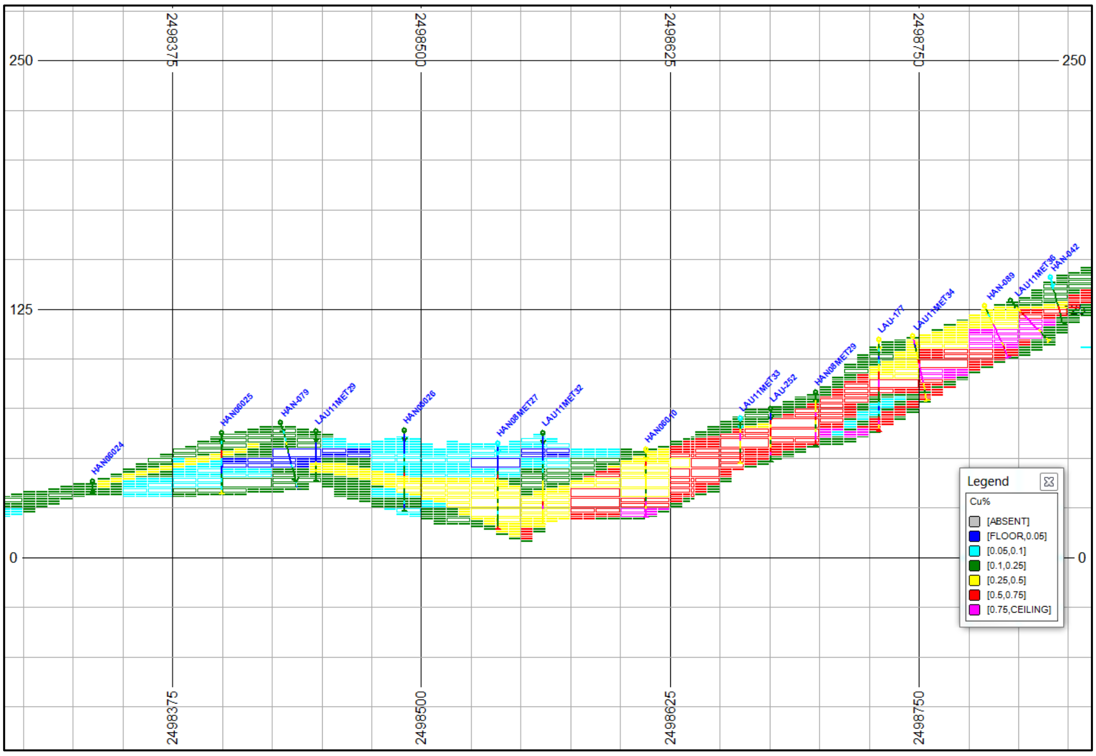


Figure 13-44: XS6800 cross-section through Laurinoja showing block model and composite file coloured by %Cu (Source: SRK Oct 2012)

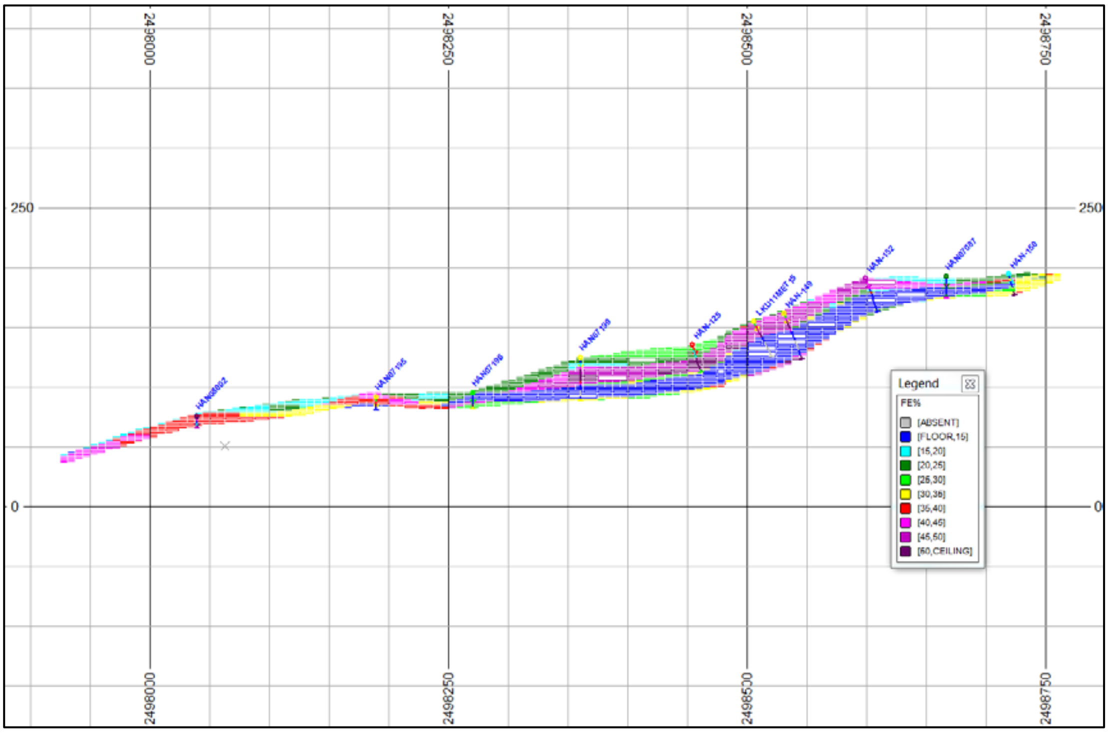


Figure 13-45: XS7550 cross-section through Lauku showing block model and composite file coloured by %Fe Total (Source: SRK Oct 2012)

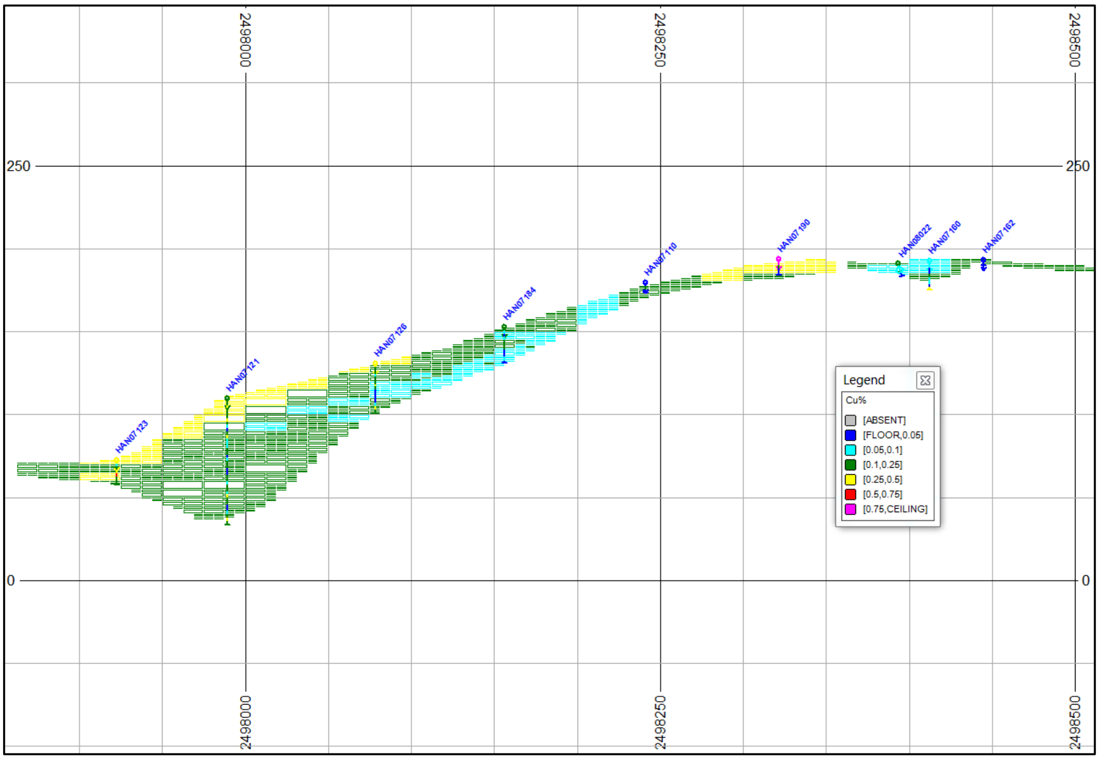


Figure 13-46: XS7750 cross-section through Lauku showing block model and composite file coloured by %Cu (Source: SRK Oct 2012)

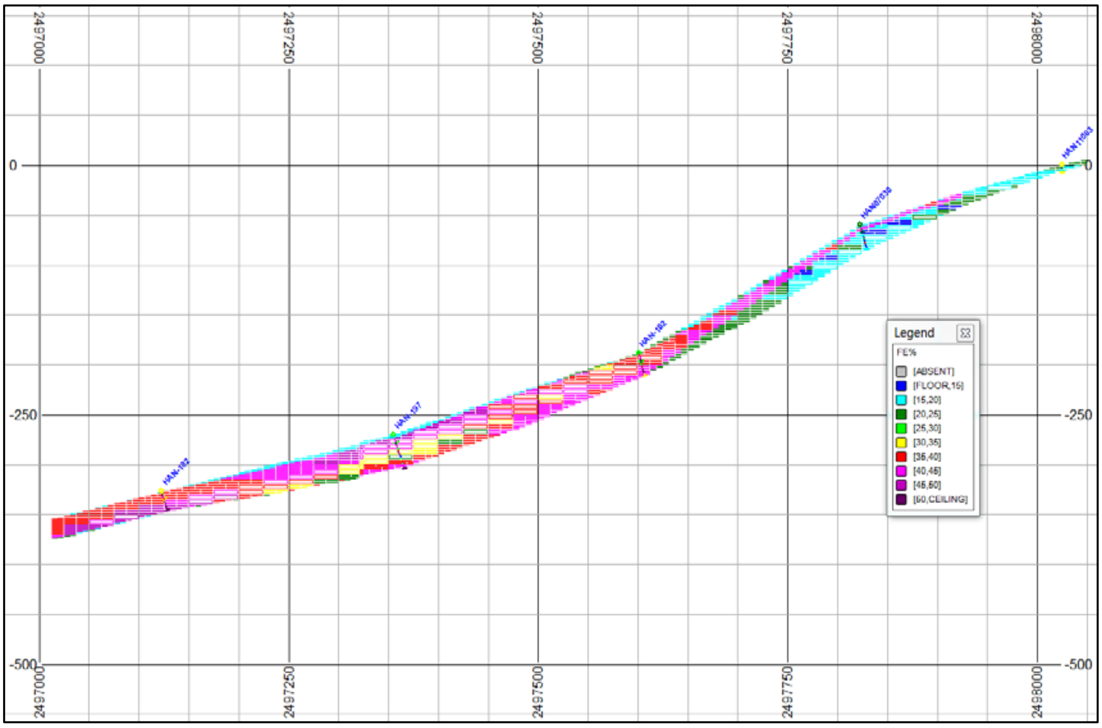


Figure 13-47: XS6800 cross-section through Kivivuopio showing block model and composite file coloured by %Fe Total (Source: SRK Oct 2012)

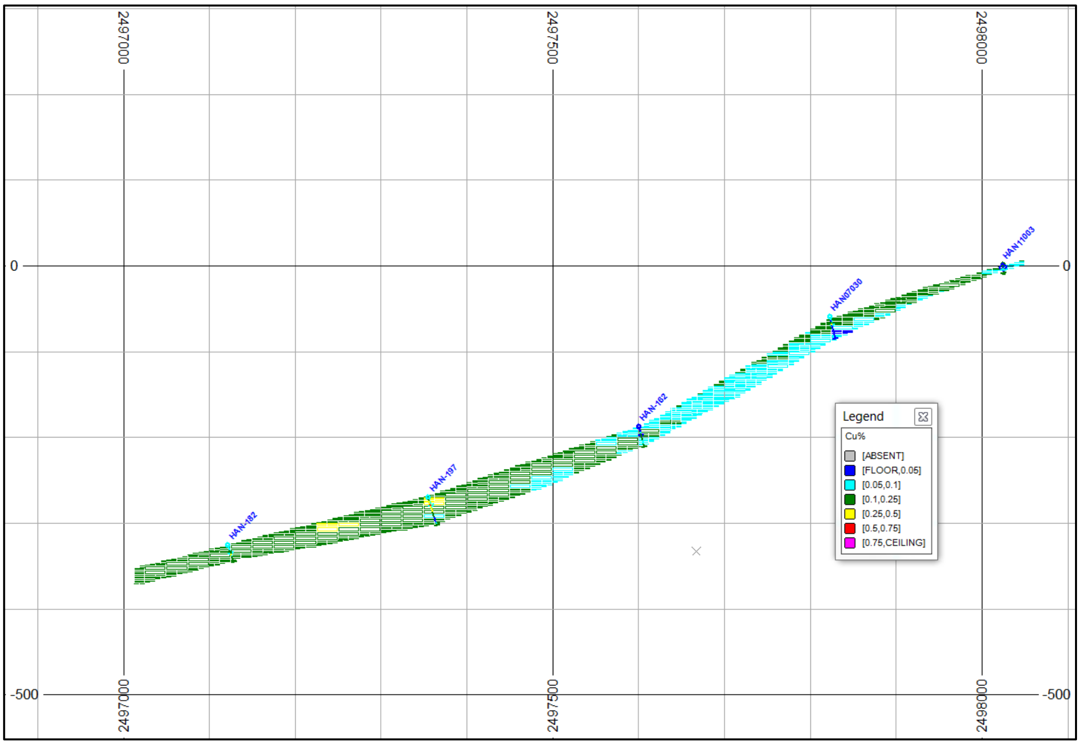


Figure 13-48: XS6800 cross-section through Kivivuopio showing block model and composite file coloured by %Cu (Source: SRK Oct 2012)

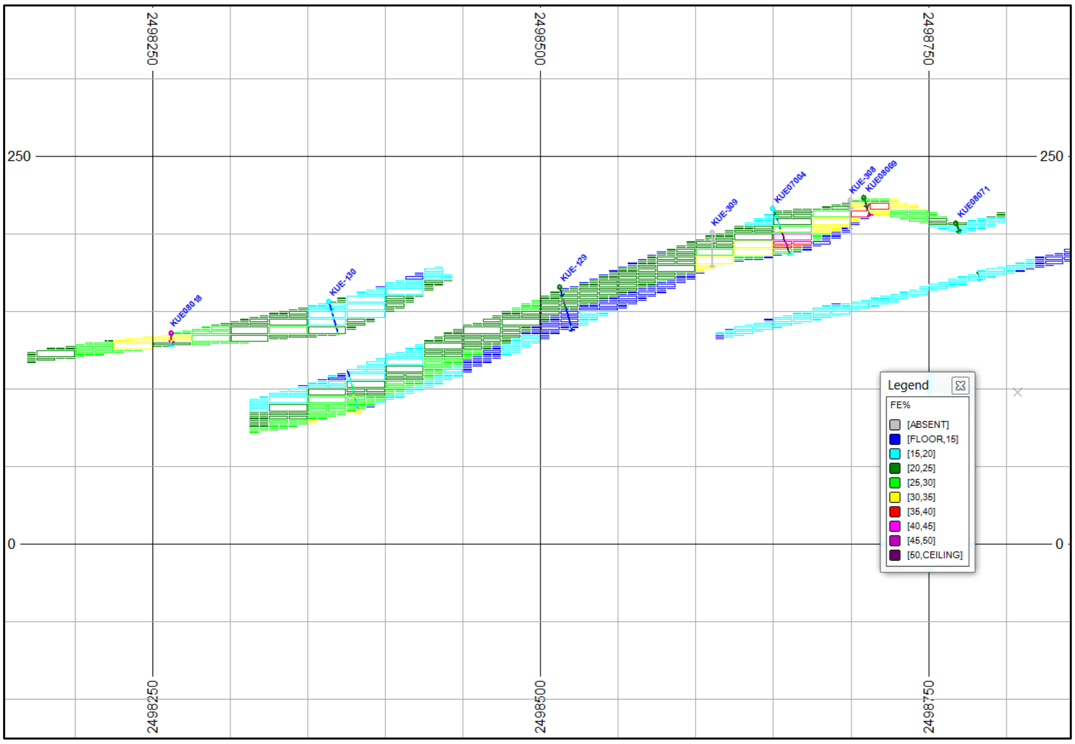


Figure 13-49: XS10600 cross-section through Kuervitikko showing block model and composite file coloured by %Fe Total (Source: SRK Oct 2012)

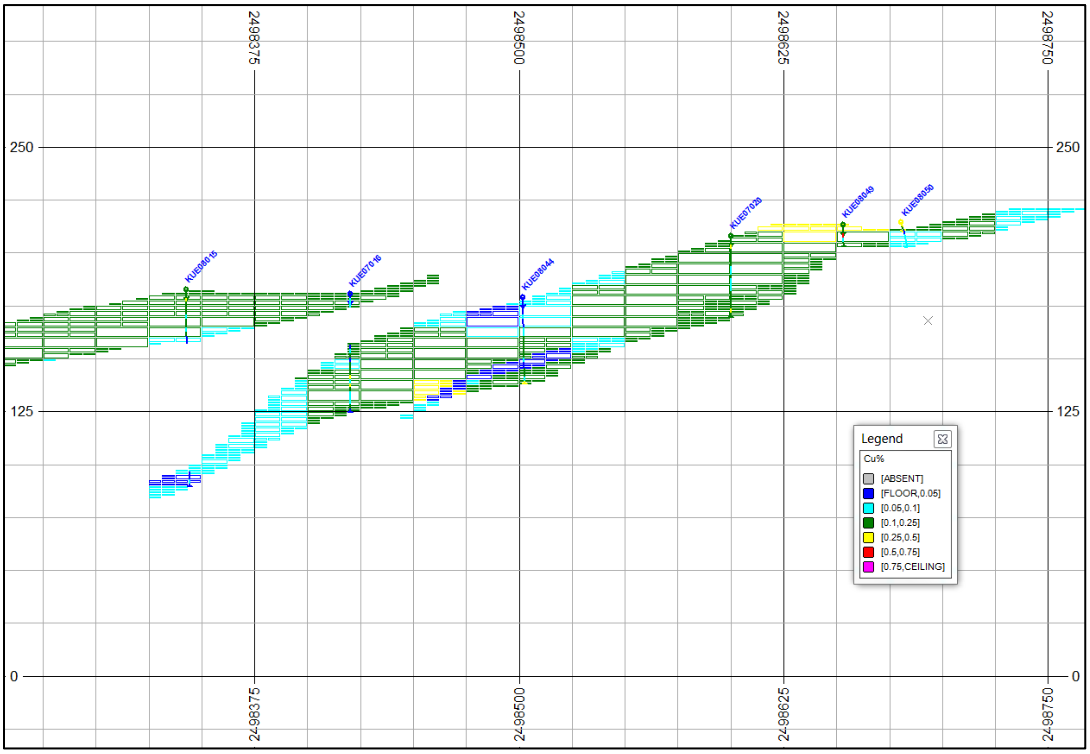


Figure 13-50: XS10750 cross-section through Kuervitikko showing block model and composite file coloured by %Cu (Source: SRK Oct 2012)

Mean Grade Comparisons

Along with visual comparisons, global mean grades for each domain were compared and are shown in Table 13-15. The % difference values over 10 are highlighted red, showing where the estimate has produced differing grades in the model to the declustered composite sample file. These higher values are mainly restricted to the minor elements, and occur in domains with the fewest samples. The elements of main economic interest, Fe and Cu generally show excellent correlation to the composite sample means.

Whilst there are some large discrepancies in percentage terms, these relate to cases where the values themselves are very low and the percentage discrepancy is accentuated. Overall, SRK is confident that the interpolated grades are a reasonable reflection of the available sample data with the key grade fields being well within acceptable limits.

Table 13-15: Comparison of global block model grade and declustered composite grade (search volume 1 only)

ZONE	Assay	Standard Deviation	Model Mean	CoV	Composite Mean	Absolute Difference	% Difference
110	FE	3.57	21.16	0.17	21.42	0.26	1.23
110	SIO2	3.85	37.75	0.10	38.29	0.54	1.42
110	MGO	1.10	5.78	0.19	5.74	0.04	0.70
110	AL2O3	1.53	6.95	0.22	6.76	0.20	2.89
110	CAO	2.15	12.41	0.17	12.48	0.07	0.58
110	P	0.05	0.07	0.68	0.07	0.00	0.02
110	MN	0.09	0.16	0.55	0.17	0.00	2.51
110	TIO2	0.11	0.39	0.29	0.38	0.00	0.82
110	K2O	0.32	0.58	0.56	0.57	0.01	2.22
111	FE	5.77	39.74	0.15	40.31	0.56	1.40
111	SIO2	4.80	23.17	0.21	22.49	0.68	3.03
111	MGO	0.68	4.43	0.15	4.38	0.05	1.20
111	AL2O3	1.28	3.54	0.36	3.54	0.01	0.25
111	CAO	1.99	8.29	0.24	8.03	0.26	3.21
111	P	0.02	0.06	0.39	0.06	0.00	2.81
111	MN	0.09	0.15	0.56	0.14	0.02	12.35
111	TIO2	0.09	0.22	0.39	0.22	0.00	1.17
111	K2O	0.16	0.31	0.51	0.28	0.02	8.02
119	FE	2.42	10.55	0.23	10.06	0.50	4.94
119	SIO2	4.14	45.96	0.09	47.09	1.13	2.40
119	MGO	1.00	5.23	0.19	5.07	0.16	3.09
119	AL2O3	1.98	11.54	0.17	11.69	0.15	1.30
119	CAO	3.86	13.32	0.29	13.34	0.02	0.13
119	P	0.03	0.10	0.32	0.11	0.00	1.93
119	MN	0.08	0.14	0.54	0.14	0.00	1.26
119	TIO2	0.14	0.61	0.22	0.62	0.00	0.71
119	K2O	0.68	0.97	0.70	1.01	0.04	3.74
120	FE	4.59	21.06	0.22	22.26	1.20	5.41
120	SIO2	4.37	40.06	0.11	40.65	0.60	1.47
120	MGO	1.46	4.85	0.30	5.08	0.23	4.55
120	AL2O3	2.25	6.23	0.36	5.90	0.34	5.75
120	CAO	3.81	11.77	0.32	11.08	0.69	6.19
120	P	0.03	0.06	0.53	0.07	0.01	7.67
120	MN	0.08	0.21	0.40	0.20	0.01	4.11
120	TIO2	0.15	0.29	0.52	0.27	0.02	7.38
120	K2O	0.54	0.77	0.70	0.78	0.01	1.39
121	FE	5.13	42.50	0.12	42.83	0.33	0.78
121	SIO2	4.65	22.95	0.20	22.63	0.32	1.42
121	MGO	0.77	4.28	0.18	4.22	0.05	1.25
121	AL2O3	1.30	3.23	0.40	3.14	0.09	2.84
121	CAO	1.87	6.10	0.31	6.02	0.08	1.39
121	P	0.03	0.05	0.49	0.05	0.00	1.69
121	MN	0.16	0.21	0.73	0.22	0.00	1.37
121	TIO2	0.08	0.17	0.45	0.16	0.01	4.35
121	K2O	0.31	0.51	0.62	0.49	0.02	3.71
122	FE	3.96	12.20	0.32	12.96	0.76	5.87
122	SIO2	5.44	50.68	0.11	51.17	0.49	0.97
122	MGO	1.58	4.02	0.39	3.93	0.09	2.40
122	AL2O3	2.77	11.16	0.25	11.15	0.01	0.13

ZONE	Assay	Standard Deviation	Model Mean	CoV	Composite Mean	Absolute Difference	% Difference
122	CAO	2.63	7.42	0.35	7.65	0.23	2.98
122	P	0.03	0.08	0.41	0.08	0.00	4.52
122	MN	0.05	0.11	0.44	0.11	0.00	0.92
122	TIO2	0.18	0.42	0.42	0.42	0.00	0.81
122	K2O	0.56	1.43	0.39	1.39	0.03	2.44
129	FE	3.17	8.38	0.38	8.91	0.52	5.89
129	SIO2	4.11	52.84	0.08	53.98	1.14	2.11
129	MGO	1.26	3.82	0.33	4.06	0.25	6.10
129	AL2O3	2.41	13.02	0.18	12.15	0.87	7.15
129	CAO	3.03	8.14	0.37	9.37	1.24	13.19
129	P	0.04	0.10	0.34	0.10	0.01	8.89
129	MN	0.06	0.11	0.53	0.13	0.02	17.81
129	TIO2	0.20	0.58	0.34	0.52	0.06	11.25
129	K2O	0.89	1.53	0.58	1.49	0.03	2.33
130	FE	5.24	23.97	0.22	23.76	0.21	0.90
130	SIO2	8.62	41.70	0.21	42.12	0.42	1.00
130	MGO	2.22	3.18	0.70	3.32	0.14	4.10
130	AL2O3	2.88	7.57	0.38	7.75	0.18	2.35
130	CAO	5.56	5.35	1.04	5.07	0.28	5.52
130	P	0.01	0.04	0.33	0.04	0.00	2.06
130	MN	0.15	0.15	1.01	0.13	0.02	13.14
130	TIO2	0.21	0.45	0.48	0.47	0.02	4.91
130	K2O	0.53	0.76	0.70	0.74	0.01	1.97
132	FE	3.44	14.13	0.24	14.22	0.09	0.64
132	SIO2	11.96	46.04	0.26	47.08	1.03	2.19
132	MGO	1.54	4.86	0.32	4.84	0.03	0.56
132	AL2O3	2.55	7.85	0.32	8.33	0.48	5.78
132	CAO	6.75	9.48	0.71	8.61	0.87	10.08
132	P	0.01	0.05	0.31	0.05	0.00	2.78
132	MN	0.18	0.20	0.91	0.14	0.06	39.22
132	TIO2	0.21	0.44	0.49	0.49	0.05	10.51
132	K2O	0.52	0.97	0.53	1.02	0.05	4.54
139	FE	1.30	9.77	0.13	10.17	0.41	3.98
139	SIO2	11.83	31.91	0.37	34.60	2.69	7.79
139	MGO	1.20	5.17	0.23	4.89	0.28	5.78
139	AL2O3	2.97	6.25	0.48	7.17	0.92	12.86
139	CAO	11.45	25.56	0.45	21.29	4.27	20.06
139	P	0.01	0.05	0.17	0.05	0.00	2.17
139	MN	0.22	0.49	0.45	0.40	0.09	22.11
139	TIO2	0.17	0.34	0.48	0.40	0.06	14.68
139	K2O	0.45	1.03	0.44	1.21	0.18	15.11

CUZONE	Assay	Standard Deviation	Model Mean	CoV	Composite Mean	Absolute Difference	% Difference
1	CU_PPM	534.02	1,124	0.48	1,046	77.72	7.43
1	AU_PPB	23.79	26	0.92	25	0.80	3.22
1	S	1.08	2.9	0.37	2.9	0.01	0.29
1	CO_PPM	35.46	144	0.25	144	0.65	0.45
2	CU_PPM	778.74	1,385	0.56	1,401	15.72	1.12
2	AU_PPB	130.30	49	2.68	45	3.19	7.01
2	S	0.72	1.9	0.37	2.0	0.06	2.83
2	CO_PPM	50.02	136	0.37	142	6.06	4.26
3	CU_PPM	1567.62	5,017	0.31	5,320	302.25	5.68
3	AU_PPB	423.46	394	1.07	336	58.43	17.41
3	S	0.79	2.5	0.31	2.7	0.16	5.89
3	CO_PPM	59.02	177	0.33	181	3.32	1.84
4	CU_PPM	771.63	1,845	0.42	1,998	153.11	7.66
4	AU_PPB	147.94	204	0.73	204	0.45	0.22
4	S	1.30	2.5	0.52	2.5	0.05	1.81
4	CO_PPM	51.77	115	0.45	116	1.11	0.96
5	CU_PPM	324.31	438	0.74	566	128.28	22.67
5	AU_PPB	28.02	17	1.62	22	4.33	20.03
5	S	0.72	0.9	0.80	0.9	0.04	4.26
5	CO_PPM	20.32	53	0.38	57	3.89	6.85

Validation slices

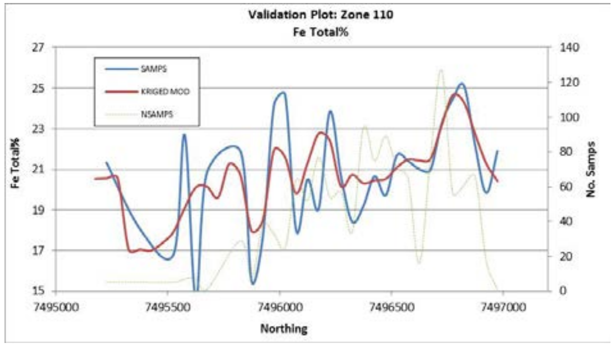
As part of the validation process, the block model and input samples that fall within defined sectional or elevation criteria were compared and the results displayed graphically to check for visual discrepancies between grades. Due to the near north-south strike of the mineralisation, the Y axis (Northing) slices show excellent cross-section comparisons of sample and block model grades.

Whilst this process does not truly replicate the samples used in the estimation of each block, the process of sectional validation quickly highlights areas of concern within the model and enables a more thorough and quantifiable check to be undertaken in specific areas of the model. Each graph also shows the number of samples available for the estimation. This provides information relating to the support of the blocks in the model. Only those blocks estimated within search volume one were compared, as this represents the estimated data using the optimum sample criteria.

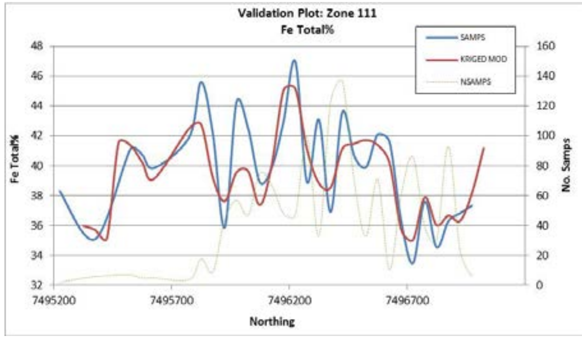
Figure 13-51 to Figure 13-52 show the Fe and Cu validation slices through each of the mineralisation zones. They show generally excellent correlation to the sample data, with a smoothing effect on the large outliers, but also showing good correlation to subtle changes (in areas of numerous samples).

SRK is confident that the block model grades reconcile well to the composite sample grades

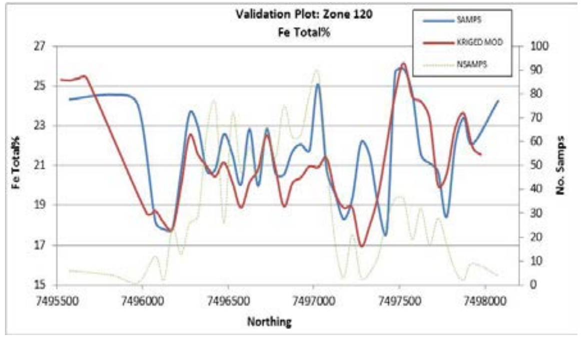
ZONE 110



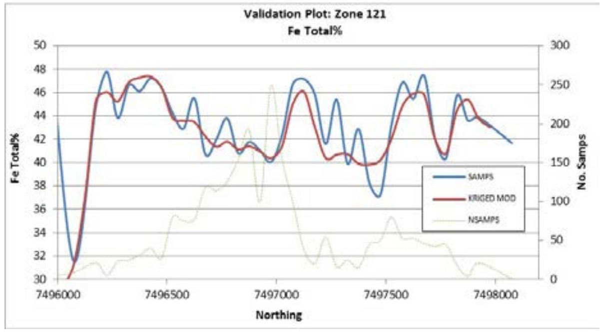
ZONE 111



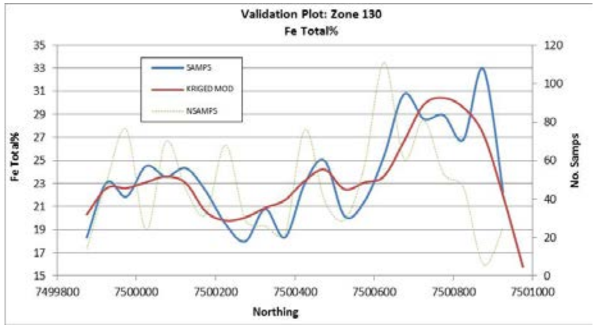
ZONE 120



ZONE 121



ZONE 130



ZONE 132

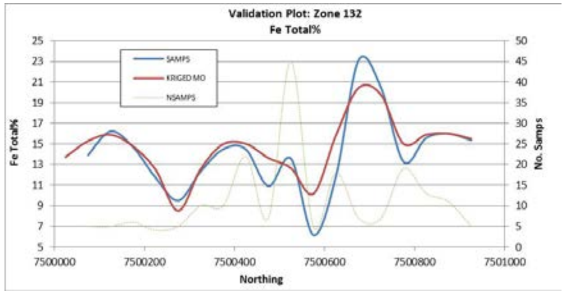


Figure 13-51: %Fe Total Validation slices per zone along the Y axis (northing)

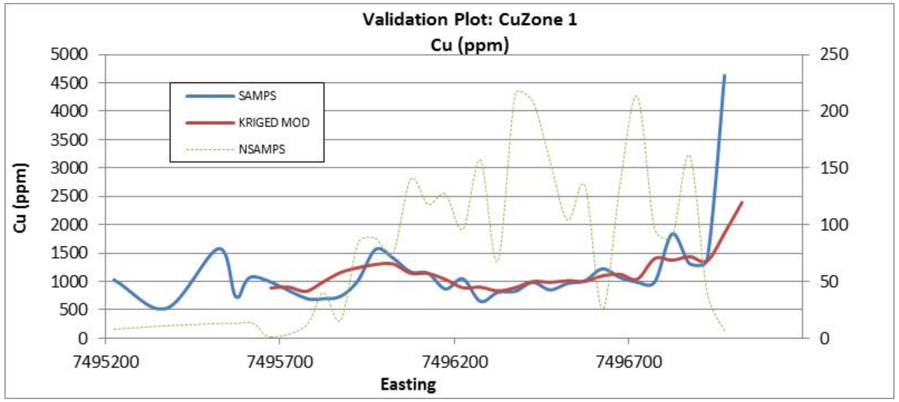
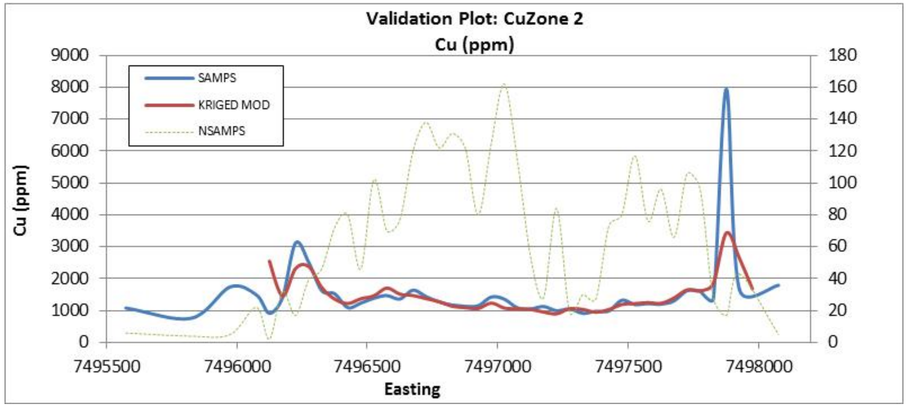
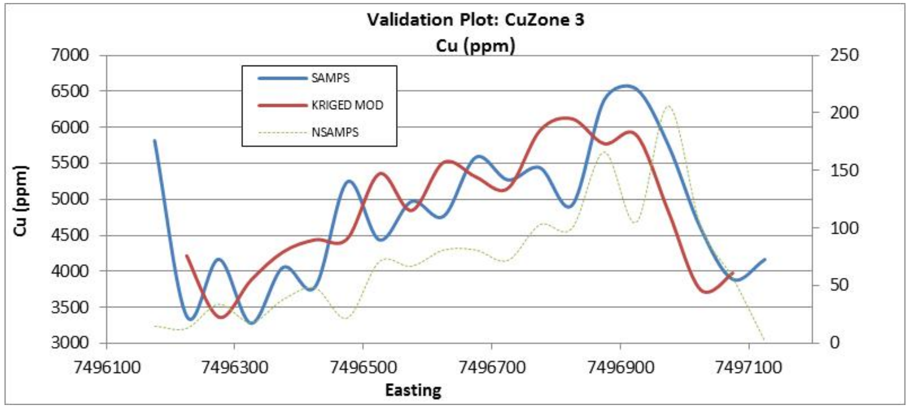
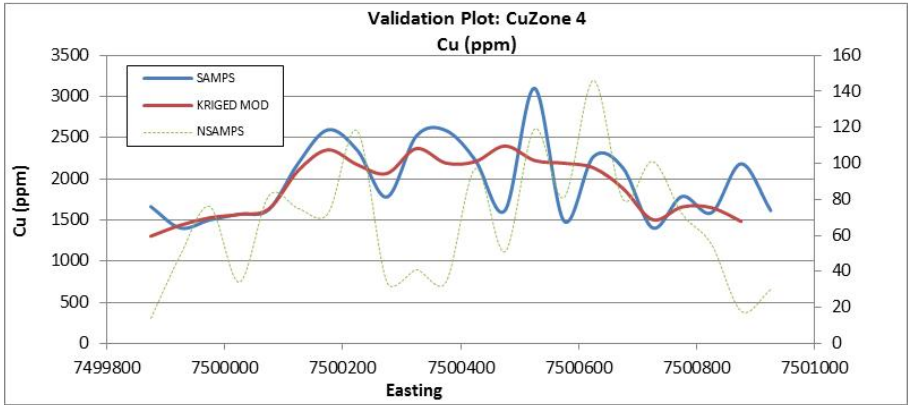
CUZONE 1	CUZONE 2
	
CUZONE 3	**CUZONE 4**
	

Figure 13-52: Cu (ppm) Validation slices per zone along the Y axis (northing)

13.9 Mineral Resource Classification

13.9.1 CIM Definitions

Mineral Resource

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

The term Mineral Resource covers mineralisation and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase 'reasonable prospects for economic extraction' implies a judgement by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralisation that, under realistically assumed and justifiable technical and economic conditions, might become economically extractable. These assumptions must be presented explicitly in both public and technical reports.

Inferred Mineral Resource

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.

Due to the uncertainty which may attach to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Indicated Mineral Resource

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

Mineralisation may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralisation. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the Project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough to confirm both geological and grade continuity.

Mineralisation or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade of the mineralisation can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

13.9.2 Hannukainen Classification

Introduction

To classify the Hannukainen deposit, the following key indicators were used:

- geological complexity;
- quality of data used in the estimation:
    - QAQC, historical vs modern data;
- results of the geostatistical analysis:
    - variography;
    - QKNA results; and
- quality of the estimated block model.

Geological Complexity

Due to the high density of drill data, it is possible to see clear geological continuity between sections and deduce a clear geological model for the Project. The Fe-mineralisation zones are skarn-hosted magnetite deposits, with almost all of the mineralisation occurring within the skarn unit which lies at the contact zone between intrusive igneous units and metavolcanics. The drill spacing has allowed for the interpretation of continuous internal high grade zones and the Cu-mineralisation generally occurs in chalcopyrite veins and disseminations, and occurs generally within defined units within the magnetite skarn unit, with patches of mineralisation occurring in the magnetite footwall and hangingwall zones. Zones of internal waste material have also been domained with a high level of accuracy. Hangingwall and footwall contacts of the magnetite orebody are clearly defined using a 15% Fe Total grade cut-off in addition to the evaluation of Satmagan readings, magnetic susceptibility and sulphur data.

It is the opinion of SRK that the associated risk relating to geological complexity is low.

Quality of Data

QAQC checks were implemented throughout the Northland assaying period that included the insertion of standards, blanks, laboratory duplicates and the use of an umpire laboratory. In general, the results of the QAQC checks provided reasonable results although it is recognised that the certified standards used show a possible underestimate in %Fe Total grade. Accurate inter-laboratory and umpire laboratory checks were completed, but it is the opinion of SRK that an unsuitable blank sample source is being used. The historical assays, from the Rautaruukki drilling in the 1970s and 1980s did not contain any QAQC data; however, it has been verified by Northland with twinned drilling and re-assaying. SRK is confident that the two datasets are compatible and no bias has been introduced using the historical data.

Overall, SRK is confident that the results of the QAQC analysis have validated the accuracy of the database being used to generate the Mineral Resource Estimate.

A comprehensive dataset of density has also been generated by Northland throughout the sampling periods. These have been analysed and all recent density measurements have been deemed appropriate for use in the Mineral Resource Estimation via a regression formula of density calculated against the %Fe Total grade. Average density has been assigned to the host lithologies due to limited data in these geological domains.

Results of the Geostatistical Analysis

The data used in the geostatistical analysis resulted in robust down-hole variograms. This enabled the nugget and short-scale variation in grade to be determined with a high level of confidence. The along-strike and down dip variograms also provided robust results and could be modelled effectively with ranges greater than the drillhole spacing. The detailed variography allowed for the determination of appropriate search ellipse parameters to be determined through the application of multiple QKNA tests prior to the grade interpolation.

Quality of the Estimated Model

The validation tools used show that the input data used to estimate the model is replicated in the estimation. The block model grades are smoothed around the input composites as shown in the validation slices and the mean grades of the block model and composites are comparable for all modelled domains.

Classification

The Project has been classified as containing Measured, Indicated and Inferred Mineral Resources. The classified model is shown in Figure 13-54 and Figure 13-55 for the Hannukainen and Kuervitikko deposits respectively.

Measured Mineral Resources have been assigned in well supported blocks as highlighted by the following criteria:

- low geological complexity;
- drillhole spacing less than the 2/3rd geostatistical range; and
- all blocks estimated in search volume one, using the optimum search parameters determined.

SRK often uses the slope of regression probability plot to assist with the classification. However, in this instance, the plot shown in Figure 13-53 shows a smooth curve up until around 0.9, when a break in population is defined. This break coincides with a probability of approximately 0.95, leaving very little data above 0.9 slope. It was therefore deemed inappropriate for SRK to use the slope of regression as a classification tool in this instance.

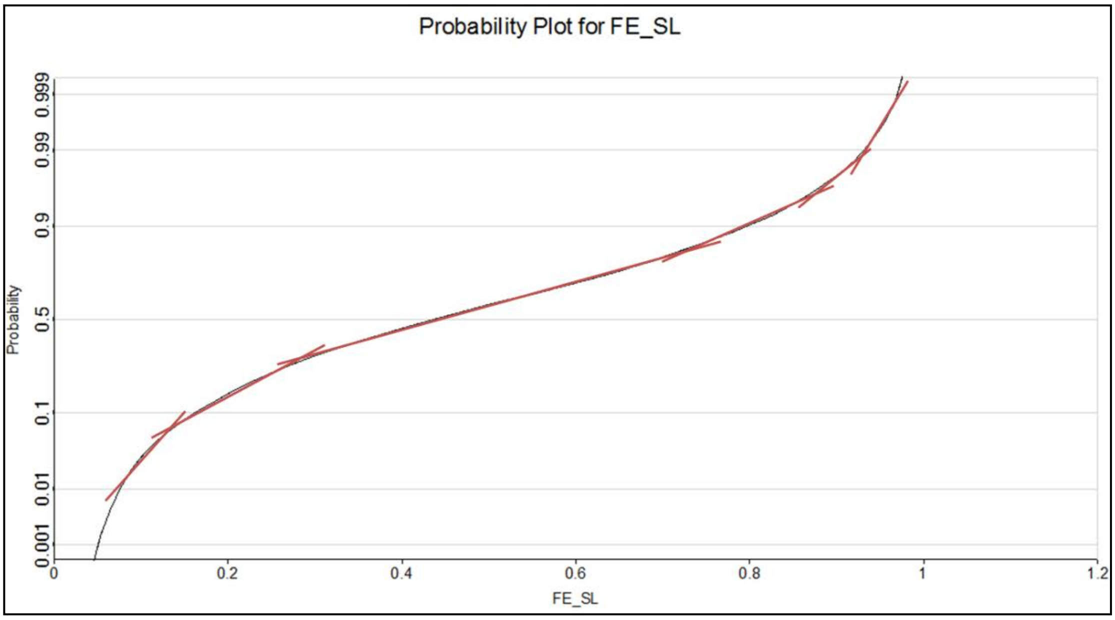


Figure 13-53: Probability plot of Fe Slope of Regression

Indicated Mineral Resources were assigned to blocks estimated within search volume 2 that demonstrated continuity between sections. This search used distances double the first search, where the drillholes are still spaced closely enough for geological and grade continuity to be reasonably assumed.

Inferred Mineral Resources were assigned to the remainder of the mineralisation in the model. The geological and grade continuity remains good despite the sparse drilling and can be still reasonably assumed.

Figure 13-54 and Figure 13-55 show the Hannukainen and Kuervitikko models coloured by classification. To determine the final Mineral Resource Statement, and so as to comply with the CIM guidelines, the resulting blocks have been subjected to a Whittle pit optimisation exercise to determine the proportion of the material defined that has a reasonable prospect of eventual economic extraction. This exercise is not intended to generate a Mineral Reserve and is purely used to assist in determining the possible extent of the Mineral Resource model.

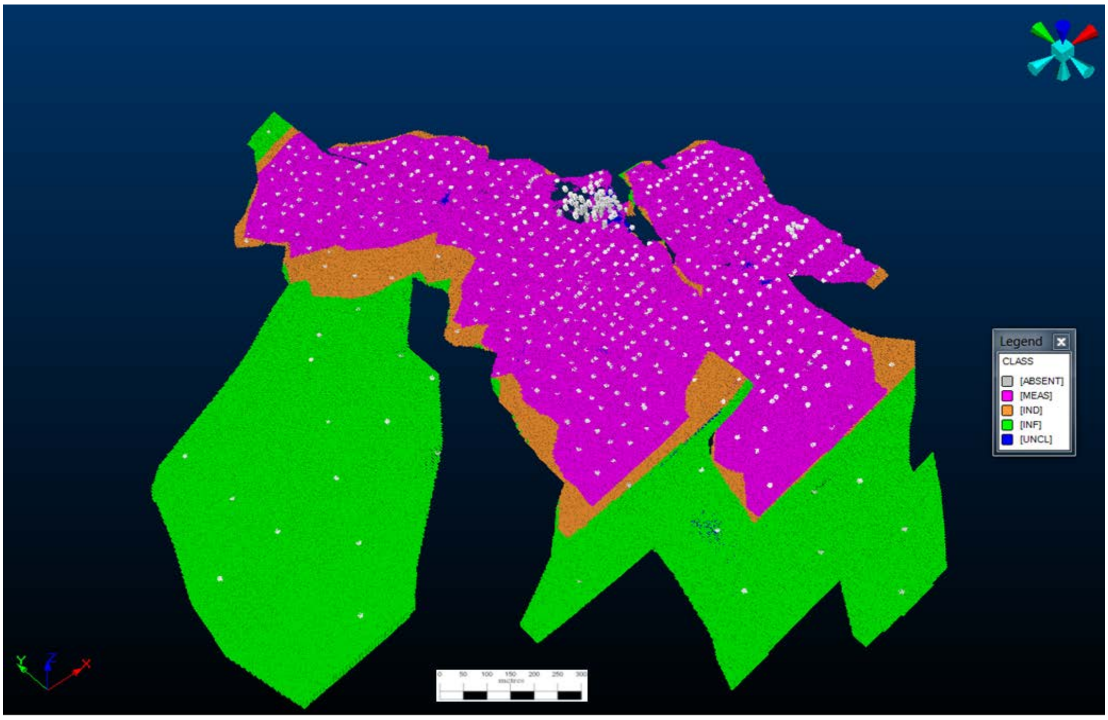


Figure 13-54: Hannukainen classified model (Source: SRK Oct 2012)

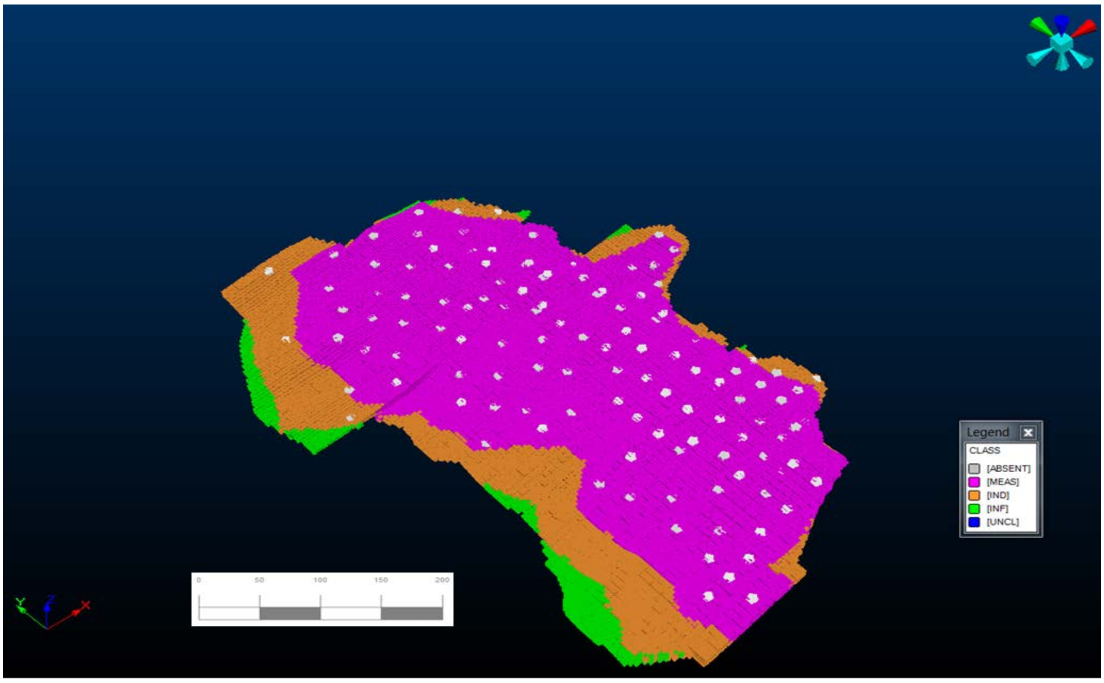


Figure 13-55: Kuervitikko classified model (Source: SRK Oct 2012)

13.10 Whittle Pit Optimisation Assumptions

The Whittle optimisation requires the input of reasonable processing and mining cost parameters in addition to appropriate pit slope angles and processing recoveries. Table 13-16 shows the assumptions applied in the Whittle optimisation used to determine the material which SRK considers suitable for open pit mining activities.

Table 13-16: Whittle optimisation parameters

Parameters for Whittle Run 23 – Revenue Factor =1	
Slope Angles	
Overburden	27°
Hannukainen North	47°
Hannukainen Central & South	45
Kuervitikko	48
Mining & Processing	
Mining Recovery	100
Mining Dilution	0
Fe Processing Recovery	=98.5*(1-exp(--0.06038*(Fe-6)))*(-1.962744*(S/Fe)+1)
Cu Processing Recovery	=15.2487213896*ln(x) + 106.8504693385
Au Processing Recovery	=5.8861814790*ln(Au)+38.9178069302
Product Specifications	
Fe Concentrate	68.9% Fe
Cu Concentrate - Cu Grade	25% Cu
Cu Concentrate - Au Grade	7.7 g/t Au
Costs	
Base Mining Cost	USD1.78 /t
Average Mining Cost	USD2.06 /t
Incremental Cost	USD0.02 /t/5m
Processing Cost	USD6.78 /t processed
Transport Cost Fe concentrate	USD19.22 /t
Transport Cost Cu concentrate	USD23.24 /t
Industrial Area	USD0.68/ t processed
General & Administrative Cost	USD1.33/ t processed
Royalty	0.15%
Cu Selling Cost	USD0.27 /lb Cu in concentrate
Metal Prices	
Fe Selling Price	USD1.50/dmtu
Cu Selling Price	USD3.35/lb
Au Selling Price	USD1375/oz

13.10.1 Processing Recoveries

SRK was supplied with mineralisation recovery factors for Fe, Cu and Au as highlighted in Table 13-16.

13.10.2 Underground Mineral Resource cut-off grade calculation

In addition to the Mineral Resources included within the calculated Whittle pit shell, SRK has undertaken an underground study to produce a cut-off grade for the determination of underground Mineral Resources with the potential for eventual economic extraction.

The Hannukainen deposits are considered suitable for high underground production rates in excess of 2 Mtpa with longhole open stoping and/or a combination of room and pillar mining methods. The underground mining cost used in the analysis is USD14.4/t of ore and was based on an analysis of underground production rates and order of magnitude cost models from the Infomine database subscription. All other relevant assumptions are the same as the Whittle optimisation study.

The underground study produced a cut-off grade of 35.6% Fe equivalent to be used to restrict the underground Mineral Resources, where:

Fe equivalent=(FE/100+(CU_PPM/1000000*82.8833)+(AU_PPB/1000*0.13237512))*100

13.11 Mineral Resource Statement

The Mineral Resource Statement generated by SRK is divided into two categories: open-pit and underground. The open pit Mineral Resource has been restricted to all classified material falling within a Whittle shell representing a metal price of USD1.50/dmtu for magnetite concentrate, USD3.35/lb for copper and USD1375/oz for gold and through the application of the parameters outlined in section 13.10. In addition, the open pit Mineral Resource were reported above a Fe equivalent cut-off grade of 13.3%, however, this represented almost all material within the pit. This represents the material which SRK considers has reasonable prospect for eventual economic extraction potential via open pit mining methods based on the above Whittle optimisation analysis. The underground Mineral Resource has been calculated using a Fe equivalent cut-off grade of 35.6%, calculated using the same metal prices as the Whittle shell. This represents the material which SRK considers has reasonable prospect for eventual economic extraction potential via underground mining methods.

Table 13-17 shows the resulting Mineral Resource Statement for the Project. The statement has been classified by Qualified Person Howard Baker (MAusIMM(CP)) in accordance with CIM Guidelines. It has an effective date of 24 October 2012 and incorporates all drilling undertaken to date. Mineral Resources that are not mineral reserves do not have demonstrated economic viability. SRK is not aware of any factors (environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors) that have materially affected the Mineral Resource Estimate.

The quantity and grade of reported Inferred Mineral Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral Resources as an Indicated or Measured Mineral Resource; and it is uncertain if further exploration will result in upgrading them to an indicated or Measured Resource category.

Table 13-17: Mineral Resource Statement

Deposit	Resource Category	Tonnes (Mt)	%Fe Total	%Cu	Au(g/t)	%S
Open Pit						
Deposit	**Resource Category**	**Tonnes (Mt)**	**%Fe Total**	**%Cu**	**Au(g/t)**	**%S**
Hannukainen	Measured	120	32.25	0.18	0.083	2.47
	Indicated	3	24.80	0.19	0.064	2.15
	Meas+Ind	**123**	**32.08**	**0.18**	**0.082**	**2.46**
	Inferred	0.9	27.16	0.19	0.024	2.86
Kuervitikko	Measured	34	23.22	0.19	0.217	2.49
	Indicated	3	23.36	0.15	0.174	1.99
	Meas+Ind	**36**	**23.23**	**0.19**	**0.213**	**2.45**
	Inferred	0.1	19.37	0.15	0.129	2.48
TOTAL	Measured	154	30.28	0.18	0.112	2.47
	Indicated	6	24.09	0.17	0.118	2.07
	Meas+Ind	**159**	**30.06**	**0.18**	**0.112**	**2.46**
	Inferred	1.0	26.24	0.19	0.036	2.82
Underground						
Deposit	**Resource Category**	**Tonnes (Mt)**	**%Fe Total**	**%Cu**	**Au(g/t)**	**%S**
Hannukainen	Measured	14	32.13	0.18	0.150	2.4
	Indicated	8	32.44	0.16	0.072	1.9
	Meas+Ind	**22**	**32.24**	**0.17**	**0.123**	**2.2**
	Inferred	61	32.33	0.15	0.044	2.3
Kuervitikko	Measured	3	17.79	0.19	0.140	2.8
	Indicated	3	20.27	0.17	0.169	2.7
	Meas+Ind	**6**	**19.15**	**0.18**	**0.156**	**2.7**
	Inferred	1	23.21	0.15	0.203	2.3
TOTAL	Measured	17	29.88	0.18	0.149	2.4
	Indicated	11	28.82	0.17	0.101	2.2
	Meas+Ind	**28**	**29.46**	**0.18**	**0.130**	**2.3**
	Inferred	62	32.14	0.15	0.047	2.3
Combined Open Pit and Underground (Total Resource)						
Deposit	**Resource Category**	**Tonnes (Mt)**	**%Fe Total**	**%Cu**	**Au(g/t)**	**%S**
Hannukainen	Measured	154	32.24	0.18	0.090	2.5
	Indicated	6	30.37	0.17	0.070	2.0
	Meas+Ind	**159**	**32.17**	**0.18**	**0.089**	**2.4**
	Inferred	61	32.25	0.15	0.044	2.3
Kuervitikko	Measured	36	22.82	0.19	0.210	2.5
	Indicated	6	21.69	0.16	0.172	2.4
	Meas+Ind	**42**	**22.66**	**0.19**	**0.205**	**2.5**
	Inferred	1	22.87	0.15	0.196	2.3
TOTAL	Measured	171	30.44	0.18	0.113	2.5
	Indicated	17	25.88	0.17	0.122	2.2
	Meas+Ind	**187**	**30.04**	**0.18**	**0.114**	**2.4**
	Inferred	63	32.05	0.15	0.047	2.3

(1) The effective date of the Mineral Resource is 24 October 2012

(2) The open pit Mineral Resource Estimate for the Hannukainen deposit was constrained within grade based solids and within a Lerchs-Grossman pit shell defined by the following assumptions: selling price of USD1.50/dmtu for iron, USD3.35/lb for copper and USD1,375/oz for gold; slope angles of 45º (Hannukainen South and Central), 47º (Hannukainen North) and 48º (Kuervitikko); a base case mining cost of USD1.78/t and an incremental cost will be applied to reflect the haulage at various depths - the incremental cost above and below the reference level will be USD0.02/t/block height, where the block height is 5m; onsite process operating costs of USD6.78/t ore feed; transport costs for iron concentrate of USD 19.22/t and copper concentrate of USD23.24/t; G&A costs of USD1.33/t ore feed; royalty of 0.15%; copper selling cost of USD0.27/lb.

(3) The underground Mineral Resource Estimate for the Hannukainen deposit was reported above an Fe-equivalent cu-off grade of 35.6% for everything beneath the Whittle shell. The Fe equivalent cut-off calculation is defined by the assumptions above, but with an underground mining cost of USD14.4/t.

(4) Mineral Resources for the Hannukainen deposit has been classified according to the "CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines (December 2005) by Howard Baker (MAusIMM(CP)) an independent Qualified Person as defined by CIM.

The Project has a combined Measured and Indicated open pit Mineral Resource of 159 Mt grading 30.06% Fe Total, 0.18% Cu, and 0.112 g/t Au. Of this, 154 Mt grading 30.28% Fe Total, 0.18% Cu, and 0.112 g/t Au is in the Measured category, and 6 Mt grading 24.09% Fe Total, 0.17% Cu, and 0.118 g/t Au is in the Indicated category. In addition, 1 Mt grading 26.24% Fe Total, 0.19% Cu and 0.036 g/t Au is in the Inferred category.

The Project also has a combined Measured and Indicated underground Mineral Resource of 28 Mt grading 29.46% Fe Total, 0.18% Cu, and 0.130 g/t Au. Of this, 17 Mt grading 29.88% Fe Total, 0.18% Cu, and 0.149 g/t Au is in the Measured category, and 11 Mt grading 28.82% Fe total, 0.17% Cu, and 0.101 g/t Au is in the Indicated category. In addition, 62 Mt grading 32.14% Fe Total, 0.15% Cu, and 0.047 g/t Au is in the Inferred category.

In total, the Project has a joint underground and open pit combined Measured and Indicated Mineral Resource of 187 Mt grading 30.04% Fe Total, 0.18% Cu and 0.114 g/t Au. In addition, there is a total of 63 Mt grading 32.05% Fe Total, 0.15% Cu and 0.047 g/t Au in the Inferred category.

In addition to the Mineral Resource statement above, 1 Mt of underground Inferred material above the cut-off grade falls outside of the current Northland exploration licence claim area. This material cannot be reported as a Mineral Resource.

Figure 13-56 and Figure 13-57 show the Whittle pit shells generated using a metal price of USD1.50/dmtu for magnetite concentrate USD3.35 /lb for copper and USD1,375/oz for gold for Hannukainen and Kuervitikko, respectively.

Figure 13-58 and Figure 13-59 show the underground Mineral Resource blocks (>35.6% Fe equivalent) coloured by Classification category for Hannukainen and Kuervitikko, respectively.

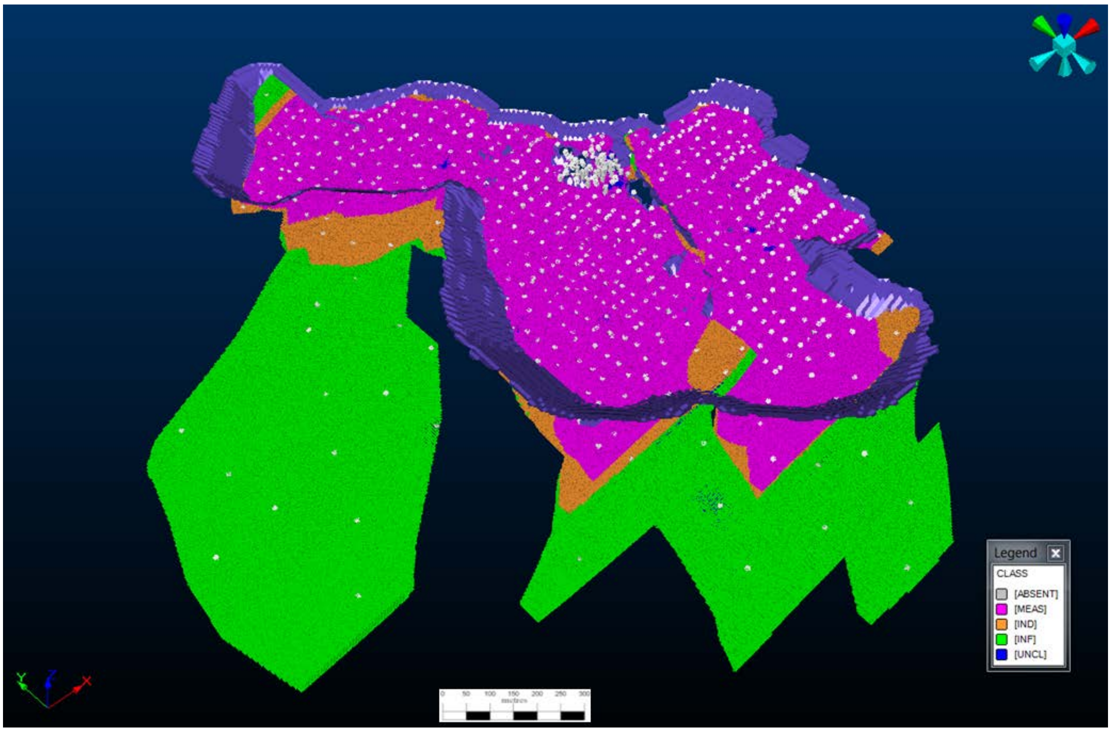


Figure 13-56: Hannukainen Whittle pit shell and classified model (Source: SRK Oct 2012)

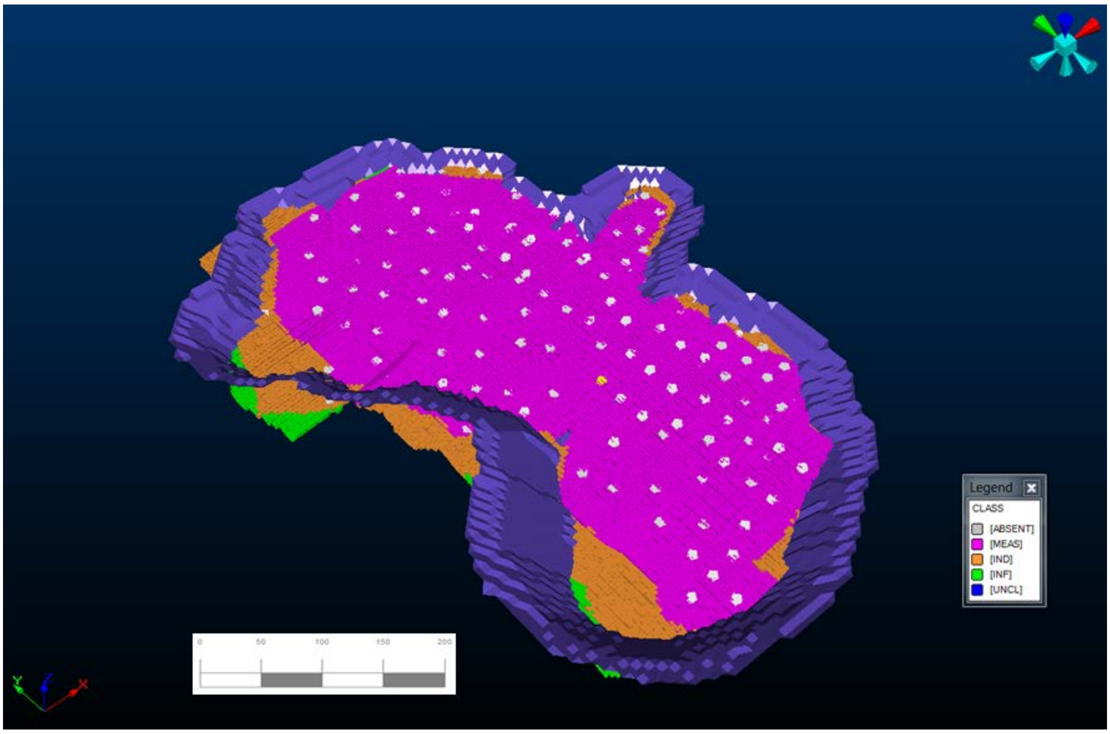


Figure 13-57: Kuervitikko Whittle pit shell and classified model (Source: SRK Oct 2012)

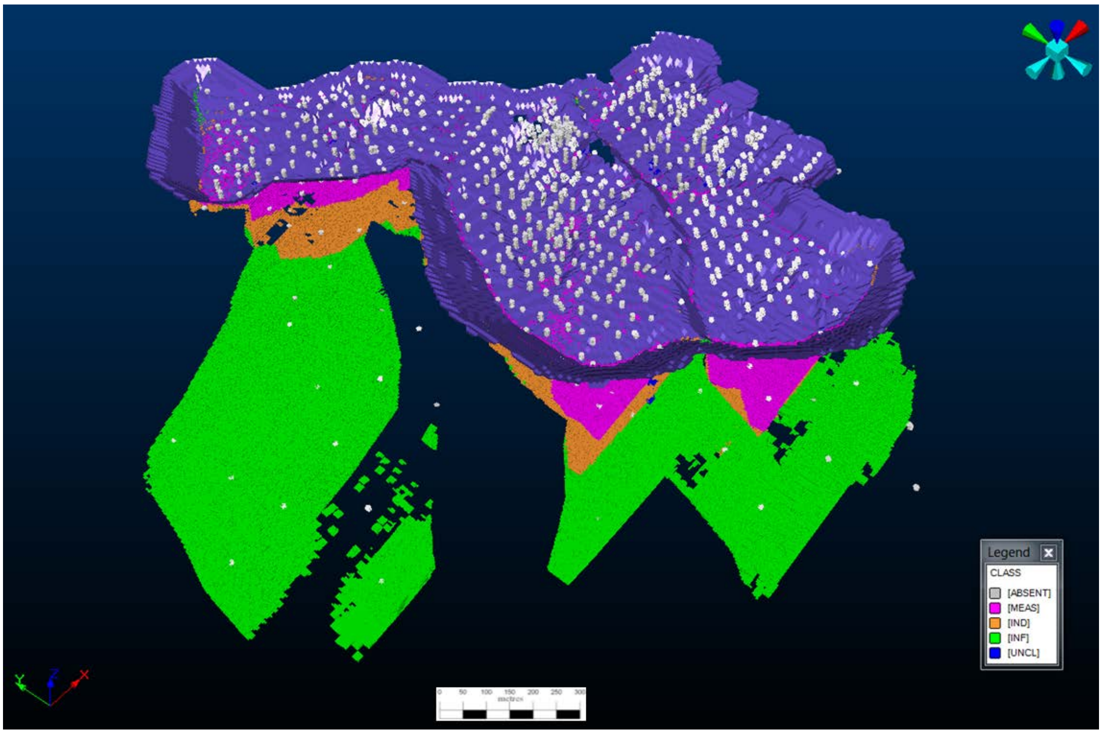


Figure 13-58: Underground Mineral Resource blocks at Hannukainen (Source: SRK Oct 2012)

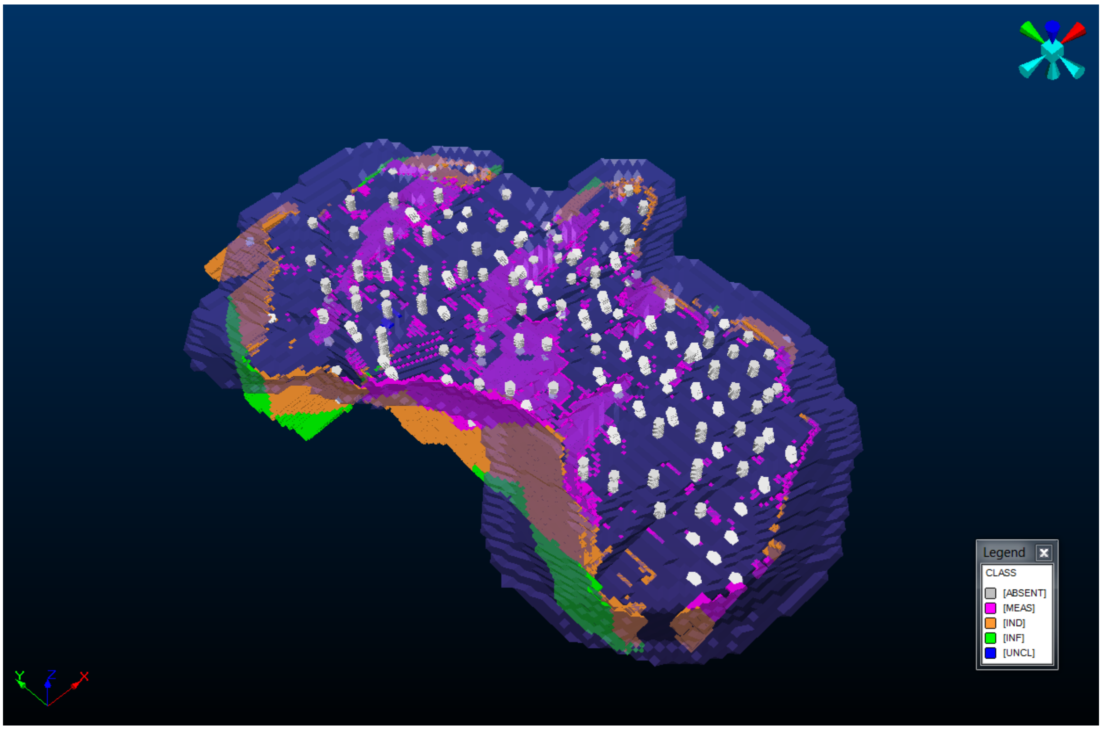


Figure 13-59: Underground Mineral Resource blocks at Kuervitikko (Source: SRK Oct 2012)

13.12 Grade Tonnage Curves

13.12.1 Open Pit

Hannukainen and Kuervitikko grade-tonnage curves for the mineralisation domains within the Whittle shell are shown in Figure 13-60 and Figure 13-61 for %Fe Total. The curve shows the relationship between the modelled tonnage and grade at increasing %Fe Total cut-offs.

The Hannukainen grade-tonnage curve, Figure 13-60, shows three stages. The first is the low-grade population with steadily decreasing tonnage with increasing grade. The second shows the curve shallowing, representing the lack of tonnage between 25-40% Fe Total. The last is the high grade population, with a steepening of the curves representing higher loss of tonnage with increasing grade.

The Kuervitikko grade-tonnage curve, Figure 13-61, shows a smoother curve, with a single rise and fall indicating one continuous population. This is due to the lack of internal high-grade Fe domaining at Kuervitikko.

Confidence can be taken when viewing the grade-tonnage curves with the actual prediction at elevated cut-offs due to the high level of confidence in the variograms with the ranges being longer than the average drillhole spacing.

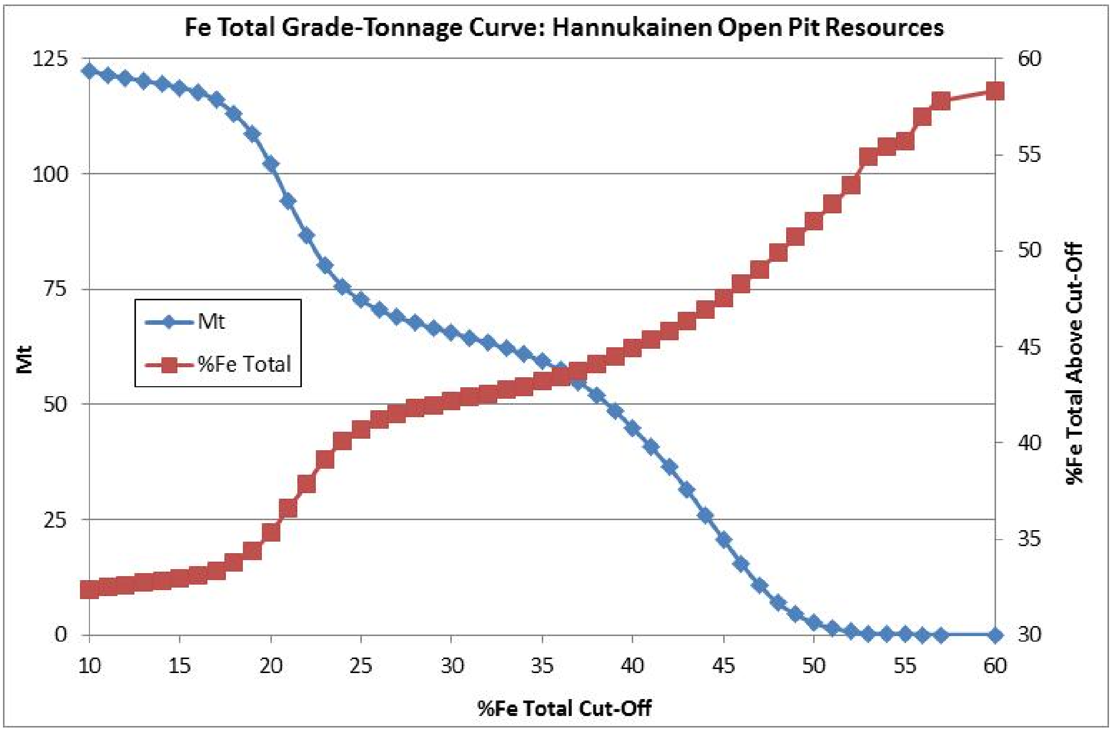


Figure 13-60: %Fe Total Grade-Tonnage Curve for Hannukainen open pit Mineral Resources

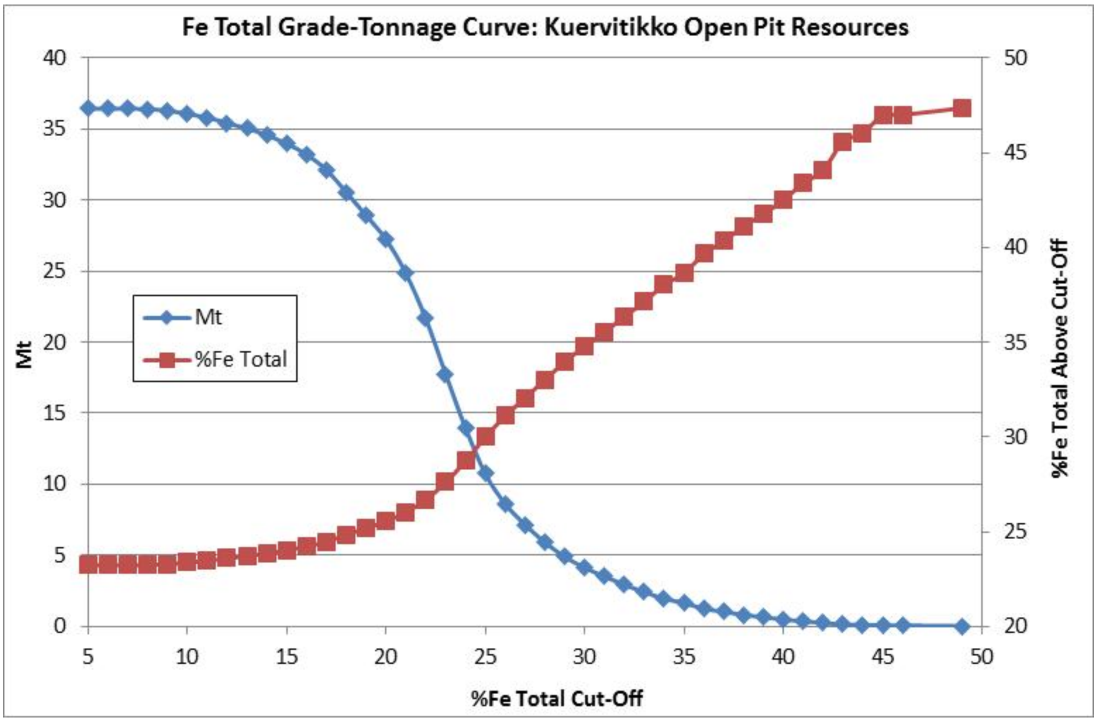


Figure 13-61: %Fe Total Grade-Tonnage Curve for Kuervitikko open pit Mineral Resources

13.12.2 Underground

The grade-tonnages curves for the underground Mineral Resources are shown in Figure 13-62 and Figure 13-63. The same structures as the open pit Mineral Resources are seen in the underground Mineral Resources, with a stepped curve for Hannukainen, and a smoothed curve for Kuervitikko.

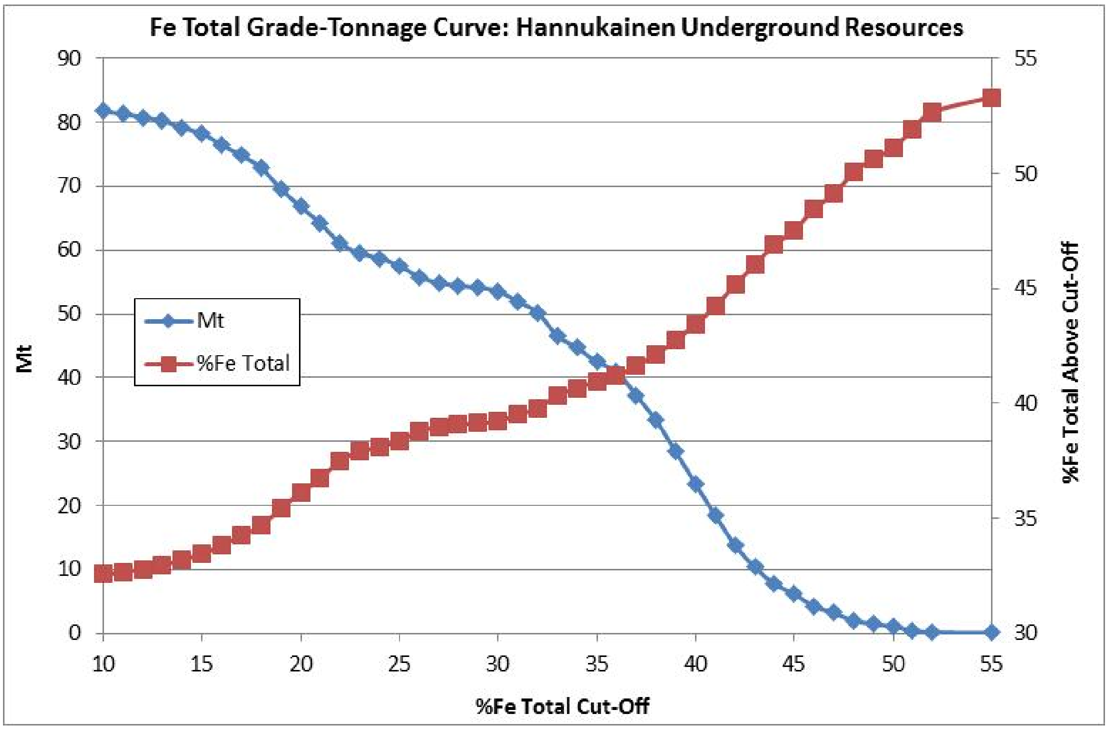


Figure 13-62: %Fe Total Grade-Tonnage Curve for Hannukainen underground Mineral Resources

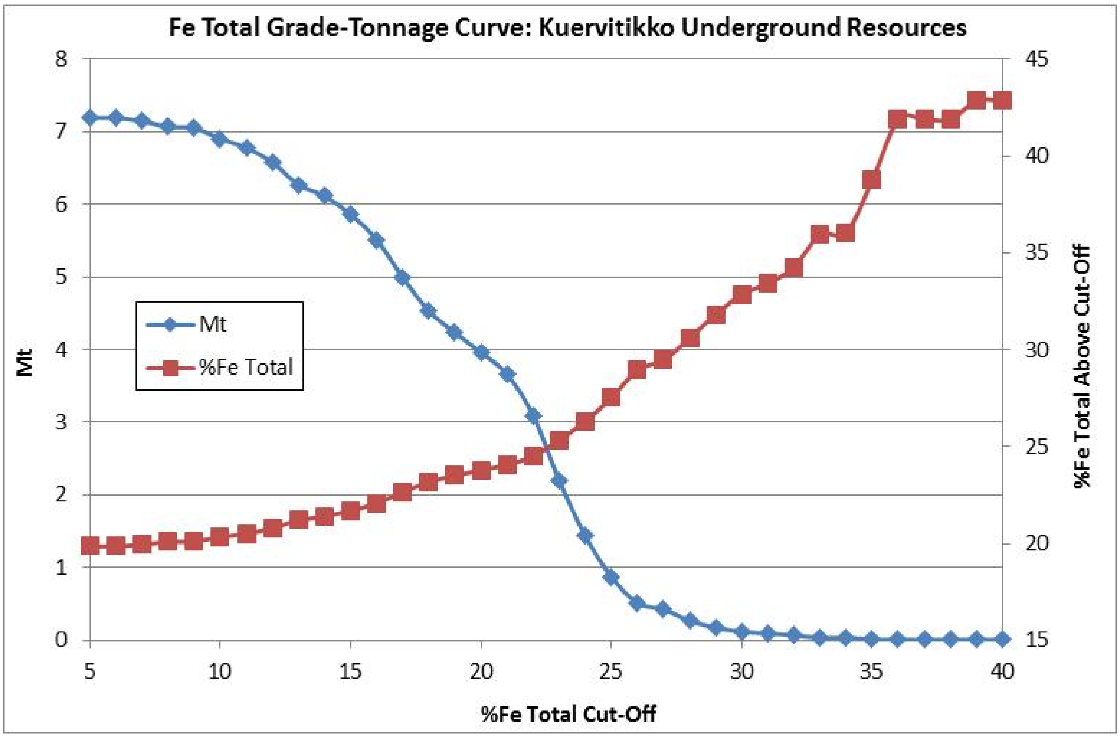


Figure 13-63: %Fe Total Grade-Tonnage Curve for Kuervitikko underground Mineral Resources

13.13 Exploration Potential

The majority of the modelled deposit is included in the Mineral Resource statement due to the inclusion of open pit and underground Mineral Resources. SRK has not assigned any mineralisation to an unclassified category due to the density of drilling and quality of the estimation.

The majority of the underground Mineral Resources, especially in the Kivivuopio area, have been classified as Inferred due to the sparse drilling at depth. Additional drilling was completed in areas suggested by SRK in 2011, which included infill drilling at Kivivuopio and deeper drilling down-plunge towards the southeast. These holes confirmed the continuation of mineralisation in areas of sparse data, but were not drilled on a sufficiently tight grid to allow for large resource category upgrading. Additional drilling to create a <100 x 100 m grid would likely allow for resource upgrading, although this is not guaranteed.

In addition to the Mineral Resource Estimate completed for the mineralisation domains, sporadic Cu and Au mineralisation is found in the hangingwall and footwall zones adjacent to the domained mineralisation. It is not considered of economic interest due to the dispersed nature of the mineralisation.

13.14 Comparison to 2010 WGM Estimate

WGM completed a Mineral Resource Estimate for the Project in June 2010.

Table 13-18 shows the 2010 WGM Mineral Resource statement. The Hannukainen statement is reported within a Whittle shell generated by WGM (in addition to some Inferred Mineral Resources reported below the Whittle shell), whereas the Kuervitikko statement is reported above a 15% Fe Total cut-off only.

Table 13-18: WGM 2010 Mineral Resource statement

HANNUKAINEN (reported within a Whittle shell)				
CLASS	MT	FE(%)	CU(%)	AU(g/t)
MEASURED	101	33.80	0.17	0.067
INDICATED	9	35.00	0.13	0.023
MEAS+IND	110	33.90	0.17	0.064
INFERRED (above Whittle Shell)	1	31.30	0.09	0.020
INFERRED (below Whittle Shell)	88	31.70	0.13	0.041
KUERVITIKKO (reported to a 15% Fe Total cut-off)				
CLASS	MT	FE(%)	CU(%)	AU(g/t)
MEASURED	-	-	-	-
INDICATED	26	23.80	0.17	0.175
MEAS+IND	26	23.80	0.17	0.175
INFERRED	19	21.70	0.15	0.165
TOTAL				
CLASS	MT	FE(%)	CU(%)	AU(g/t)
MEASURED	101	33.80	0.17	0.067
INDICATED	35	26.68	0.16	0.136
MEAS+IND	136	31.97	0.17	0.085
INFERRED	20	21.95	0.15	0.161

Note: the 2010 WGM Resource Estimate was NI43-101 compliant

It is difficult to directly compare the two Mineral Resource statements due to the differing Resource reporting techniques used. SRK has used a combination of a Whittle shell (for both deposits) and an underground cut-off Fe equivalent grade, whereas WGM used a Whittle shell for Hannukainen, and a Fe cut-off for Kuervitikko.

The SRK open pit Mineral Resource for Hannukainen can be compared to the WGM Hannukainen Mineral Resource, which are both constrained to Whittle shells. The minimal differences are due to the differing domaining methods used by SRK and WGM. This resulted in differing wireframe geometry and estimation populations.

14 MINERAL RESERVE ESTIMATES

14.1 Mineral Reserve Statement

The HFS reports an open pit Mineral Reserve for the Project which is based on the Mineral Resource Estimate described herein. As is the case with the Mineral Resource, this has been reported using the CIM Code and has an effective date of 25 November, 2013.

Specifically, SRK derived an optimised pit for the Project based on the material reported as Measured and Indicated Mineral Resources. The operating costs assumed for the optimisation were agreed between SRK and Northland while the metallurgical recoveries were provided by external consultants and reviewed by SRK as part of the review process. SRK conducted all geotechnical studies, water management studies and Waste Rock Dump design studies to enable the mining study to be completed. All technical reports completed by SRK can be found in Appendix 4 of the HFS with the key findings summarised herein.

Two separate designed pits were developed from the optimised pits, Hannukainen and Kuervitikko.

Table 14-1 presents the Mineral Reserve Statement derived following this process.

These tonnages have been based on a long term price forecast of USD1.25/dmtu for Fe with modifying factors of ore loss and dilution being built in to the regularized block model. The average mining recovery and dilution for the regularised block model are 97.7% and 6.5%, respectively.

Table 14-1: Mineral Reserve Estimate

	Quantity Mt	Fe %	Cu %	Au g/t	S %
Hannukainen					
Proven	91.8	32.2	0.186	0.088	2.4
Probable	0.8	32.6	0.148	0.060	2.4
Kuervitikko					
Proven	21.9	23.6	0.183	0.216	2.5
Probable	0.3	23.8	0.177	0.194	2.5
Total					
Proven	113.7	30.5	0.185	0.112	2.4
Probable	1.1	30.0	0.157	0.100	2.4
TOTAL	**114.8**	**30.5**	**0.185**	**0.112**	**2.4**

SRK has reported a Proven Mineral Reserve of 113.7 Mt grading 30.5% Fe, 0.185% Cu, 0.112 g/t Au and 2.4% S and a probable Mineral Reserve of 1.1 Mt grading 30.0% Fe, 0.157% Cu, 0.1 g/t Au and 2.4% S for a total of 114.8 Mt grading at 30.5% Fe, 0.185% Cu, 0.112 g/t Au and 2.4% S. Confidence in the modifying factors has resulted in classifying all Measured material as a Proven Ore Reserve and all Indicated material as a Probable Ore Reserve.

The Qualified Person with overall responsibility for the reporting of Mineral Reserves is Ms Colleen MacDougall, BEng, MAusIMM(CP), who is a Senior Consultant (Mining Engineering) with SRK. All work has been reviewed by Mr Rick Skelton, CEng, MSc (Mining), MIMMM, MSAIMM, who is an employee of SRK. Rick Skelton is a mining engineer with over 30 years' experience in the mining industry and has been involved in the review and reporting of Mineral Reserves on various iron ore properties in Europe, Africa and South America during the past five years.

SRK is confident that sufficient geological work has been undertaken, and sufficient geological understanding gained, to enable the construction of an orebody model suitable for the derivation of Mineral Resource and Mineral Reserve estimates. SRK considers that both the modelling and the grade interpolation have been carried out in an unbiased manner and that the resulting grade and tonnage estimates should be reliable within the context of the classification applied. In addition, SRK is not aware of any metallurgical, infrastructural, environmental, legal, title, taxation, socio-economic or marketing issues that would impact on the Mineral Resource or Mineral Reserve statements as presented.

14.1.1 Block Model Regularization

The regularization of the block model has been carried out on the Hannukainen Mineral Resource model. This Mineral Resource model is sub-celled to reflect the geometry of the geological boundaries. The parent block size is 25 mX x 25 mY x 5 mZ.

The SMU size for the Project has been decided based the following criteria:

- multiple of the parent block size of the Mineral Resource block model;
- minimum X and Y dimensions reflect the bucket width of the loading units;
- bench height; and
- minimise resultant diluting factors.

A SMU size of 6.25 mX x 6.25 mY x 5 mZ was deemed appropriate for the Project.

The process of regularising the block model incorporates recovery and dilution into the model. Within a regularised model, blocks can be categorised as ore or waste given a CoG.

The waste blocks have been assigned 0% grades for all elements. This conservative approach has been used to prevent any erroneous introduction of payable metal into the block model.

The CoG is variable due to processing regressions; therefore, an average CoG of 13% Fe has been applied to the regularised and Mineral Resource models to determine the ore loss and mining dilution resulting from regularisation.

The average mining recovery and dilution for the regularised block model are 97.7% and 6.5%, respectively with zero dilutant for all grades.

15 MINING METHODS

15.1 Introduction

The Project, when constructed, will comprise two conventional open pit mines. The following section provides a summary of the relevant information used to establish the amenability of the mineral reserve to the proposed mining method.

15.2 Pit Optimisation

The objective of the pit optimisation has been to select the optimal pit shell to take forward to detailed pit design. A sensitivity analysis has been conducted to identify the key components that affect the value of the Project.

For the purposes of this study, SRK has considered only measured and indicated classified material in the optimisation which forms the basis for estimating Ore Reserves. The classification has been carried through to the mining schedule and the economic estimate. The pit optimisation results form the basis of the mine design and schedule.

A summary of the parameters used in the pit optimisation is shown in Table 15-1.

Table 15-1: Optimisation Parameters

	Units	Value
Overall Slope Angle		
Overburden	°	27
Hannukainen North Host Rock	°	47
Hannukainen Central & South Host Rock	°	45
Kuervitikko Host Rock	°	48
Mining and Processing		
Mining Recovery	%	100
Mining Dilution	%	0
Fe Processing Recovery	$98.5 \times (1 - e^{-0.06038\times(Fe-6)} \times (1.962744 \times \frac{S}{Fe} + 1)$	
Cu Processing Recovery	$15.2487213896 \times \ln Cu + 106.8504693385$	
Au Processing Recovery	$5.8861814790 \times \ln Au + 38.9178069302$	
Costs		
Base Mining Cost	USD/t	1.78
Incremental Vertical Cost	USD/t/5 m	0.02
Processing Cost	USD/t processed	6.78
Transport Cost Fe concentrate	USD/t Fe conc.	19.22
Transport Cost Cu concentrate	USD/t Cu conc.	23.24
Industrial Area	USD/t processed	0.68
General & Administrative Cost	USD/t processed	1.33
Royalty	% of selling price	0.15
Cu Selling Cost	USD/lb Cu conc.	0.27
Price		
Fe Selling Price	USD/dmtu	1.40
Cu Selling Price	USD/lb	2.86
Au Selling Price	USD/t.oz	1,250

15.2.1 Optimisation Results

Results

The optimisation process produces a series of nested pit shells based on various selling prices expressed as revenue factors. The revenue factors have been applied to each element selling price. The cash flow for each shell is calculated using the input selling prices and provides an indication of the economic changes for the various pit shells.

The Whittle results for cash flow variation for an iron ore price between USD0.40/dmtu and USD2.88/dmtu with pit size are shown in Figure 15-1.

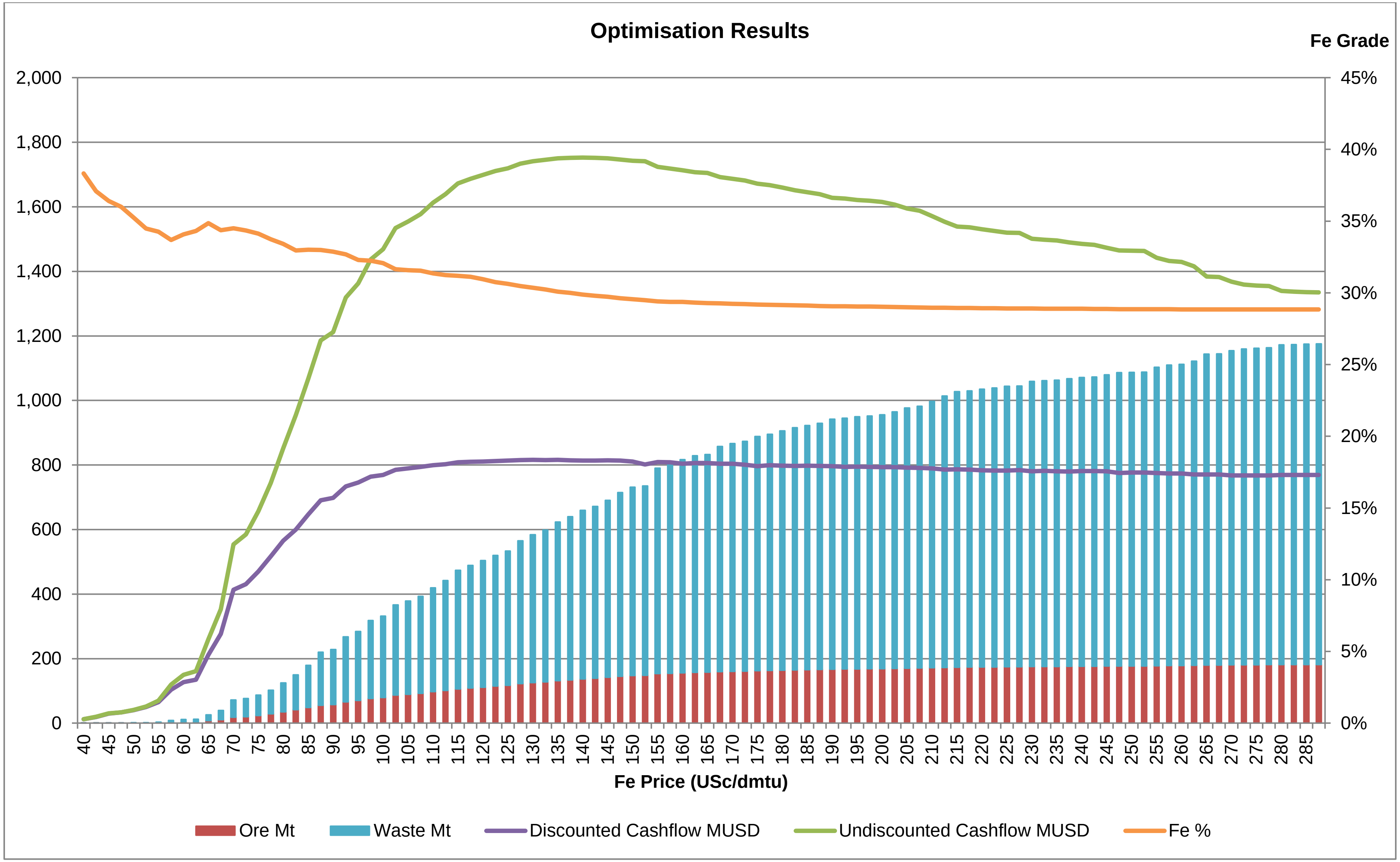


Figure 15-1: Optimisation Cashflow Results

Sensitivity Analysis

The metal price sensitivity of the optimisation is shown in Figure 15-2.

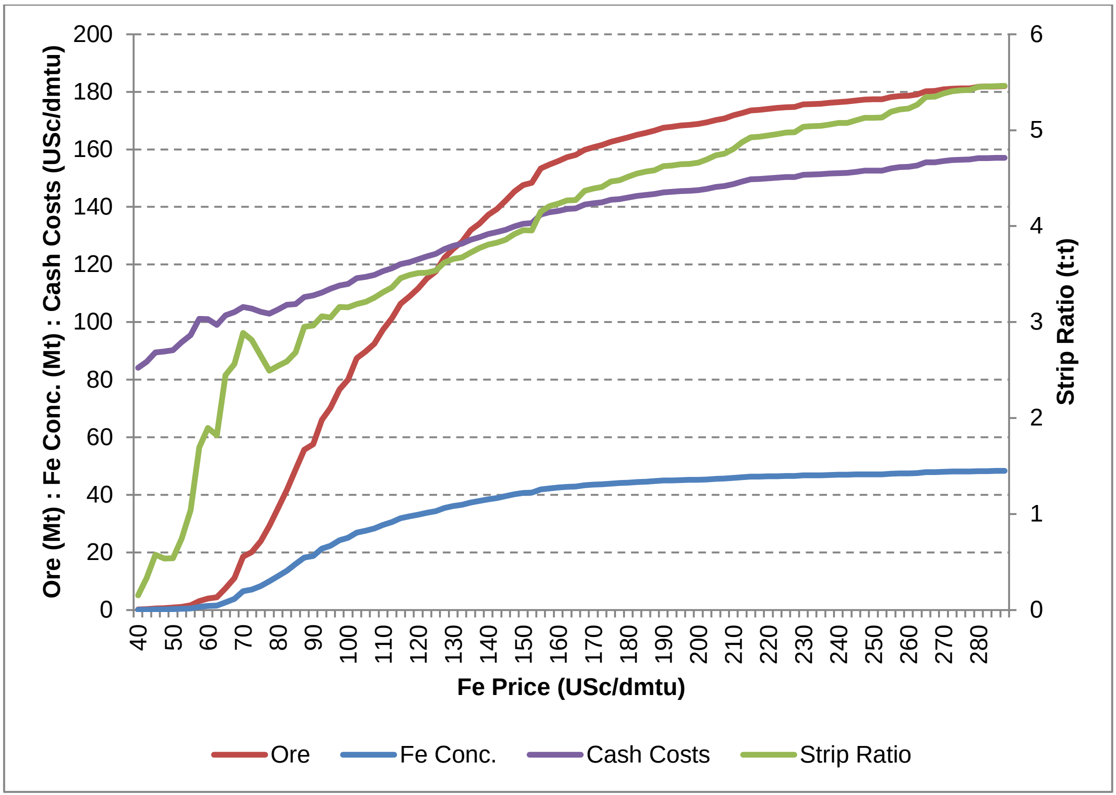


Figure 15-2: Metal Price Sensitivity Analysis

The sensitivity to slope angle, mining cost, processing cost, transport cost, selling cost, Fe processing recovery and selling prices is shown in Table 15-2. The sensitivity analysis shows that the optimisation is most sensitive to changes in Fe selling price followed by the Fe processing recovery as shown in Figure 15-3.

Table 15-2: Pit Optimisation Sensitivity

	Units	-40%	-30%	-20%	-10%	0%	+10%	+20%	+30%	+40%
Slope Angle Sensitivity										
Change in Slope Angle	°	-8	-6	-4	-2	0	+2	+4	+6	+8
Ore	Mt	133	134	134	136	137	137	139	138	139
Waste	Mt	541	530	520	521	523	509	507	495	496
Undisc. Cashflow	mUSD	1,632	1,668	1,702	1,728	1,752	1,783	1,815	1,832	1,851
Mining Cost Sensitivity										
Base Mining Cost	USD/t	1.07	1.25	1.42	1.60	1.78	1.96	2.14	2.31	2.49
Ore	Mt	161	157	152	144	137	131	126	119	117
Waste	Mt	727	687	643	565	523	481	451	411	401
Undisc. Cashflow	mUSD	2,305	2,148	2,009	1,875	1,752	1,638	1,532	1,438	1,343
Processing Cost Sensitivity										
Processing Cost	USD/t ore	4.07	4.75	5.42	6.10	6.78	7.46	8.14	8.81	9.49
Ore	Mt	159	154	149	144	137	131	125	119	129
Waste	Mt	585	571	555	541	523	504	484	466	531
Undisc. Cashflow	mUSD	2,156	2,049	1,947	1,848	1,752	1,662	1,574	1,493	1,391
Transport Cost Sensitivity										
Fe Concentrate	USD/t conc.	11.53	13.45	15.38	17.30	19.22	21.14	23.06	24.99	26.91
Ore	Mt	147	145	142	139	137	134	132	129	127
Waste	Mt	572	558	547	531	523	508	499	480	470
Undisc. Cashflow	mUSD	2,056	1,979	1,902	1,827	1,752	1,679	1,607	1,535	1,465
Cu Concentrate	USD/t conc.	13.94	16.27	18.59	20.92	23.24	25.56	27.89	30.21	32.54
Ore	Mt	137	137	137	137	137	137	137	137	137
Waste	Mt	523	523	523	523	523	523	522	522	521
Undisc. Cashflow	MUSD	1,760	1,758	1,756	1,754	1,752	1,751	1,749	1,747	1,745
Selling Cost Sensitivity										
Cu Concentrate	USD/t conc.	0.16	0.19	0.22	0.24	0.27	0.30	0.32	0.35	0.38
Ore	Mt	139	139	138	137	137	137	136	135	134
Waste	Mt	527	526	524	523	523	521	512	510	508
Undisc. Cashflow	mUSD	1,803	1,789	1,776	1,766	1,752	1,739	1,730	1,716	1,702
Processing Recovery Sensitivity										
Fe Recovery	% of Rec.	-40%	-30%	-20%	-10%	0%	10%	20%	30%	40%
Ore	Mt	88	101	115	127	137	147	156	160	164
Waste	Mt	305	358	420	470	523	572	637	667	684
Undisc. Cashflow	mUSD	718	940	1,192	1,464	1,752	2,057	2,373	2,702	3,037
Selling Prices Sensitivity										
Fe Selling Price	USc/dmtu	85.0	97.5	112.5	125.0	140.0	155.0	167.5	182.5	195.0
Ore	Mt	68	91	110	124	137	148	157	163	169
Waste	Mt	228	316	398	460	523	578	643	681	727
Undisc. Cashflow	mUSD	529	771	1,064	1,395	1,752	2,134	2,536	2,952	3,377
Cu Selling Price	USD/lb	1.74	1.99	2.30	2.55	2.86	3.17	3.42	3.73	3.98
Ore	Mt	113	122	126	132	137	141	148	151	158
Waste	Mt	422	461	475	498	523	535	564	579	630
Undisc. Cashflow	mUSD	1,263	1,376	1,499	1,622	1,752	1,887	2,020	2,162	2,303
Au Selling Price	USD/t.oz	759	871	1,004	1,116	1,250	1,384	1,496	1,629	1,741
Ore	Mt	133	135	136	136	137	138	138	138	139
Waste	Mt	503	514	517	519	523	524	524	526	527
Undisc. Cashflow	mUSD	1,679	1,697	1,715	1,734	1,752	1,771	1,771	1,809	1,828

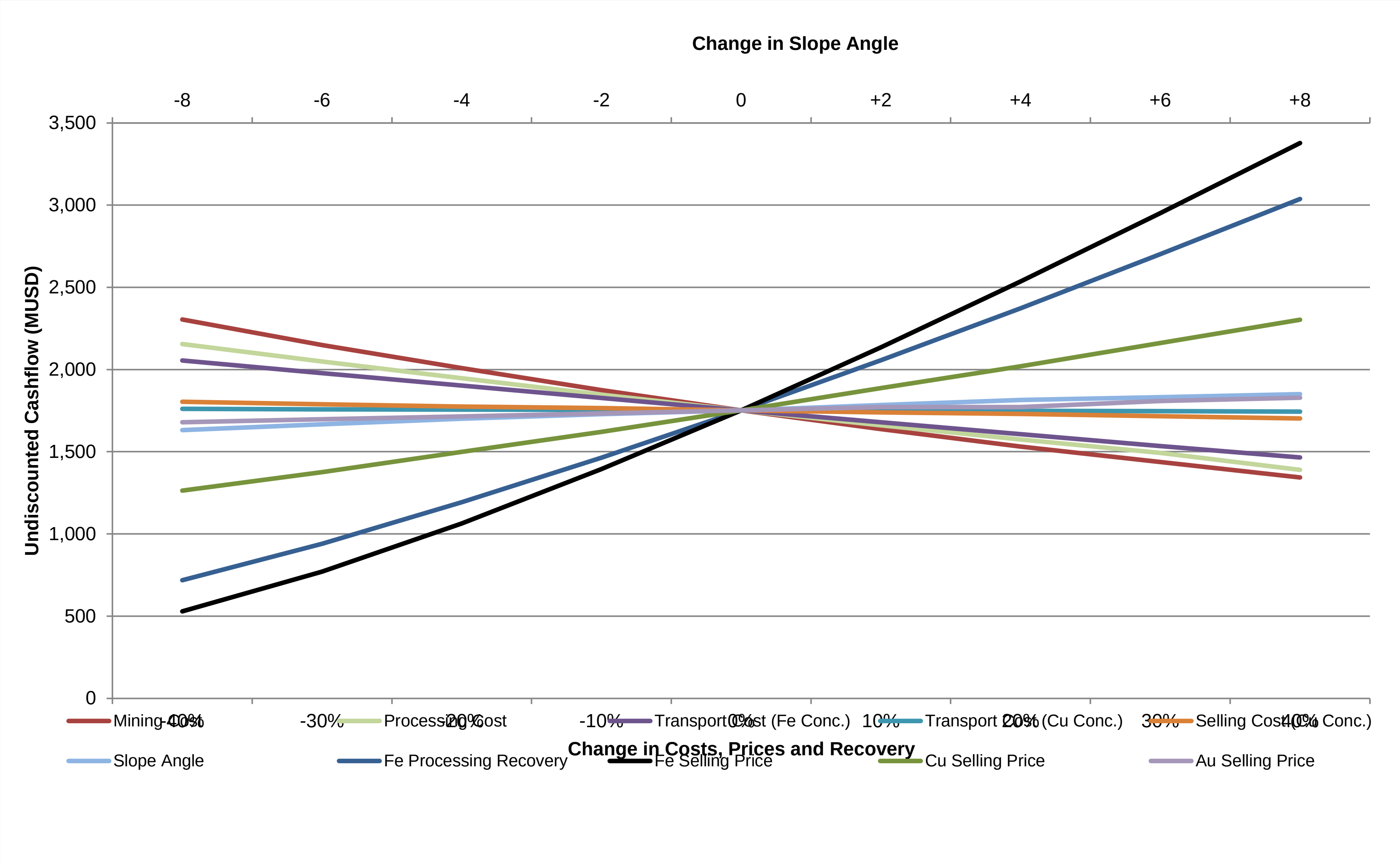


Figure 15-3: Pit Optimisation Sensitivity

15.2.2 Pit Shell Selection

Selection Criteria

The following criteria have been used for the pit shell selection for the Project:

- provide sufficient ore for a 19 year LoM at 6.5 Mtpa; and
- maximise undiscounted cashflow.

Selected Pit Shell

Table 15-3 shows that pit shell 39 provides the highest discounted cashflow at a 10% discount rate. Taking into account approximately 5% ore loss in the pit engineering process, 17 years of ore production at 6.5 Mtpa would require at least 116 Mt of ore. This would indicate the minimum shell to be selected is pit shell number 35. The difference between pit shell 35 and 39 in cashflow terms is USD2.4M (0.3%) and between ore tonnages is 14.3 Mt (10.8%).

Table 15-3: Excerpt from Whittle Optimisation Discounted Results

Revenue Factor	Price Fe USD/dmtu	Processed Ore Mt	Processed Fe %	Processed Cu %	Processed Au g/t	Strip Ratio t:t	Discounted Cashflow MUSD	Cash Costs USD/dmtu
0.71	1.000	80.0	32.1	0.204	0.128	3.2	769.5	113.3
0.73	1.025	87.5	31.6	0.200	0.127	3.2	785.1	115.3
0.75	1.050	89.8	31.6	0.199	0.126	3.2	789.4	115.8
0.77	1.075	92.5	31.5	0.197	0.124	3.3	793.7	116.4
0.79	1.100	97.4	31.4	0.195	0.123	3.3	799.4	117.7
0.80	1.125	101.4	31.2	0.193	0.122	3.4	802.4	118.8
0.82	1.150	106.4	31.2	0.191	0.119	3.5	808.2	120.1
0.84	1.175	108.9	31.1	0.190	0.118	3.5	809.7	120.8
0.86	1.200	111.7	30.9	0.189	0.118	3.5	810.6	121.8
0.88	1.225	115.1	30.7	0.188	0.117	3.5	812.2	122.8
0.89	1.250	117.6	30.6	0.187	0.116	3.5	813.7	123.7
0.91	1.275	122.4	30.5	0.186	0.115	3.6	815.4	125.4
0.93	1.300	125.4	30.4	0.186	0.114	3.7	816.0	126.4
0.95	1.325	128.0	30.2	0.185	0.113	3.7	815.5	127.2
0.96	1.350	131.9	30.1	0.184	0.113	3.7	816.1	128.6
0.98	1.375	134.2	30.0	0.183	0.113	3.8	814.6	129.5
1.00	1.400	137.3	29.9	0.182	0.112	3.8	813.7	130.7
1.02	1.425	139.2	29.8	0.182	0.112	3.8	814.0	131.3
1.04	1.450	142.2	29.7	0.180	0.110	3.9	814.4	132.1
1.05	1.475	145.4	29.6	0.179	0.110	3.9	813.7	133.3
1.07	1.500	147.6	29.6	0.179	0.110	4.0	811.0	134.1
1.09	1.525	148.4	29.5	0.178	0.110	4.0	801.6	134.4
1.11	1.550	153.3	29.4	0.179	0.111	4.2	809.3	137.3
1.13	1.575	154.7	29.4	0.179	0.113	4.2	808.4	138.2
1.14	1.600	155.9	29.4	0.179	0.112	4.2	804.0	138.6

SRK has selected pit shell 35 (revenue factor 0.89) to use as the basis for the pit designs.

This provides a minimum of 17 years of ore production without significant negative impact on the undiscounted cashflow. This also reduces the risk to the Project by selecting a lower revenue factor with an iron ore price of USD1.25/dmtu. Figure 15-4 displays the selected pit shell 35.

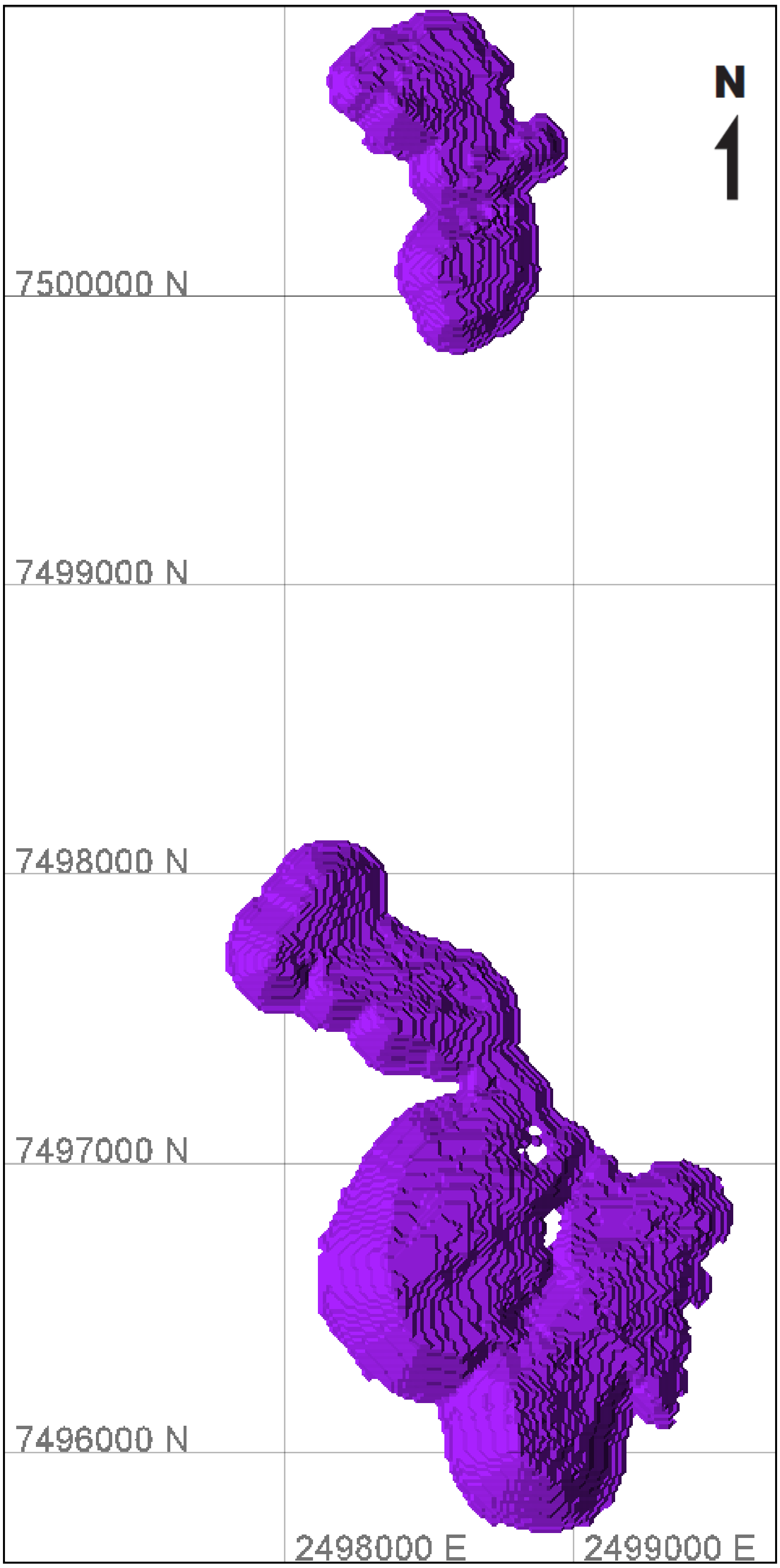


Figure 15-4: Whittle Pit Shell 35

15.3 Pit Design

The final and cutback designs have been completed to verify the engineering feasibility of the optimised pit shells. The objectives of the design work are to:

- provide practical engineered pits and cutbacks for mining;
- incorporate geotechnical design parameters;
- provide a design which can be used as a basis for geotechnical assessment;
- honour equipment operating limitations;
- honour the interface between the cutbacks; and
- provide a basis for detailed mine scheduling.

15.3.1 Design Parameters

Cut-Off Grade

In order to establish an Fe CoG for the Project, Cu and Au were converted to an Fe Equivalent Grade ("FeEq"). The FeEq calculation is shown below:

$$FeEq = \frac{\left((FeRec \times Fe \times (FePr \times (1 - Roy) - FeTr)\right) + \left(CuRec \times Cu \times (CuPr \times (1 - Roy) - CuTr - CuSC)\right) + \left(AuRec \times Au \times AuPr \times (1 - Roy)\right)\Big)}{((FePr \times (1 - Roy) - FeTr) \times FeRec)}$$

where:

FeRec = Fe Recovery (%)
Fe = Fe Grade (%)
FePr = Fe Selling Price (USD/t)
Roy = Royalty (%)
FeTr = Fe Transport Cost (USD/t)
CuRec = Cu Recovery (%)
Cu = Cu Grade (%)
CuPr = Cu Selling Price (USD/t)
CuTr = Cu Transport Cost (USD/t)
CuSC = Cu Selling Cost (USD/t)
AuRec = Au Recovery (%)
Au = Au Grade (%)
AuPr = Au Selling Price (USD/t)

The marginal and break-even CoG are typically used to differentiate between waste, ore and low grade ore. However, as directed by Northland, no long term stockpiling can occur due to the decrease in ore recovery when the ore is left exposed for significant periods of time, therefore only the marginal CoG has been applied for this Project.

Due to the variable nature of the Fe processing recovery, the CoG varies for each block. The CoG and FeEq have been calculated for each block in the model, and if the FeEq is equal to or greater than the CoG it has been classified as ore.

Geotechnical Parameters

Table 15-4 lists the design parameters used to create the pit designs for Hannukainen and Kuervitikko. A cross section schematic of the geotechnical parameters in the different material types is represented in Figure 15-5.

Table 15-4: Geotechnical Parameters

Parameter	Units	Hannukainen Fresh Material	Kuervitikko Fresh Material	Weathered Material
Inter-Ramp Slope Angle	°	53	55	27
Face Angle	°	70	70	35
Berm Width	m	8	10	5
Bench Height	m	20	30	10

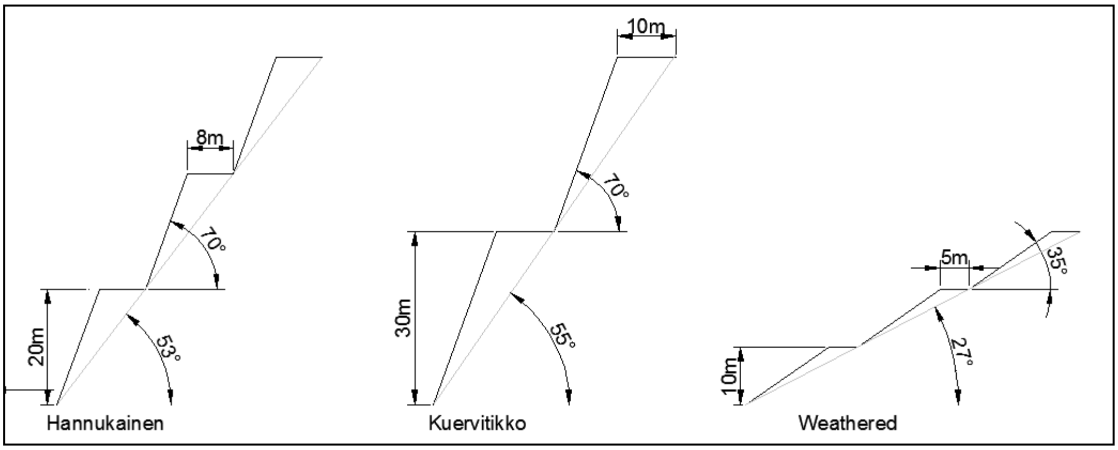


Figure 15-5: Geotechnical Parameters Cross Section

Design Parameters

The design parameters shown in Table 15-5 were used for the pit designs.

Table 15-5: Design Parameters

Parameter	Units	Value
Ramp Width – Dual Lane	m	27
Ramp Width – Single Lane	m	19
Ramp Gradient	%	10
Turning Circle	m	30
Minimum Mining Width	m	50

A Caterpillar 789 truck (181 t capacity) has been used to estimate the ramp width, gradient and turning circle. The dual lane ramp width has been determined as 3.5 times the width of the overall haul truck width. The final benches in the pit designs have been designed using single lane access, at 2.5 times the truck width, which allows the retrieval of extra ore at the base of the pits.

During the production scheduling of the Hannukainen Project, it was determined that larger trucks (227 t) would be more economical. The ramp width was not changed in the pit designs, resulting in dual lane ramps 3.25 times the width of a Caterpillar 793 truck, which is within acceptable design standards.

Cutback Parameters

CAE's NPV Scheduler ("NPVS") has been used to split the selected pit shells into practical mining cutbacks with the following criteria being used:

- pit value (cashflow assessment of pit shell);
- minimum mining widths;
- contained ore; and
- practical staging of the cutbacks.

A minimum distance of 100 m between cutbacks has been considered to allow for multiple bench mining.

The mining tonnages resulting from the sequencing exercise in NPVS are shown in Table 15-6. These tonnages are based on the optimised selected pit shell.

Table 15-6: NPVS Proposed Cutback Quantities

	Total Mt	Waste Mt	Ore Mt	Fe %	Cu %	Au g/t	Strip Ratio t:t
Hannukainen North							
HN1	36.1	27.2	8.9	32.9%	0.124%	0.029	3.1
HN2	52.6	44.1	8.4	33.0%	0.168%	0.100	5.2
Hannukainen Central							
HC1	19.7	13.9	5.8	32.8%	0.336%	0.191	2.4
HC2	24.2	19.1	5.1	35.6%	0.296%	0.148	3.8
HC3	37.9	30.2	7.7	34.0%	0.296%	0.186	3.9
HC4	55.0	45.5	9.5	32.9%	0.274%	0.166	4.8
HC5	58.2	51.8	6.5	33.3%	0.310%	0.185	8.0
Hannukainen South							
HS1	21.8	10.3	11.5	30.3%	0.100%	0.023	0.9
HS2	32.8	24.0	8.9	29.9%	0.121%	0.025	2.7
HS3	53.0	41.5	11.5	32.7%	0.106%	0.034	3.6
HS4	54.5	44.8	9.7	32.6%	0.124%	0.032	4.6
Kuervitikko							
KU1	26.4	17.7	8.7	23.9%	0.214%	0.217	2.0
KU2[1]	58.0	42.8	15.3	23.0%	0.173%	0.222	2.8
Total	**530.3**	**412.8**	**117.5**	**29.5%**	**0.178%**	**0.102**	**3.5**

[1] *KU2 has been split into two pits during the design process*

Constraints

No topographical, environmental or other constraints have been considered when designing the pit designs.

15.3.2 Pit Designs

Cutback Staging

The staging of the cutbacks follows the mining cutbacks established in NPVS, although changes have been made to allow for minimum mining widths and practical interaction between the cutbacks.

The Hannukainen deposit has three mining areas: HN, HC and HS. There are five pits in the central area (HC1, HC2, HC3, HC4, and HC5), four pits in the southern area (HS2, HS2, HS3, and HS4), and two in the northern area (HN1 and HN2), as shown in Figure 15-6.

The Kuervitikko deposit has one initial pit (KU1) and then one cutback in the north (KU2) and one in the south (KU3), as shown in Figure 15-7.

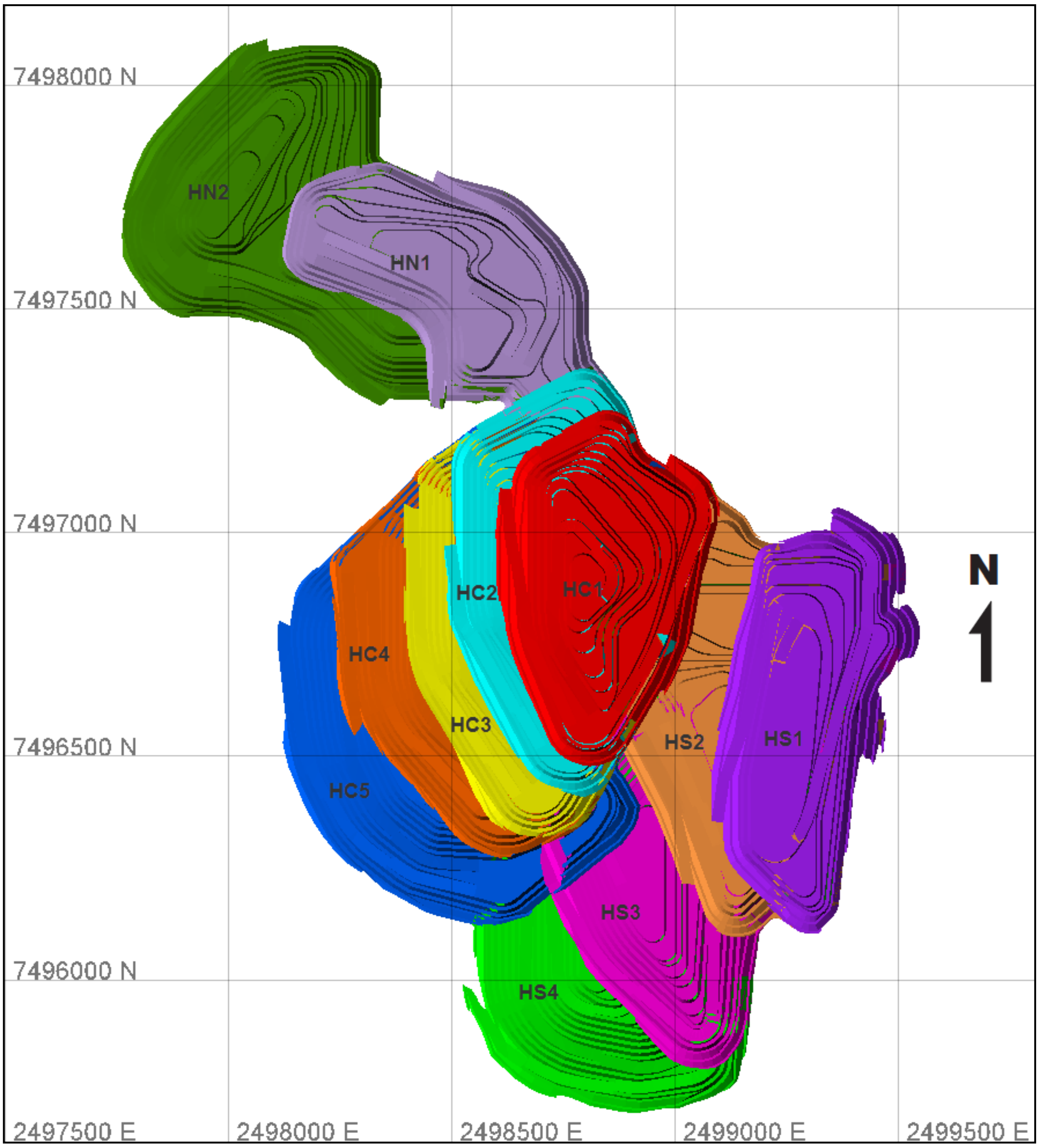


Figure 15-6: Hannukainen Cutback Staging

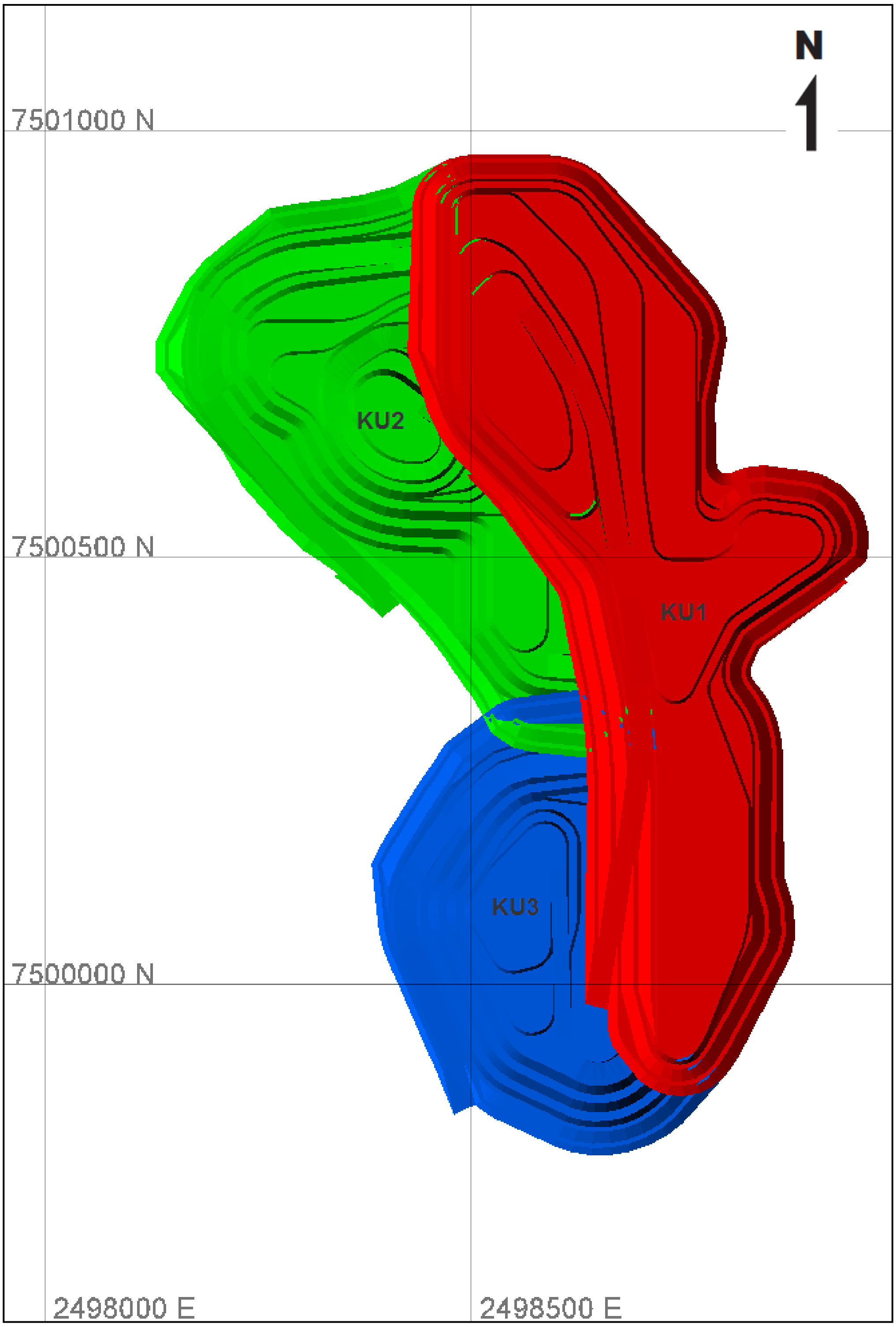


Figure 15-7: Kuervitikko Cutback Staging

Mineable Tonnage

The mineable tonnages for the pit designs are shown in Table 15-7 and the Mineral Reserves are listed in Table 15-8. The tonnages and grades are inclusive of modifying factors and have been evaluated using the regularised block model with a variable CoG. Confidence in the modifying factors has resulted in classifying all Measured material as a Proven Ore Reserve and all Indicated material as a Probable Ore Reserve.

Table 15-7: Mineable Tonnage

Pit / Cut Back	Total Mt	Total Ore Mt	Measured Mt	Indicated Mt	Total Waste Mt	OVB Mt	PAF Mt	NAF Mt	Strip Ratio t:t	Fe %	Cu %	Au g/t	S %
Hannukainen North													
HN1	28.2	5.7	5.7	0.0	22.5	8.4	7.1	7.0	4.0	33.4	0.119	0.026	1.9
HN2	63.1	9.0	8.7	0.3	54.1	5.8	9.6	38.7	6.0	33.2	0.157	0.080	2.3
Hannukainen Central													
HC1	26.7	6.2	6.1	0.1	20.5	5.8	9.2	5.5	3.3	31.9	0.328	0.188	2.1
HC2	32.2	7.1	7.1	0.0	25.1	4.6	11.3	9.2	3.5	34.3	0.277	0.155	2.1
HC3	38.1	6.5	6.5	0.0	31.6	2.6	16.2	12.8	4.8	34.2	0.302	0.182	2.1
HC4	40.2	6.2	6.2	0.0	34.1	1.4	16.1	16.6	5.5	34.4	0.293	0.177	2.1
HC5	76.4	8.7	8.7	0.0	67.7	3.2	26.3	38.1	7.8	32.5	0.285	0.168	2.0
Hannukainen South													
HS1	27.9	13.4	13.4	0.0	14.5	6.8	6.1	1.6	1.1	29.3	0.099	0.023	3.1
HS2	32.8	9.4	9.4	0.0	23.4	5.3	11.6	6.5	2.5	29.3	0.125	0.026	3.1
HS3	51.8	10.9	10.9	0.0	40.9	5.7	23.7	11.5	3.8	32.4	0.106	0.034	2.3
HS4	58.8	9.5	9.1	0.4	49.4	3.5	30.0	15.8	5.2	32.4	0.123	0.032	2.5
Kuervitikko													
KU1	25.0	6.6	6.5	0.1	18.4	12.1	4.7	1.6	2.8	25.2	0.201	0.208	2.6
KU2	34.4	8.6	8.4	0.2	25.8	5.0	11.5	9.4	3.0	23.8	0.170	0.174	2.5
KU3	26.0	7.1	7.1	0.0	18.9	4.8	7.1	7.0	2.7	21.9	0.182	0.274	2.3
Total	**561.6**	**114.8**	**113.7**	**1.1**	**446.8**	**74.8**	**190.5**	**181.5**	**3.9**	**30.5**	**0.185**	**0.112**	**2.4**

Table 15-8: Mineral Reserves

	Quantity	Fe	Cu	Au	S
	Mt	**%**	**%**	**g/t**	**%**
Hannukainen					
Proven	91.8	32.2	0.186	0.088	2.4
Probable	0.8	32.6	0.148	0.060	2.4
Kuervitikko					
Proven	21.9	23.6	0.183	0.216	2.5
Probable	0.3	23.8	0.177	0.194	2.5
Total					
Proven	113.7	30.5	0.185	0.112	2.4
Probable	1.1	30.0	0.157	0.100	2.4
TOTAL	**114.8**	**30.5**	**0.185**	**0.112**	**2.4**

Engineered Design

The final pit designs are shown in Figure 15-8 for Hannukainen and Kuervitikko.

The final Hannukainen pit design is approximately 2.5 km long and reaches a maximum depth of 230 m. The final Kuervitikko pit design is approximately 1.2 km long and reaches a maximum depth of 140 m.

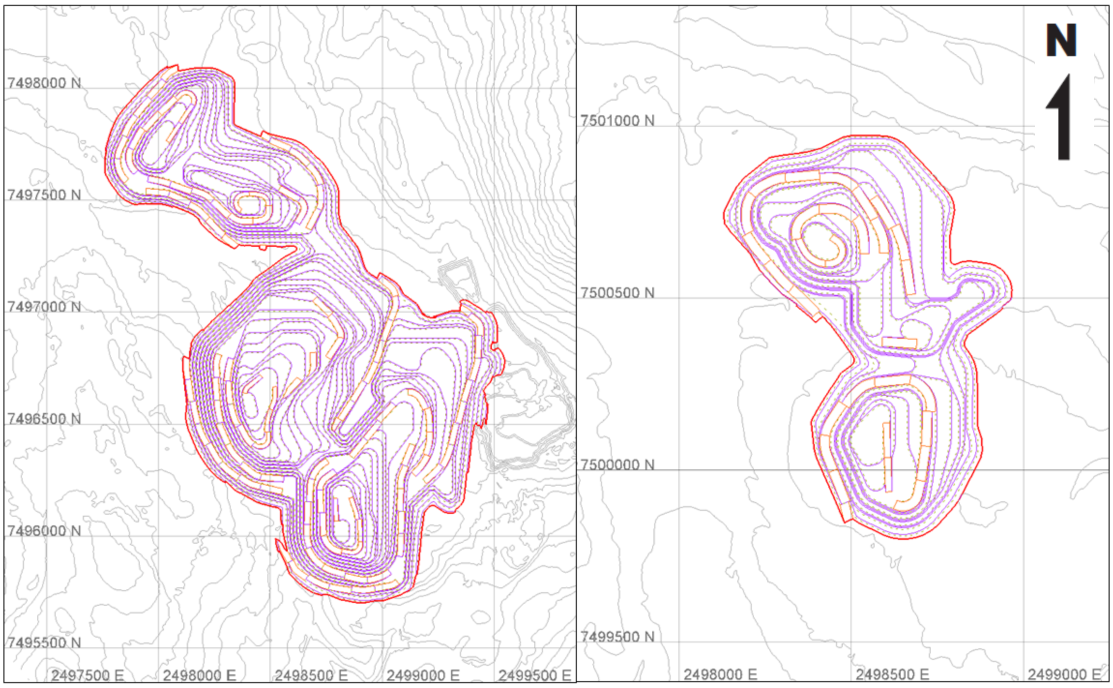


Figure 15-8: Hannukainen (left) and Kuervitikko (right) Final Pit Design

Selected Pit Shell Comparison

A comparison between the engineered pit design and the selected optimisation shell is shown in Table 15-9. The engineered pit design and selected optimisation shell have been evaluated in Maptek's Vulcan at a variable CoG, dependent on the processing recoveries.

Table 15-9: Pit Design Comparison with Selected Optimisation Shell

Pit	Total Mt	Waste Mt	Ore Mt	Fe Grade %	Strip Ratio t:t
Pit Shell 35					
Hannukainen	447.6	353.9	93.7	32.4	3.8
Kuervitikko	85.6	61.6	24	23.4	2.6
Total	**533.2**	**415.5**	**117.7**	**30.6**	**3.5**
Pit Design					
Hannukainen	476.2	383.7	92.5	32.2	4.1
Kuervitikko	85.4	63.2	22.3	23.6	2.8
Total	**561.6**	**446.8**	**114.8**	**30.5**	**3.9**
Difference					
Hannukainen	28.6	29.7	-1.2	-0.3	0.4
Kuervitikko	-0.2	1.5	-1.7	0.2	0.3
Total	**28.4**	**31.3**	**-2.9**	**-0.1**	**0.6**
Difference (%)					
Hannukainen	6%	8%	-1%	-1%	11%
Kuervitikko	0%	2%	-7%	1%	12%
Total	**5%**	**8%**	**-2%**	**0%**	**17%**

The differences in ore and waste tonnages are mainly due to the inability to recover ore from the lowest benches of the optimised shell while honouring minimum mining widths. A minor amount of ore was also lost on the footwall due to the orientation of the orebody and the face angle used in the pit design. SRK believes that this material could be retrieved operationally by mining directly to the ore contact.

The resulting mineable tonnages from the engineered pit design are 114.8 Mt of ore with 446.8 Mt of waste. The engineered pit designs resulted in a 2.9 Mt reduction in ore and 31.3 Mt increase in waste.

The engineered pit designs:

- are the basis of the mineable inventory for the schedule;
- are used a basis for the hydrological and hydrogeological studies; and
- are used to develop the haulage network for the production schedule.

15.4 Waste Dump Design

15.4.1 Objectives

The waste dump designs were designed based on the following criteria:

- contain sufficient capacity for the waste inventories within the engineered pit design;
- honour waste dump design parameters;
- provide three distinct sections for the different material types: overburden (“OVB”), potentially acid forming (“PAF”) and non-acid forming (“NAF”);
- ensure PAF material is contained within the groundwater catchment area;
- ensure waste dumps are located within mining lease boundary;
- locate as much NAF material to be within the groundwater catchment area as possible;
- minimise haulage distances;
- develop a basis for equipment destination scheduling for waste
- develop a basis for the road layouts and infrastructure design; and
- establish a basis for dump scheduling to demonstrate the dump development.

15.4.2 Waste Dump Design Parameters

The design criteria shown in Table 15-10, shown by material type, have been used to create the waste dumps. A cross-section of the waste dump design parameters is shown in Figure 15-9.

Table 15-10: Waste Dump Design Parameters

Parameter	Unit	PAF & NAF	OVB
Maximum Rehabilitation Angle		1:3	1:3
Lift Height	m	20	20
Rill Angle	°	35	26.5
Berm Width	m	30	30
Ramp Width	m	27	27
Ramp Grade	%	10	10

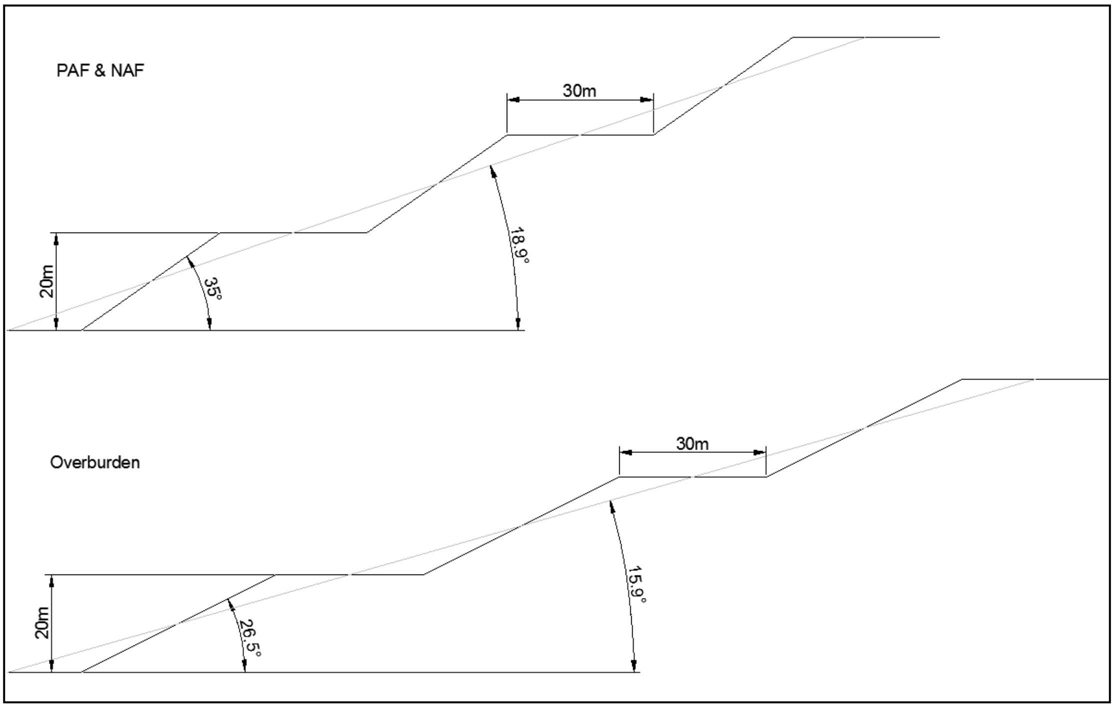


Figure 15-9: Cross Section – Waste Dump Design Parameters

15.4.3 Constraints

The following constraints were used when designing the waste dumps (Figure 15-10):

- PAF material must be contained within the groundwater catchment area;
- where possible, the NAF material must be placed within the groundwater catchment area;
- an offset of 100 m was maintained from the larger of the pit design edge and the Mineral Resource shell, which is the optimised pit shell at a Mineral Resource price of USD1.50/dmtu including inferred material;
- the area between the western dump and the overburden dump, where the crusher and workshops have been designed, have been avoided as advised by the Company; and
- no in pit dumping has been considered in this study as advised by the Company, due to the potential for underground mining.

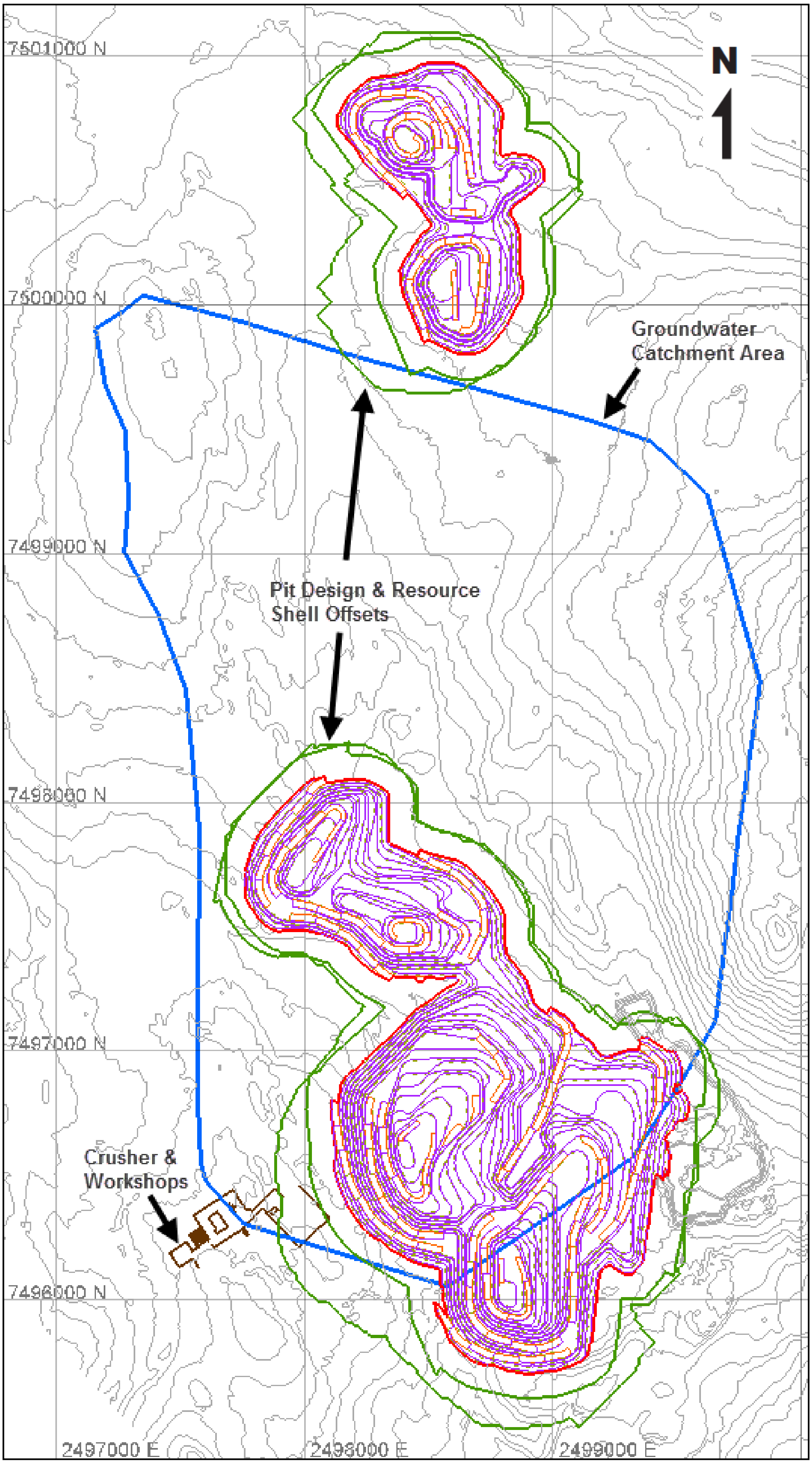


Figure 15-10: Waste Dump Constraints

15.4.4 Waste Dump Design

The waste dump have been designed to have excess capacity to allow for destination selectivity during production scheduling. Table 15-11 shows the waste pit inventories compared to the waste dump capacities by material type. Table 15-12 shows the waste dump capacities by waste dump. A swell factor of 30% was applied to all rock types to the in situ waste volume (bank cubic meters (“bcm”)) in the pit inventories to estimate the loose cubic metres (“lcm”) required for dump volumes.

Table 15-11: Waste Dump Capacities by Material Type

Material	Pit Inventory M bcm	Pit Inventory M lcm	Capacity M lcm
OVB	33.5	43.5	69.2
PAF	64.9	84.3	104.2
NAF	63.7	82.8	102.4

Table 15-12: Waste Dump Capacities by Waste Dump

Dump	Capacity M lcm
East Dump	198.9
West Dump	50.8
West Overburden Dump	26.0
Total	**275.7**

The location restrictions placed on the PAF material and, to a lesser degree, the NAF material has limited the space available for storage and has forced the waste dumps to be built higher than initially anticipated. This has increased haulage distances and has required some waste exiting on the western side of the pit to be hauled to the eastern side where storage is available. The ultimate waste dump design and layout of the waste dumps by material type is shown in Figure 15-11.

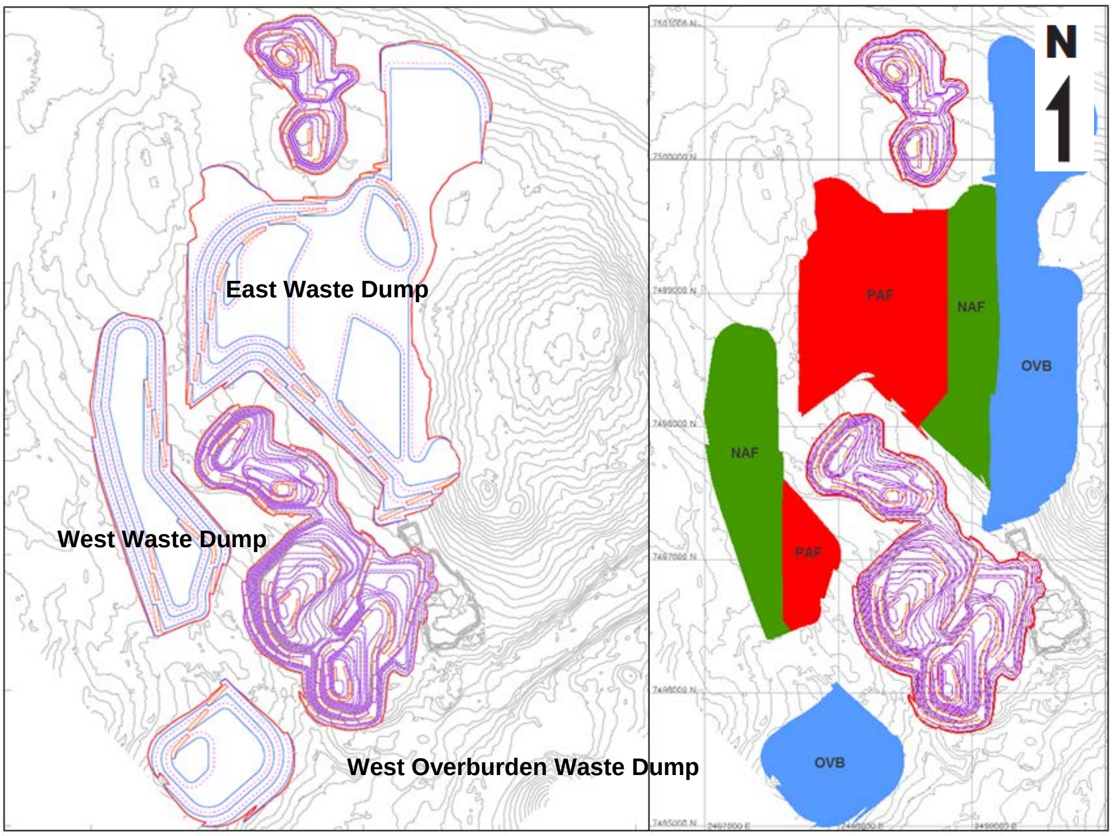


Figure 15-11: Waste Dump and material type locations

15.5 Mine Dewatering

15.5.1 Objectives

Groundwater and surface water inflows to an open pit mining operation can create saturated conditions and standing water within the pit. The objectives of the mine dewatering are to:

- prevent loss of access to areas of the pit;
- reduce explosives failure or the need to use more expensive explosives due to wet blast holes;
- reduce equipment wear;
- prevent inefficient loading and hauling; and
- allow for safe working conditions.

15.5.2 Pre-operational Dewatering

Requirements

There are currently two pits located at the Hannukainen site; Laurinoja and Kuervaara. These pits were allowed to flood following cessation of mining in 1995 and it was estimated in the PEA that they contain approximately 3.4 Mm^3 of water.

The pits will need to be dewatered before operation mining can commence. A dewatering permit was awarded to Northland by the Finnish – Swedish Transboundary Water Commission “BRC”) in 2007 which would enable the pits to be dewatered to the Äkäsjoki in less than a year.

It should be noted that the original BRC dewatering permit was valid until 31 December 2010 and it is understood by SRK that this permit has been extended to 31 December 2015. It should also be noted that in the Company intends to discharge water to the clarification pond rather than the Äkäsjoki River.

Dewatering of Historic Pits

The volume of water currently held in the pits was re-calculated based on topography received from Northland and the pit water levels recorded during a site visit in June 2011 (195 m above ordinance datum (“AOD”) for both pits). The calculated volume of water in the Laurinoja and Kuervaara pits was 3.7 Mm^3 and 0.3 Mm^3 respectively, giving a total volume in June 2011 of approximately 4.0 Mm^3.

Geochemical samples were collected from both pits in June 2011 and analysis shows that:

- the Laurinoja pit water quality is reasonably good though at depths greater than 30 m the water is virtually anoxic with slightly elevated concentrations of uranium; and
- the water of Kuervaara pit is poorer quality. It is anoxic at depth and has a high concentration (300 mg/L) of ferrous iron (SRK, 2011).

Given the constraints of the BRC permit, it was calculated that the dewatering of both pits can be completed in the summer and flooding months. The pre-operational dewatering plan will have water pumped to the clarification pond located at Hannukainen.

Table 15-13 presents the estimated pumping rates required to dewater the pits within six months, which has been specified in the mining schedule as the time allocated to dewater the pits. It is still necessary to ensure that discharge from Laurinoja constitutes 90% of the total discharge to the sedimentation pond, and Kuervaara pit contributes 10% of the total discharge.

Table 15-13: Estimated Dewatering Schedule

Parameter	Laurinoja (90% total discharge)	Kuervaara (10% total discharge)
Pumping rate (m^3/hr)	900	100
Time to dewater pits (days)	170	126
Time to dewater pits (months)	5.7	4.2

Dewatering Design for Historic Pits

Pre-operational pit dewatering requirements have been incorporated into the operational dewatering design such that equipment, including both pumps and pipe-work, can be utilised during both phases of the mine operation.

An example electrical, pontoon-mounted pump option and diesel option is presented in Table 15-14. The final decision on pump type and supplier will depend on cost and availability locally. Cost benefit analysis of other sump pumps should be undertaken at the engineering design phase.

Table 15-14: Example Pre-Operational Dewatering Pump

	Units	Electric	Diesel
Pumpset name		150-170SJ20-2	CF32
Head range	m	2 – 32	18 – 58
Flow range	m^3/h	0 – 360	0 – 720
RPM range	rpm	2,930	1,500 – 2,000
Motor size	kW	21	Diesel
Maximum solids size	mm	10	25
Unit cost	USD	12,100	105,700

15.5.3 Operational Requirements

Pit Water Management

The groundwater and surface water inflows predicted at Hannukainen through analytical and numerical modelling, if properly managed, are unlikely to cause significant operational problems. However, it is important that dewatering infrastructure, including sumps, pumps and pipelines, are adequately sized such that they can easily handle the predicted groundwater and surface water inflows with a suitable capacity in reserve to accommodate uncertainty. Uncertainty in the pit groundwater inflows arises from the assumptions and limitations of the models as well as the unpredictability of a fractured/anisotropic aquifer. Uncertainty in the surface water inflows arises from the unpredictability of precipitation.

Perhaps most importantly, it is essential that any dewatering system is designed to be as flexible as possible such that additional infrastructure can be easily added or removed to adapt to changes in predicted inflows. In this regard, a modular system with several pumps used in unison can be far more useful than a single dedicated pump designed to deal with all inflows to a particular sump.

The final excavation will consist of two main pits: Hannukainen and Kuervitikko. The southern end of Hannukainen, (Hannukainen South or HS), is initially the shallower of the pits, but will eventually reach a maximum depth of approximately 235 m below ground surface and be the lowest point in the Hannukainen pit. The central area, (Hannukainen Central or HC), will reach a maximum depth of approximately 220 m below ground surface but is the deeper of the two main pits until year 15 of mining. Water from Hannukainen North (“HN”) will be pumped to a tank located at the crest between the two pits from where it will be moved to ground surface.

The final excavation at Kuervitikko (“KU”) will also comprise two main pits with a shallower pit in the central area. The southern section of the excavation will be the shallower of the pits and will reach a depth of 120 m below ground surface. The northern section will reach a depth of about 140 m below ground surface. Water from the satellite sump will be pumped to a tank located on the crest between the central and northern pits, from where it will be pumped to ground surface.

Water from the collection sumps and tanks at Hannukainen and Kuervitikko will be pumped to ground surface by means of a series of in-line booster pumps. Ideally, the in-line boosters will be located at similar height intervals so they share the pressure head equally. The use of in-line booster pumps increases the number of pumps used for dewatering; however, this is offset by:

- increased flexibility;
- reduced static head, and therefore pressure, on the pit floor sump pump;
- reduced pressure in the discharge hose due to decreased static head above each pump and therefore lower specification hose required; and
- lower specification pump requirements.

Geotechnical Requirements

The pits are situated in hard rock where joint set intersections are more significant than pore water pressures in determining slope stability. A wedge stability analysis conducted by SRK in September 2011 concluded that:

- from the four identified joint sets there are three combinations of joint set intersections which have the potential to form unstable wedges in some pit sectors;
- intersection of any of these three sets with the main set – the foliation – is unlikely to form unstable wedges;
- using a 70° bench face angle is preferred over using 80° bench face angle as the risk of wedge type instabilities is significantly lower; and
- risk of wedge failures affecting multiple benches is generally regarded as low to very low with the exception of one pit sector where analysis indicates a higher potential for wedge type slope instabilities.

Given the current understanding of pit slope geology and structures, depressurisation of the pit slopes is not considered necessary. However, the need for pit slope depressurisation, for example through the use of horizontal drains, should be reviewed based on mapped structures, monitored groundwater levels and pit inflows recorded during operation mining.

Mine Development Requirements

Surface water inflow from direct precipitation into the pit has been modelled on the basis of a 24-hour storm event being removed by pumping over a nominal 30-day period. A 1 in 50 year storm event has been used to define the surface water dewatering requirements, which is considered a conservative approach in the light of the estimated LoM (19 years).

Water management, including in-pit storage capacity, will be an important factor during the spring melt when significant quantities of snow and ice thaw will generate significant quantities of water over a short period of time.

Water Treatment Requirements

SRK understands that all water from the pit water management infrastructure (sump, protection ditches, etc) will be pumped to the oil and sand separator. This separating pond must be located sufficiently far from the pit edge to avoid recirculation of water in to the pit. Water treatment of managed water is outside the scope of this report.

Typical open-pit dewatering equipment is only guaranteed down to pH 5 and a specified suspended load. Therefore it is desirable to have information about likely pit sump water quality in order to select suitable dewatering equipment. Geochemical modelling predicted pH to be in the range of 5.5 to 6.0 at Hannukainen and 5.3 to 7.2 at Kuervitikko. Pit sump water was predicted to have a low suspended solids load.

For the purposes of the current dewatering plan it is assumed that all in-pit waters will be pH 5 or greater and that the suspended solids load is within the normal range of a typical mine dewatering pump. These assumptions should be verified by ongoing monitoring during operational mining.

In the case that pit wall geochemistry leads to pH lower than 5, some form of in-sump pH control will be required. This can be achieved through dosing with hydrated lime or sodium hydroxide pellets. Sodium hydroxide pellets are more expensive but much easier to administer.

Operational Dewatering Options

The current pit design has three pits in the Hannukainen excavation (HS, HC and HN) and three at Kuervitikko (KU1, KU2 and KU3). Each pit will require a separate sump while each pit is being developed, with two sumps at HN.

It is proposed that water from HN be pumped to a collection tank and subsequently moved to the surface. Similarly at Kuervitikko, water from the sump in KU3 and KU2 should be pumped to a collection tank and then moved to the surface. This system will reduce the pumping and piping requirements and offers a flexible solution to predicted groundwater and surface water inflows.

The discharge lines have been designed to extend from the pit sumps to a point 100 m beyond the pit edge. Subsequent dewatering infrastructure is beyond the scope of this study and is included in the infrastructure design developed by Pöyry. Water will be discharged to oil and sand separators, which are assumed to be located sufficiently far from the pit edge so as to avoid recirculation of water in to the pit.

A summary of the proposed dewatering system is shown in Figure 15-12.

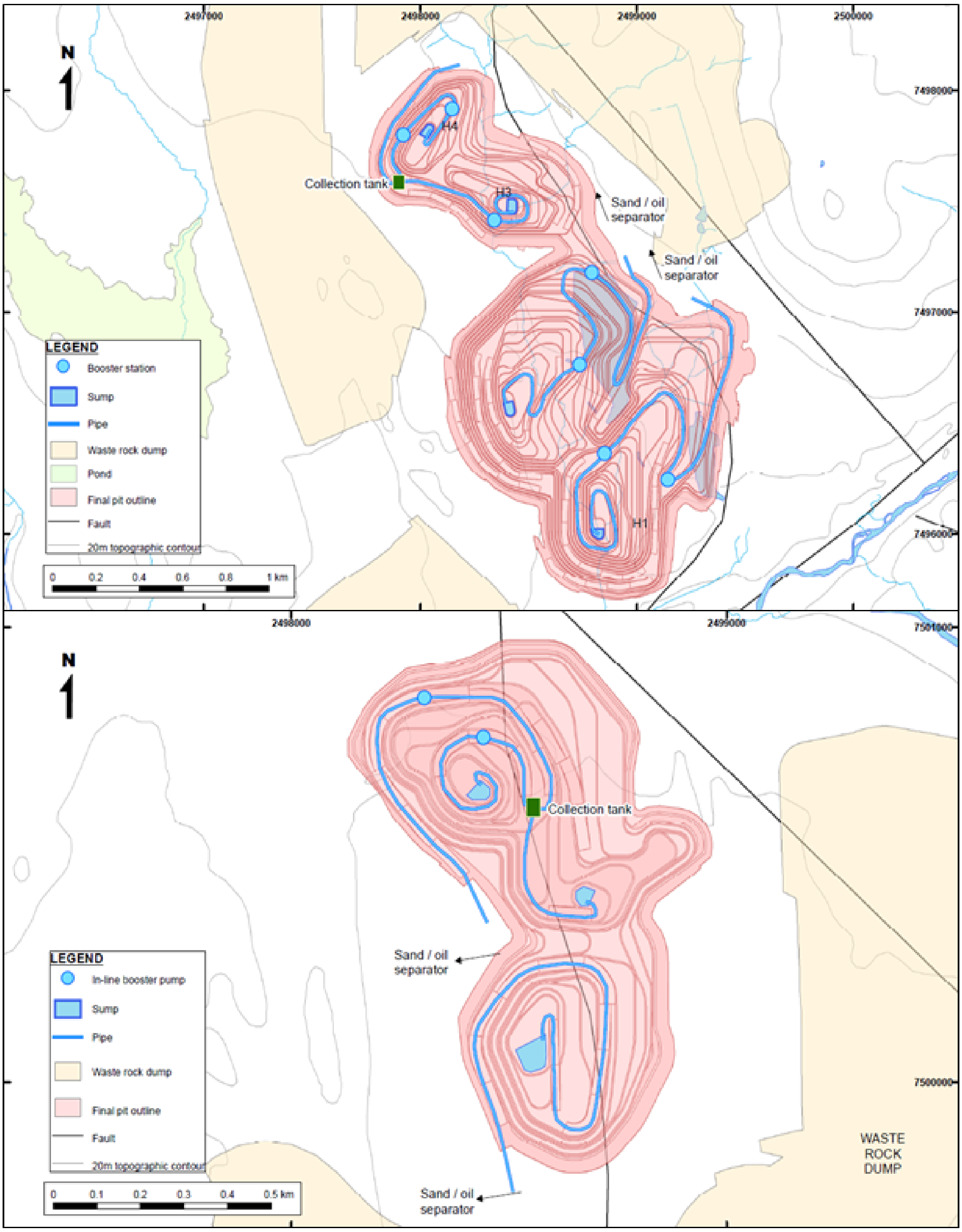


Figure 15-12: Proposed Dewatering system at the Hannukainen and Kuervitikko Pits

Dewatering Pumps

Groundwater pumping requirements have been derived from the numerical modelling. The likely groundwater pumping requirements for each sump pump during the lifetime of the mine are summarised in Table 15-15. There will be some seasonal variation in groundwater inflow but it will be insignificant in comparison to the variation in surface water inflow.

Table 15-15: Annual Groundwater Pumping Requirements

Year	Pumping volume (m^3/h)						
	HS	HC	HN1	HN2	KU3	KU1	KU2
1	10	20					
2	30	90	50				
3	10	70	50				
4	10	90	60				
5		100	40				
6		110	60				
7	10	100	50				
8	10	130	50	20			
9	20	130	40	40			
10	20	130	30	60			
11	40	110	30	70			
12	60	90	40	80			
13	70	80	40	70			
14	90	70	40	70			
15	100	70	30	70	40	40	30
16	100	60	30	70	40	30	120
17	100	60	30	70	10	80	170
18	100	70	30	70		70	200
19	90	90	30	70	20	40	210
20	90	80	30	70	120	10	180
21	90	110	30	70	160		160

Water derived from snowmelt and precipitation within the Hannukainen and Kuervitikko pits will be directed towards the main sumps and then pumped to the surface. There are two aspects that need to be considered when assessing the total amount of water that will need to be extracted from the pit sumps:

- average precipitation and snow-melt (including seasonal variation); and
- large rainfall events.

Table 15-16 illustrates the total volume of water (rainfall and snowmelt) that will require extraction from the pits in year 21 under average climatic conditions. The maximum amount of pumping capacity required to remove water occurs in May during the spring thaw, when a capacity of 410 m^3/h may be required at Hannukainen and 110 m^3/h at Kuervitikko (about four times that produced by precipitation in other months). In addition to average monthly precipitation and snowmelt, large rainfall events (>50 mm/d) have the potential to severely impact mining operations.

Table 15-16: Monthly Rainfall and Snowmelt into Total Pit Areas

Month	Rainfall + snowmelt (mm/month)	Pit inflows in year 21 Hannukainen (m^3/h)	Kuervitikko (m^3/h)
Jan			
Feb			
Mar			
Apr	10.4	19	5
May	224.9	412	112
Jun	53.0	97	26
Jul	74.0	135	37
Aug	70.8	130	35
Sep	50.9	93	25
Oct			
Nov			
Dec			

Table 15-17 illustrates the volume of water likely to be produced by 24-hour rainfall events of different magnitudes and return periods (illustrated as total volume for each pit at LoM). It can be seen that there is a potential for significant volumes of water to be produced. Whilst operationally speaking it would be beneficial to remove water immediately to allow mining to continue unhindered, it would not be cost effective to size the pump set for peak events.

Table 15-17: Extreme Rainfall Magnitudes for LoM

	Units					
Return Period	years	5	10	20	50	100
Rainfall	mm	39.2	47.8	56.4	67.7	76.3
Hannukainen	m^3/month	56,450	68,820	81,180	97,530	109,890
Kuervitikko	m^3/month	15,290	18,640	21,980	26,410	29,760

It is standard to size a pump set to pump out a 1 in 50 year storm event within 30 days. However, it has been demonstrated that at Hannukainen the annual spring thaw also generates a significant volume of surface water runoff. At the neighbouring Tapuli open pit mine, Arundon has suggested that the dewatering system be designed to remove the additional load associated with the spring thaw over a 60 day period. For this purpose, the mine design should provide storage for the surplus water to accommodate pumping at the required rate. For this reason, however, any required feed to plant stockpiles should be topped up at the start of the thaw season to prevent loss of production.

It is plausible that a 1 in 50 storm event could coincide with the spring thaw; however, analysis of the historical climate record at Pajala climate station shows that between 1988 and 2007:

- there have been three 1 in 10 year storm events, one 1 in 20 year storm event and no 1 in 50 year storm events;
- the 1 in 20 year storm event occurred in August, while the 1 in 10 year events occurred in July; and
- high magnitude rainfall events (greater than the 99th percentile of the rainfall data set) generally occur in July and August.

Consequently, it has been assumed that a storm event is unlikely to coincide with the spring thaw in May and so pump sets need not be sized to accommodate both events simultaneously.

The pumping capacity required to dewater the May thaw in 60 days is sufficient to dewater a 1 in 50 storm event and therefore the May thaw dewatering requirements have been used to size the appropriate pump set. The likely maximum surface water pumping requirements for each sump pump during the lifetime of the mine are summarised in Table 15-18.

Table 15-18: Summary of Stand-By Pumping Requirements

Year	Surface water pumping volume (m^3/h) (Spring thaw)						
	HS	HC	HN1	HN2	KU3	KU1	KU2
1		60					
2	40	60	40				
3	40	60	40				
4	40	60	40				
5	40	80	40				
6	40	100	40				
7	70	90	40	30			
8	70	90	50	40			
9	100	90	50	40			
10	120	100	50	40			
11	120	100	50	40			
12	120	100	50	40			
13	120	110	50	40			
14	120	120	50	40			
15	120	120	50	40	10	10	20
16	120	120	50	40	20	20	40
17	120	120	50	40	20	20	40
18	120	120	50	40	30	20	40
19	120	120	50	40	30	20	40
20	120	120	50	40	30	20	40
21	120	120	50	40	30	20	40

A dewatering system that makes use of multiple pumps and in-line boosters is recommended for both pits at Hannukainen. Although some pumps are capable of pumping over 300 m head, the drivers for using a system of sump pumps and multiple in-line boosters are flexibility, and to minimise pressure for the pipe requirements. Similar specification pumps can be added to the system with depth if inflows are as predicted, but smaller or larger capacity pumps can be added to the system as and when required in order to upgrade or downgrade the overall pumping capacity to suit updated demand.

The use of a modular system gives rise to a progressive cost as the dewatering demand increases throughout the mine life rather than a large initial capital requirement at the start of mining.

Pump selection depends on pumping volume and dynamic head. It is assumed that pipes run from the sump to 100 m beyond the pit edge, using the access ramps, wherever possible.

The proposed system utilises electrically driven pumps. The units will be hard wired into the power supply system (400 V, 50 Hz). The pump selection includes five sizes of motor, 21, 45, 75, 110 and 132 kW, with the same frame size in order to allow maximum flexibility within the system.

Submersible pumps operate on level controls. All submersibles will be positioned above the deepest section of the sump, supported on a flotation unit.

Multiple in-line booster pumps will be used to move water from the pit sumps to ground surface. Pumps will be located at regular height intervals (approximately 50 to 100 m) to ensure the load on each unit is approximately equal. The in-line booster pumps utilise the variable frequency drive ("VFD") technology which allows the operating rpm of the pumps to be electrically controlled and varied either on the unit or remotely. This enables the pump to operate over a wider range of pumping rates than traditional single speed pumps. The use of VFD units results in a cost effective dewatering solution that minimises the infrastructure required to manage water in the pits. Two collection tanks will also be used, one each at Hannukainen and Kuervitikko.

Each dewatering line will have suitable valves in the tanks and pipe work so all pipe lines can be drained when not in use. Draining the lines minimises the pressure rating of the pipe work (16 bar required) alleviates the need for none return valves and prevents freezing within the pipe which is critical given the temperatures experienced at Hannukainen during the winter months.

An example electrical sump and stage pump option for Hannukainen is presented in Table 15-19.

Table 15-19: Summary of Electrical Sump and Stage Specifications

	Units	Submersible pump	Stage pump
Pumpset name		100-230SJ20-2	125PC-DWU + 110
Head range	m	10 – 74	20 – 140
Flow range	m^3/h	0 – 130	0 – 400
RPM range	rpm	2,930	1,200 – 2,950
Motor size	kW	21	110
Maximum solids size	mm	10	32
Unit cost	USD	12,100	68,700 (including VFD)

The final decision on pump type and supplier will depend on cost and availability locally. A cost benefit analysis of other sump pumps should be undertaken at the engineering design phase.

It is recommended that one spare submersible and one spare in-line booster of each size are kept on site for emergency and general dewatering support. In addition, a diesel engine, auto prime portable pump would provide flexible, instantaneous dewatering capability in the case of an emergency and should be included in the dewatering contingency. This style of diesel pump has been included in the pre-operational dewatering design.

Pipework

The discharge lines have been designed to extend from the pit sump to 100 m beyond the pit edge. Water will be discharged to an oil and sand separator. Flexible armoured pipeline has been used in the cost estimate, as it provides a cost benefit at the lengths required.

Due to low temperatures during the winter months, it is important that a constant flow is maintained in the discharge line to prevent freezing. This is an important planning consideration when selecting discharge line diameters. In addition, the pipes should be able to drain fully to avoid freezing when not in use.

Discharge lines will be routed, where possible, along the access ramps to facilitate maintenance of the line. A single diameter of pipe work (6 in.) has been used throughout the site. This offers significant benefits when maintaining the system and managing spares.

Sump sizing

The pit sumps should be sized according to three main factors: the volume of water that will need to be contained (as defined by expected surface and groundwater inflows), operating conditions, and amount of space at the bottom of the pit.

Table 15-20 shows the volume of water each sump would need to accommodate in year 21 following a 1 in 50 year storm event. Sump depth has been altered between pit areas to account for available space at the bottom of each area.

Table 15-20: Surface Water Inflow to Pit Sumps

Year 21 inflow	Proportion of total pit area (%)	1 in 50 year event (m^3)	Depth (m)	Area (m^2)	Length (m)	Width (m)
HS	38	37,000	10	3,700	60	62
HC	36	34,860	10	3,486	60	58
HN1	15	14,610	5	2,922	55	53
HN2	11	11,050	5	2,210	45	49
KU3	37	9,660	5	1,932	45	43
KU1	19	5,070	5	1,014	30	34
KU3	44	11,680	5	2,336	45	52

Surface Water Control

Surface water runoff during the spring thaw will have a significant influence on operations and inflows to the pits should be limited as far as practically possible. Management should include an engineered surface water diversion system. This will include a bund to prevent inflow and/or ditches diverting flow away from the pits. Protection should extend around the whole perimeter of the pits. Runoff from the waste dumps should be managed at the toe of the dumps and diverted away from the pits to avoid the cost of pumping this water out of the pits.

Dewatering Schedule

It is likely that a sump would be engineered into the bottom bench as part of the mine planning. An in-line pump would be added to the system approximately every 50 to 100 m of pit depth to serve as booster stations. A discharge line would be extended as required. Such a dewatering system would be able to accommodate any changes in mine planning with ease.

15.6 Mine Production Schedule

15.6.1 Objectives

The mine production schedule for the Project has used the pit inventories from the pit designs. The objectives of the production schedule are to:

- develop a planning model suitable for modelling the development constraints of the Project;
- achieve annual quantity and quality targets;
- honour crusher capacity constraints;
- determine pre-stripping requirements;
- strip waste to ensure sufficient quantities of ore are available to maintain production targets; and
- develop a production schedule suitable for developing HFS mining capital and operating cost estimates.

15.6.2 Scheduling Parameters

The key scheduling parameters for the production schedule were to:

- achieve 6.5 Mtpa of ore production;
- not to exceed 2.3 Mtpa of Fe concentrate;
- not to exceed 50 ktpa of Cu concentrate;
- Fe concentrate grade is 70% Fe;
- Cu concentrate grade is 25% Cu;
- feed highest quality ore at the start of the schedule;
- minimise initial waste stripping;
- no long term ore stockpiling due to a decrease in recoverability when the ore is left exposed; and
- no RoM stockpile has been modelled; however, it has been assumed it will located adjacent to the crusher to accommodate the short term differences between the loading unit and crusher production rates.

The production schedule has been developed with the following calendar periods:

- quarterly for the initial five years; and
- annually for year 6 onwards.

15.6.3 Scheduling Model

The schedule model was developed using Runge's XPAC scheduling software. The schedule model has been developed to allow for:

- development of a practical mining sequence;
- a range of capacity constraints and sequencing dependencies; and
- destination scheduling for the crusher feed and waste material.

Scheduling Methodology

The scheduling methodology undertaken in the XPAC model is:

- a main database in XPAC is populated from the mining block model and mining dump model;
- a haulage database in XPAC is populated based on the results from the haulage network simulation using Runge's TALPAC engine;
- equipment resources, objectives, constraints, and targets are defined for the given scheduling scenario;
- a calendar database is used to control mining constraints and targets on a period by period basis;
- scheduling iterations are performed until an acceptable schedule is completed given the planning objectives; and
- results are output to a csv format file for further assessment in Microsoft Excel.

The mining blocks have been scheduled on 10 m benches to limit the number of blocks imported into XPAC. To achieve the SMU size the ore blocks must be mined to 5 m selectivity, SRK believes that this can be handled operationally and will not impact the production schedule.

15.6.4 Scheduling Inventories

The scheduling inventory in the schedule model is shown in Table 15-21.

Table 15-21: Scheduling Inventories

	Units	Total	HC1	HC2	HC3	HC4	HC5	HS1	HS2	HS3	HS4	HN1	HN2	KU1	KU2	KU3
Total																
Quantity	Mt	**561.6**	26.7	32.2	38.1	40.2	76.4	27.9	32.8	51.8	58.8	28.2	63.1	25.0	34.4	26.0
Volume	Mm^3	**193.7**	9.3	11.0	12.9	13.7	26.4	9.5	11.0	17.5	19.8	10.1	22.0	9.6	11.8	9.2
Density	t/m^3	**3.0**	2.9	3.0	3.0	3.0	2.9	3.1	3.0	3.0	3.0	2.9	2.9	2.7	3.0	2.9
Waste																
Quantity	Mt	**446.8**	20.5	25.1	31.6	34.1	67.7	14.5	23.4	40.9	49.4	22.5	54.1	18.4	25.8	18.9
Volume	Mm^3	**162.0**	7.6	9.1	11.2	12.0	24.0	5.7	8.4	14.5	17.2	8.6	19.6	7.7	9.3	7.1
Density	t/m^3	**2.8**	2.7	2.8	2.8	2.8	2.8	2.6	2.8	2.9	2.9	2.7	2.8	2.4	2.8	2.7
Waste Types																
OVB	Mt	**181.5**	5.5	9.2	12.8	16.6	38.1	1.6	6.5	11.5	15.8	7.0	38.7	1.6	9.4	7.0
PAF	Mt	**190.5**	9.2	11.3	16.2	16.1	26.3	6.1	11.6	23.7	30.0	7.1	9.6	4.7	11.5	7.1
NAF	Mt	**74.8**	5.8	4.6	2.6	1.4	3.2	6.8	5.3	5.7	3.5	8.4	5.8	12.1	5.0	4.8
Ore																
Quantity	Mt	**114.8**	6.2	7.1	6.5	6.2	8.7	13.4	9.4	10.9	9.5	5.7	9.0	6.6	8.6	7.1
Volume	Mm^3	**31.7**	1.7	1.9	1.7	1.7	2.4	3.7	2.6	3.0	2.6	1.5	2.4	1.9	2.5	2.1
Density	t/m^3	**3.6**	3.7	3.7	3.7	3.7	3.7	3.6	3.6	3.7	3.7	3.7	3.7	3.5	3.4	3.4
Ore Grades																
Fe	%	**30.5**	31.9	34.3	34.2	34.4	32.5	29.3	29.3	32.4	32.4	33.3	33.2	25.2	23.8	21.9
Cu	%	**0.18**	0.33	0.28	0.30	0.29	0.29	0.10	0.13	0.11	0.12	0.12	0.16	0.20	0.17	0.18
Au	g/t	**0.11**	0.19	0.16	0.18	0.18	0.17	0.02	0.03	0.03	0.03	0.03	0.08	0.21	0.17	0.27
S	%	**2.4**	2.1	2.1	2.1	2.1	2.0	3.1	3.1	2.3	2.5	1.9	2.3	2.6	2.5	2.3
Co	%	**0.01**	0.02	0.02	0.02	0.02	0.01	0.02	0.01	0.01	0.01	0.01	0.02	0.01	0.01	0.02
Ore Classification																
Measured	Mt	**113.7**	6.1	7.1	6.5	6.2	8.7	13.4	9.4	10.9	9.1	5.7	8.7	6.5	8.4	7.1
Indicated	Mt	**1.1**	0.1	0.0				0.0	0.0	0.0	0.4	0.0	0.3	0.1	0.2	0.0
Strip Ratio	**t:t**	**3.9**	**3.3**	**3.5**	**4.8**	**5.5**	**7.8**	**1.1**	**2.5**	**3.8**	**5.2**	**4.0**	**6.0**	**2.8**	**3.0**	**2.7**

15.6.5 Equipment Parameters

Loading

The production rates used to determine the loading fleet requirements are shown in Table 15-22.

Table 15-22: Loading Rates

Loading Unit	Ore (tph)	Waste (tph)	OVB (tph)
26 m^3 Shovel	3,690	3,250	2,570
19 m^3 Front end loader	3,020	2,620	2,050

The loading unit bucket size has been used to denote a class of equipment. The actual bucket capacities used in the equipment productivity analysis have been reduced to match the maximum suspended load ratings for the equipment, considering the high density of the ore.

Haulage

The parameters used for the haulage fleet are shown in Table 15-23. The flat bench and lift speeds have been used to calculate the travel time between the pit bench blocks to the pit ramp exit and the dump ramp entrance to the dump blocks. A job efficiency factor has been applied to the TALPAC travel time estimates. The production schedule is reported in dry tonnes; however wet tonnes have been used for the equipment estimates.

Table 15-23: Haulage Parameters

Parameter	Units	Value
Truck Capacity	t	227
Truck Capacity	m^3	176
Loading & Spot Time	min	2.5
Dump & Spot Time	min	1.1
Moisture Content		
Rock	%	5
Overburden	%	15
Rolling Resistance	%	3
Maximum Speed	kph	Unconstrained
Pit Bench Flat Speed	kph	20
Dump Lift Flat Speed	kph	20
Operator Efficiency	%	80

The following methodology has been undertaken for the haulage fleet estimate:

- develop a haulage network in Maptek's Vulcan comprised of start nodes (pit ramp exits), end nodes (dump ramp entrances), and haulage strings;
- create a haulage database of travel times between all node combinations using XPAC's TALPAC module;
- during scheduling combine the pit centroid to pit ramp, node to node travel times and dump ramp entrances to dump block travel times to estimate a total travel time value for each scheduling record;

- add spot, loading, and dump times to the travel time to calculate a total cycle time; and
- use the cycle time estimate and truck capacity to estimate the haulage operating hours.

Figure 15-13 represents HC1 haulage strings and nodes which were used for the haulage fleet estimates. A similar network has been developed for all pit stages.

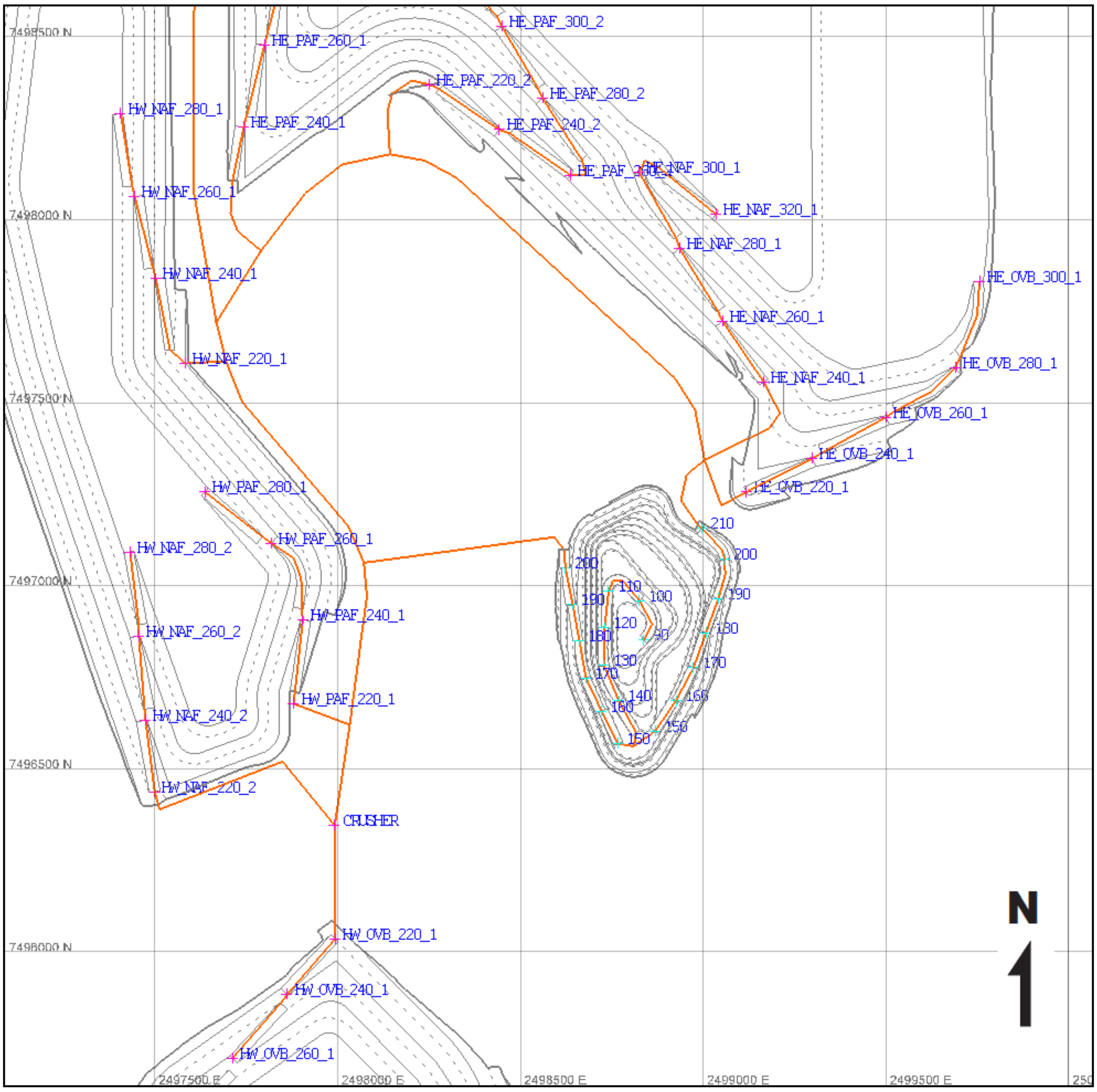


Figure 15-13: HC1 Haulage Strings and Nodes

15.6.6 Mine Development Sequence

The resulting pit and dump development from the production schedule are shown in Table 15-24 and Table 15-25.

Table 15-24: Annual Pit Development Sequence – Total Material Movement

	Units	Total	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Hannukainen North																							
HN1	Mt	**28.2**		5.7	7.3	7.7		7.4															
HN2	Mt	**63.1**							3.0	2.4	13.6	10.8	12.8	13.7	6.8								
Total	**Mt**	**91.3**		**5.7**	**7.3**	**7.7**		**7.4**	**3.0**	**2.4**	**13.6**	**10.8**	**12.8**	**13.7**	**6.8**								
Hannukainen Central																							
HC1	Mt	**25.4**	2.4	0.9	6.0	13.4	2.8																
HC2	Mt	**32.2**	0.2	2.3		1.6	13.4	9.2	4.9	0.5													
HC3	Mt	**38.1**					7.8	10.4	4.0	12.8	3.2												
HC4	Mt	**40.2**						2.5	13.4	10.0	6.4	6.1	1.9										
HC5	Mt	**76.4**													2.8	6.4	5.5	12.2	12.8	13.3	17.2	3.8	2.4
Total	**Mt**	**212.4**	**2.6**	**3.2**	**6.0**	**14.9**	**24.0**	**22.1**	**22.3**	**23.2**	**9.6**	**6.1**	**1.9**		**2.8**	**6.4**	**5.5**	**12.2**	**12.8**	**13.3**	**17.2**	**3.8**	**2.4**
Hannukainen South																							
HS1	Mt	**27.9**		2.7		3.5	6.0	2.8	4.3	2.6	3.1	3.0											
HS2	Mt	**32.8**							2.7	3.1	4.4	7.1	10.1	4.2	1.2								
HS3	Mt	**51.8**									1.9	7.3	7.9	8.4	9.9	11.8	4.7						
HS4	Mt	**58.8**									0.0	2.3	4.3	7.2	10.9	10.7	13.1	6.2	2.0	2.1			
Total	**Mt**	**171.3**		**2.7**		**3.5**	**6.0**	**2.8**	**7.0**	**5.8**	**9.3**	**19.7**	**22.4**	**19.8**	**21.9**	**22.5**	**17.8**	**6.2**	**2.0**	**2.1**			
Kuervitikko																							
KU1	Mt	**25.0**														4.0	9.8	6.8	4.4				
KU2	Mt	**34.4**																10.2	12.6	8.7	3.0		
KU3	Mt	**26.0**																		4.1	8.4	10.2	3.3
Total	**Mt**	**85.4**														4.0	9.8	16.9	17.0	12.8	11.3	10.2	3.3

Table 15-25: Annual Dump Development Sequence

	Units	Total	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
East OVB	**Mt**	**34.9**	**2.5**	**4.8**	**2.4**	**0.4**	**0.1**		**2.6**	**0.3**						**4.0**	**7.5**	**5.6**	**0.0**	**3.6**	**1.2**		
220	Mt	**0.0**	0.0																				
240	Mt	**7.0**	2.4	0.3												0.0	1.4	1.9	0.0	0.8	0.1		
260	Mt	**24.8**		4.5	1.9	0.1	0.1		0.6							4.0	6.1	3.8		2.8	1.0		
280	Mt	**3.1**			0.5	0.3			2.0	0.3													
East NAF	**Mt**	**98.3**		**0.0**	**2.3**	**3.4**	**2.7**	**1.7**	**1.7**	**2.4**	**0.7**	**0.0**	**1.6**	**12.2**	**7.6**	**7.7**	**9.3**	**16.2**	**14.3**	**7.0**	**5.6**	**1.9**	**0.1**
220	Mt	**0.0**		0.0																			
240	Mt	**3.2**		0.0	1.9	1.1	0.2	0.0															
260	Mt	**25.9**			0.4	2.4	2.5	1.7	1.7	2.4	0.7	0.0	0.7	2.8	3.1	5.6	1.9						
280	Mt	**40.2**											0.8	9.4	4.5	2.1	5.0	12.1	6.2	0.1			
300	Mt	**29.1**															2.4	4.1	8.1	6.9	5.6	1.9	0.1
East PAF	**Mt**	**160.9**		**0.1**	**0.7**	**2.3**	**0.4**	**9.3**	**3.3**	**13.5**	**6.0**	**8.7**	**13.0**	**14.7**	**16.4**	**13.5**	**9.0**	**6.8**	**11.0**	**11.1**	**15.3**	**4.3**	**1.3**
220	Mt	**0.2**		0.1	0.1	0.0		0.0															
240	Mt	**30.3**			0.6	2.3	0.4	9.3	3.3	7.1	2.0	2.1	1.3	0.8	0.7	0.1							
260	Mt	**56.8**								6.4	4.0	6.6	10.9	8.4	7.6	5.5	3.5	3.8					
280	Mt	**48.3**											0.7	5.5	8.0	7.9	5.6	3.0	2.6	6.2	8.2	0.6	
300	Mt	**25.4**																0.0	8.4	4.9	7.1	3.7	1.3
West OVB	**Mt**	**39.9**	**0.1**	**6.4**	**2.9**	**3.2**	**4.9**	**1.3**	**3.2**	**3.0**	**4.4**	**5.8**	**1.5**	**0.0**	**1.1**	**1.1**	**0.9**	**0.2**					
220	Mt	**6.0**	0.1	1.0	0.9	1.3	1.5	0.6	0.6	0.0													
240	Mt	**15.8**	0.1	5.3	1.9	1.7	2.4	0.7	2.1	1.3	0.4												
260	Mt	**13.9**					0.9		0.5	1.7	4.0	5.8	1.1		0.0								
280	Mt	**3.6**											0.4	0.0	1.1	1.1	0.9	0.2					
West NAF	**Mt**	**83.1**		**0.1**	**3.2**	**3.2**	**9.9**	**7.7**	**11.9**	**3.8**	**13.9**	**14.7**	**14.6**										
220	Mt	**6.7**		0.1	3.2	1.4	1.9	0.0															
240	Mt	**30.0**				1.8	8.0	7.7	7.0	1.9	3.5	0.1											
260	Mt	**28.2**							4.9	1.9	9.9	10.9	0.6										
280	Mt	**18.3**									0.6	3.8	14.0										
West PAF	**Mt**	**28.2**		**0.1**	**1.0**	**7.9**	**6.8**	**5.8**	**3.1**	**1.9**	**1.0**	**0.7**											
220	Mt	**2.0**		0.1	0.3	0.8	0.8																
240	Mt	**12.1**			0.7	7.1	4.4																
260	Mt	**8.7**					1.6	5.8	1.3														
280	Mt	**5.4**							1.8	1.9	1.0	0.7											

Mining Areas

The Hannukainen production schedule requires mining in multiple pit stages concurrently to achieve targets while honouring constraints. The number of mining areas by year is shown in Figure 15-14. The key outcomes from the mining intensity are shown below:

- overburden and waste stripping in the first two years in all areas;
- an increase in mining areas in the next five years to honour ore targets and strip enough waste for future ore requirements; and
- a decrease in mining areas for the last 10 years as the number of areas available decrease.

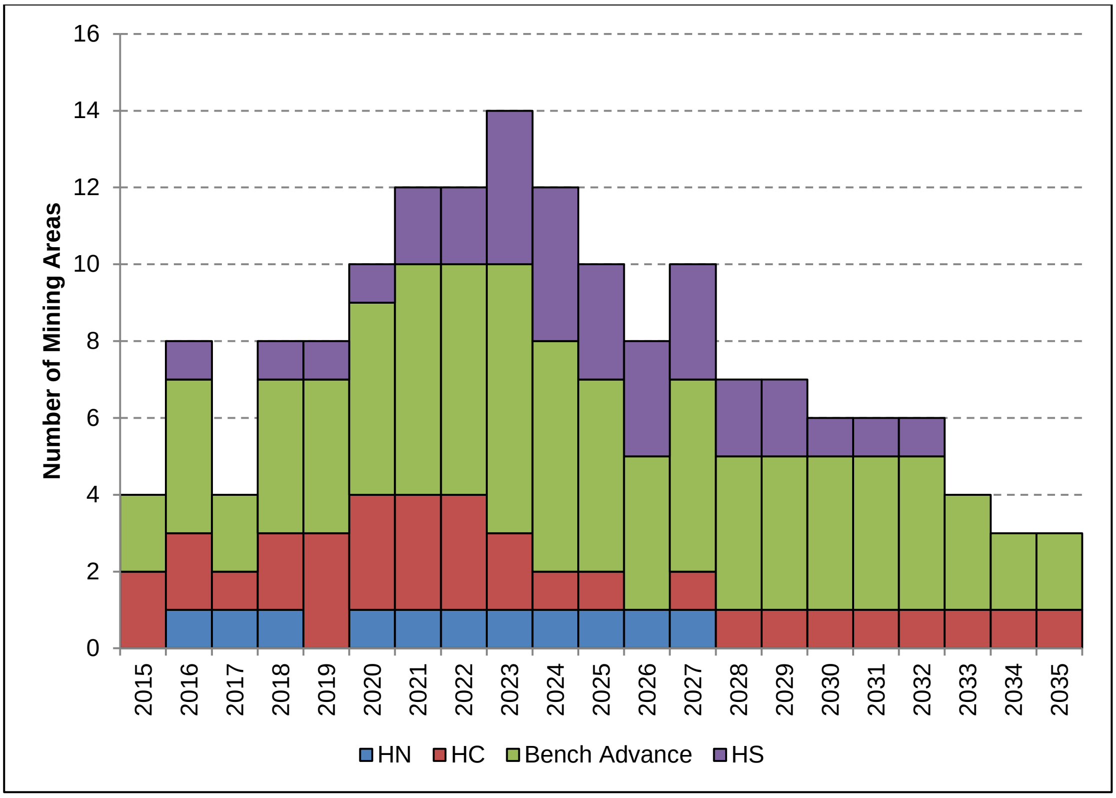


Figure 15-14: Number of Mining Areas

Bench Advance

The vertical advance rate by region is shown in Figure 15-15 and can be used as a measure of practical mining. The schedule has been constrained to limit the vertical advance to a maximum of six benches per year to limit bench turnover to two benches per year. The period where the maximum vertical advance rate has been exceeded is a result of relaxed constraints to achieve ore targets. The key outcomes from the production schedule include:

- the vertical advance rates are well below the upper limit for most periods;
- the lower vertical advance rate is due to the blending of the ore between the pits to achieve a constant Fe grade; and
- the lower the vertical advance rate indicates that there is potential to increase rates in the pits if ore stockpiling were permitted.

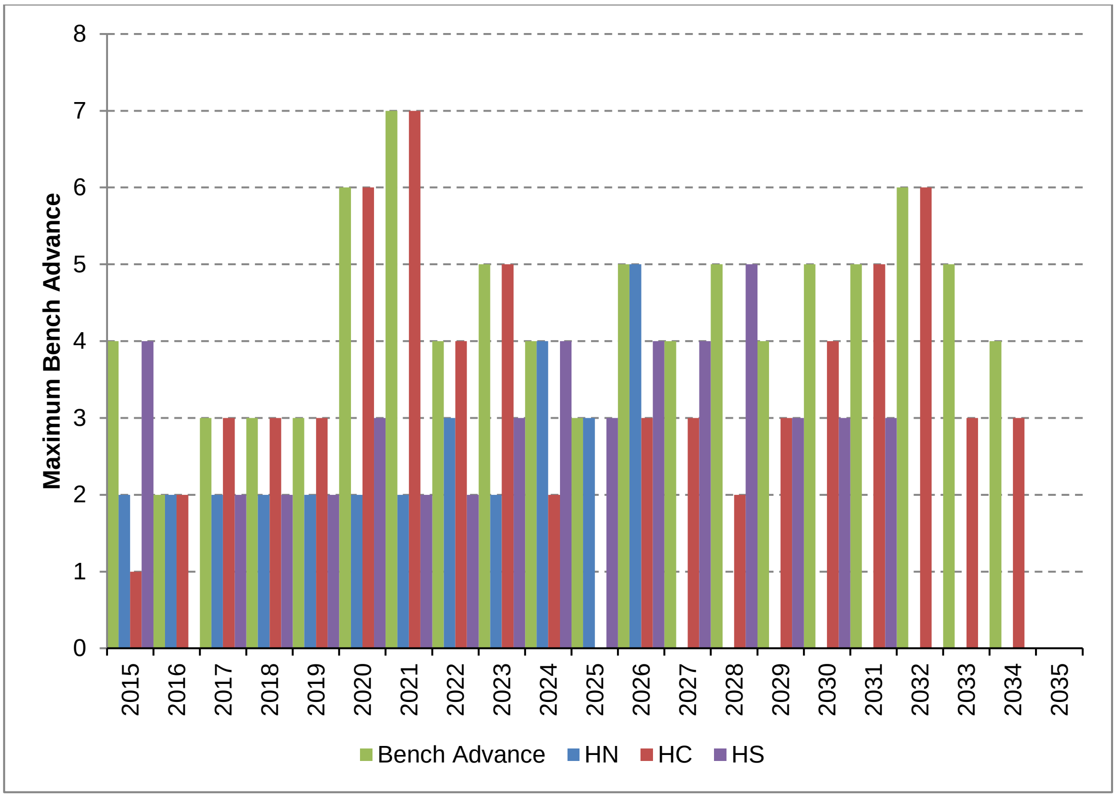


Figure 15-15: Maximum Vertical Advance Rate

15.6.7 Material Movements

Total Material Movement

The total material movement profile (exclusive of RoM re-handle) is shown in Figure 15-16 and the findings are summarised below:

- there are two and a half years of pre-stripping prior to ore production;
- approximately 350 kt of ore is required to be stockpiled for less than a year at the beginning of the schedule to facilitate waste stripping, these are small pockets of ore in mostly waste regions;
- once full ore production is achieved the total material movement profile remains steady until the end of the mine life; and
- overburden material mining has been deferred until the mining area is to be mined.

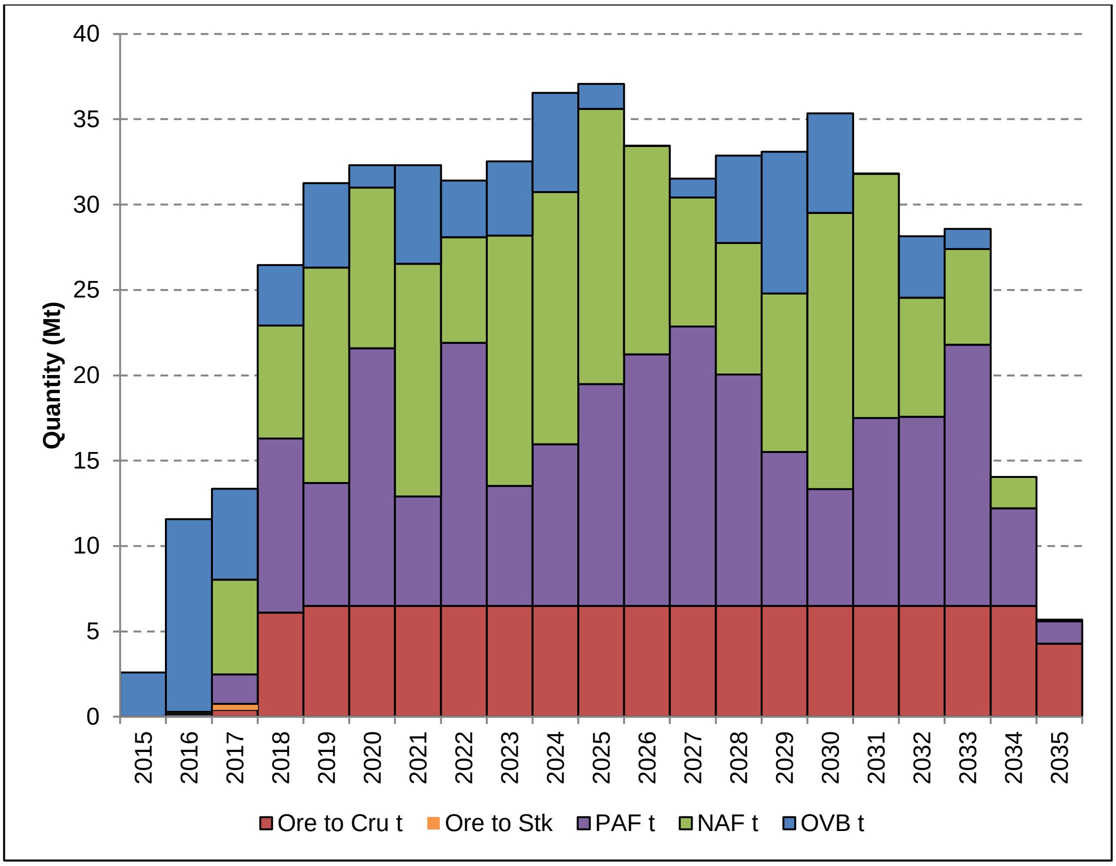


Figure 15-16: Total Material Movement Profile

Total Material Movement by Region

The total material movement (exclusive of RoM re-handle) by region is shown in Figure 15-17 and the findings are summarised below:

- high mining rates are required in the HC region as the pit is developed to expose high grade Fe as early as possible;
- HN and HS are mined in conjunction with HC in the initial years to keep the strip ratio low;
- high mining rates in the HS region are required in the middle of the schedule to access the high grade Fe in the lower benches;
- high mining rates in the HC region at the end of the schedule are due to the final cutback being developed to access high grade Fe to be blended with KU lower grade Fe.

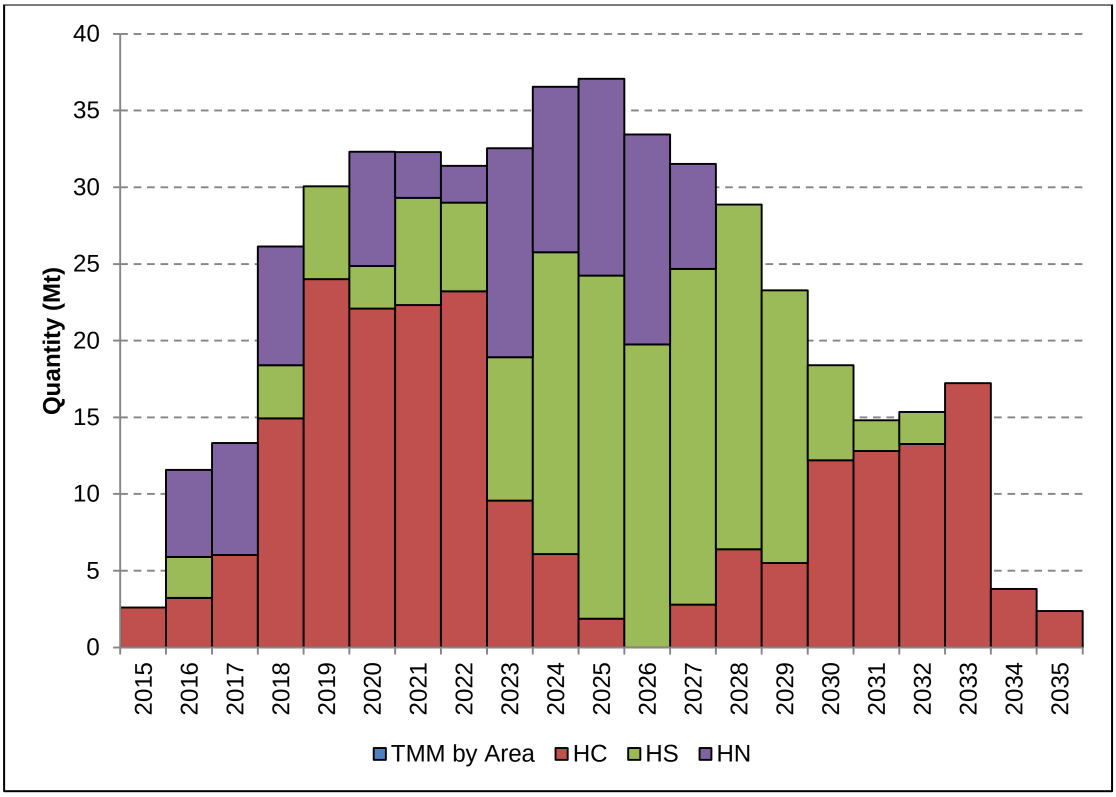


Figure 15-17: Total Material Movement by Region

Ore Movement by Region

The total material movement (exclusive of RoM re-handle) by region is shown in Figure 15-18 and the findings are summarised below:

- HC is the primary source of ore for the initial years;
- HS and HN are used to blend with HC in the initial years to maintain a constant Fe grade and to lower the strip ratio;
- once the initial four stages of HC are completed HS and HC are mined simultaneously as the ore source;
- KU is left until the end of the mine life to the lower Fe grades, mining beginning in 2028 and is mined with the remainder of HS;
- once HS is completed, the final HC stage is mined with KU until the end of the schedule;
- the final HC stage could be mined earlier to increase Fe grades towards the middle to end of the schedule, however should KU be mined by itself at the end of the mine life the Fe grades would be lowered significantly.

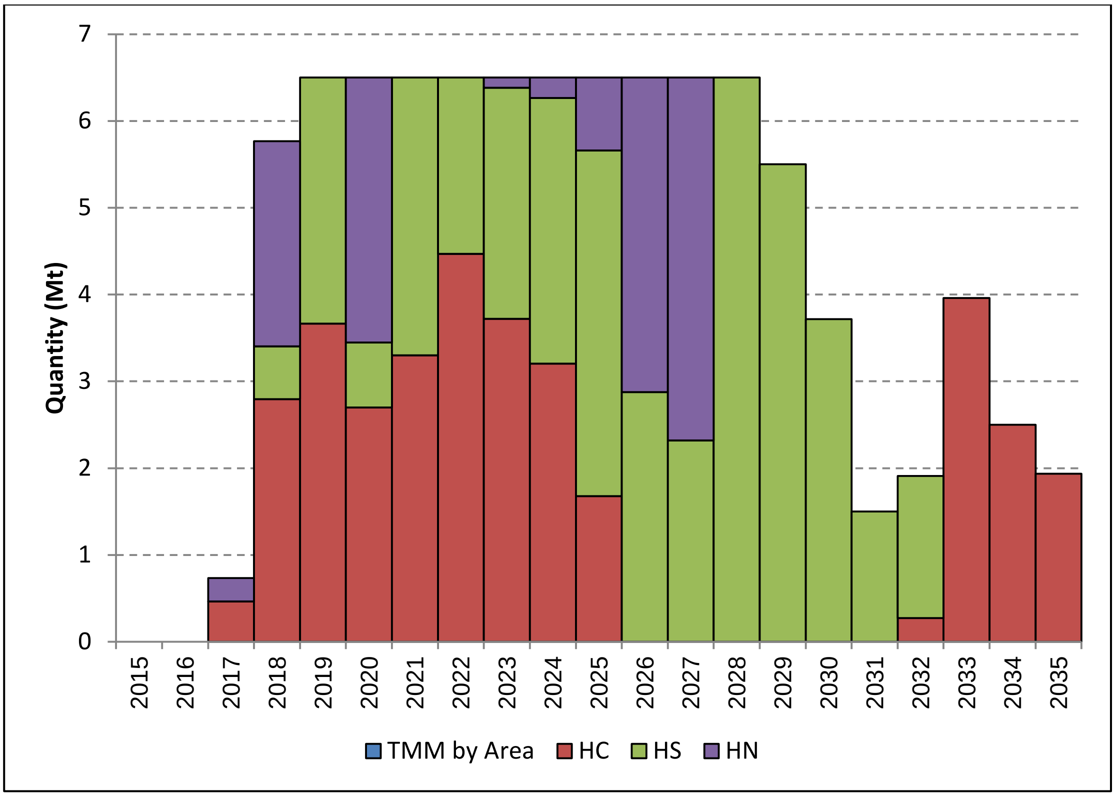


Figure 15-18: Ore Movement by Region

15.6.8 Crusher Feed and Product

Ore Feed by Metallurgical Type

The ore feed is shown by metallurgical type in Figure 15-19 which are detailed in Table 15-26.

Table 15-26: Metallurgical Ore Types

Area	Metallurgical Type	Description
Laurinojo	1	15<=Fe<35 & S<3
Laurinojo	2	35<=Fe<43 & S<3
Laurinojo	3	Fe>=43 & S<3
Laurinojo	4	Fe>=15 & S>=3
Kuervaara	10	Fe>=31
Kuervaara	11	Fe>=15
Lauku	20	Fe>=15
Kivivuopio	30	Fe>=15
Kuervitikko	40	Fe>=15
Other	0	Fe<15

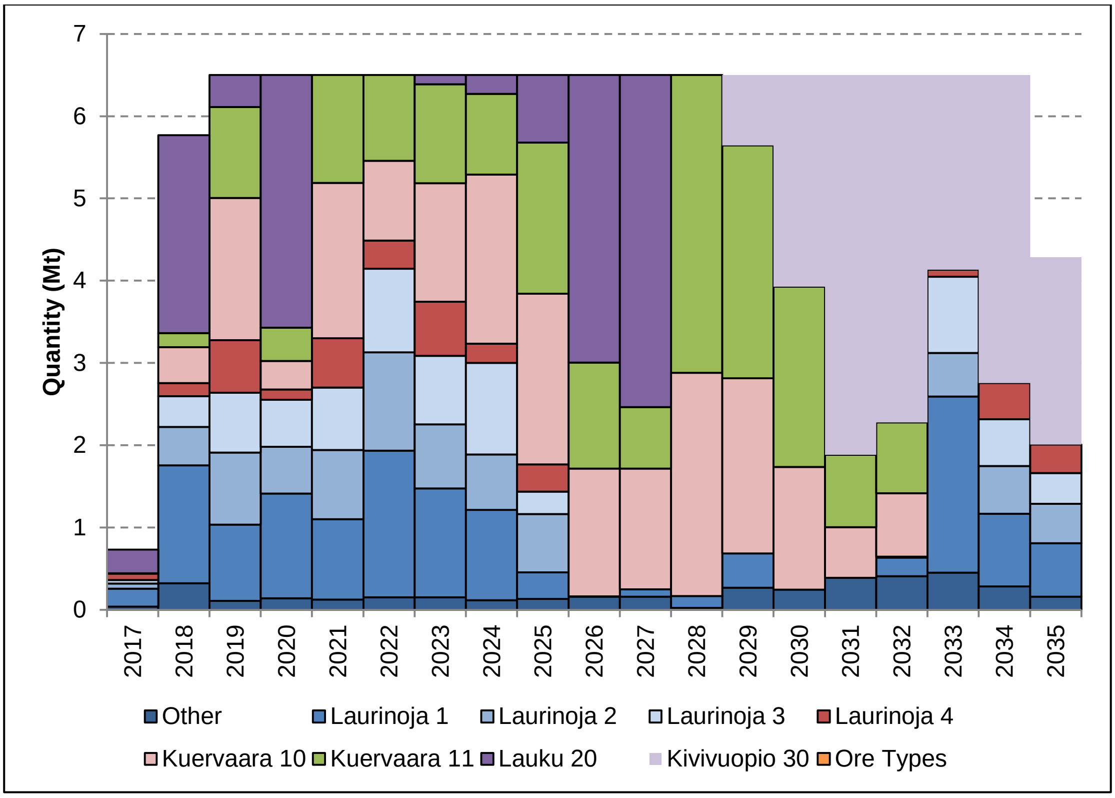


Figure 15-19: Ore Feed by Metallurgical Type

Ore Feed Quality

The ore feed Fe grades are shown in Figure 15-20 along with the S grades. Figure 15-21 shows the Cu and Au grades for the ore feed. The findings for the ore feed quality are summarised below:

- Fe grades ramps up to approximately 32%-34% for the first 10 years and then slowly decrease with the introduction of lower quality ore, especially when KU ore starts to be fed in 2029;
- S grades fluctuate between 2.0%-2.8% depending on the ore source, with the lowest S grades in HC region; and
- Cu and Au grades follow similar trends, with high grades in the HC and gradually decreasing until the KU region begins.

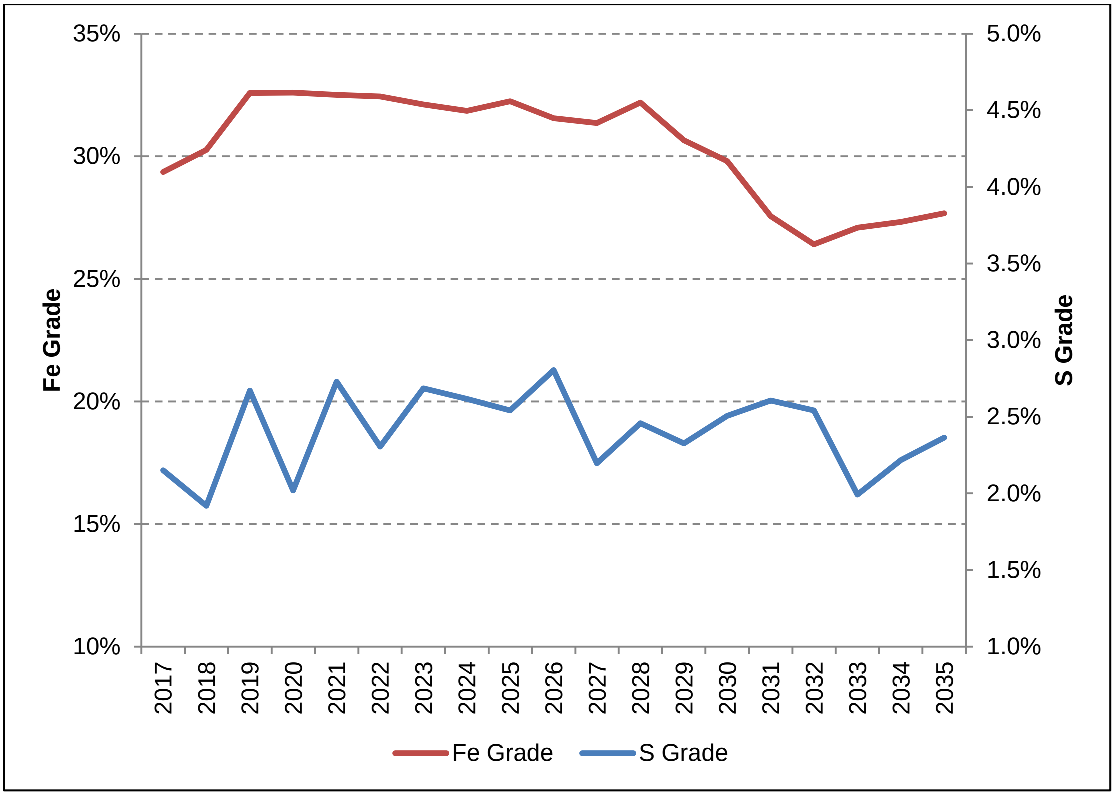


Figure 15-20: Ore Feed Quality – Fe and S Grades

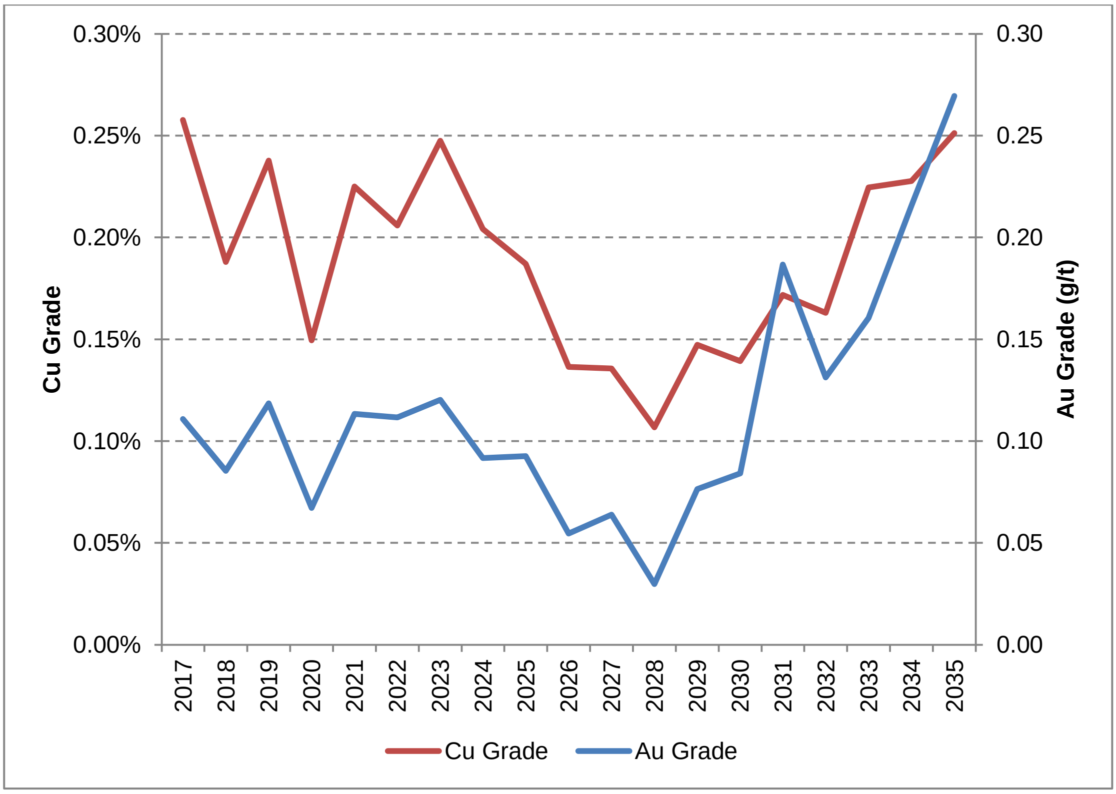


Figure 15-21: Ore Feed Quality – Cu and Au Grades

Product Output

The product output is shown in Figure 15-22 and the findings are summarised below:

- the Fe concentrate ranges between 1.9 to 2.2 Mtpa for the initial 13 years and then drops away as ore feed quality drops; and
- the Cu concentrate fluctuates with the Cu grade in the ore feed and does exceed the threshold of 50 ktpa on a few occasions. However to adjust the Cu throughput would decrease the Fe grade, therefore the increased Cu throughput has not been deccreased, as requested by the Client.

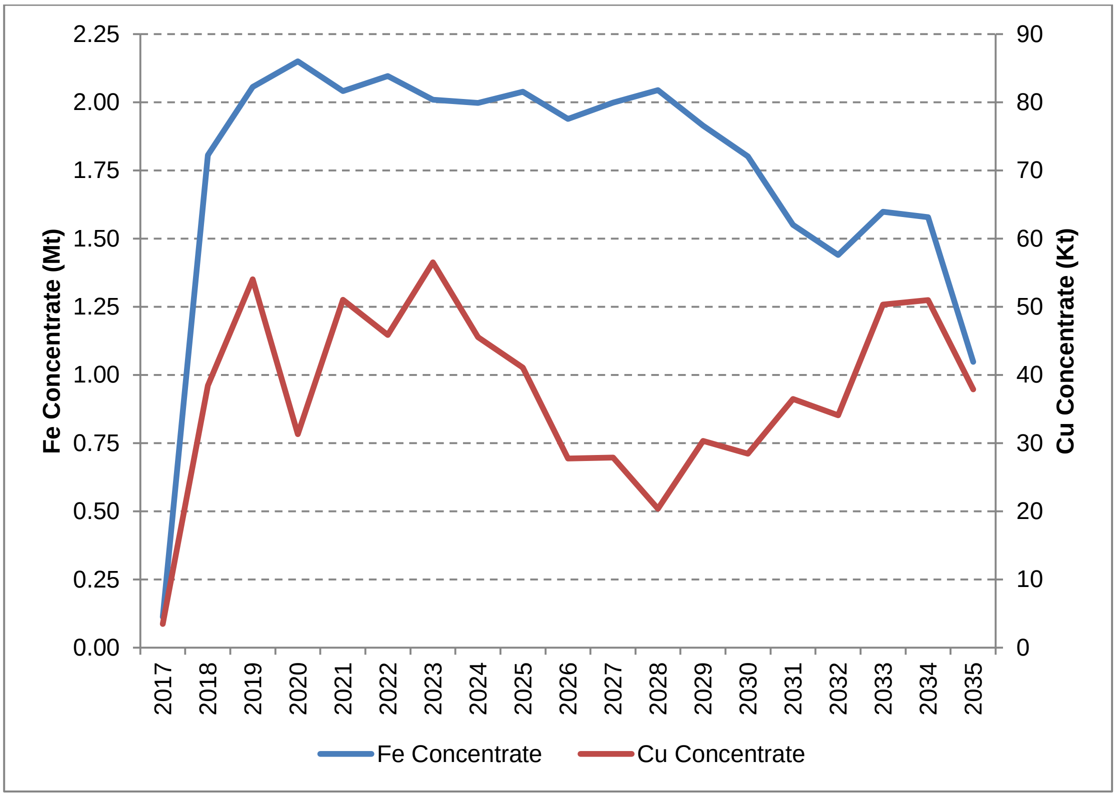


Figure 15-22: Product Output

Product Recovery

The product recoveries are shown in Figure 15-23 and follow the trends in the ore feed grades.

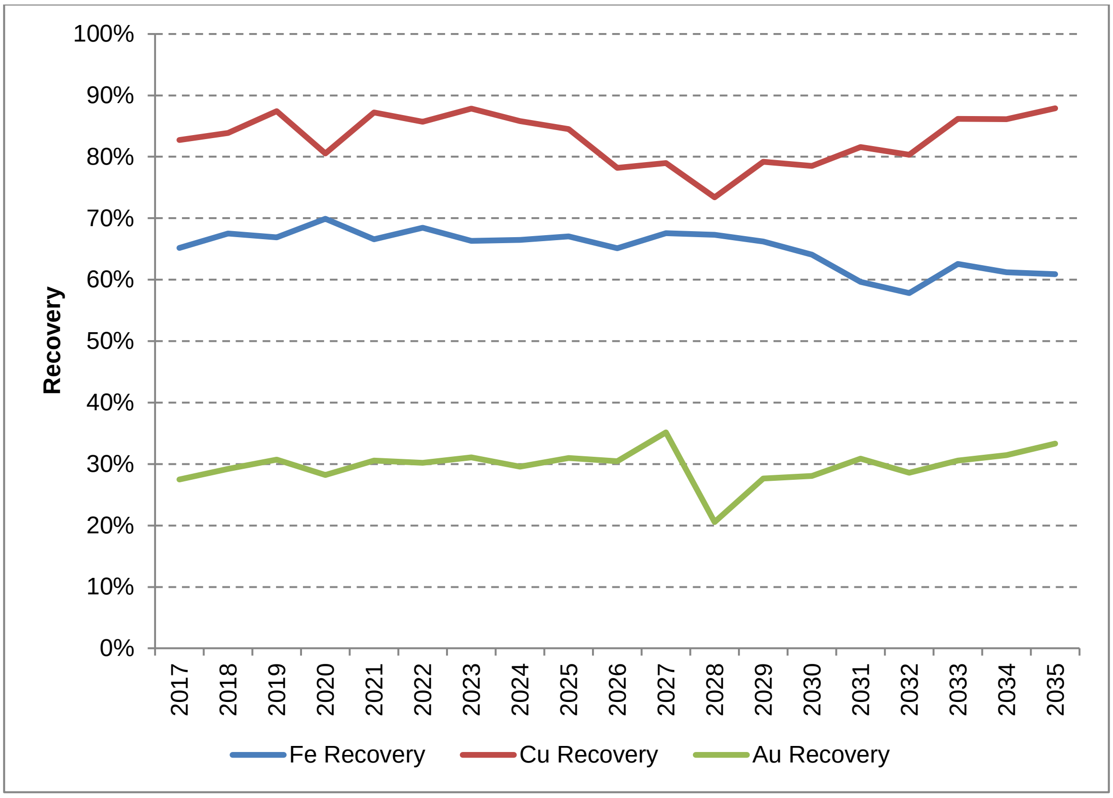


Figure 15-23: Product Recoveries

15.6.9 Topsoil Clearing

A topsoil clearing schedule for the pits and dumps has been developed from the results of the production schedule. It has been assumed that contractors will be used for all topsoil stripping and the stripping will therefore be grouped into three campaigns, each beginning in 2015 Q3, 2019 Q1 and 2020. Table 15-27 shows the topsoil clearing schedule by period.

Table 15-27: Topsoil Clearing Schedule

Pit	Units	Total	2015Q3	2015Q4	2016Q1	2019Q1	2020
HC1	000 m^2	**280**	280				
HC2	000 m^2	**124**	124				
HC3	000 m^2	**105**	105				
HC4	000 m^2	**122**				122	
HC5	000 m^2	**193**					193
HS1	000 m^2	**281**		281			
HS2	000 m^2	**125**				125	
HS3	000 m^2	**146**				146	
HS4	000 m^2	**142**				142	
HN1	000 m^2	**243**			243		
HN2	000 m^2	**297**				297	
KU1	000 m^2	**301**					301
KU2	000 m^2	**149**					149
KU3	000 m^2	**107**					107
Dump	0	**0**	0	0	0	0	0
East OVB	000 m^2	**2,052**	1,132				920
East NAF	000 m^2	**799**		799			
East PAF	000 m^2	**1,635**			1,635		
West OVB	000 m^2	**744**	744				
West NAF	000 m^2	**998**		998			
West PAF	000 m^2	**330**		330			
Total	**000 m^2**	**9,173**	**2,385**	**2,408**	**1,878**	**831**	**1,671**

15.6.10 Equipment Scheduling

The mining equipment requirements have been derived from the production schedule based on the following:

- use of contractors until end of 2016 Q3 for all material movement;
- use of contractors for all topsoil stripping;
- truck haulage for overburden, waste and ore feed from 2016 Q4 onwards;
- waste material movement by 26 m^3 face shovels from 2016 Q4 onwards;
- ore feed and extra waste material movement by 19 m^3 front end loader ("FEL") from 2017 Q2 onwards; and
- surface pit haul roads and crusher location provided by Pöyry Finland Oy.

The loading unit bucket size has been used to denote the class of equipment. The actual bucket capacities used in the equipment productivity analysis have been reduced to match the maximum suspended load ratings for the equipment.

The truck haulage has been evaluated on the basis of a Caterpillar 793F (227 t capacity) truck. The productivity rates for the loading units have been developed using Caterpillar Fleet Production and Cost ("FPC") software.

Loading

The annual loading fleet requirements are shown in Figure 15-24. A maximum of three loading units are required, comprised of one 19 m^3 FEL and two 26 m^3 face shovels:

- the first face shovel is required in 2016 Q4 and the second in 2018 Q1 to strip waste;
- the FEL will be required starting in 2017 Q4 to mine ore and occasionally to mine waste; and
- it has been assumed that all RoM re-handle will be handled by a small FEL and has not been estimated as part of the loading units.

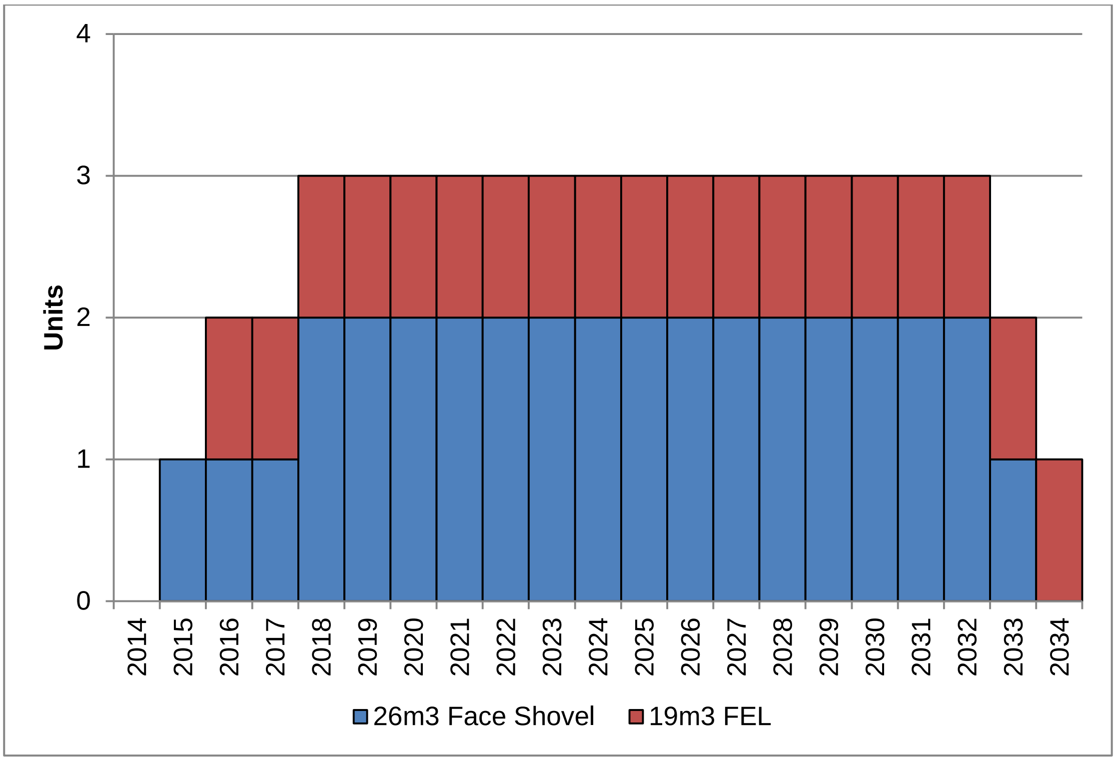


Figure 15-24: Annual Loading Fleet Requirements

Hauling

The annual haulage fleet requirements are shown in Figure 15-25 and the findings are summarised below:

- a maximum of 12 haul trucks are required (year 2025);
- it has been assumed that all RoM re-handle will be handled by a small FEL (load, and carry) and has not been estimated as part of the haulage units; and
- the haulage fleet follows the general trend of the total material movement profile, with an increase in trucks required towards the end of the schedule due to the deepening of the pit and the increasing height of the waste dumps.

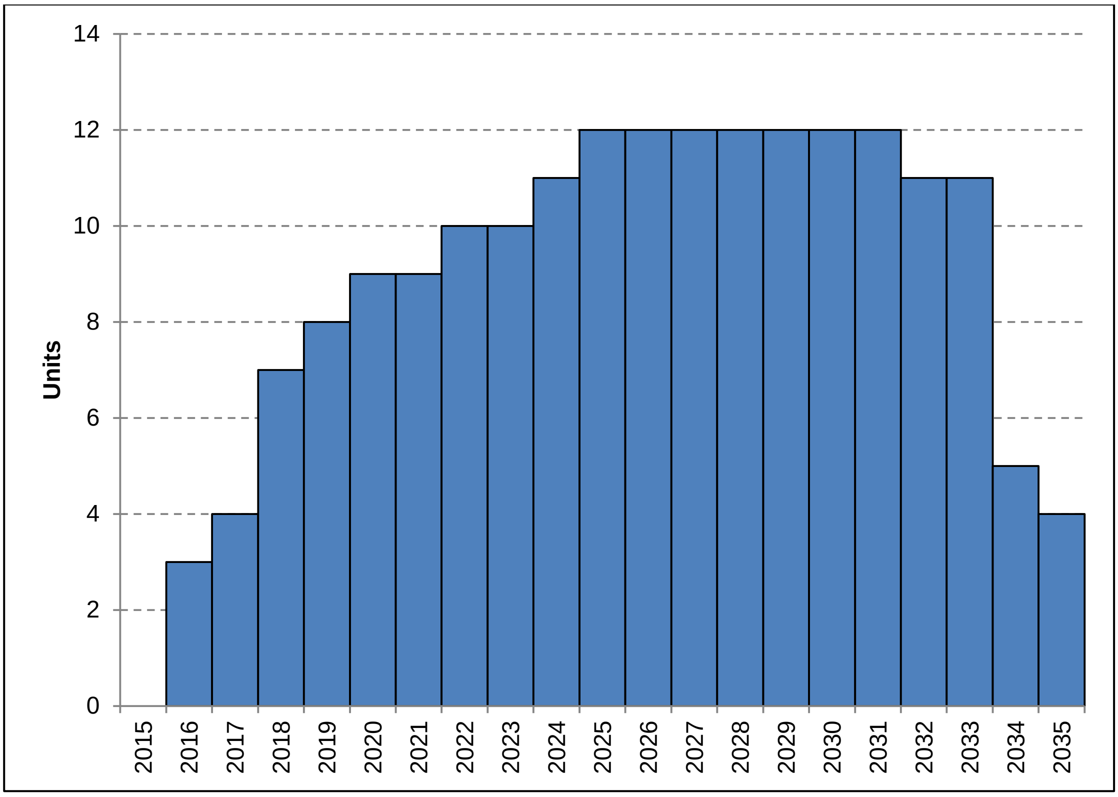


Figure 15-25: Annual Haulage Unit Requirements

The truck productivities and average annual cycle times are shown in Figure 15-26.

The productivities per truck are relatively high at the start of the mine life, due to shorter cycle times. Productivities decrease as the pits become deeper and the haul distance to the waste dump tipping points become longer. There are some spikes in the productivities and dips in the cycle times towards the end of the schedule where there is a majority of KU material being mined, which has shorter cycle times.

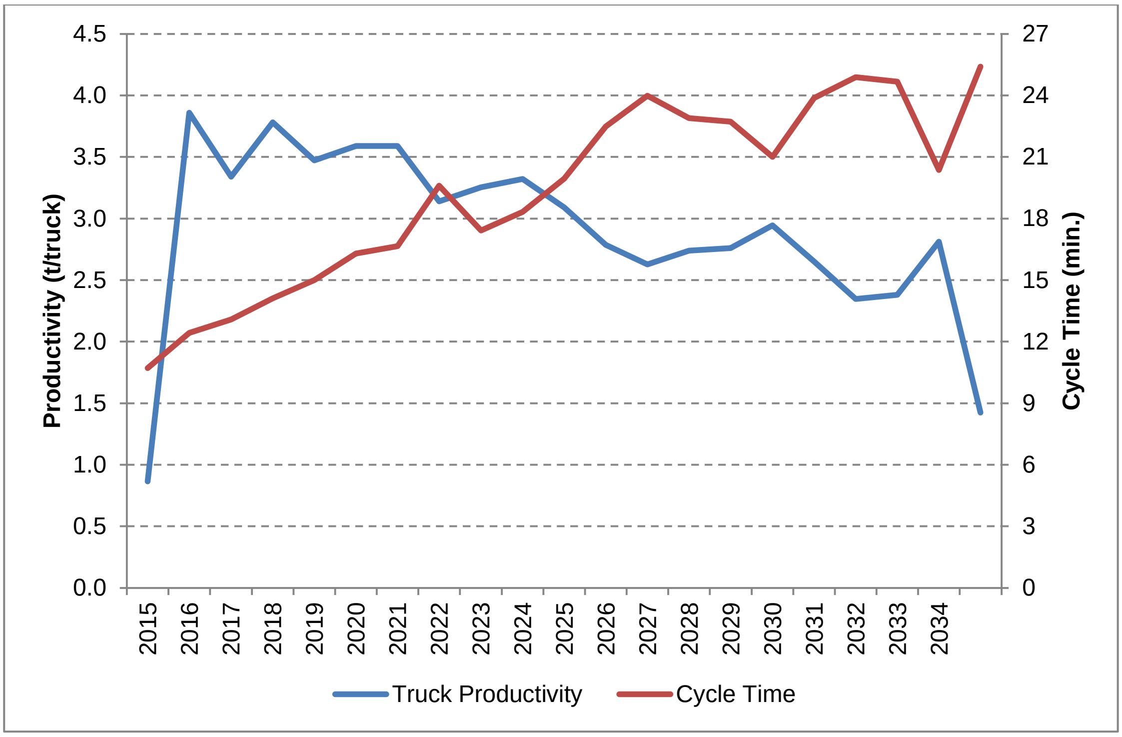


Figure 15-26: Truck Productivities and Cycle Times

15.6.11 Ancillary Equipment

The ancillary equipment estimates are based on the following factors:

- material movement rate;
- labour levels; and
- and number of working areas.

The total equipment estimate is shown in Table 15-28.

Table 15-28: Equipment Schedule

	Max	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Shovel (26 m^3)	**2**	0	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
FEL (15 m^3)	**1**	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Truck (227t)	**12**	0	3	4	7	8	9	9	10	10	11	12	12	12	12	12	12	12	11	11	5	4
Drill (229 mm)	**2**	0	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Drill (172 mm)	**1**	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Track Dozer (580 hp)	**3**	0	1	1	2	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	1	1
Grader (300 hp)	**3**	0	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Wheel Loader (6.5 m^3)	**1**	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
IT (5m^3)	**1**	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Rockbreaker (2.4 m^3)	**1**	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck	**1**	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel/Lube Truck	**1**	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tyre Handler	**1**	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lighting Plant	**9**	0	4	5	8	9	9	9	9	9	9	9	9	9	9	9	9	9	8	8	5	5
Light Vehicle	**18**	7	16	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18
Light Vehicle (inpit only)	**7**	1	3	4	6	6	6	6	7	7	7	7	7	7	7	7	7	7	7	7	4	4

15.6.12 Mining Labour Requirements

Labour Rosters

The Hannukainen labour requirements have been estimated based on three 8 hour shifts, 365 days per year for equipment operators and maintenance operators. A total of five crews are required for 24 hour roles due to labour laws in Finland[2]. Office based roles have been assumed to be five days a week with 8 hour shifts.

Labour Estimate

The labour requirements have been estimated for the mine operations, mine technical services and mine maintenance groups. The labour requirements have been estimated from the outcomes of the production schedule and the equipment levels. The labour estimates have been based on the following criteria according to the position:

- material movement rates;
- equipment fleet levels; and
- number of shifts.

It has been assumed that all material will be mined by contractors until the end of 2016 Q3, therefore minimal mining production personnel are required prior to this date. The mine manager, chief mining engineer, mine surveyor and the water management team are employed prior to production to ensure effective start-up of the operation, contractor management and short term scheduling.

The mining maintenance and mining operation groups are determined through the equipment requirements.

The technical services department, including mining engineers, geologists, surveyors etc, has a slight ramp up in the first two years and then remains quite constant.

It has been assumed that all blasting activities will be carried out by a licensed contractor and therefore only supervisory positions have been assumed for blasting requirements. It has been assumed that maintenance contracts will be used for major equipment and therefore only a small number of maintenance labourers will be used for the remaining equipment. Equipment training will be provided by the equipment manufacturers when the equipment arrives onsite.

The mine personnel requirements by group are shown in Figure 15-27.

[2] *National Labour Law Profile: Republic of Finland, June 17, 2011. International Labour Organization. [online] Available at: <http://www.ilo.org/ifpdial/information-resources/national-labour-law-profiles/WCMS_158896/lang--en/index.htm> [Accessed 09 November 2012].*

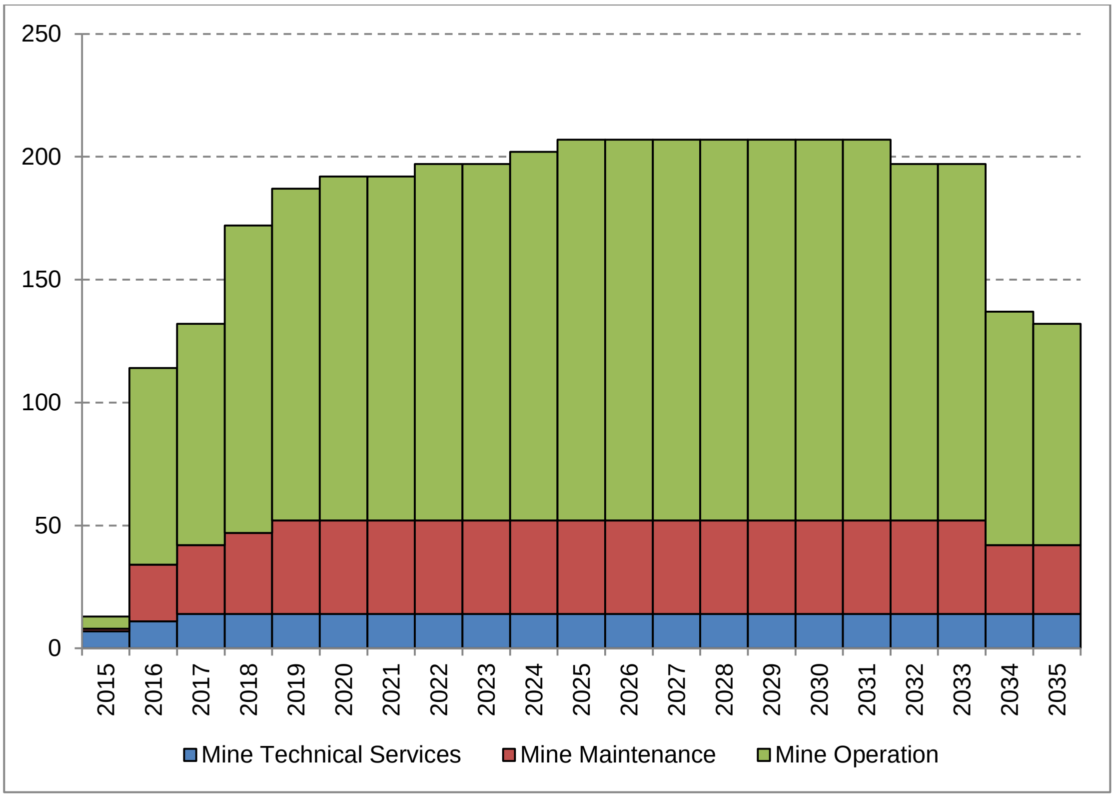


Figure 15-27: Mine Labour Requirements

15.7 Operating Strategy

15.7.1 Objectives

A general operating strategy is required for the equipment selection and estimated productivities in order to justify the production schedule. The objectives of the operating strategy are summarised below:

- justification of a truck and shovel match;
- development of shovel loading productivities;
- development of equipment operating times; and
- development of a drill and blast strategy.

15.7.2 Mining

Mining Fleet Selection

The mining fleet selection has been based on the outcomes of various scheduling scenarios, mineable pit tonnages and annual ore production. Based on initial production schedules, which used a 26 m^3 face shovels and two 19 m^3 FEL with 227 t haul trucks, it was determined that smaller equipment would be more feasible for this Project. The first stages of this study were completed under the assumption that 22 m^3 face shovels and a 19 m^3 FEL would be used with 181 t haul trucks. However, due to a larger pit size and increased annual ore production, the equipment size was increased to 26 m^3 face shovels and a 19 m^3 FEL with 227 t haul trucks at the production schedule stage of the Project. The capital and operating cost estimates have been adjusted accordingly.

It has been assumed that the face shovels will be for waste stripping, due to their limited flexibility. The FEL will be used for ore feed, as it has more manoeuvrability. It will also be used to mine excess waste material as it presents near the ore feed. The loading fleet productivities are shown in Table 15-29 by material type and have been calculated using Caterpillar's FPC software.

Ancillary equipment has been selected to support the loading and hauling fleet and to match a bulk iron ore mining operation based on industry experience.

Although this study has used various makes and models of equipment to develop operating costs, this report does not recommend one particular manufacturer or equipment model over any others. Where specific equipment models or manufacturers have been referred to, it is merely to acknowledge where information has been derived, or to provide the reader with an example of the type of equipment being discussed.

Table 15-29: Equipment Productivities

Material Type	Units	Ore	Waste	OVB	Ore	Waste	OVB
Loading							
Configuration		**6050FS**	**6050FS**	**6050FS**	**994H**	**994H**	**994H**
Bucket Size Required	m^3	18.0	18.0	18.0	15.0	15.0	15.0
Loading Spot Time	min	0.75	0.75	0.75	0.70	0.70	0.70
Loading Cycle Time	min	0.48	0.48	0.48	0.58	0.58	0.58
First Bucket Dump	min	0.05	0.05	0.05	0.10	0.10	0.10
Hauling							
Unit		**CAT793**	**CAT793**	**CAT793**	**CAT793**	**CAT793**	**CAT793**
Dump & Spot Time	min	1.15	1.15	1.15	1.15	1.15	1.15
Capacity	t	227	227	227	227	227	227
Capacity	m^3	176	176	176	176	176	176
Loading Parameters							
Bucket Fill Factor (%)	%	90%	90%	90%	100%	100%	100%
Bank Density	t/m^3	3.62	2.89	2.24	3.62	2.89	2.24
Swell Factor	m^3:m^3	1.40	1.30	1.30	1.40	1.30	1.30
Loose Density	t/m^3	2.58	2.23	1.72	2.58	2.23	1.72
Passes	#	5.4	6.3	8.2	5.9	6.8	8.8
Passes (Int)	#	5.0	6.0	8.0	6.0	7.0	9.0
Loaded Quantity	t	209.3	216.4	222.8	232.5	233.7	232.1
Loaded Quantity	LCM	81.0	97.2	129.6	90.0	105.0	135.0
Loaded Quantity	BCM	57.9	74.8	99.7	64.3	80.8	103.8
Fill (% of Rated Payload)	%	92%	95%	98%	102%	103%	102%
Fill (% of Rated Volume)	%	46%	55%	74%	51%	60%	77%
Cycle Time							
Load Spot Time	min	0.75	0.75	0.75	0.70	0.70	0.70
Load Time	min	1.97	2.45	3.41	3.00	3.58	4.74
Dump and Spot Time	min	1.15	1.15	1.15	1.15	1.15	1.15
Shovel Productivity							
Shovel Operator Efficiency	%	80%	80%	80%	80%	80%	80%
Loading Productivity	t/doh	3,690	3,250	2,570	3,020	2,620	2,050
Loading Productivity	BCM/doh	1,020	1,123	1,150	835	905	917
Loading Productivity	LCM/doh	2,636	2,500	1,977	2,157	2,015	1,577
Shovel Utilisation	%	59.9%	59.9%	59.9%	60.5%	60.5%	60.5%
Loading Productivity	Mtpa	19.4	17.1	13.5	16.0	13.9	10.9
Loading Productivity	kbcmpa	4,282	4,714	4,827	3,541	3,840	3,890

Fleet Operating Time

Fleet operating time estimates have been developed on a first principle approach; building up the availability losses, operating standby and operating delays. The operating time estimate for the major mining equipment is shown in Table 15-30 and is based on three 8 hour shifts / 365 days per year.Table 15-30 provides the operating hours associated with each type of equipment.

Table 15-30: Equipment Operating Time Summary

	Unit	Shovel (26m3)	FEL 15m3)	Truck (227t)	Drill (229mm)	Drill (172mm)	Track Dozer (580hp)	Grader (300hp)	Wheel Loader (6.5m3)	IT (5m3)	Rock-breaker (2.4m3)	Water Truck	Fuel/Lube Truck	Tyre Handler	Lighting Plant	Light Vehicle
Calendar Time																
Annual Working Time	**(h)**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**
Availability	**(%)**	**87.5%**	**87.5%**	**87.5%**	**87.5%**	**87.5%**	**85.4%**	**87.5%**	**87.5%**	**87.5%**	**87.5%**	**87.5%**	**92.2%**	**93.6%**	**95.6%**	**93.6%**
Planned Loss	**(h)**	**910**	**910**	**910**	**910**	**910**	**910**	**910**	**910**	**910**	**910**	**910**	**497**	**378**	**348**	**378**
Scheduled Maintenance	(h/year)	576	576	576	576	576	576	576	576	576	576	576	346	317	288	317
Capital Works	(h/year)	67	67	67	67	67	67	67	67	67	67	67	34			
Cleaning for Scheduled Maintenance	(h/year)	16	16	16	16	16	16	16	16	16	16	16	9	8	7	8
Scheduled Over Runs	(h/year)	8	8	8	8	8	8	8	8	8	8	8	4	1	1	1
Inspections/PM	(h/week)	5	5	5	5	5	5	5	5	5	5	5	2	1	1	1
Breakdown Loss	**(h)**	**183**	**183**	**183**	**183**	**183**	**366**	**183**	**183**	**183**	**183**	**183**	**183**	**183**	**37**	**183**
Waiting for Parts/Labour	(day/year)	1	1	1	1	1	3	1	1	1	1	1	1	1	0	1
Breakdown	(day/year)	1	1	1	1	1	3	1	1	1	1	1	1	1	0	1
Repair Time	(day/year)	5	5	5	5	5	10	5	5	5	5	5	5	5	1	5
Cleaning	(day/year)	0.1	0.1	0.1	0.1	0.1	0.3	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.0	0.1
Available Time																
Use of Availability	**(%)**	**83.4%**	**83.4%**	**83.4%**	**79.9%**	**79.9%**	**75.7%**	**69.1%**	**69.1%**	**69.1%**	**26.3%**	**69.1%**	**50.4%**	**32.2%**	**46.8%**	**32.2%**
Operating Standby	**(h)**	**1,270**	**1,270**	**1,270**	**1,544**	**1,544**	**1,818**	**2,365**	**2,365**	**2,365**	**5,650**	**2,365**	**4,008**	**5,559**	**4,452**	**5,559**
Not manned	(h/shift)				0.25	0.25	0.50	1.00	1.00	1.00	4.00	1.00	2.50	4.00	4.00	4.00
Safety	(h/month)	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00		1.00
Meal Break	(h/shift)	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50		0.50
Other (Blast, etc)	(h/day)	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25			
Shift Change	(h/shift)	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50		0.50
Weather	(day/year)	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00
Utilised Time																
Operating Efficiency	**(%)**	**82.0%**	**82.9%**	**83.8%**	**88.1%**	**88.1%**	**77.5%**	**88.0%**	**81.1%**	**81.1%**	**59.3%**	**84.5%**	**92.2%**	**87.9%**	**93.0%**	**87.9%**
Operating Delays	**(h)**	**1,150**	**1,095**	**1,034**	**730**	**730**	**1,277**	**638**	**1,003**	**1,003**	**821**	**820**	**319**	**319**	**274**	**319**
Travelling/Walking	(h/shift)	0.30	0.25		0.25	0.25	0.75	0.17	0.50	0.50	0.50				0.25	
Face Preparation	(h/shift)	0.25	0.25													
Wait for cleanup	(h/shift)															
Scaling	(h/week)															
Wait for Truck	(h/shift)	0.25	0.25													
Pre-Start Checks	(h/shift)	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25		0.25
Wait for Shovel	(h/shift)			0.33												
Wait at Dump/Crusher	(h/shift)			0.25												
Fuelling/Water	(h/week)			2.33	3.50	3.50	3.50	3.50	3.50	3.50		10.50	0.88	0.88		0.88
Operating Time																
Effective Utilisation	**(%)**	**59.9%**	**60.5%**	**61.2%**	**61.6%**	**61.6%**	**50.1%**	**53.2%**	**49.1%**	**49.1%**	**13.6%**	**51.2%**	**42.8%**	**26.5%**	**41.7%**	**26.5%**
Direct Operating Time	**(h)**	**5,247**	**5,301**	**5,363**	**5,393**	**5,393**	**4,389**	**4,663**	**4,298**	**4,298**	**1,195**	**4,482**	**3,753**	**2,321**	**3,650**	**2,321**

Material Handling

The mining operation consists of a conventional load and haul operation with material hauled to waste dumps, ROM stockpile or directly tipped at the crusher. The overall layout of the operation including pits, waste dumps, haul roads, RoM stockpile and crusher is shown in Figure 15-28.

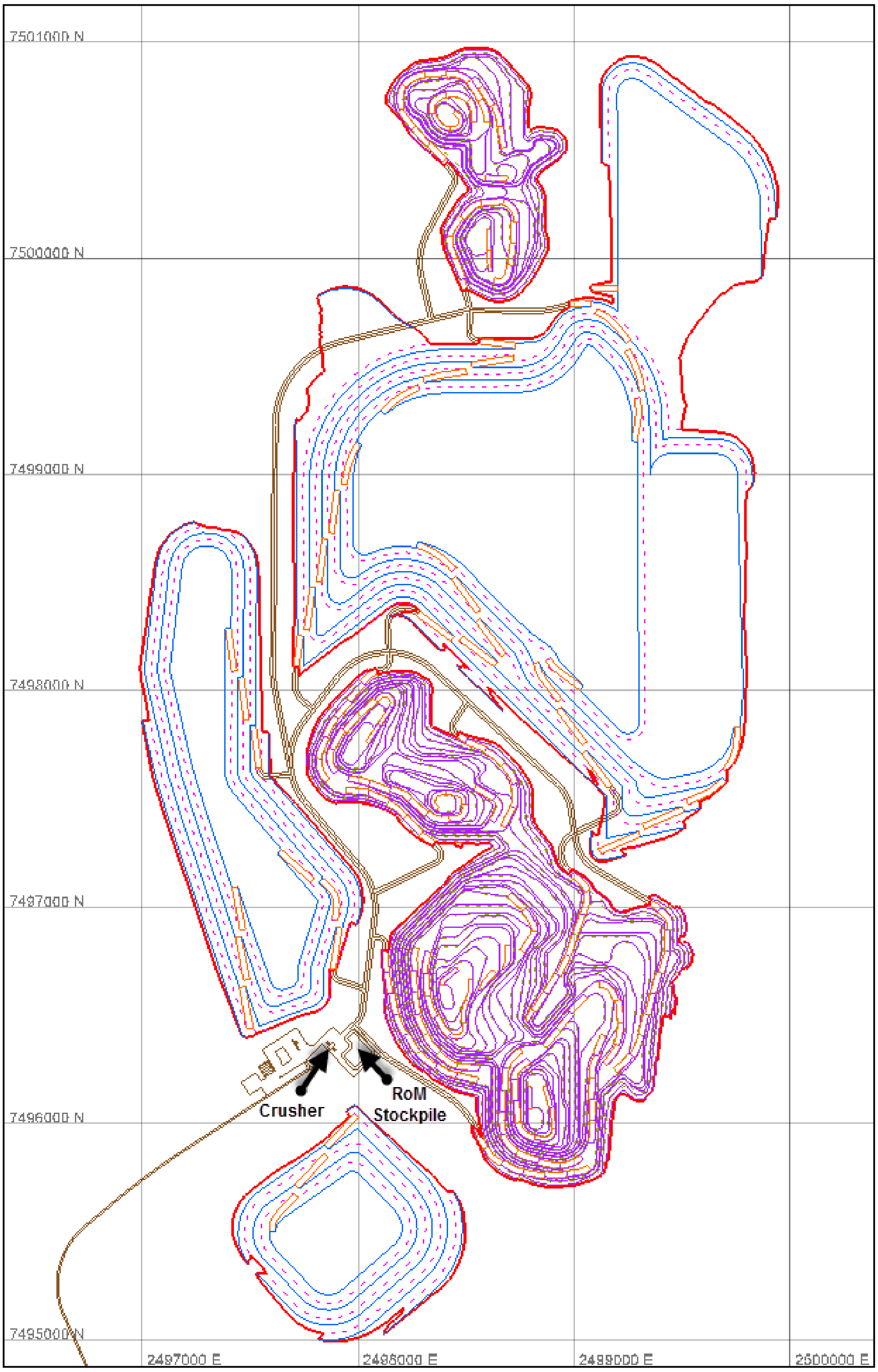


Figure 15-28: General Arrangement for Mining and Infrastructure

Mining Method

The mining method will vary between the material types and loading units. The proposed methods are summarised below:

- waste material (OVB, PAF and NAF) will be mined with 22 m^3 face shovels on 10 m benches;
- ore will be mined with the 19 m^3 FEL in 5 m flitches with track dozer support on the footwall to help achieve the ore-waste contact with minimal dilution and ore loss;
- the 19 m^3 FEL will be used for any excess waste movement, mining full 10 m benches, where possible; and
- excess ore material that cannot be directly fed to the crusher will be stockpiled short term in the RoM stockpile, this material will be fed to the crusher by a small loader when required.

Excavatability

The overburden material is mineable with free digging with no drill and blast required. The overburden material will be mined with a contractor fleet. Drill and blast is required for the fresh material (ore, PAF and NAF).

Mine Dispatch System

The Hannukainen fleet will require a mine dispatch system to manage the equipment and for recording and reporting purposes. SRK has incorporated this system into the mining cost estimate.

15.7.3 Drill and Blast

Drilling Requirements

The drill and blast estimate is based on the production schedule ex-pit material movement and material types. Drill and blast activities are not anticipated for the overburden material but are required for the ore, PAF and NAF. The drilling requirements have been split into three categories:

- production drilling;
- trim (wall control) drilling; and
- in-fill drilling.

Production drilling will be undertaken on a 6.0 m by 5.2 m spacing on 10 m benches with a 229 mm blasthole diameter. The 229 mm blasthole drill accounts for the majority of the drilling production.

Trim drilling will be undertaken with 5.0 m by 4.3 m spacing on 10 m benches with a 172 mm blasthole. The 172 mm blasthole drill will be used for all secondary blasting requirements.

Grade control sampling will be conducted on all production and trim holes in ore and 10% of the waste to establish the ore-waste contact.

In-fill drilling is required in all material types to provide information for planning and grade control purposes. The in-fill drilling will be undertaken on 25 x 25 m spacing on 10 m benches with the 229 mm blasthole drill. All ore material will be in-fill drilled along with 30% of the waste to account for the ore-waste boundary.

The drilling requirements are shown in Table 15-43. SRK recommends that estimated penetration rates are benchmarked with operating information and validated by the equipment suppliers.

15.8 Mining Operating Costs

The mining cost estimate has been developed to a HFS level of confidence. The mining equipment capital costs for the Project have been sourced from the manufacturers. A sensitivity analysis has been conducted on the critical cost components.

The cost estimate has been calculated in USD. Due to the location of the Project some quotes have been provided in other currencies. The exchange rates used in this cost estimate have been provided by Northland and are shown in Table 15-31.

Table 15-31: Exchange Rates

Currency	Units	Exchange Rate
SEK	USD	0.14
EUR	USD	1.28
GBP	USD	1.59

15.8.1 Equipment Operating Cost Estimate

The mining equipment operating costs for the Project have been sourced from the manufacturers and were provided by Arundon. The mining equipment cost estimates have been developed using the unit cost data shown in Table 15-32.

Table 15-32: Supply Quotes

Quotes	Units	Cost
Diesel	EUR/l	0.809
Lube	EUR/l	2.62
Power	EUR/kWh	0.051

Table 15-33 shows the manufacturer quotes used in the operating cost estimate. The equipment unit operating costs are shown inTable 15-34. All operating costs were provided in EUR except for drill wear parts in SEK and the FEL tyre costs in USD.

Table 15-33: Manufacturer Quote Sources

Equipment	Manufacturer	Model/Type	Description
Shovel (26 m^3)	CAT	6050	Electric 18 m^3
FEL (15 m^3)	CAT	994F	15 m^3
Truck (227 t)	CAT	793F	227 t
Track Dozer (580 hp)	CAT	D10T	580 hp
Grader (300 hp)	CAT	16M	16 ft
Drill (229 mm)	AtlasCopco	PV271	
Drill (172 mm)	AtlasCopco	ROC D65LF	
Wheel Loader (6.5 m^3)	CAT	988H	6.5 m^3
IT (5 m^3)	CAT	980H	5 m^3
Water Truck	CAT	773	Includes tank
Fuel/Lube Truck	CAT	740	
Rockbreaker (2.4 m^3)	CAT	336E	Includes CAT140 hammer
Tyre Handler	CAT	988	used
Light Vehicle	Volkswagon	Amarok DC Trendline	5 seat pick-up
Lighting Plant	NA	NA	2012/2013 Costmine data

Table 15-34: Equipment Unit Operating Costs

Equipment	Total	Maint.	Wear Parts	Tyre Cost	Fuel	Electricity	Lube
	USD/h	USD/h	USD/h	USD/h	USD/h	USD/h	USD/h
Shovel (26 m^3)	**719.3**	254.7	371.2		0.0	92.2	1.2
FEL (15 m^3)	**388.3**	140.8	59.8	24.4	160.5		2.8
Truck (227 t)	**368.2**	134.4	5.1	55.5	170.9		2.4
Drill (229 mm)	**224.3**	123.7	73.3		0.0	27.3	-[3]
Drill (172 mm)	**223.0**	74.6	64.5		83.9		-[4]
Track Dozer (580 hp)	**150.4**	56.3	21.1		72.5		0.5
Grader (300 hp)	**65.3**	34.6	5.6	2.0	22.8		0.3
Wheel Loader (6.5 m^3)	**122.9**	44.8		25.9	51.8		0.5
IT (5 m^3)	**75.0**	37.1		11.6	25.9		0.3
Rockbreaker (2.4 m^3)	**0.0**	32.0	3.8		0.0		0.0
Water Truck	**73.0**	43.5			36.2		0.9
Fuel/Lube Truck	**117.4**	19.2		25.8	46.6		1.4
Tyre Handler	**45.9**	44.8		0.5	25.9		0.3
Lighting Plant	**82.1**	0.9		0.5	36.2		0.5
Light Vehicle	**2.0**	3.0			1.6		0.1

15.8.2 Labour

The labour rates for the cost estimate have been provided by the Client in EUR and are inclusive of on-costs. The annual salaries are shown in Table 15-35.

[3] *Lube costs are included in the Atlas Copco maintenance cost.*
[4] *Lube costs are included in the Atlas Copco maintenance cost.*

Table 15-35: Salary Rates

	Salary USD/y
Mine Site - Mining Operations	
Mine Manager	189,485
Superintendents	126,974
Supervisors	85,952
Trainer	62,511
Equipment Operator	68,371
Crusher Operator	62,511
Dewatering Crew	54,697
Mine Site - Mining Maintenance	
Maintenance Superintendent	126,974
Maintenance Supervisor	85,952
Maintenance Planner	62,511
Maintenance Crew	58,604
Mine Site - Technical Services	
Chief Engineer/Geologist	136,743
Senior Engineer/Geologist	117,207
Planning Engineer	84,068
Mine Surveyor	72,278
Mine Geologist	85,952
IT Geologist	83,998
Geology Technicians	62,511
Consultant (Hydro, Geotech, etc)	83,998
Administrative Assistant	58,604

15.8.3 Blasting

It has been assumed that all charging and blasting activities will be provided by a contractor, who will provide a bulk emulsion product and deliver it into the drillholes. The unit costs shown in Table 15-36 have been used for the blasting cost estimate and were provided in EUR.

Table 15-36: Blasting Unit Costs

Cost Parameter	Units	Cost
Bulk Emulsion	USD/t	602
Primer	USD/unit	2.80
Detonator	USD/unit	2.96
Surface Delay	USD/unit	2.40

15.8.4 Dewatering

The dewatering operating costs have been provided by SRK. The average power and fuel consumptions are shown in Table 15-37. The dewatering operating costs are shown in Table 15-38.

Table 15-37: Average Dewatering Power and Fuel Consumption

	Units	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Power Consumption																						
Hannukainen																						
South	KWh/day	101	67	54	49	38	37	41	42	45	46	99	112	124	135	139	202	200	199	194	192	192
Central	KWh/day	101	56	83	162	157	152	153	163	164	161	223	214	196	183	178	176	175	182	197	193	209
Satellite 1	KWh/day			43	44	41	47	32	40	38	36	33	36	63	61	61	60	60	60	59	59	59
Satellite 2	KWh/day								22	25	29	47	47	62	62	62	62	62	61	61	61	61
Collection Sump	KWh/day								24	25	26	26	27	28	27	27	27	27	27	27	26	26
Kuervitikko																						
South	KWh/day															117	120	77	37	83	129	133
Central	KWh/day																		18	16	11	5
North	KWh/day																	112	115	211	208	204
Collection Sump	KWh/day																		245	234	199	160
Diesel Consumption																						
Pre- Mining	**L/h**	20																				

Table 15-38: Dewatering Operating Costs

	Units	Total	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Maintenance Spares																							
Hannukainen																							
South	USDk	**283**	14.5	7.2	7.2	7.2	7.2	7.2	7.8	7.8	7.8	8.5	12.8	13.4	14.0	14.3	15.9	20.2	21.1	22.1	22.1	22.1	22.1
Central	USDk	**338**	13.9	6.9	6.9	10.4	13.1	13.7	14.0	14.0	15.0	15.0	19.0	19.0	19.0	19.0	19.0	19.6	19.6	19.6	19.6	20.2	21.1
Satellite 1	USDk	**153**			4.8	4.8	4.8	5.7	5.7	6.3	6.3	6.3	6.3	6.3	10.4	10.4	10.4	10.4	10.4	11.0	11.0	11.0	11.0
Satellite 2	USDk	**126**								4.2	4.2	4.2	7.0	7.0	10.7	10.7	10.7	10.7	10.7	11.3	11.3	11.3	11.3
Collection Sump	USDk	**87**								5.6	5.6	5.6	5.6	5.6	6.2	6.2	6.2	6.2	6.2	6.8	6.8	6.8	6.8
Kuervitikko																							
South	USDk	**51**															6.6	6.6	6.6	6.9	6.9	8.8	8.8
Central	USDk	**9**																		2.3	2.3	2.3	2.3
North	USDk	**57**																	8.2	8.8	13.4	13.4	13.4
Collection Sump	USDk	**20**																		5.1	5.1	5.1	5.1
Power Cost																							
Hannukainen																							
South	USDk	**1,248**	28.8	38.1	30.8	27.8	21.4	21.1	5.9	5.9	25.6	26.2	56.5	63.8	70.7	76.5	79.1	114.7	113.5	113.0	110.3	109.4	109.1
Central	USDk	**1,819**	35.1	31.7	47.3	92.0	89.2	86.3	21.8	23.2	93.2	91.5	126.5	121.9	111.5	104.2	101.1	99.8	99.2	103.2	111.8	109.5	118.8
Satellite 1	USDk	**500**			24.3	25.1	23.5	26.5	4.6	5.7	21.8	20.5	18.6	20.6	36.0	34.8	34.5	34.4	34.3	34.1	33.8	33.6	33.4
Satellite 2	USDk	**403**								3.1	14.5	16.7	26.6	26.9	35.4	35.2	35.1	35.1	35.0	34.9	34.8	34.7	34.8
Collection Sump	USDk	**200**								3.3	14.2	14.9	14.8	15.5	15.7	15.4	15.3	15.3	15.2	15.2	15.1	15.0	15.0
Kuervitikko																							
South	USDk	**395**															66.6	68.4	43.7	21.1	47.1	73.1	75.4
Central	USDk	**28**																		10.0	9.0	6.1	2.7
North	USDk	**483**																	63.8	65.2	120.0	118.1	116.2
Collection Sump	USDk	**476**																		139.2	132.9	113.1	90.7
Diesel Cost																							
Total	USDk	**45**	45.4																				
Total Cost	**USDk**	**6,721.0**	**137.6**	**84.0**	**121.4**	**167.3**	**159.3**	**160.6**	**59.8**	**79.2**	**208.1**	**209.3**	**293.6**	**300.1**	**329.6**	**326.8**	**400.5**	**441.4**	**487.7**	**629.9**	**713.4**	**713.6**	**698.0**

15.8.5 Miscellaneous

Miscellaneous costs include: mining software and dispatch system. These costs have been provided by Arundon and are shown in Table 15-39.

Table 15-39: Miscellaneous Operating Costs

	Units	Cost
Mining Software Package		
Maintenance	% of unit cost	1.5
Dispatch System		
Maintenance	USD/y	128,000

15.8.6 Operating Cost Estimate

The average unit operating costs are shown in Table 15-40. The total annual operating costs are shown in Table 15-41. Labour, diesel and maintenance are the largest contributors to the operating costs by category. By function, hauling is the largest contributor to the operating costs, which increases throughout the mine life due to increasing haulage distances.

Table 15-40: Average Unit Operating Costs

Unit Operating Costs	Units	Cost
Unit Operating Costs by Category		
Labour	USD/t moved	0.46
Diesel	USD/t moved	0.46
Electricity	USD/t moved	0.03
Oil/Lubricants	USD/t moved	0.01
Tyres	USD/t moved	0.13
Wear Parts	USD/t moved	0.17
Maintenance	USD/t moved	0.49
Blasting	USD/t moved	0.24
Dewatering	USD/t moved	0.01
Miscellaneous	USD/t moved	0.01
Total	**USD/t moved**	**2.00**
Unit Operating Costs by Equipment		
Labour	USD/t moved	0.46
Loading	USD/t moved	0.27
Hauling	USD/t moved	0.75
Drilling	USD/t moved	0.08
Blasting	USD/t moved	0.24
Ancillary Equipment	USD/t moved	0.18
Dewatering	USD/t moved	0.01
Miscellaneous	USD/t moved	0.01
Total	**USD/t moved**	**2.00**

Table 15-41: Operating Cost Summary

	Units	Total	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Total Operating Costs by Category																							
Labour	**USDm**	**259.7**	0.5	4.2	9.0	12.4	13.3	13.7	13.7	14.0	14.0	14.4	14.7	14.7	14.7	14.7	14.7	14.7	14.7	14.0	14.0	10.0	9.7
Diesel	**USDm**	**256.4**	0.0	1.2	5.0	9.8	12.0	13.0	13.4	14.5	14.1	15.8	16.9	16.7	16.5	16.6	16.6	16.7	16.7	14.9	15.0	7.3	3.9
Electricity	**USDm**	**19.5**	0.0	0.1	0.6	1.0	1.2	1.2	1.1	1.1	1.0	1.3	1.3	1.2	1.1	1.2	1.2	1.3	1.1	1.0	1.0	0.4	0.1
Oil/Lubricants	**USDm**	**4.7**	0.0	0.0	0.1	0.2	0.2	0.2	0.2	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.2	0.1
Tyres	**USDm**	**72.4**	0.0	0.3	1.5	2.7	3.3	3.6	3.7	4.0	3.9	4.4	4.8	4.7	4.7	4.7	4.7	4.7	4.7	4.4	4.4	2.1	1.1
Wear Parts	**USDm**	**93.7**	0.0	0.7	2.7	4.5	5.6	5.6	5.4	5.2	5.2	6.0	6.1	5.6	5.5	5.7	5.8	6.0	5.4	5.0	5.0	2.0	0.7
Maintenance	**USDm**	**272.9**	0.0	1.5	6.1	11.3	13.7	14.5	14.6	15.4	14.9	16.9	17.9	17.4	17.1	17.3	17.3	17.6	17.2	15.5	15.6	7.5	3.7
Blasting	**USDm**	**135.6**	0.0	2.9	5.1	9.7	10.4	11.3	6.4	6.7	6.7	7.2	8.1	7.7	7.2	6.6	6.1	7.0	7.4	6.0	6.6	4.0	2.4
Dewatering	**USDm**	**6.7**	0.1	0.1	0.1	0.2	0.2	0.2	0.1	0.1	0.2	0.2	0.3	0.3	0.3	0.3	0.4	0.4	0.5	0.6	0.7	0.7	0.7
Miscellaneous	**USDm**	**3.4**	0.0	0.1	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
Total	**USDm**	**1,125.1**	**0.7**	**11.1**	**30.4**	**51.9**	**60.0**	**63.5**	**58.8**	**61.5**	**60.5**	**66.6**	**70.5**	**68.8**	**67.5**	**67.6**	**67.2**	**68.9**	**68.1**	**61.9**	**62.8**	**34.3**	**22.6**
Total Operating Costs by Equipment																							
Labour	**USDm**	**259.7**	0.5	4.2	9.0	12.4	13.3	13.7	13.7	14.0	14.0	14.4	14.7	14.7	14.7	14.7	14.7	14.7	14.7	14.0	14.0	10.0	9.7
Loading	**USDm**	**149.3**	0.0	1.1	4.3	7.2	8.8	8.6	8.9	8.4	8.7	10.0	9.7	8.7	8.3	9.1	9.5	9.8	8.2	7.7	7.6	3.4	1.2
Hauling	**USDm**	**421.4**	0.0	1.6	7.2	15.1	18.8	20.8	21.1	23.5	22.1	25.9	28.2	28.2	28.1	28.1	28.1	28.1	28.2	25.9	26.1	10.9	5.3
Drilling	**USDm**	**45.3**	0.0	0.0	0.7	2.1	2.5	2.9	2.5	2.6	2.6	2.8	3.3	3.1	2.8	2.6	2.3	2.7	2.9	2.3	2.6	1.4	0.6
Blasting	**USDm**	**135.6**	0.0	2.9	5.1	9.7	10.4	11.3	6.4	6.7	6.7	7.2	8.1	7.7	7.2	6.6	6.1	7.0	7.4	6.0	6.6	4.0	2.4
Ancillary Equip.	**USDm**	**103.6**	0.0	1.1	3.8	5.1	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.1	5.1	3.8	2.5
Dewatering	**USDm**	**6.7**	0.1	0.1	0.1	0.2	0.2	0.2	0.1	0.1	0.2	0.2	0.3	0.3	0.3	0.3	0.4	0.4	0.5	0.6	0.7	0.7	0.7
Miscellaneous	**USDm**	**3.4**	0.0	0.1	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
Total	**USDm**	**1,125.1**	**0.7**	**11.1**	**30.4**	**51.9**	**60.0**	**63.5**	**58.8**	**61.5**	**60.5**	**66.6**	**70.5**	**68.8**	**67.5**	**67.6**	**67.2**	**68.9**	**68.1**	**61.9**	**62.8**	**34.3**	**22.6**

15.8.7 Operating Cost Sensitivity

A sensitivity analysis has been undertaken on the diesel price, labour rates, maintenance costs and haulage costs. The sensitivity results are shown in Table 15-42 and Figure 15-29. The diesel price in Figure 15-29 (red line) is behind the blue line (Labour Rates) and can be difficult to see. The operating cost estimate is most sensitive to changes in haulage costs.

Table 15-42: Operating Cost Sensitivity Analysis

	Units	60%	70%	80%	90%	100%	110%	120%	130%	140%
Diesel Price	EUR/L	0.49	0.57	0.65	0.73	0.81	0.89	0.97	1.05	1.13
Diesel Unit Cost	USD/t	0.27	0.32	0.36	0.41	0.46	0.50	0.55	0.60	0.64
Total Operating Cost	USD/t	1.82	1.86	1.91	1.96	2.00	2.05	2.09	2.14	2.19
Labour Rates	% Change	-40%	-30%	-20%	-10%	0%	10%	20%	30%	40%
Labour Unit Cost	USD/t	0.28	0.32	0.37	0.42	0.46	0.51	0.55	0.60	0.65
Total Operating Cost	USD/t	1.82	1.86	1.91	1.96	2.00	2.05	2.09	2.14	2.19
Maintenance Costs	% Change	-40%	-30%	-20%	-10%	0%	10%	20%	30%	40%
Maintenance Unit Cost	USD/t	0.29	0.34	0.39	0.44	0.49	0.53	0.58	0.63	0.68
Total Operating Cost	USD/t	1.81	1.86	1.91	1.95	2.00	2.05	2.10	2.15	2.20
Haulage Costs	% Change	-40%	-30%	-20%	-10%	0%	10%	20%	30%	40%
Haulage Unit Cost	USD/t	0.45	0.52	0.60	0.67	0.75	0.82	0.90	0.97	1.05
Total Operating Cost	USD/t	1.70	1.78	1.85	1.93	2.00	2.08	2.15	2.23	2.30
Blasting Costs	% Change	-40%	-30%	-20%	-10%	0%	10%	20%	30%	40%
Blasting Unit Cost	USD/t	0.14	0.17	0.19	0.22	0.24	0.27	0.29	0.31	0.34
Total Operating Cost	USD/t	1.91	1.93	1.95	1.98	2.00	2.03	2.05	2.07	2.10

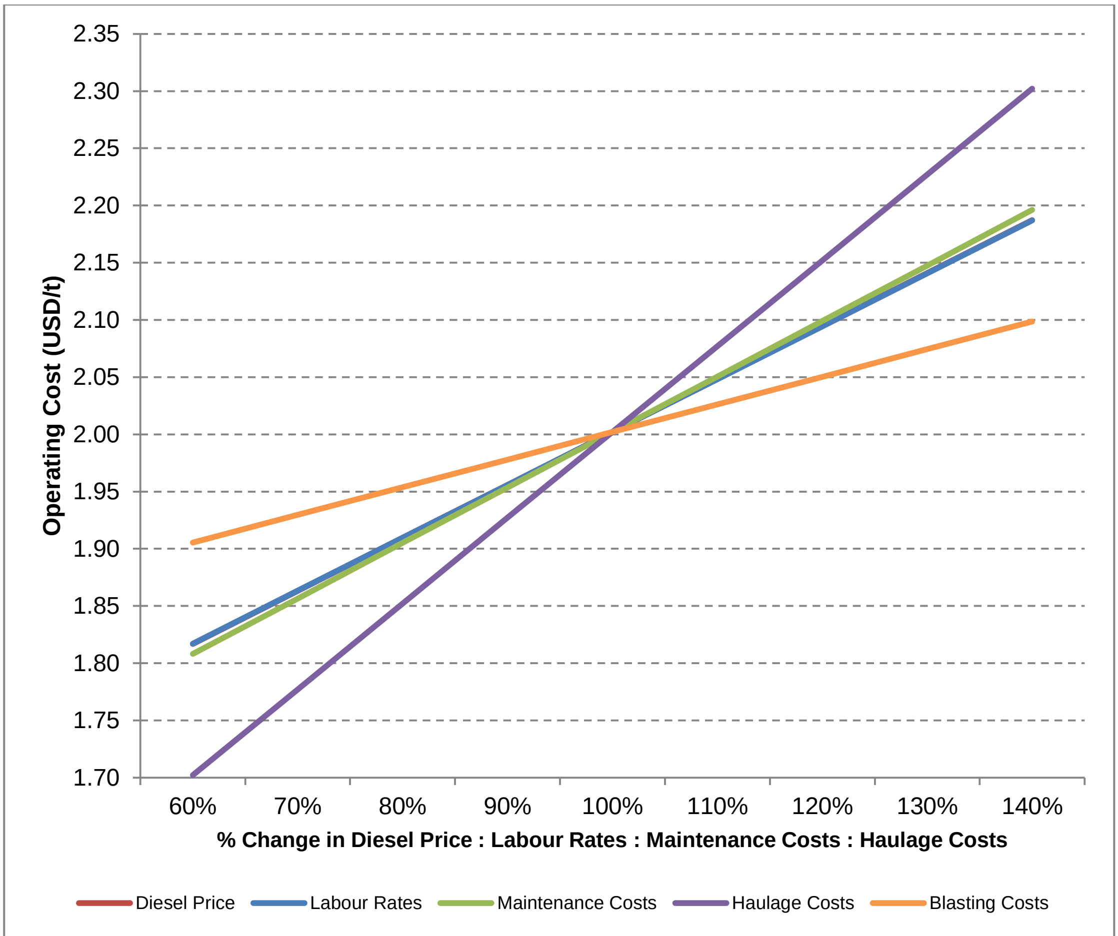


Figure 15-29: Operating Cost Sensitivity Analysis

Table 15-43: Drill and Blast Requirements

	Units	Total	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Drilling Required	**m**	**5,495,674**	2,981	87,089	256,367	297,837	347,512	300,360	316,817	317,812	344,783	396,381
Production	m	**4,292,424**	2,335	68,204	200,172	232,580	271,616	234,550	247,510	248,256	269,433	309,960
Trim	m	**1,090,781**	593	17,332	50,867	59,103	69,022	59,603	62,897	63,086	68,468	78,766
In-Fill	m	**112,470**	53	1,553	5,327	6,154	6,873	6,207	6,410	6,469	6,883	7,655
Holes Required	**holes**	**978,764**	532	15,552	45,644	53,033	61,934	53,482	56,438	56,608	61,436	70,678
Production	holes	**390,220**	212	6,200	18,197	21,144	24,692	21,323	22,501	22,569	24,494	28,178
Trim	holes	**99,162**	54	1,576	4,624	5,373	6,275	5,418	5,718	5,735	6,224	7,161
In-Fill	holes	**489,382**	266	7,776	22,822	26,517	30,967	26,741	28,219	28,304	30,718	35,339
Blasting Requirements												
Bulk Emulsion	t	**139,712**	81	2,310	6,481	7,547	8,904	7,615	8,066	8,092	8,828	10,236
Detonators	units	**978,764**	532	15,552	45,644	53,033	61,934	53,482	56,438	56,608	61,436	70,678
Surface Delays	units	**489,382**	266	7,776	22,822	26,517	30,967	26,741	28,219	28,304	30,718	35,339
Primers	units	**978,764**	532	15,552	45,644	53,033	61,934	53,482	56,438	56,608	61,436	70,678
Stemming	m3	**80,089**	44	1,273	3,735	4,340	5,068	4,376	4,618	4,632	5,027	5,783

	Units	Total	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Drilling Required	**m**	**5,495,674**	373,355	341,423	313,147	281,830	331,919	356,248	279,404	309,523	168,036	72,852
Production	m	**4,292,424**	291,899	266,847	244,647	220,019	259,282	278,369	218,032	241,696	130,651	56,367
Trim	m	**1,090,781**	74,177	67,810	62,169	55,911	65,888	70,738	55,406	61,419	33,201	14,324
In-Fill	m	**112,470**	7,279	6,766	6,331	5,900	6,750	7,141	5,966	6,408	4,184	2,161
Holes Required	**holes**	**978,764**	66,559	60,847	55,785	50,169	59,122	63,474	49,716	55,112	29,791	12,853
Production	holes	**390,220**	26,536	24,259	22,241	20,002	23,571	25,306	19,821	21,972	11,877	5,124
Trim	holes	**99,162**	6,743	6,165	5,652	5,083	5,990	6,431	5,037	5,584	3,018	1,302
In-Fill	holes	**489,382**	33,280	30,423	27,892	25,085	29,561	31,737	24,858	27,556	14,896	6,426
Blasting Requirements		**0**										
Bulk Emulsion	t	**139,712**	9,609	8,738	7,966	7,110	8,475	9,138	7,041	7,864	4,005	1,605
Detonators	units	**978,764**	66,559	60,847	55,785	50,169	59,122	63,474	49,716	55,112	29,791	12,853
Surface Delays	units	**489,382**	33,280	30,423	27,892	25,085	29,561	31,737	24,858	27,556	14,896	6,426
Primers	units	**978,764**	66,559	60,847	55,785	50,169	59,122	63,474	49,716	55,112	29,791	12,853
Stemming	m3	**80,089**	5,446	4,979	4,565	4,105	4,838	5,194	4,068	4,510	2,438	1,052

Blasting Requirements

The blasting parameters have been based on average rock densities and pattern type. The powder factors have been assumed based on operational experience for the ore and waste given the geotechnical parameters. The drill and blast parameters are shown in Table 15-44. Emulsion content has been estimated at 20% and 40% for waste and ore material, respectively.

Table 15-44: Drill and Blast Parameters

	Units	Waste Prod.	Waste Trim	Ore Prod.	Ore Trim
Drill Pattern					
Density	t/m^3	2.89	2.9	3.62	3.6
Bench Height	m	10.0	10.0	10.0	10.0
Hole Diameter	mm	229	172	229	172
Subdrill	m	1	1	1	1
Hole Depth	m	11.0	11.0	11.0	11.0
Stemming Height	m	4.50	3.50	4.50	3.50
Burden	m	5.17	4.31	5.17	4.31
Spacing	m	6.00	5.00	6.00	5.00
Volume Rock per Hole	m^3	310	216	310	216
Quantity Rock per Hole	t	897	623	1123	780
Re-drill/Drilling Overlap Factor	%	10%	10%	10%	10%
Blasting					
Charge Height	m	6.5	7.5	6.5	7.5
Charge per Hole	kg	253.4	164.6	328.1	213.1
Powder Factor	kg/m^3	0.82	0.76	1.06	0.99
Powder Factor	kg/t	0.28	0.26	0.29	0.27
AN	%	75.2%	75.2%	56.4%	56.4%
FO	%	4.8%	4.8%	3.6%	3.6%
Emulsion	%	20.0%	20.0%	40.0%	40.0%
Primer:Hole Ratio	units:holes	2	2	2	2
Drilling Penetration Rate					
Penetration Rate (m/min)	m/min	0.80	0.50	0.80	0.50
Hoisting Rate (m/min)	m/min	39	54	39	54
Rod Length	m	7.60	6.10	7.60	6.10
Assumed Cleaning Hole Time	min	1.50	1.50	1.50	1.50
Retract Jacks	min	0.50	0.50	0.50	0.50
Tramming	min	0.50	0.50	0.50	0.50
Level Drill	min	0.75	0.75	0.75	0.75
Add rods	min	1.00	1.00	1.00	1.00
Remove Rod	min	1.50	1.50	1.50	1.50
Drilling Time per Hole	min	19.78	27.95	19.78	27.95
Productivity	m/doh	33.4	23.6	33.4	23.6
Productivity	t_{rock}/doh	2,720	1,337	3,407	1,675
Drill Effective Utilisation	%	61.6%	61.6%	61.6%	61.6%
Annual Productivity	ktpa	14,671	7,210	18,377	9,031

It has been assumed that all blasting activities will be provided by a contractor, who will provide a bulk emulsion product and deliver it into the drillholes. The bulk explosive requirements are shown in Table 15-43.

It has been assumed that the average blast size will be 100,000 bcm, which indicates that blasting will take place every third day on average.

SRK notes that changes in pattern design and powder factors will be required in certain areas. SRK recommends the blasting requirements are benchmarked with an explosives supplier.

15.9 Mining Capital Costs

15.9.1 Introduction

The mining cost estimate has been developed to a HFS level of confidence. The mining equipment capital costs for the Project have been sourced from the manufacturers.

15.9.2 Equipment

The mining equipment capital costs for the Project have been sourced from the manufacturers. The equipment capital costs are shown in Table 15-45.

Table 15-45: Equipment Capital Costs

Equipment	Purchase Cost USDk
Shovel (26 m^3)	9,458.6
FEL (15 m^3)	5,298.9
Truck (227 t)	4,351.2
Drill (229 mm)	3,443.7
Drill (172 mm)	1,054.7
Track Dozer (580 hp)	1,563.3
Grader (300 hp)	1,100.7
Wheel Loader (6.5 m^3)	895.9
IT (5 m^3)	558.8
Rockbreaker (2.4 m^3)	460.8
Water Truck	1,269.3
Fuel/Lube Truck	870.3
Tyre Handler	473.6
Lighting Plant	20.0
Light Vehicle	62.7
Light Vehicle (inpit only)	40.6

Replacement costs throughout the mine life are based on the estimated equipment life for each unit. It has been assumed that the haul trucks would have certified rebuilds once they reached the end of their life at a cost of 60% of the replacement cost. All other equipment was assumed to be replaced. No allowance for salvage values of the mining equipment have been considered in this study.

The equipment fleet purchases and the replacement/rebuilds schedule are shown in Appendix L of the SRK Mining report in the HFS.

15.9.3 Contractor Cost Estimate

The initial waste stripping and all topsoil clearing is to be undertaken by contractor. The contractor unit rates used in the cost estimate have been provided by Arundon and are shown in Table 15-46.

Table 15-46: Contractor Costs

Cost Parameter	Units	Cost
Topsoil Stripping	USD/bcm	1.28
Waste Stripping		
Haulage <=2.05 km	USD/bcm	3.43
Incremental Haulage >2.05 km	USD/bcm/km	0.29

15.9.4 Dewatering

The dewatering costs have been provided by SRK and are shown Table 15-47.

Table 15-47: Pit Dewatering Costs

Component	Units	Hannukainen	Kuervitikko	Total
Sump and stage pumps	USD	904,635	346,740	1,251,375
Flotation units	USD	14,230	11,385	25,615
Booster tanks	USD	12,890	12,890	25,780
In-line flow meters	USD	22,800		22,800
Discharge line (including couplings)	USD	514,955	220,695	735,650
Total	**USD**	**1,446,710**	**591,710**	**2,653,395**

15.9.5 Miscellaneous

The miscellaneous capital cost estimate is based on the unit costs shown in Table 15-48. The mining software purchases are based on the personnel requirements, with five Geology licences: three Design, two Schedule and one Survey. Two survey equipment units have been estimated for costing purposes, which are replaced every three years.

Table 15-48: Miscellaneous Costs

	Units	Cost
Mining Software Package		
Geology	USD/unit	29,694
Design	USD/unit	24,165
Schedule	USD/unit	14,335
Survey	USD/unit	27,441
Dispatch System		
Initial Setup	USD	1,313,384
Survey Equipment		
Initial Setup	USD	178,357
GPS System	USD/unit	76,283
LV GPS System	USD/unit	1,745
Labour Costs	USD/h	110

15.9.6 Capital Cost Estimate

The mining capital cost estimate is shown in Table 15-49 including Engineering, Procurement and Construction Management ("EPCM") costs.

Table 15-49: Capital Cost Summary

	Units	Total	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Mining Capital Costs																							
Equipment Capital	**USDm**	**97.4**	0.5	34.9	9.8	28.8	5.9	4.4		4.4		4.4	4.4										
Equip. Replacement /Rebuilds	**USDm**	**65.2**					0.4	0.6	0.2	0.6	0.6	1.1	2.8	12.7	13.6	9.2	3.7		7.4	1.0	9.7	1.6	
Contractor	**USDm**	**26.7**	9.4	14.0			1.1	2.1															
Dewatering	**USDm**	**2.65**	0.9		0.1	0.1	0.1	0.0	0.0	0.2	0.0	0.0	0.2	0.0	0.2	0.0	0.2	0.1	0.2	0.2	0.1	0.0	0.0
Miscellaneous	**USDm**	**2.9**	0.5	1.4	0.1	0.2			0.2			0.2			0.2			0.2			0.2		
Subtotal	**USDm**	**194.7**	**11.2**	**50.4**	**10.0**	**29.0**	**7.5**	**7.1**	**0.3**	**5.2**	**0.6**	**5.6**	**7.4**	**12.7**	**13.9**	**9.2**	**3.9**	**0.3**	**7.5**	**1.3**	**10.0**	**1.6**	**0.0**
EPCM Costs	**USDm**	**3.0**	0.3	1.5	0.3	0.9																	
Total	**USDm**	**197.7**	**11.6**	**51.9**	**10.3**	**29.9**	**7.5**	**7.1**	**0.3**	**5.2**	**0.6**	**5.6**	**7.4**	**12.7**	**13.9**	**9.2**	**3.9**	**0.3**	**7.5**	**1.3**	**10.0**	**1.6**	**0.0**

15.10 Tailings Management

The tailings management facility ("TMF") was designed by Pöyry Finland Oy and is documented in Pöyry's design report no. 16FCI1018-E0036 (Pöyry 2013). The following sections provide an overview of the design.

15.10.1 Background

The initial phase of the Project considered depositing the tailings at the Hannukainen site. This site was located within an environmentally sensitive area and implied high costs. The Rautuvaara site was subsequently selected as an alternative site as it had already been disturbed from previous mining activities. The Rautuvaara site was considered as a potential site during the initial phase of the Project. It was not selected on the basis of potential liabilities associated with the previous operations. The Rautuvaara site eventually became the preferred site given the environmental and cost issues associated with the Hannukainen site. The Rautuvaara site also had features that made it less environmentally sensitive and had the potential for lower costs.

15.10.2 Site Location and Description

The Rautuvaara site is located about 6.5 km south of the Hannukainen site and is within the valley of the Niesajoki River. The Rautuvaara site was formerly used by Rautaruukki Oy for mining and processing Fe, Cu-Fe and Au-Cu ore between 1975 and 1996. Waste rock and tailings were produced and disposed at the site over that period. The mining activities resulted in excavating two open pits and some underground mine workings, which are currently flooded. Figure 15-30 shows the location of the Rautuvaara site relative to the Hannukainen site, while Figure 15-31 shows the main features of the Rautuvaara site.

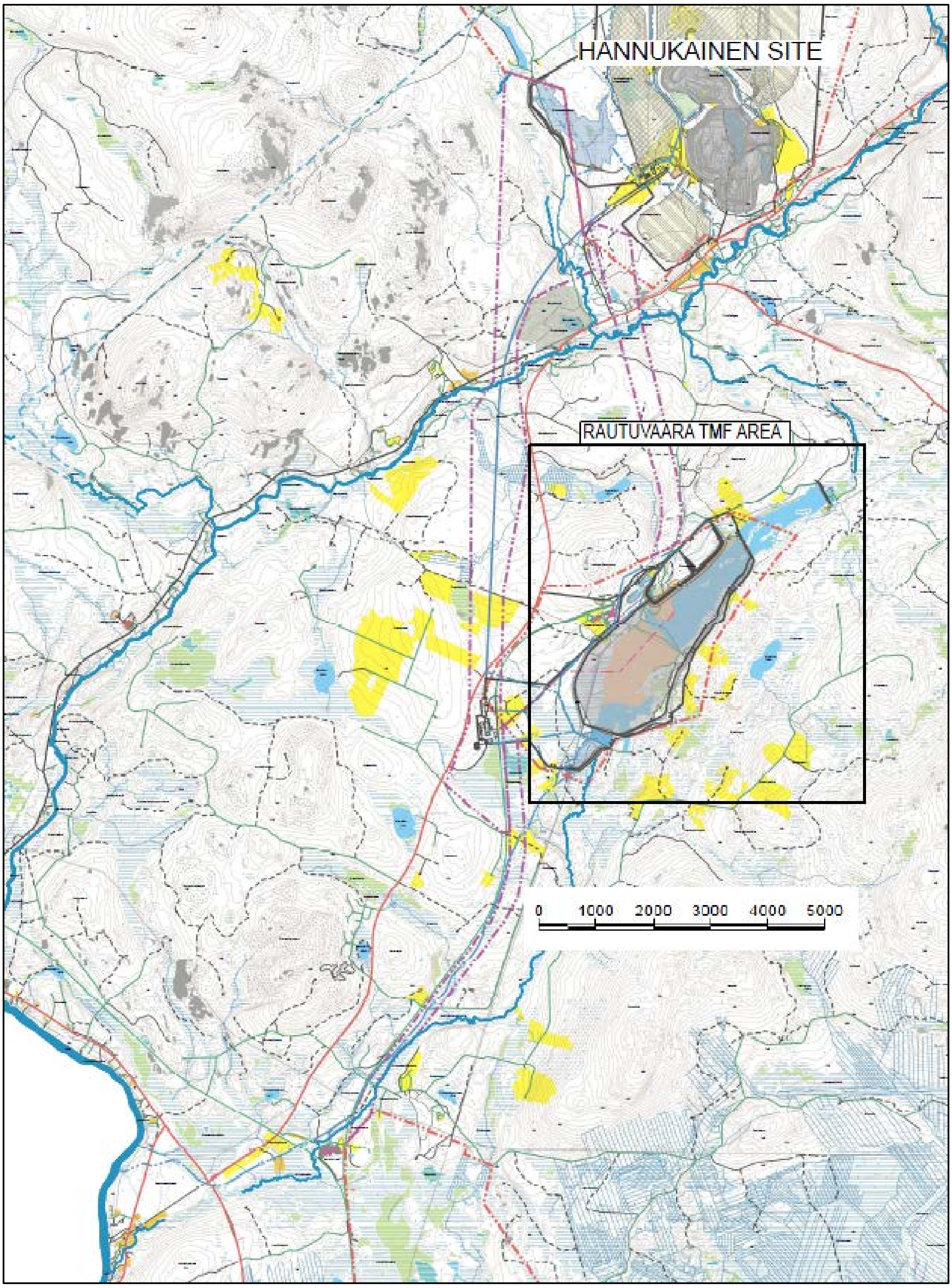


Figure 15-30: Location of Rautuvaara site relative to the Hannukainen site

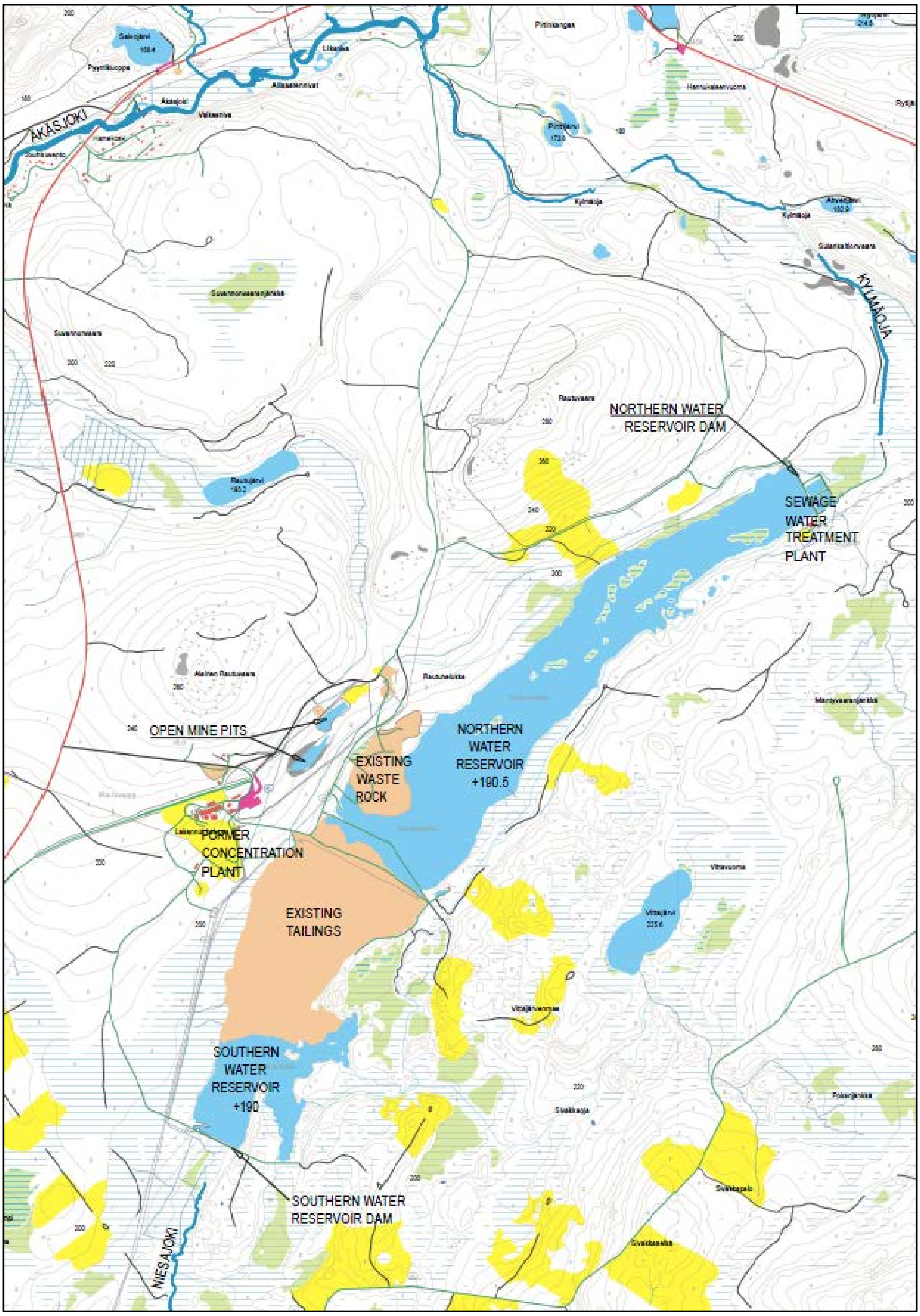


Figure 15-31: General site plan of the Rautuvaara site

There is an operating sewage water treatment plant (“SWTP”) owned and operated by Ylläksen Yhdyskuntatekninen Huolto Oy (“YYTH”) located at the northern end of the site. The valley currently has two water retaining dams at both ends of the valley, resulting in two water ponds at both ends (Northern and Southern Water Reservoirs). The middle portion of the valley is covered with deposited tailings and waste rock from previous mining operations. The Niesojoki River originally flowed through the valley, but was diverted to the Kylmäoja River when the SWTP dam was constructed. The overflow from the current Rautuvaara TMF discharges the Niesajoki River to the south and then eventually connects to the Muonio River. The Kylmäoja River flows to the Äkäsjoki River and then to the Muonio River. The Hannukainen site is part of the watershed of the Äkäsjoki River and is located upstream of the junction with the Kylmäoja River.

15.10.3 Site Conditions

Ground and foundation conditions

The overburden at the Rautuvaara consists generally of moraine soils with variable amounts of sand and fine particles (clay and silt). The site also has areas with well graded sand and gravel deposits, mainly along the river valleys. Peat layers are also present in some areas. The flooded areas have soft fine grained sediments at the bottom. Most of the area will be subject to freeze/thaw cycles, thus requiring the design to include frost protection and/or mitigation measures. A complete geotechnical site investigation was performed to support the design of the TMF and a summary of the field programme is summarized in Table 15-50.

Table 15-50: Summary of geotechnical investigation

Investigation type	Quantity
Soil Radar surveys (km)	21.9
Swedish weight sounding (points)	78
Ram drilling (points)	9
Drill sounding (points)	3
Disturbed soil sampling (points)	69
Monitoring pipes (points)	6
Field vane test (points)	2

The area has sufficient borrow sources to support the construction of the dams required at the Rautuvaara site. Crushed rock will be produced from rock excavated from the Hannukainen site.

The water depth of the ponded water is in general less than 4 m deep and some small areas approaching a maximum depth of 7 m.

Existing tailings and waste rock dump

The existing deposited tailings at Rautuvaara are originated from Fe-Cu-Au ores (Rautuvaara and Hannukainen mines) and Au-Cu ores (Saattopora Mine), together with thin interlayers of tailings from other sources. The total thickness of the tailings varies from 4 m to 15 m. It is estimated that the total amount of existing tailings is 6.6 Mt and that the current tailings area is around 85 ha, 30 ha of which is covered with water.

There is also about 700,000 to 800,000 m^3 of waste rock present at the Rautuvaara site, from historic operations. The waste rock consists of largely segregated micacious quartz schists and crystalline alkaline magmatic rocks. These rocks have a significant sulphide mineral content. Following the deposition of these waste rocks, the weathering of the majority of surficially exposed fines within the dump have been oxidised, thereby releasing acidity, sulphate and dissolved metals. Beneficial acid neutralising minerals have been consumed and this has resulted in the precipitation of secondary minerals.

The geochemical characterisation of the existing tailings and waste rock indicates that the sulphidic material is oxidising and producing acidic leachates containing contaminants. The results show that the more prominent contaminants present in the pore water are likely sulphate ($So4_2$), uranium, zinc, manganese, nickel, cadmium, arsenic, cobalt and copper, in addition to cyanide from the tailings. Concentrations of metals emanating from the TMF and WRD coupled with calculated volumetric seepage rates were used to calculate future water quality within the proposed clarification ponds. Tests performed on tailings and waste rock material also indicated the presence of fibrous minerals, but asbestos fibres were not present. Further details are provided in the Rautuvaara waste rock and tailings characterisation report (SRK, 2013b) and the HIA Report (SRK 2013a).

Groundwater

The depth to the groundwater table is generally shallow over the entire site. The groundwater flow is generally following the local topography, thus converging towards the bottom of the valley of the Niesajoki River.

15.10.4 Tailings production

The LoM is currently set at 19 years. Two types of tailings will be produced, namely the LIMS and the high sulphur ("High-S") tailings. The High-S tailings will be produced from the pyrite ("Py") and pyrrhotite ("Po") streams. These streams are mixed at the process plant and pumped and deposited as one combined stream.

The mass balance of the tailings streams is shown in Table 15-51 (nominal) and in Table 15-52 (design). The design flow rates are based on nominal throughputs of +15%. Tailings deposition designs use the annual total tailings production rates with +20% safety margin to allow for tailings consolidation. The quantities are listed in Table 15-53 and the tailings annual production over the LOM is detailed in Table 15-54.

Table 15-51: Nominal case mass balance of the tailings streams (Jacobs 2013)

Stream	Solids		Water	Slurry		Specific gravity		Percent solids	
	tph	m^3/hr	m^3/h	tph	m^3/h	solids	slurry	mass	volume
LIMS	466,2	139,0	163,8	630,0	302,8	3,36	2,1	74,0	45,9
Py (High-S)	31,7	8,1	15,8	47,4	23,9	3,9	2,0	66,8	33,9
Po (High-S)	47,6	9,3	33,9	81,6	43,3	5,1	1,9	58,4	21,6
Total High-S	79,3	17,4	53,1	132,3	70,5	4,6	1,878	59,9	24,7

Table 15-52: Design case mass balance of the tailings streams (Jacobs 2013)

Stream	Solids		Water	Slurry		Specific gravity		Percent solids	
	tph	m^3/h	m^3/h	tph	m^3/h	solids	slurry	mass	volume
LIMS	536,2	159,8	188,4	724,5	348,2	3,36	2,081	74,0	45,9
Py (High-S)	36,4	9,3	17,8	54,2	27,1	3,9	2,000	67,2	34,3
Po (High-S)	54,8	10,7	38,7	93,5	49,4	5,1	1,890	58,6	21,7
Total High-S	91,2	20,0	59,9	151,0	79,9	4,550	1,891	60,4	25,1

Table 15-53: Tailings total production rates for LoM

Stream	Production (at full 6,5 Mtpa ore production rate)		Total production		Total production +20% - consolidated
	Mtpa	Mm^3/a	Mtonnes	Mm^3	Mm^3
LIMS	3,68	1,89	65,2	33,4	42,9
Py	0,25	0,13	4,4	2,2	-
Po	0,38	0,19	6,7	3,4	-
Total High-S	0,63	0,32	11,1	5,6	5,9

Table 15-54: Detailed mass balance of the tailings production

		Year																			
		2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
	Mton	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Ore	115	0.00	1.63	4.83	6.50	6.50	7.21	6.74	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.5	6.5	3.7
Conc	38	0.00	0.53	1.58	2.13	2.13	2.36	2.20	2.13	2.13	2.13	2.13	2.13	2.13	2.13	2.13	2.13	2.13	2.13	2.13	1.21
Tailings	77	0.00	1.10	3.25	4.37	4.37	4.85	4.54	4.37	4.37	4.37	4.37	4.37	4.37	4.37	4.37	4.37	4.37	4.37	4.37	2.49
Pyrite	4	0.00	0.06	0.18	0.25	0.25	0.27	0.26	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.14
Pyrrhotite	7	0.00	0.09	0.28	0.38	0.38	0.42	0.39	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.21
LIMS	65	0.00	0.92	2.73	3.68	3.68	4.08	3.81	3.68	3.68	3.68	3.68	3.68	3.68	3.68	3.68	3.68	3.68	3.68	3.68	2.09

15.10.5 Tailings properties

Geotechnical properties

The current design for the thickening of the tailings is 74% of solids by mass for the LIMS tailings and 60% for the High-S tailings.

Rheological properties were determined from samples collected from the concentration test plant testing designed process for Hannukainen ore (lab scale test plant). Ore sample GEM-5 (average sample) was used to determine the rheological properties of both tailings. The test results are presented in Ketek (2012). Table 15-55 summarises the rheological properties for both tailings types.

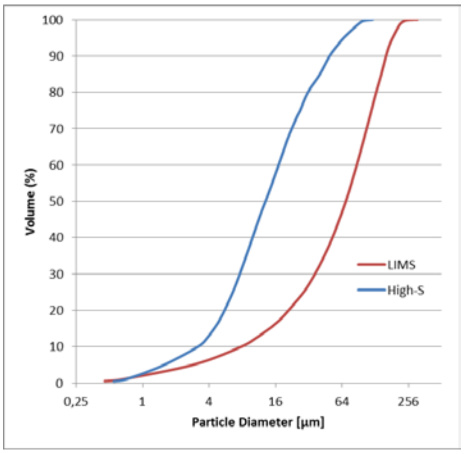
Particle size distributions ("PSD") from both tailings types were measured and are presented by GTK (2012). The PSD were not representative of the target process design, as the PSD for the LIMS was coarser than expected and the High-S was finer. Further testing will be required once samples from the pilot plant become available. The results are shown in Table 15-56.

Table 15-55: Tailings rheological properties for LIMS and total High-S tailings

	Yield stress (average from Bingham approximation) [Pa]	Plastic viscosity [Pa·s]
LIMS	32.5	1.05
Total High-S	5.1	0.18

Table 15-56: Tailings particle size distribution for LIMS and Total High-S tailings

Cumulative particle size, passing %	LIMS Particle size (µm)	HIGH-S Particle size (µm)
100	309,6	121,8
90	160,8	49,5
85	145,75	40,465
80	130,7	31,43
75	118,5	26,72
50	69,35	12,97
25	28,77	6,555
20	18,3	4,8655
10	7,83	3,176



KETEK (2012) reports fast particle settling rates and poor test repeatability for both the High-S and LIMS tailings samples.

Consolidation of the High-S tailings will result in a reduction in the porosity from the initial 67% to a final porosity estimated at 24%. This represents a reduction in porosity of the order of 50%. The LIMS tailings are expected to consolidate by about 20% by volume after deposition. Further testing is, however, required when pilot plant testing is performed.

Geochemical properties

The proposed new tailings geochemical properties have been evaluated as part of an earlier stage of the Project (Eriksson, 2012). However, following the recently revised process design that resulted in coarser tailings particles, it is recommended that these materials also require characterisation. It is proposed that two tailings streams will be produced, a high sulphur pyrite and pyrrhotite tailings and a de-sulphurised LIMS tailings. The High-S tailings has been shown to rapidly oxidise, generating acidic leachates containing elevated concentrations of sulphate as well as elements iron, aluminium, cadmium, cobalt, copper, manganese, nickel, lead, uranium and zinc, whilst the LIMS tailings has been shown to be non-acid generating. However, the LIMS tailings is shown to leach elevated concentrations of elements uranium and molybdenum.

Based on the predicted leaching characteristics and high reactivity of the High-S tailings material, it has been necessary to encapsulate the tailings to contain the contaminated pore water.

Seepage from the LIMS and High-S tailings will ultimately make its way in to the Muonionjoki River by way of either the pipeline (during operation) or by way of the Niesajoki River post closure. In all cases, once fully mixed across the full width of the Muonionjoki River, concentrations of determinates will likely remain below the site specific Muonionjoki "Action Values" during operation and post-closure, based on baseline values defined by the Project. Some of the site specific "Trigger Values" (indicating a significant increase above baseline conditions and therefore indicating a deviation from baseline conditions) would, however, be exceeded if the effectiveness of the liner of the High-S cells is reduced to 50% during post-closure. This scenario is equivalent to removing 50% of the liner when assessing the potential impact during post-closure.

Mineralogical testing of the fresh Hannukainen tailings (Erikson, 2012) identifies the presence of the mineral tremolite. Tremolite can form an asbestos habit and therefore it is advisable that mineralogical examination of the latest tailings material be undertaken to positively identify the mineral habits of fibrous minerals present in the tailings.

15.10.6 TMF Configuration and Deposition

The configuration of the TMF over the life of mine is shown in Figure 15-32 to Figure 15-36 and will have the following key features:

- LIMS tailings deposited to the former Niesajoki valley onto the existing tailings stack. Deposition begins from the existing tailings area, which minimizes the initial CAPEX. No bottom structure required.
- Flood cutoff dam constructed against the eastern dyke to prevent possible flood from reaching the Niesajoki River in the event of a failure of the High-S dam.
- High-S cell located uphill from the LIMS tailings and adjacent to the existing two open pits. The High-S impoundment is constructed as two separate cells (Cell 1 and Cell 2) to reduce initial CAPEX and enable progressive reclamation and includes a bottom structure with drainage system to promote consolidation of the tailings and reduce water pressure on liner.
- Existing open pits at Rautuvaara used as storage for High-S deposition for the initial 12 months of operation and remaining 6 months of storage capacity kept as emergency storage.
- Clarification Pond located at south end of the TMF.
- SWTP Dam at north end of valley to be upgraded and raised to prevent potential impacts to the Äkäsjoki River.
- Clean and dirty waters kept separate using dykes and ditches.
- Clean waters from the northern areas of the TMF deposit (ponds) diverted towards the Niesajoki River.
- High concentration contaminated mine water directed to the water treatment plant and then pumped to the Clarification Pond.
- Mine water directed to the Clarification Pond and then pumped to the Muonio River.

The configurations of the dams were designed to use as much local moraine material as a measure to reduce cost. Further details are provided in the TMF Design Report (Pöyry 2013).

The tailings deposition schedule over the life of mine shown as cumulative volumes is shown in Table 15-57.

Table 15-57: Tailings deposition schedule as cumulative volumes

		Year																			
		2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
	Mm^3	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
LIMS	41.97	0.00	0.59	2.36	4.73	7.10	9.72	12.18	14.55	16.92	19.29	21.66	24.03	26.40	28.77	31.14	33.51	35.88	38.25	40.62	41.97
High S																					
Pit	0.13	0.00	0.08	0.13																	
Cell 1	3.55			0.19	0.51	0.83	1.18	1.51	1.83	2.15	2.47	2.79	3.11	3.43	3.55						
Cell 2	1.99														0.33	0.52	0.84	1.16	1.48	1.80	1.99

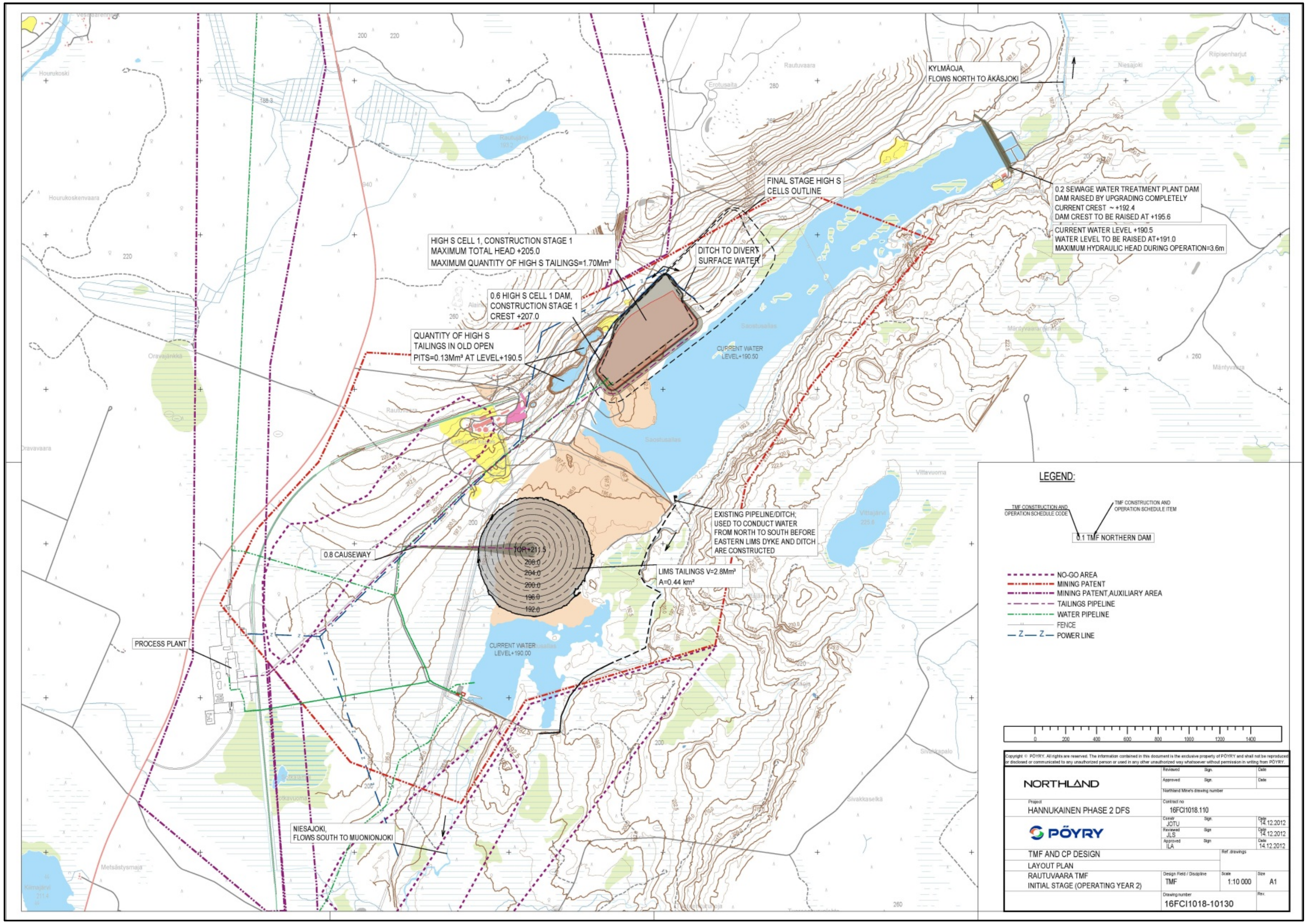


Figure 15-32: Configuration of TMF, initial deposition

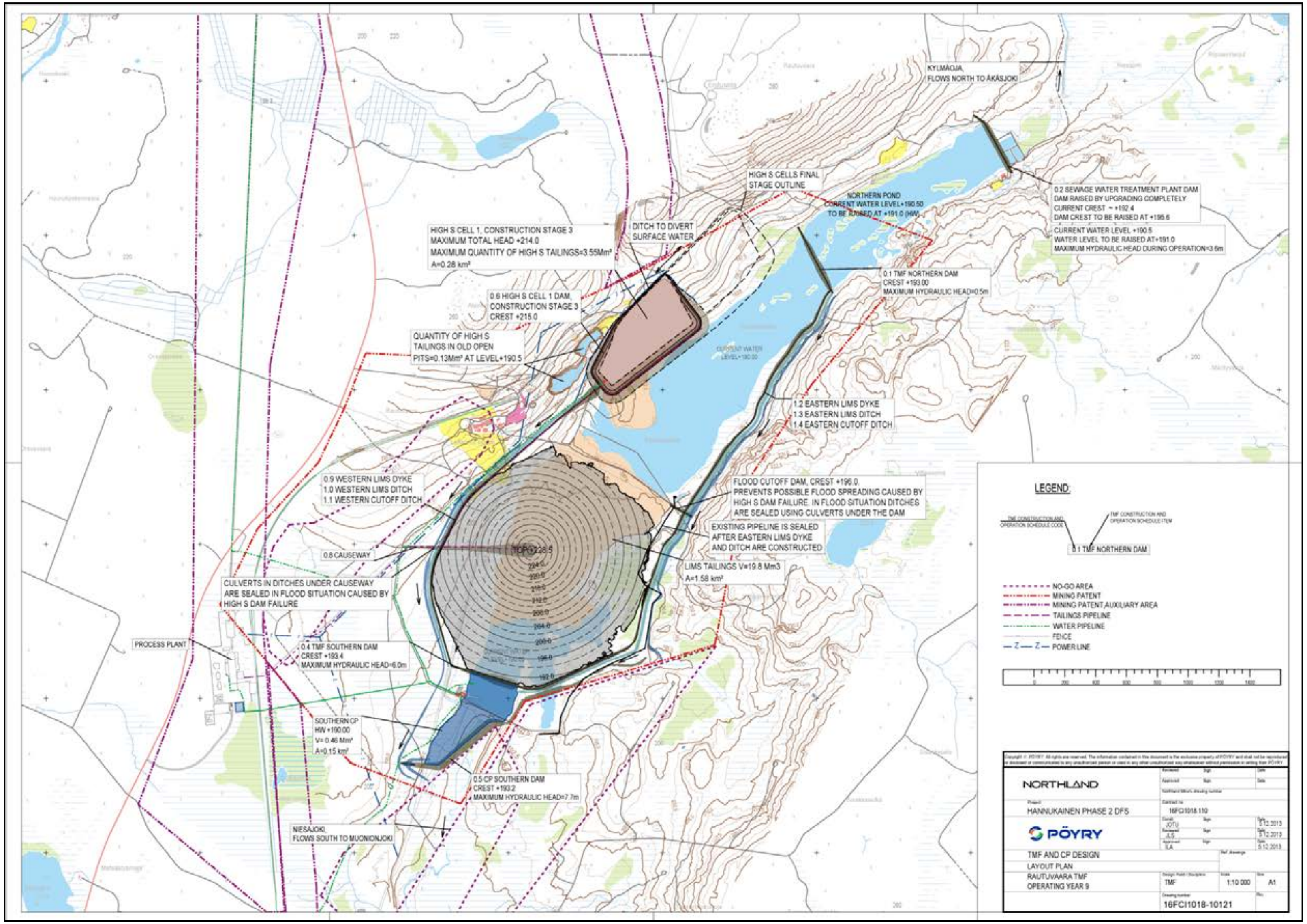


Figure 15-33: Configuration of TMF, deposition after year 9

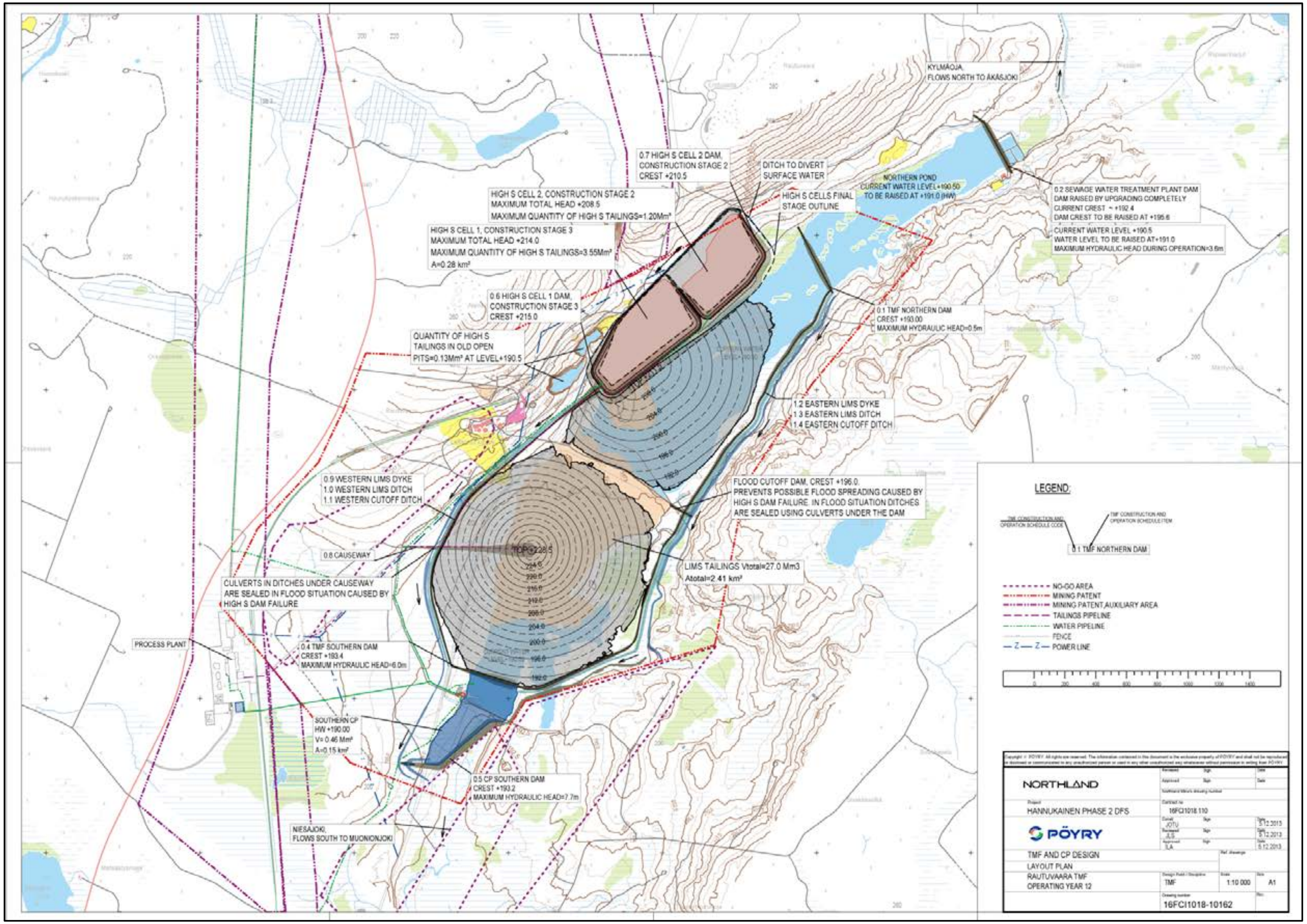


Figure 15-34: Configuration of TMF, deposition after year 12

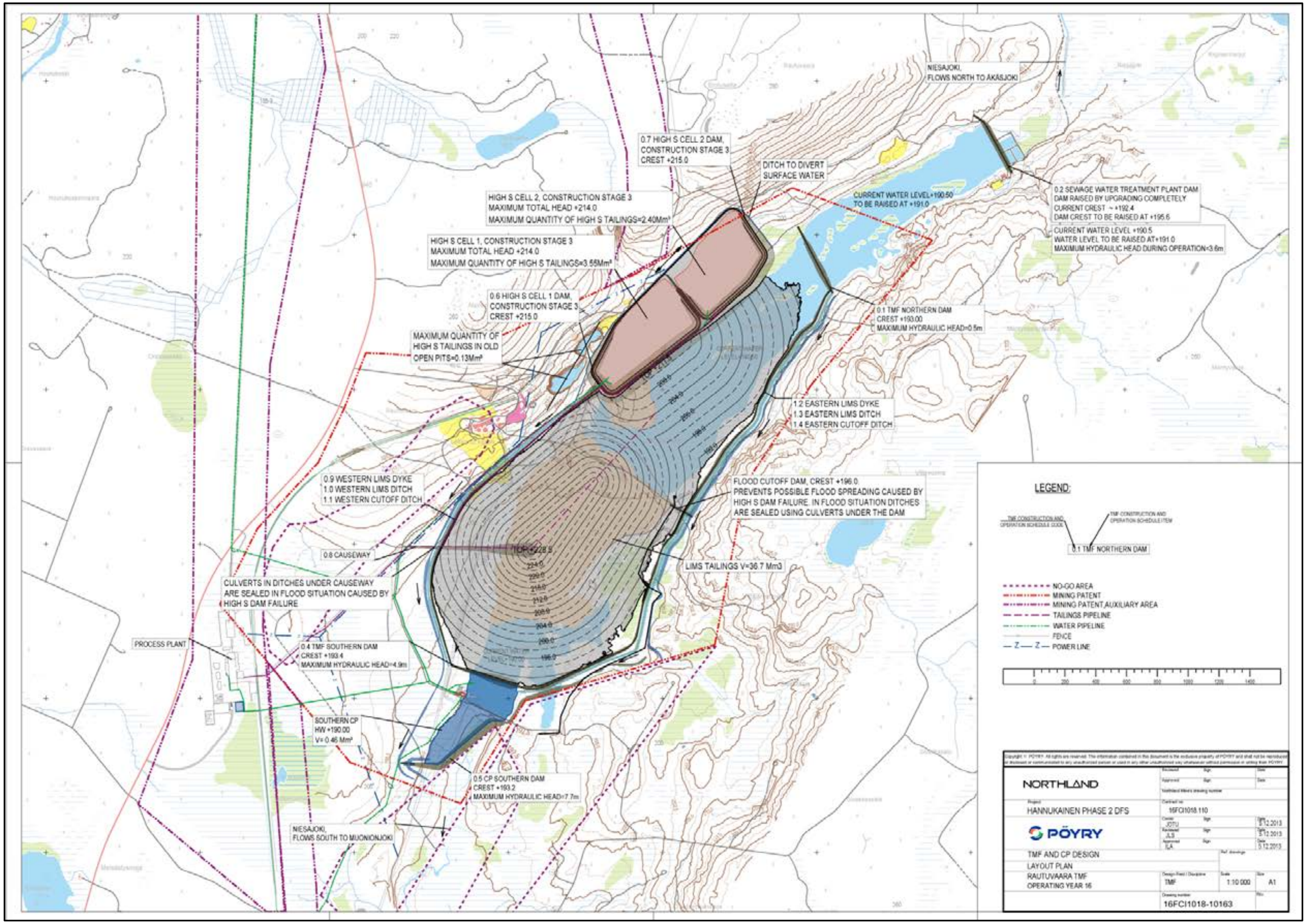


Figure 15-35: **Configuration of TMF, deposition after year 16**

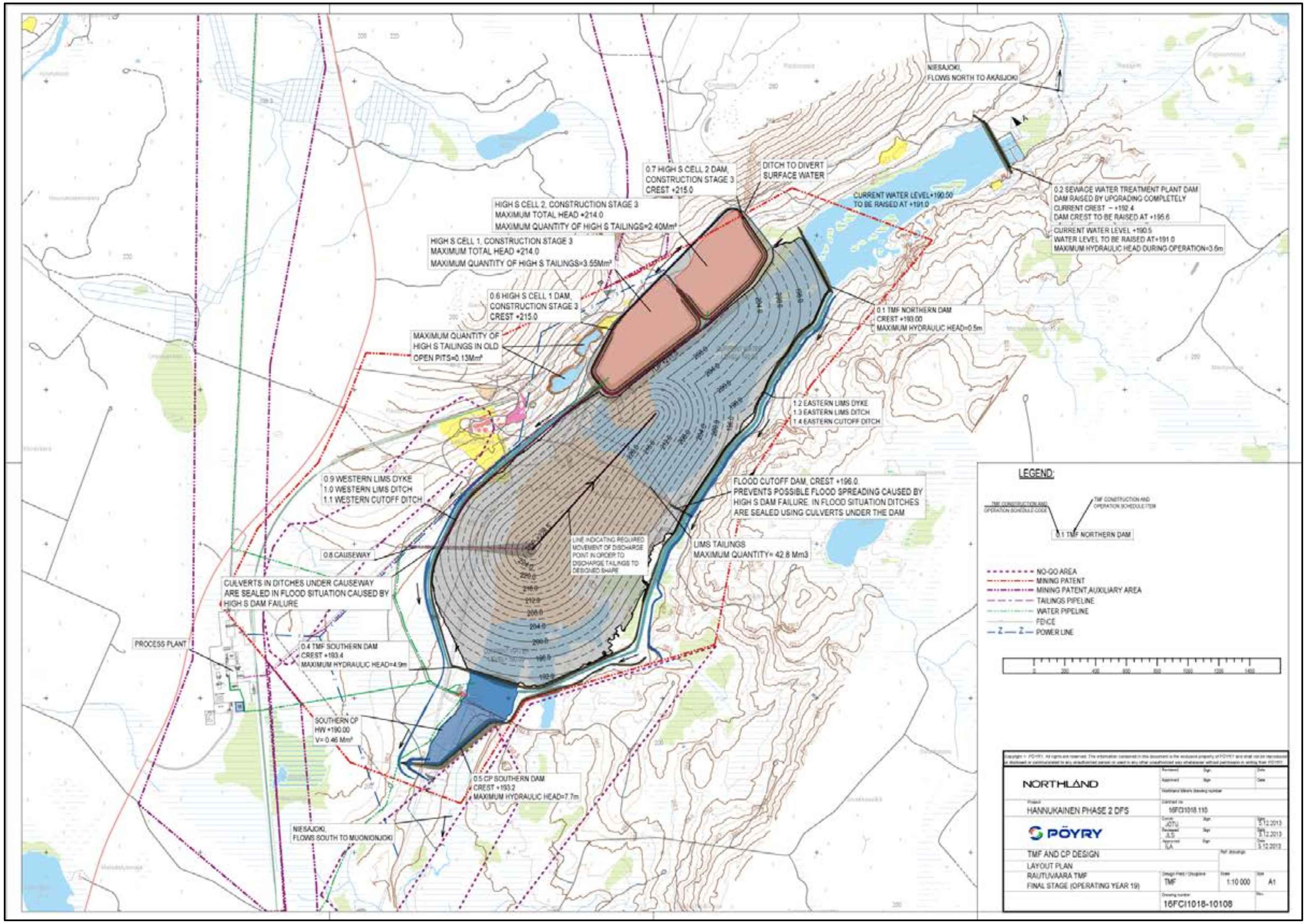


Figure 15-36: **Configuration of TMF, final deposition at end of mine life**

High-S Tailings

The High-S will first be discharged in the two flooded pits at the Rautuvaara site. These two pits have a storage capacity of about 18 months of operation (about 0.13 Mm^3). The proposed plan is to dispose High-S in the pits for the initial 12 months of operation while the High-S cell is being constructed. The remaining 6 months of storage will be kept as emergency storage during the life of mine. The excess water will be collected and be pumped either to the water treatment plant or to the clarification pond depending on the quality of the water over-spilling from the pits.

The High-S will subsequently be stored in two fully lined storage cells that will include a drainage system at the base immediately below the tailings. The two cells will be separated by an internal dyke and the dams will be raised in three stages using a downstream construction. Figure 15-37 shows the final configuration of the High-S cell and Figure 15-38 shows the details of the impervious barrier at the bottom the High-S cells.

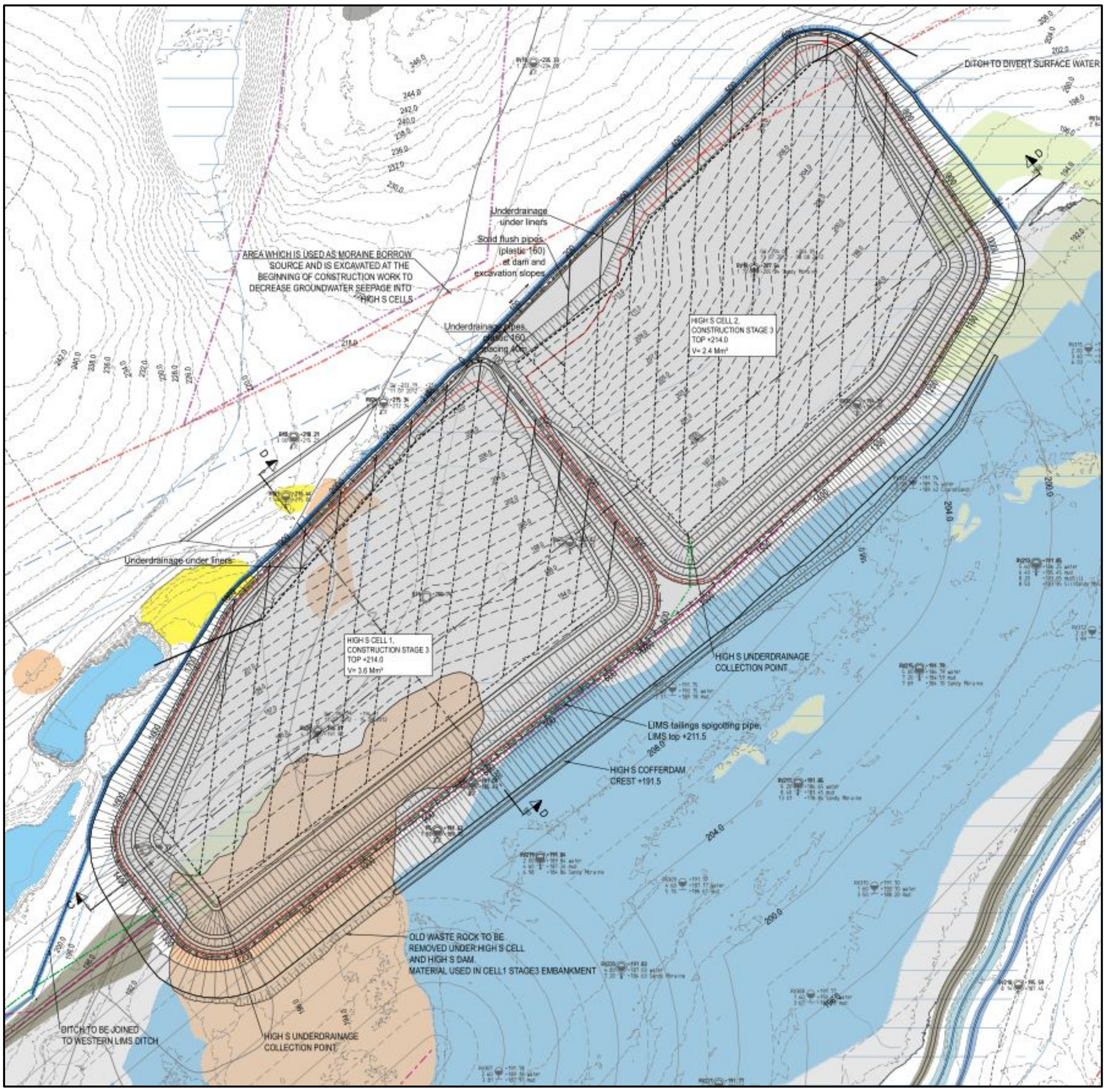


Figure 15-37: High-S cell, final configuration

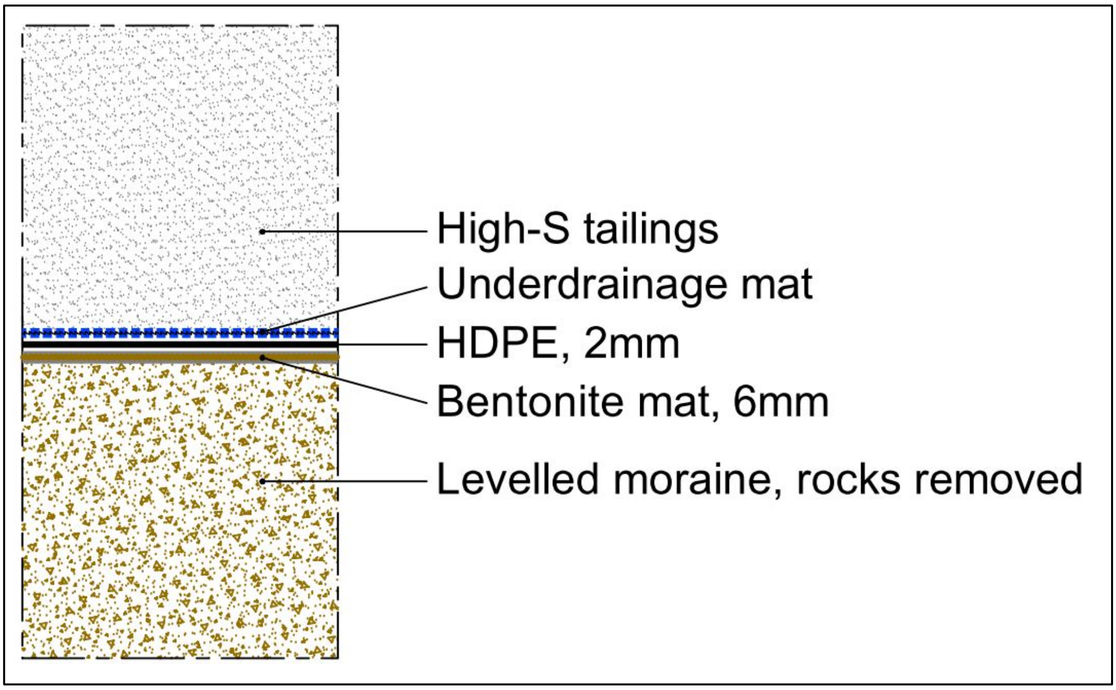


Figure 15-38: High-S cell, details of the bottom structure

The High-S tailings will be deposited sub-aqueously using spigotting and/or single point discharge. The excess water will be pumped to the water treatment plant using a floating barge. The emergency spillway will be located at the south-eastern corner and the overflow will report to the LIMS tailings storage area.

The storage capacity calculations did not include any beach slope (that is, assumed a flat deposit).

The dams will have a lined upstream face and will be constructed using moraine for the first two stages and then waste rock material will be used for the third stage. The waste rock material used for dam construction will be coming from the on-site waste rock that will need to be excavated for preparing the surface to construct the base liner of the High-S cell.

The dam length is about 1,900 m for Cell 1 and about 1,686 m for Cell 2. The dams will reach a maximum height of 24.2 m in Cell 1 and 23.4 m in Cell 2 as shown in Figure 15-39.

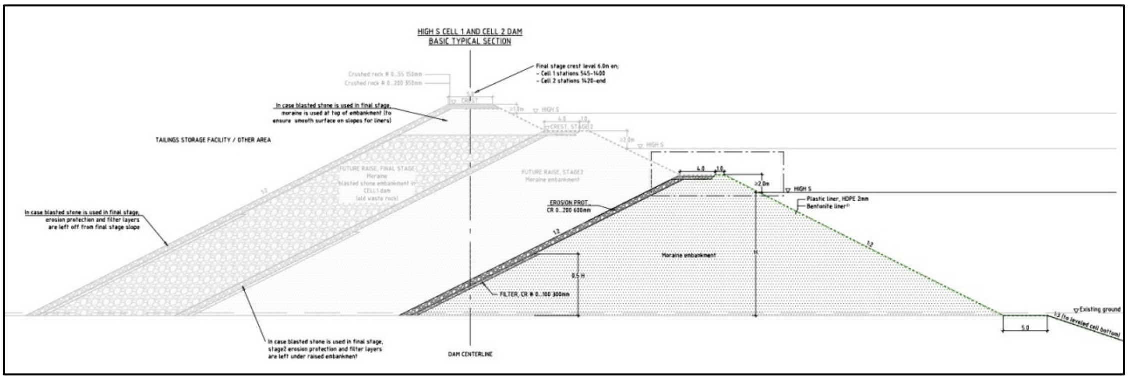


Figure 15-39: High-S cell, typical dam section with the three stage construction sequence

The construction of the High-S dams will first use moraine that will be sourced uphill of the High-S cell as a measure to intercept shallow groundwater and reduce dewatering for the construction of the bottom structure. The construction of the bottom structure will require the excavation of waste rock that is located within the footprint of the High-S cell. The excavated waste rock material will be stockpiled locally until it is used for constructing the third stage of the High-S dams.

LIMS Tailings

The sequence of deposition for the LIMS tailings will start on top of the exposed tailings and gradually progress towards the north. The initial period of deposition will not require any dam construction and will need only the construction of a causeway for bringing the tailings pipeline to the desired location. The causeway will be gradually raised as the level of tailings rises. Once the causeway reaches maximum level, the discharge point will gradually move towards the north and to the High-S cells. The discharge operation will be able to alternate between spigotting on the main stack and on the crest of High-S tailings dams. The progression of deposition is shown in Figure 15-32 to Figure 15-36.

The beach slope profile was assigned a concave shape according to the priori method developed by Fitton (2006) and is shown in Figure 15-40. Given the uncertainties associated with slope predictions of tailings beach, the storage capacity and the height of the dams was also assessed using half of the design values.

The deposition plan should, however, be reviewed during operation to enable further optimization of the discharge arrangements.

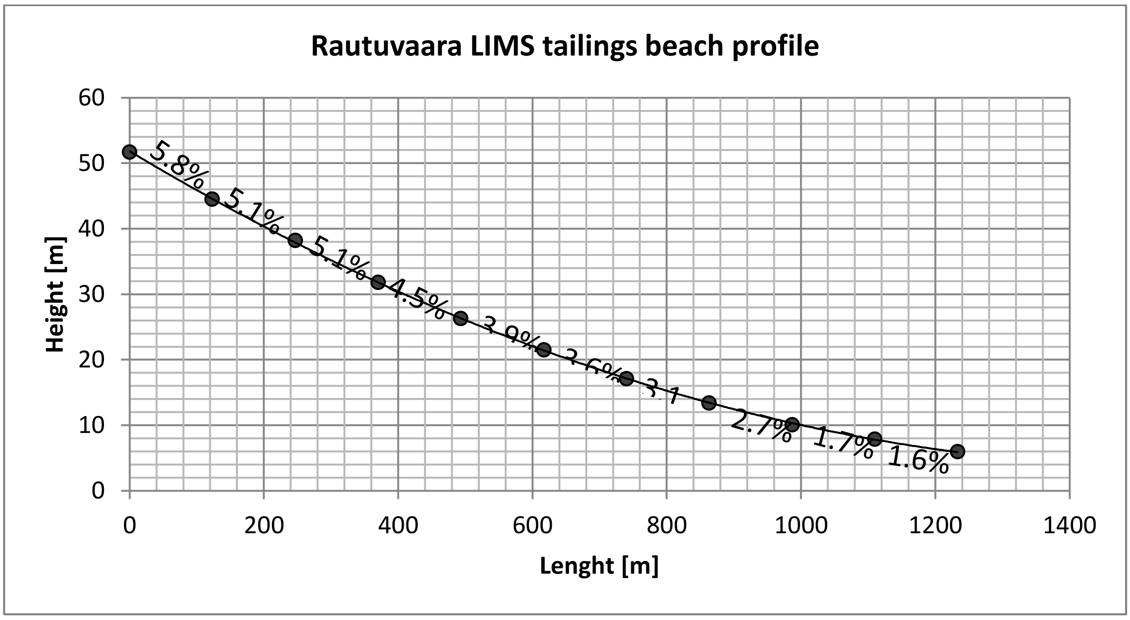


Figure 15-40: LIMS estimated beach slope

Dykes will be required to retain tailings and to provide road access, but would act as water retaining structure (so-called leaky dykes). They will also be used to separate clean and mine water. They will be constructed with unqualified coarse moraine and will include base and sub-base layers plus a top traffic layer. Crushed rock/blasted rock will be used for erosion protection on slopes. A typical section of TMF LIMS tailings dyke is presented in Figure 15-41.

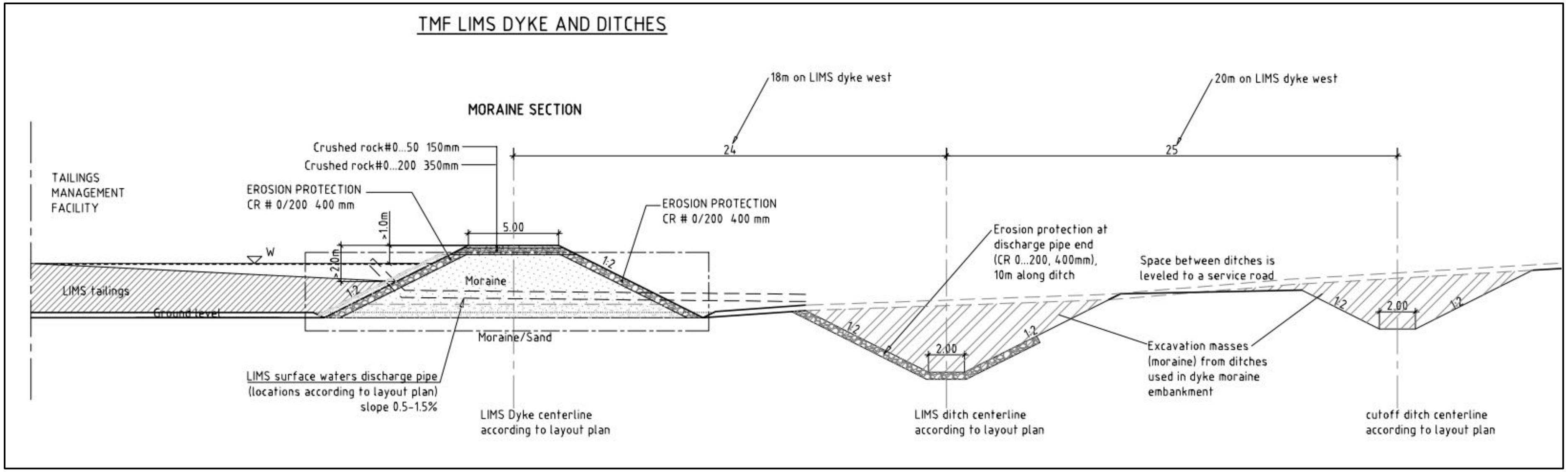


Figure 15-41: Typical section of TMF LIMS tailings dyke

The purpose of TMF Northern Dam is to separate the tailings from the northern pond. The length of the dam is 506 m and has a maximum height of about 6.5 m. The maximum hydraulic head across the dam will be limited to 0.5 m under normal operating conditions. Figure 15-42 shows a typical section of the TMF Northern Dam.

The portion of the dam footprint located below water will require the excavation of the old riverbed sediments of the Niesajoki River and any peat layers that may be present. The soft layers will be excavated from the dam base along its total width and replaced with a low permeability moraine (ksat ≤ 3x10-7 m/s). Only the topsoil will be removed where moraine is present. The low permeability core will be constructed with moraine and will extend 1.5 m above the high water-level.

The placement of moraine embankment below water will not be compacted until it is about 1.0 m above the water level; only then will the moraine be compacted. To ensure optimal construction conditions, the water level of the pond is lowered as much as possible. There is a potential for additional settlement after the compaction of the submerged material, but this was taken into consideration in the design of the dam. The proposed construction method is based on experience from other projects where it was effective. If placement of moraine under water is not satisfactory, alternative dewatering methods are possible, such as temporary cofferdams constructed in sections to enable local dewatering of the foundations before fill placement.

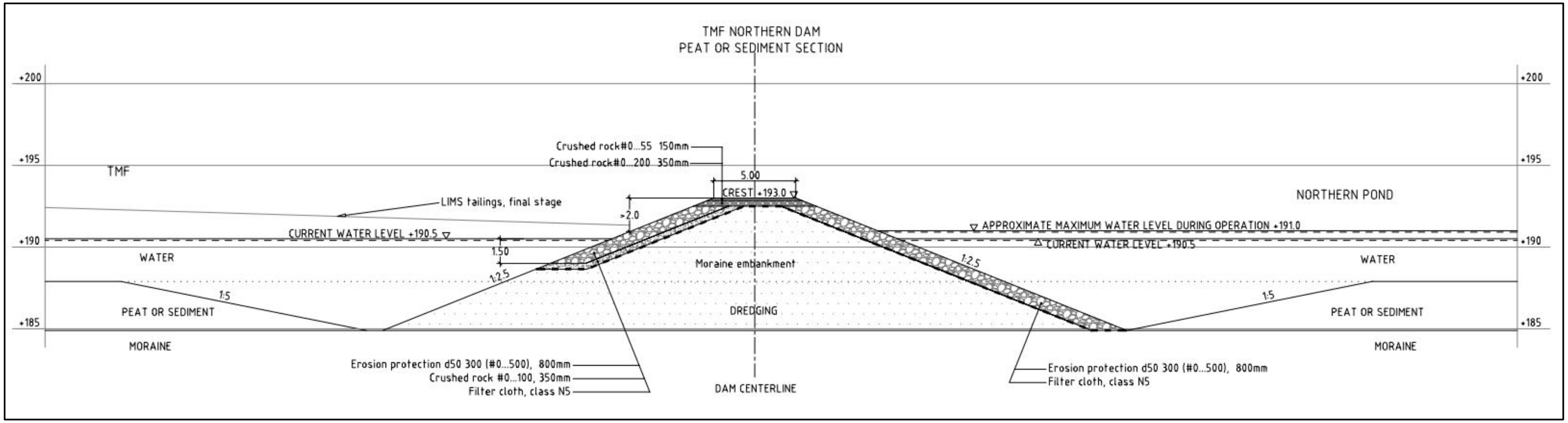


Figure 15-42: **Typical section of TMF Northern Dam**

The TMF Southern Dam consists of an earthfill structure with a low permeability core, and will have a similar configuration as the Northern Dam, with minor differences. The purpose of TMF southern dam is to prevent the migration of tailings towards the Clarification Pond to the south. The length of the dam is 1,252 m and the maximum height is about 8 m. As the Clarification Pond is regulated, the maximum hydraulic head can be up to 6 m on the Southern Dam, although it is intended to retain tailings only. It will be operated as a leaky structure where the seepage will flow to the Clarification Pond.

The Southern Dam will be constructed using the same construction method as for the Northern Dam. The water level in the adjacent pond can however be lowered via the old spillway structures, which will facilitate placement and compaction of the dam material.

The SWTP Dam consists of an earthfill structure with a low permeability core that will be upgraded to accommodate the operation of the TMF. The hydraulic head applied to the dam will only be increased by 0.5 m but most of the upgrade is to raise the crest of the dam to obtain sufficient storage for the hypothetical failure of both High-S dams. The additional storage capacity requires a rise of about 4 m above the current crest elevation. The dam is currently operating with a hydraulic head of about 3 m, which will be raised to 3.5 m once the Northern Dam is constructed. The maximum dam height will be about 8.6 m as shown in Figure 15-43.

The low permeable moraine core will be raised to 4.0 m above the high water-level and 0.4 m above the level estimated for the High-S dam break situation. The raised SWTP Dam has horizontal seepage collection filter structures under the downstream slope. The side slopes will include protective rock armour. The upgrade of the SWTP Dam will require the excavation and fill placement under water. The fill placement will follow the same general procedure as mentioned above for the Northern Dam. Additionally, temporary cofferdams may be required to enable local dewatering of the foundations before fill placement. The SWTP Dam will include an emergency spillway.

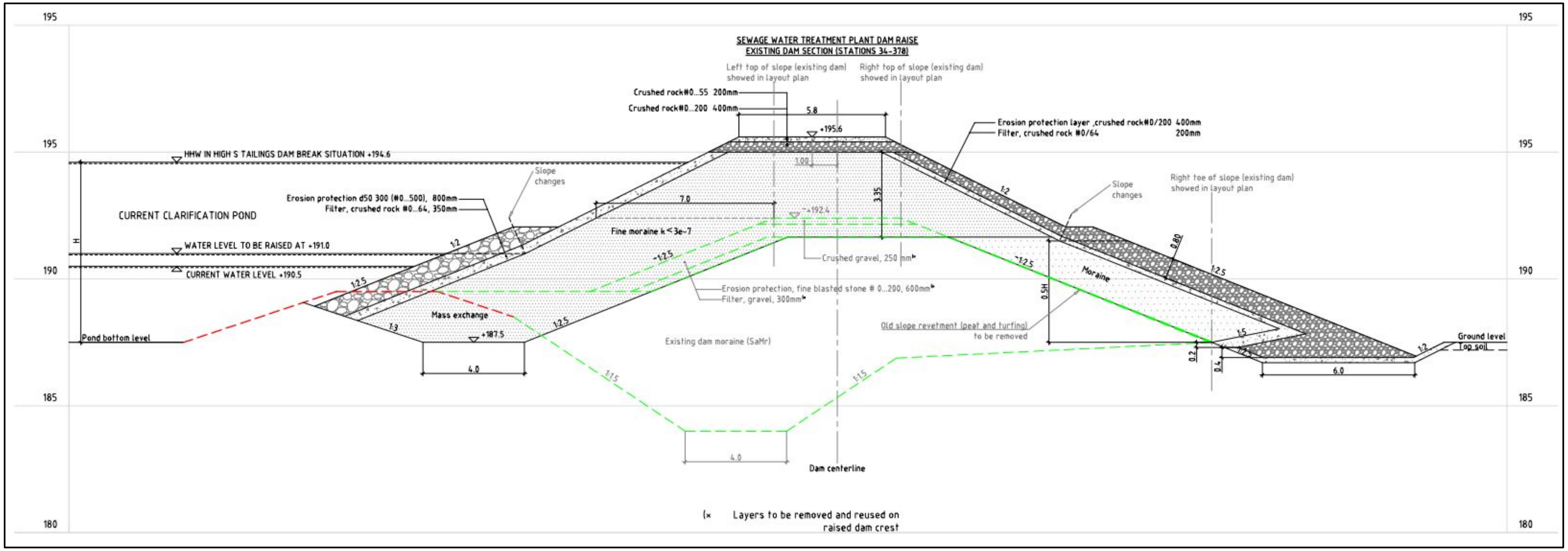


Figure 15-43: Typical section of upgraded SWTP Dam

The Clarification Pond ("CP") Southern Dam will consist of an earthfill structure with a low permeability core. The dam was designed and will operate as a water retaining structure. The general layout of the CP is shown Figure 15-44.

The freeboard of the dam was set to 3.2 m and is based on a frost depth with a 10 year return period. The low permeable moraine core extends 0.8 m above the high water level, which is more than the wave height protection height. The core is made of moraine with a saturated permeability of 3x10-7 m/s or less. The CP Southern Dam has horizontal seepage collection filter structures under the downstream slope. Both side slopes will include rock fill armour for erosion protection. The geometry of the dam will be dependent on the type of foundation material, namely moraine or peat/soft soils. The configuration also includes an emergency spillway. Figure 15-45 shows the typical sections for the CP Southern Dam.

The CP Southern Dam can be constructed mostly under dry conditions. This will require that the current pond is emptied prior to the construction.

A flood cutoff dam will be constructed along the eastern LIMS ditch to prevent uncontrolled discharge toward the Niesajoki River in the event of a failure of the High-S dam. The flood cutoff dam is 164 m long and the crest set at elevation 196.0 masl. The flood cutoff dam is composed of moraine with crushed rock/blasted rock for erosion protection as indicated in Figure 15-46.

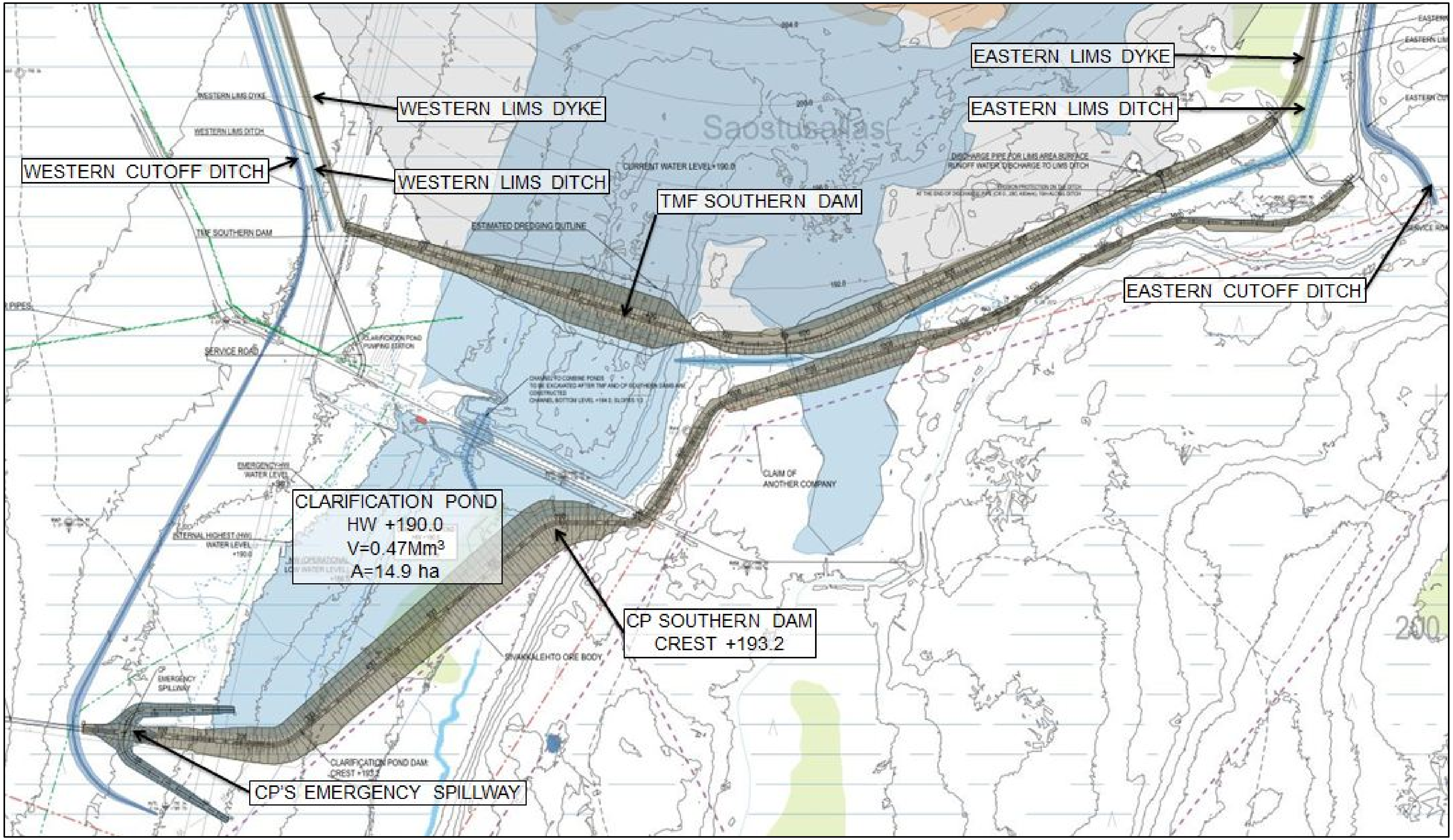


Figure 15-44: **Layout plan of the CP and related dams**

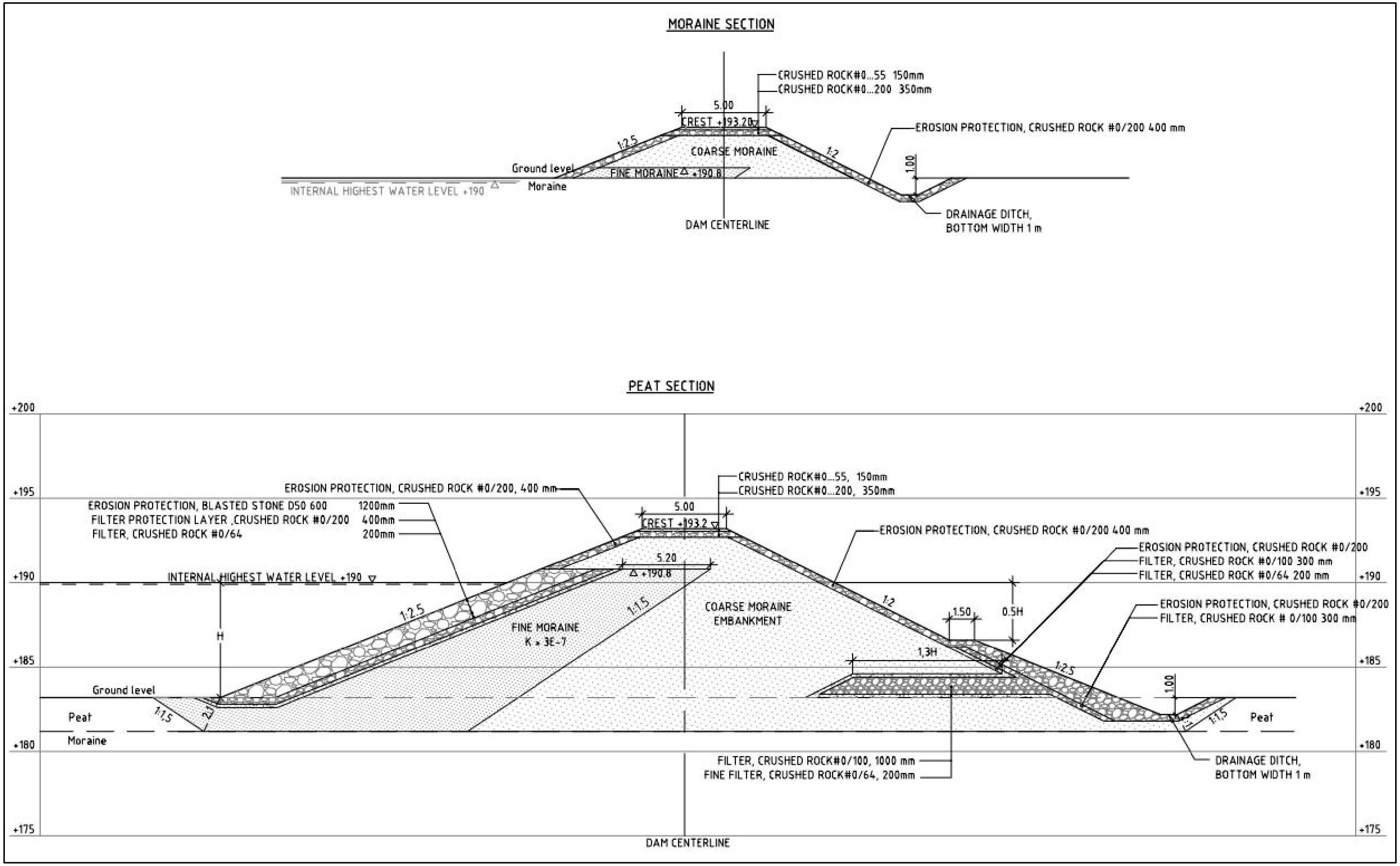


Figure 15-45: Typical section of the TMF Clarification Pond

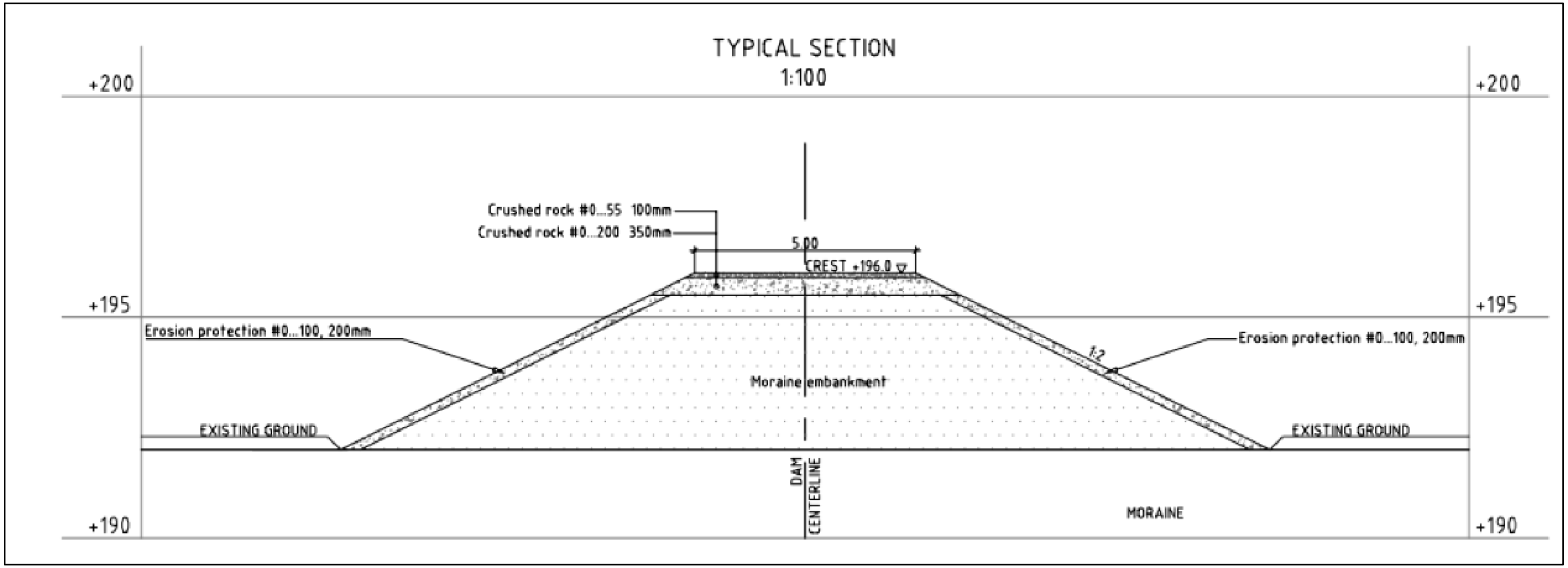


Figure 15-46: Typical section of the TMF Flood Cutoff Dam

15.10.7 Tailings handling

Pipelines

The tailings will be pumped from the mill to the TMF via pipelines. The pipeline will be placed on the ground over most of the route and will be adjacent to a service road to provide access for inspection and maintenance. The pipelines will be buried at the railway crossing. Purging basins with equipment and valves will be included along the pipe route.

The LIMS tailings will be pumped via carbon steel pipe (P355GH material) with an outside diameter of 267 mm and an inner HDPE lining 5.5 mm thick. Corrosion protection outside the pipeline will consist of epoxy painting (240 microns).

The pipeline route will enable the use of unreinforced plastic pipe material over the last portion of the pipeline, mainly inside the TMF area. The plastic pipelines can either be HDPE slurry pipes or standard HDPE pipes.

The High-S tailings will also be pumped via carbon steel pipes but a smaller outside diameter (133 mm) and an 8 mm thick inner lining. Outer corrosion protection will also be epoxy paint. The last 800 m of the High-S tailings pipeline can be unreinforced plastic pipe, as for the LIMS tailings.

Pumps

Piston pumps were chosen for LIMS and High-S tailings pumping on the basis of:

- up to 93 % mechanical efficiency (for large piston diaphragm pumps);
- low wear due to low internal velocities;
- higher pressure yield;
- better constant flow rate maintenance;
- better management of higher density and higher viscosity slurries; and
- lower operating costs (although higher capital costs).

The pumps will need to have a maximum discharge pressure of 120 bar and the pipeline designed to sustain a maximum operation pressure of at least 120 bar.

Spigotting

The spigot system for the LIMS tailings will include spigot pipes equipped with a minimum of six shut-off valves. At least two to four spigot pipe valves will be opened at any given time during normal operation. The spigot pipes will be made of HDPE plastic material.

The spigot system for the High-S will be similar to the LIMS tailings. It will consist of spigot pipes attached to the tailings pipeline with T-collars. A shut-off valve for alternation of the spigot points will be installed after each T-collar. One or two discharge valves will be kept open during normal operation.

15.10.8 Monitoring

Tailings

Tailings beach slope shape and consolidation rates have to be monitored continuously during operation to confirm the consolidation of the tailings, the bulk density and the storage capacity.

Dams

The performance and operation of the dams will be monitored during operation and post-closure. The monitoring program would include items/tasks such as:

- phreatic surface and/or pore pressure within the dams;
- seepage rates and quantities;
- deformation and consolidation; and
- dam inspection by a competent dam engineer.

The above monitoring would include periodic review of the collected data and the assessment of the dams in relation to the original dam designs.

Water

Discharge water quality from the LIMS and High-S tailings will be monitored periodically as part of the mine environmental and operational monitoring program. Further details are provided in the HIA report (SRK 2013a).

15.10.9 Closure

The LIMS tailings will be reclaimed by placing a cover that will consist of a bentonite mat placed directly on the deposited tailings and covered with 1.0 m of soil to protect the bentonite liner and to support vegetation. Progressive reclamation of the LIMS tailings will be possible during operation.

The High-S tailings will be reclaimed by placing a qualified cover over the consolidated tailings. The cover will include a bentonite mat on top of the tailings, an HDPE liner, a protective geotextile, and a 1.0 m thick soil cover at the surface. The placement of the cover will require that the High-S tailings are dewatered sufficiently and consolidated to minimise the risk of deformation of the liners. The dual cell approach for the deposition of the High-S tailings will also enable progressive closure during operation.

15.11 Tailings Operating Costs

The operating cost was estimated as per the project cost template and includes the following elements:

- labour;
- fuel / power;
- maintenance / lubrication;
- spares / consumables / tyres; and
- equipment rentals.

The cost for environmental monitoring has been covered elsewhere.

Operating personnel required for tailings management is:

- tailings and dam technician (WC-9; annual salary EUR76,312 or USD97,673); and
- water treatment and pump technician (WC-7; annual salary EUR65,628 or USD83,998).

The labour costs will include salary, allowances, benefits and a payroll burden to cover pensions, health insurance, taxes and other fees. The OPEX calculation covers only the labour costs once plant production has started. It is assumed that the employee costs will be capitalized within the Owners Costs prior to production start-up.

Power consumption consists mainly of power consumption for tailings pumping. Also included are power consumption costs of smaller water pumping stations for dewatering during dam construction. Yearly power consumptions are estimated with average tailings flow rates.

The OPEX energy costs were calculated using the project energy unit rate of EUR0.0507/kWh.

The tailings pumping availability/utilisation is estimated to be 90% (operating 7,884 hours per year).

The tailings pumping costs for LIMS tailings pumping are estimated to be USD326,000 per year for the first six years of operation. The energy consumption subsequently increases because the tailings will be deposited with a longer pipeline that increases the pressure drop in the pipeline. After year 6, the annual energy consumption of LIMS tailings pumping is estimated at USD540 000 per year.

The annual tailings pumping costs for High-S tailings are estimated at USD114,000 per year and remain constant over the entire LoM.

Pipeline maintenance costs are based on annual maintenance of 3% of capital costs. Pump maintenance costs are included in the spare part costs. Pipeline maintenance costs including spare parts for pipelines (flanges, replacement of deteriorated pipe sections) vary between USD60,000 and USD65,000 per year per pipeline, depending on the pipeline section.

The cost for pump spare parts is based on the Tamflow budgetary proposal for service and spare parts to support LIMS and High-S tailings pumping. The annual cost for spare parts for tailings pumping is estimated at USD134,000 per year for the entire LoM.

The cost for new spigotting pipe (300 m) for every third year is presented as a spare cost. The cost is estimated to be total USD153,000 for the LoM.

The equipment cost item includes the cost for one FEL to support the operation and maintenance of the TMF. For budget purposes, the FEL will be provided by a local contractor and is expected to work 2.5 hours/day annually over the entire life of mine.

The OPEX costs related to the TMF are summarised in Table 15-58.

Table 15-58: Summary OPEX for TMF

Category	USD1,000
Labour	3,361
Fuel / Power	10,767
Maintenance / Lubrication	6,062
Spares / Consumables / Tyres	2,560
Equipment rentals	1,385
TOTAL	**24,135**

15.12 Tailings and Clarification Pond Capital Costs

The initial and sustaining CAPEX are allocated altogether to four WBS-coding areas as presented below in Table 15-59. This estimate excludes any contingency for Project risks which have been assessed elsewhere. Costs allocated to PS and PP are detailed in their respective sections.

Table 15-59: Tailings CAPEX costs

Area code	Description	Estimated Initial CAPEX (USD1,000)	Estimated Sustaining CAPEX (USD1,000)	Comment
PS	Project Support	1,565	4,309	Investigations, engineering and construction management
IA	Industrial Area	12,241	53,874	Dams, liners, earthworks and piping
PP	Process Plant	4,576		Tailings pumping
Total		**18,382**	**58,169**	

15.12.1 Basis of the CAPEX Estimate

Dams and earthworks

The cost estimates presented in this section were determined by Pöyry. The unit prices were derived from actual budgetary proposals, discussion with suppliers and in part from Pöyry in-house cost database.

The contractors contacted by Pöyry were Lemminkäinen Infra, NCC Rakennus Oy, Tapojärvi Oy and E. Hartikainen Oy.

All the cost estimates are originally calculated in EUR, including the requested budgetary proposals.

The construction costs are based mostly on average unit prices derived from the budgetary proposals submitted by the contractors. The cost for blasted rock, crushed rock and locally sourced materials such as moraine and waste rock include the entire cost (excavation, processing, handling, hauling and placement).

The costs for design, field investigations and construction management are included in the costs presented herein.

Tailings handling

The cost estimates are based on quantities calculated by Pöyry and the unit prices were derived from actual budgetary proposals, discussion with equipment suppliers and in part from Pöyry in-house cost database.

Equipment quotes for tailings pumps, pipes and valves were provided by several suppliers. Pump costs are based on average prices from the budgetary proposals received from Schwing and Putzmeister for piston pumps (November 2013).

Pipeline costs are calculated from pipeline unit prices for the chosen pipe material as per the budgetary proposals received from KWH Pipe (November 2013) and Fineweld (December 2012). The unit prices were adjusted in proportion to the designed wall thicknesses. Other costs are based on Pöyry in-house estimates on corresponding materials and equipment from other projects.

15.12.2 CAPEX Summary

The construction of the various TMF components will take place throughout the LoM. For instance, the High-S tailings will only be commissioned after production is initiated, will consist of two separate cells, and with both cells to be constructed in three stages. A simplified construction schedule of the key TMF components is shown in Figure 15-47 below.

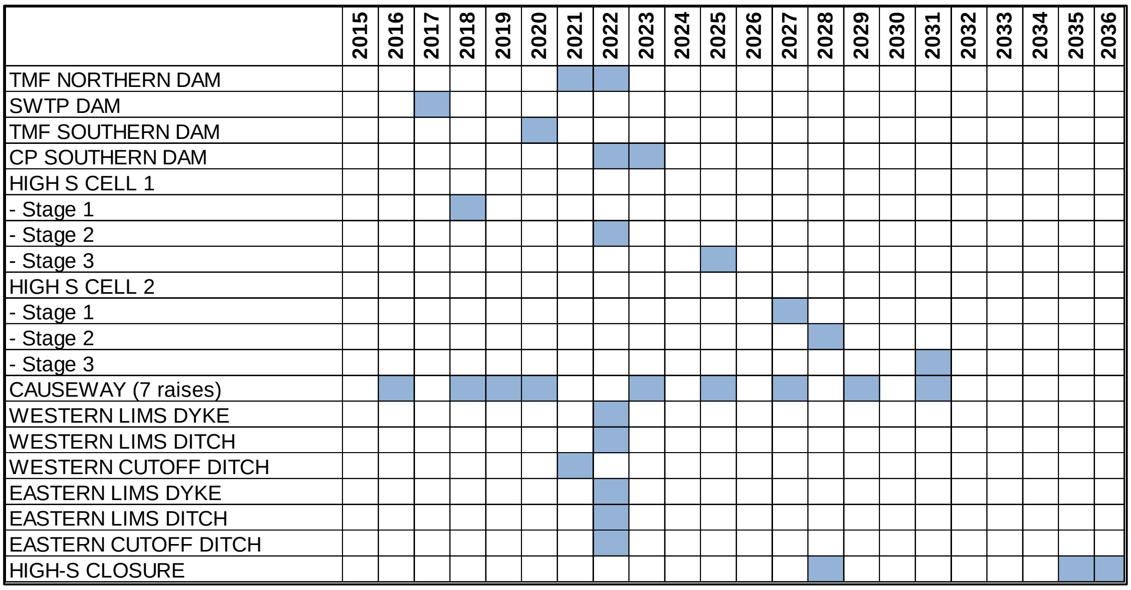


Figure 15-47: TMF construction schedule

The initial and sustaining CAPEX related to the Industrial Area TMF can be summarized as shown in Table 15-60.

Table 15-60: TMF CAPEX costs

Cost item	Initial CAPEX (USDk)	Sustaining CAPEX (USDk)	Comments
High-S Cell	-	40,080	Investment upfront (year 2018) USD11.2M
Eastern LIMS Dyke and Ditches	-	2,262	Investment year 2022
Western LIMS Dyke and Ditches	-	1,488	Investment year 2022
SWTP dam raise	908	-	Before start-up
TMF Northern Dam	-	1,478	Investment years 2021 and 2022
TMF Southern Dam	-	3,181	Investment year 2020
CP Southern Dam	-	3,450	Investment years 2022 and 2023
Causeway	1,224	2,085	Raises in seven stages, with initial stage before start-up
High-S Closure	-	6,396	Placement of a bentonite Mat and HDPE liner prior to placement of a soil cover. Cell 1 in year 2028; Cell 2 in year 2036
	2,132	**60,420**	

The Industrial Area Tailings handling initial and sustaining CAPEX are summarized in Table 15-61.

The closure costs listed above for the High-S tailings cells consist of placing geosynthetic liners over the consolidated High-S tailings. This is to provide a hydraulic barrier prior to placing a soil cover as part of the Closure Plan. The cost for the final soil cover is included in the closure costs.

Table 15-61: Summary of initial and sustaining CAPEX for Industrial Area Tailings handling

Cost item	Initial CAPEX (USDk)	Sustaining CAPEX (USDk)	Comments
TAILINGS PIPELINES			
LIMS Tailings pipelines I - IV	4,776	4,330	Investment year 2023
High-S Tailings Pipelines I and II	4,083		
OTHER COST ITEMS			
Emergency basin and rinse water connections for pumps	866		
Spigotting arrangements	134		High-S and LIMS
	9,859	**4,330**	

16 RECOVERY METHODS

16.1 Metallurgical Testwork

16.1.1 Background

The Project includes several identified ore bodies, Laurinoja (sometimes called Central Hannukainen), Kuervaara, Vuopio, Lauko, Kuervittiko and Kivivuopio. The first four are suitable for one open pit operation, the Hannukainen open pit. Kuervittiko, located about 2 km north of the Lauko ore body, will probably be mined separately at the end of the planned operation, if found to be feasible. Kuervaara and Vuopio are similar in nature and are treated as one ore body (Kuervaara-Vuopio) for metallurgical purposes. If exploited, the Kivivuopio ore body would have to be mined as an underground operation, and is reported to be similar to Kuervaara-Vuopio.

The valuable minerals are magnetite and chalcopyrite. The gold value reports with the chalcopyrite concentrate.

The iron ore will be upgraded in a 6.5 Mtpa processing plant using conventional technology. Crushing will be located at the mine site and the process plant will be located at Rautuvaara, approximately 8 km from the Hannukainen open pit. The magnetite concentrate will be shipped via train to the port of export. The copper / gold concentrate will be exported by truck from the Hannukainen site.

Over the LoM, the Hannukainen ore contains 32% to 34% Fe for the first 10 years and then slowly decreases with the introduction of lower quality ore, and the S grades fluctuate between 2% to 2.8% depending on the ore source, with the lowest S grades from the Hannukainen Central area. The average S grade over the LoM is 2.4%.

16.1.2 Historical Processing

Historically, the Hannukainen deposit was previously mined by Rautaruukki Oy in 1981 to1987 and for a short time in the period 1990 to1995 by Outokumpu Oy. The process plant incorporated grinding followed by LIMS to recover magnetite followed by a flotation circuit for recovery of copper and gold. The copper circuit was installed on the magnetite circuit tailings and incorporated regrinding to 60% passing 74 microns. The reported iron recovery was 59.9% into a concentrate containing 66.9% Fe and the reported recovery for copper was 92.6% at a concentrate grade of 22.2% Cu.

16.1.3 Mineralogy and Chemical Composition

Ore mineralogy has been studied by XRD, electron microprobe and QEMSCAN, mainly on different metallurgical composite samples.

The ore mineralogy is relatively simple. Magnetite is the only iron oxide mineral and haematite has not been observed.

Sulphides grade varies between 3 to 20%. Chalcopyrite is basically the only copper mineral. Two iron sulphides, pyrite and pyrrhotite, are present. Pyrrhotite represents about 50% of the sulphide mineralisation. As it is mainly monoclinic and therefore weakly magnetic, it reports with the magnetite and has to be removed from the concentrate by flotation to reduce the sulphide content of the product to acceptable levels.

The main silicate phases present are diopside, plagioclase, amphibole and biotite with minor phases of chlorite, quartz with apatite. Details are given in Table 16-1.

Table 16-1: Main silicate phases present

Mineral	Nominal Wt % of sample	Notes
diopside	20%	monoclinic pyroxene
plagioclase	10%	
amphibole	10%	
biotite	5%	
Chlorite, quartz, apatite	Minor amounts	

In general terms the magnetite and chalcopyrite minerals do not contain significant levels of chemical impurities. Details are given in Table 16-2.

Table 16-2: Average Chemical Composition of Magnetite and Chalcopyrite

	Magnetite		Chalcopyrite	
	Average, %	Std.Dev.	Average,%	Std.Dev.
Fe	70.4	0.407	30	0.287
TiO_2	0.06	0.051	-	-
Al_2O_3	0.283	0.184	-	-
V_2O_3	0.022	0.019	-	-
Cr_2O_3	0.012	0.017	-	-
MgO	0.199	0.297	-	-
MnO	0.14	0.08	-	-
P_2O_5	0.007	0.009	-	-
SiO_2	0.164	0.189	-	-
Co	-	-	0.056	0.078
Ni	-	-	0.004	0.008
Cu	-	-	34.7	0.348
S	-	-	34.6	0.163

Approximately 80% of the iron is in the form of magnetite, approximately 10% is associated with the gangue silicates, with the balance, 10%, as sulphides, mainly pyrrhotite.

QEMSCAN analyses have shown that the magnetite grain size is coarse, >300 microns, and in material ground to P80 70 to 100 microns the magnetite liberation is 92 to 96%. Less than 0.2% of the magnetite is associated with pyrite and chalcopyrite. Approximately 1% of magnetite is associated with pyrrhotite.

The chalcopyrite grain size is typically 20 to 30 microns. When locked the main association of chalcopyrite is with magnetite.

No significant surface oxidation has been found in Hannukainen and macroscopical observation indicates that some rusty oxidation can be seen at the surface to a depth of no more than 50 to100 mm.

16.1.4 Metallurgical Testing

Ore Classification

Over the period of the testing, the Hannukainen ore bodies have been classified into geometallurgical types in three different stages. These classifications have been used as the basis for selecting and preparing metallurgical samples for testing.

In the initial work, SGS used multivariate statistical analysis to divide the ore body into three different geometallurgical types, called clusters. Cluster 1 was high iron (average 46.8% Fe), cluster 2 was low iron (average 29.3 % Fe), and cluster 3 was material below the cut-off grade.

In general terms, the ore consisted of approximately 23% of Global Cluster 1 ore types and 77% of Global Cluster 2 ore types and this mixture was used in guiding the process development work both on individual samples, as well as on composites.

The main problem with this cluster classification was that there was no differentiation between ore types containing different levels of copper, sulphur and minerals such as pyrite and pyrrhotite. In 2011, new ore typing for metallurgical testing was established by Northland based on geographical, chemical and mineralogical factors. Generally, a CoG of 15% Fe was used. Laurinoja was divided into six types based on iron and sulphur grades, Kuervaara-Vuopio into two types by iron grades, Lauku to one ore type and Kuervitikko in to two types based on iron and copper grade. The classification was used as a basis for the selection of variability testing and the samples were called variability ore type samples, designated VOT.

In 2012, the VOT typing was re-evaluated and it was found that the recovery of iron was not only dependent on iron but also on sulphur. Additionally, the response of different ore bodies was shown to be very similar (Laurinoja, Kuervaara-Vuopio, Lauku, Kuervitikko) and therefore the geographical classification was dropped. The spatial distribution of the Fe and Cu recoveries was captured into a geometallurgical classification and the Hannukainen ore body was divided into high, medium and low recovery types for both iron and copper resulting in a total of nine classes of which six were significant in terms of ore tonnage. The new samples were termed GEM geometallurgical typing (“GEM”). This classification was more pertinent and captured the differences in both iron and copper recoveries in terms of iron, copper and sulphur contents and allowed the recovery functions, especially for iron, to be further defined to take account of the effects of pyrrhotite on overall iron recovery.

Samples

The samples used for the different stages of metallurgical testing are detailed in Table 16-3.

Table 16-3: Samples used in the metallurgical testing.

Sample	Loc	Ref	Fe%	Cu%	Au g/t	S XRF%	Fe Rec%	Cu Rec%	GEM type	Notes
SGS Testing										
HAN7009A	LO	2	37.8	0.390	0.080	2.680	72.4	92.5	3 M-H	
HAN7009B	LO	2	35.2	0.260	0.210	2.710	69.3	86.3	3 M-H	
Low-Cu-Comp	LO	2	28.0	0.069	0.020	2.330	72.1	66.1	5 M-M	
High-Cu-Comp	LO	2	22.1	0.370	0.200	2.730	59.2	91.7	3 M-H	
High-Fe-Comp	LO	2	54.3	0.190	0.050	2.710	92.5	81.5	1 H-H	
High-Au-Comp	LO	2	42.0	0.390	0.240	2.450	85.7	92.5	1 H-H	
BC1/3	K-V	3	35.9	0.088	0.020	3.970	64.4	69.8	5 M-M	
BC2	K-V	3	32.6	0.170	0.020	7.360	43.8	79.8	8 L-M	
GC1	HAN	3	47.9	0.260	0.170	2.520	81.3	86.3	1 H-H	
GC2	HAN	3	29.0	0.210	0.120	2.580	61.0	83.1	3 M-H	
GC3	HAN	3	13.8	0.110	0.030	1.670	28.2	73.2	8 L-M	
C3a	HAN	4	22.8	0.100	0.060	2.530	49.1	71.7	8 L-M	
SGSPPFeed	LO	9	43.5	0.320	0.140	2.460	78.5	89.5	1 H-H	Pilot sample
2010-2011 Testing										
PP Feed	LO	14	33.1	0.182	0.060	2.567	67.2	80.8	3 M-H	Pilot sample
VOT-1	LO	14	21.2	0.219	0.020	1.315	52.0	83.7	6 L-H	
VOT-2	LO	14	40.0	0.332	0.170	1.760	78.4	90.0	1 H-H	
VOT-3	LO	14	52.3	0.382	1.020	1.415	87.6	92.2	1 H-H	
VOT-4	LO	14	40.0	0.458	0.090	4.020	68.9	93.8	3 M-H	
VOT-10	KUE	14	24.0	0.076	0.020	2.075	54.1	67.7	8 L-M	
VOT-11	KUE	14	44.2	0.132	0.025	2.990	76.9	76.0	5 M-M	
VOT-20	LK	14	35.0	0.123	0.025	2.090	71.9	74.9	5 M-M	
VOT-30	KVI	14	26.8	0.255	0.354	3.800	50.8	86.0	6 L-H	
VOT-31	KVI	14	18.8	0.093	0.079	1.260	46.0	70.6	8 L-M	
2012 Testing										
GEM-1	LO	37	47.3	0.075	0.000	1.450	84.9	67.4	2 H-M	
GEM-2	LO	37	37.4	0.404	0.190	2.470	72.9	93.0	3 M-H	
GEM-3	LO	37	36.1	0.179	0.500	1.930	73.8	80.6	3 M-H	
GEM-4	LO	37	25.1	0.129	0.110	1.740	58.2	75.6	5 M-M	
GEM-5	LO	37	36.1	0.184	0.080	1.420	76.1	81.0	3 M-H	Average GEM-1..4
GEM-6	K-V	37	40.4	0.111	0.090	2.930	73.9	73.3	5 M-M	
GEM-7	K-V	37	29.8	0.145	0.000	2.610	62.2	77.4	5 M-M	
GEM-8	LK	37	45.7	0.099	0.000	2.150	81.3	71.6	2 H-M	
GEM-9	LK	37	35.8	0.145	0.000	2.870	69.3	77.4	5 M-M	
GEM-10	LO	37	7.7	0.063	0.000	0.820	7.4	64.7	9 L-L	Waste rock
GEM-11	LO	37	36.3	0.369	1.770	4.010	64.8	91.6	3 M-H	
GEM-12	LK	37	38.5	0.141	0.110	5.810	59.6	77.0	5 M-M	

Location: HAN=Hannukainen, KUE=Kuervaara, K-V=Kuervaara-Vuopio, KVI=Kuervitikko, LK=Lauku, LO = Laurinoja.

The initial mineralogical and metallurgical work at SGS was conducted on twin holes from Laurinoja. The samples prepared for the metallurgical testing were close to the average ore for iron, but anomalously rich in copper, 0.25 to 0.40% Cu, compared to an average 0.18% Cu. In terms of the GEM-typing, this sample corresponds to 3 M-H or medium in terms of iron recovery, but high for copper recovery. From the same drillholes, five samples for the variability testing were selected which represent the ranges for iron and copper with the exception of the very low end of the copper. With reference to GEM typing, the samples included are of types 1 H-H and 3 M-H, but 5 M-M is missing.

In the 2010, SGS testwork cluster samples were collected based on the SGS geometallurgical classification from drillholes representing the whole Hannukainen ore body. The Cluster samples covered quite a broad range in terms of GEM classification, in terms of iron recovery classification from low to high and in terms of copper recovery from medium to high.

In the 2010 testwork, including SGS piloting, some high-grade Laurinoja ore boulders from the Outokumpu operation were used. Compared to average ore, they were rich both in iron and copper. In the GEM typing, this sample is designated high Fe recovery and high Cu recovery.

In 2011, VOT samples were collected for the variability testing and pilot plant test was run with a sample prepared from large number of drill cores. These samples are more representative geographically, geologically and geometallurgically than the previous samples. The pilot plant feed sample (“PP Feed”) was generally from the medium iron recovery, high copper recovery set.

In 2012, the GEM samples were collected using the new geometallurgical model. The GEM samples cover a broader range of variation, especially for sulphides both pyrrhotite dominant and pyrite dominant. Due to limitation in the drillholes available, low copper recovery classes were not included.

The metallurgical samples have covered all the different blocks: Laurinoja, Kuervaara-Vuopio, Lauku and Kuervitikko. In terms of iron grade and iron recovery classes, the sample set covers the whole range from low to high. For copper, the samples cover the range with the exception of the low copper recovery set.

Metallurgical Testwork

Since the acquisition of the property in 2005, Northland has undertaken a number of test programs:

1. initial test work 2007 to2010 by SGS Lakefield;
2. 2010 test work by SGS for PEA;
3. 2011 test work in the HFS Phase I by SGS and GTK; and
4. 2012 test work for the HFS Phase II by Geological Survey of Finland.

The testing performed and the sample types are summarised in Table 16-4.

Table 16-4: Testwork phases and test types conducted

Testwork	Samples	Purpose	Tests	Lab	Scale
Initial testwork	HAN07009	Metallurgical characterisation, flowsheet development, ore variability	Comminution, magnetic separation, flotation tests	SGS	Lab
2010 testwork for PEA	5 cluster samples and 58 variability samples	Flowsheet development, ore variability	Comminution tests, Davis tube tests, overall flowsheet testing.	SGS	Lab
2011 testwork	Boulders (124 tonnes)	Pilot test	Fully Autogenous Grinding/Semi Autogenous Grinding comminution pilot, full flowsheet pilot	SGS	Pilot
	9 VOT	Flowsheet development, ore variability, pilot plant testing	Overall flowsheet testing, grinding mini pilot testing	GTK	Lab & pilot
2012 testwork	12 GEM	Check primary grind size P_{80}, amenability of flash flotation, ore variability	Flash flotation, overall flowsheet	GTK	Lab
2013 testwork	average ore (GEM-5) &2 high sulphur (GEM-11 & 12)	Investigate optimum grind size on Fe recovery and effect of dispersant on iron losses in pyrrhotite flotation	Laboratory grind and copper and pyrrhotite flotation tests	GTK	Lab

Over the course of the testwork, three flowsheets were considered.

Flowsheet #1 reflected the original plant with grinding followed by LIMS. The non-magnetic fraction from this stage was floated to recover chalcopyrite, whilst the magnetic stream was cleaned to remove sulphur by pyrrhotite flotation.

In Flowsheet #2, copper rougher flotation was performed after grinding, followed by sulphide flotation to remove pyrite and pyrrhotite, followed by LIMS to produce a magnetite concentrate.

Flowsheet #3 also incorporated copper rougher flotation after grinding, followed by LIMS to produce a combined magnetic concentrate containing magnetite and pyrrhotite which was then cleaned to remove the sulphur bearing pyrrhotite by flotation.

The final flowsheet was a modified flowsheet #3 with a pyrite flotation stage after copper roughing prior to LIMS and included copper regrinding in the copper cleaner circuit to produce an acceptable copper concentrate. This flowsheet resulted in better metal recoveries.

Initial Testwork

Most of the initial testwork was conducted by SGS Lakefield in Canada, starting in 2007.

Preliminary grindability testing indicated that ore would be suitable for fully autogenous grinding (“FAG”) or semi-autogenous grinding (“SAG”).

Davis tube tests showed that iron recovery correlates with iron head grade. However, it was found that the magnetic concentrate contained up to 9% sulphur indicating the presence of monoclinic pyrrhotite.

Full flowsheet tests included copper flotation, magnetite recovery by magnetic separation followed by pyrrhotite flotation to reduce the sulphur level showed that good quality iron and copper concentrates could be produced.

SGS evaluated a geometallurgical model of the ore body and prepared five cluster samples for further for metallurgical testing. Mineralogical and metallurgical testing showed that there was significant geometallurgical variation within the ore body and that iron recovery was reduced when testing samples with higher amounts of pyrrhotite.

2010 Testwork

Additional testing was conducted by SGS in 2010 for the PEA. Preliminary testing using most of the recognised testing methods were performed.

SAG mill comminution (“SMC”) tests on the five cluster samples from Hannukainen indicated that the ore is relatively soft with regards to impact breakage (mainly related to crushing and primary grinding) and medium according to Bond Work Index (related to fine grinding). It was found that competency/hardness generally decreased with increasing iron grade but did not affect the bond work index at finer sizes.

Davis tube testing on samples from Hannukainen and Kuervitikko drill cores indicated variability in the ore bodies.

The testing showed that the flowsheet #3 gives the better metallurgical results.

2011 Pilot Plant Testing

Two major testwork programmes were conducted in 2011.

The first program of work included pilot plant operation focused on providing sufficient data for a complete comminution model for FAG and SAG. In conjunction with the grinding pilot plant operation, metallurgical bench testing and a pilot plant program was undertaken using flowsheet #3.

The work was conducted by SGS in Canada and the material was collected from existing boulders on the mine site. The sample was high in both iron and copper. Pilot plant tests with the flowsheet #3 showed very good metallurgical performance: 95.0% recovery of copper to a concentrate at 25.2% Cu grade; and a magnetite concentrate at a grade of 71.3% Fe at 75.4% recovery. In addition, a final magnetite concentrate was produced for marketing evaluation purposes and tailing samples for initial environmental testing.

The second test program was undertaken at GTK and was designed to provide representative samples for variability testing, a mini pilot plant to generate data for metallurgical recovery expressions and to provide data for the HFS plant design. The testing indicated that a finer grind may be required for satisfactory recovery of copper.

During this testing, it became apparent that the primary LIMS tailing were acid generating and that the sulphur level had to be reduced. An additional flotation stage to remove pyrite was tested and found to be beneficial.

The results from the SGS pilot plant and the GTK pilot plant are given in Table 16-5 and Table 16-6.

Table 16-5: SGS Pilot Plant Results (2011) – Copper and Pyrite Concentrates

Pilot Plant Run	Date	Feed Grade (%)			% mass	Grade (%)			Recovery (%)		
		Cu	S	Au (g/t)	yield	Cu	S	Au (g/t)	Cu	S	Au
SGS PP-10	02-May-11				1.3	30.1	34.4	7.1	95.2	16.4	74.3
SGS PP-11	03-May-11				1.4	23.1	37.8	8.8	96.0	20.3	79.7
SGS PP-12	04-May-11				1.3	22.7	37.8	8.1	93.6	20.0	76.6
SGS PP-13	05-May-11				1.4	24.7	36.2	7.7	95.2	20.3	72.9
Average of SGS PP-10 to 13		0.32	2.46	0.14	1.3	25.2	36.5	7.9	95.0	19.2	75.9

Table 16-6: Overall mass balance from GTK pilot plant

	Yield	Analysis (XRF)			Recovery %		
	% of feed	% Cu	% Fe	% S	Cu	Fe	S
Feed	100.00	0.17	33.10	2.29	100.0	100.0	100.0
Final Copper concentrate	0.53	25.00	28.36	29.56	76.6	0.5	6.2
Magnetite Concentrate	31.56	0.00	71.51	0.03	0.4	68.2	0.4

2012 Laboratory Testing

In 2012, following an independent review of the historical testwork results a series of laboratory tests were performed to evaluate the effect of process changes aimed at improvement in the metallurgical performance and in capital and operational costs. Flash Flotation of copper and a coarser grind were evaluated.

The previous VOT samples were not available and 12 new GEM samples were collected based on the modified geometallurgical classification defined by Professor P Lamberg.

Three different grinds, P80 of 65, 90 and 120 microns were investigated. Tests with two samples, GEM-5 representing the average ore and GEM-11 representing the high sulphur ore, showed that iron recovery increases by 3 to 7% with a coarser grind size of P80 90 microns compared to the 65 microns grind identified in the 2011 testing. Al_2O_3+SiO_2 were below 3% in all the tests and sulphur content was generally below 0.1%. The only exception was for very high sulphur feeds.

Flash flotation and rougher flotation tests demonstrated that Hannukainen ore is amenable to flash flotation and that a circuit incorporating both flash flotation and rougher flotation gave the potential for an additional 3% copper recovery.

With the modified circuit, incorporating a grind of P80 90 microns and flash flotation of copper, updated iron and copper recovery functions with an estimated 0.25% iron recovery and 1% higher copper recovery were proposed.

Additional piloting was recommended to verify these results and to optimise copper cleaner flotation and pyrrhotite flotation.

2013 Laboratory Flotation Testing

Further laboratory scale flotation testing was performed at the GTK laboratories in 2013 using three different GEM samples, one representing average ore (GEM-5) and two high sulphur samples (GEM-11 & 12). The objective was to investigate the effect of grind size on copper and iron recovery and to test the effect of dispersion agents in pyrrhotite flotation. The tests showed that iron recovery increases with grind size, but the maximum copper recovery occurs around P80 120 microns. At this grind, the final magnetite concentrate is within specifications, even for the high sulphur ores. The sulphur grade in the final magnetite concentrate was shown to decrease at coarser grinds. While the Al_2O_3 and SiO_2 content increased, it was still well below 3% w/w. Coarsening the P80 from 90 microns, as defined in 2012 test work, 120 microns gave an increased iron recovery of approximately 3%.

Microscopical studies have shown that magnetite losses occur in pyrrhotite flotation mainly as liberated fine magnetite particles. Pyrrhotite flotation tests using a dispersant improved the recovery of iron by up to up to 5% on the GEM-5 sample.

These results are considered to be indicative and have not been incorporated in the HFS design or the financial evaluation but should be investigated further prior to final plant design.

Metallurgical Recovery Functions

Iron Recovery Function

Initial evaluation of iron recovery did not take account of the effect of pyrrhotite on iron recovery.

Hannukainen

Following the 2011 bench scale testing two iron recovery functions were proposed for feed with high sulphur grades (>2.5% S) and the other for lower sulphur feed grades (<2.5% S). The iron recovery functions are:

If the sulphur level in the feed is less than 2.5% w/w:

$$R_{Fe}\ (S<2.5\%) = 0.003Fe^3 – 0.3628\ Fe^2 + 14.6635\ Fe – 118.5826 \quad [1]$$

If the sulphur level in the feed is equal to or more than 2.5% w/w:

$$R_{Fe}\ (S>2.5\%) = –0.0908Fe2 + 6.80409\ Fe – 63.7645 \quad [2]$$

Where:

R_{Fe} = the mass recovery of iron to the magnetite concentrate, % w/w

Fe = total grade of iron in the feed, % w/w.

These iron recovery functions have been used in the Ore Reserve estimates and for the HFS design.

Further evaluation of the available testwork data, together with the additional testwork performed in 2012, resulted in a revised iron recovery function which incorporated parameters for both Fe grade and S grade. The iron recovery function was divided into two parts: magnetic separation and pyrrhotite flotation.

The equation proposed was:

$$R_{Fe}\ (\%) = \left(98.5 * \left(1 - e^{-0.06038*(Fe\,\% - 6)}\right)\right) * \left(-1.962744 * \frac{S\%}{Fe\%} + 1\right) \quad [3]$$

Where S% is the sulphur content in the feed.

Based on the 2012 testwork, the iron recovery function was modified to reflect the higher iron recovery at the coarser grind of P_{80} of 90 microns. Statistically, the iron recovery is 0.25% higher. The revised iron recovery function is:

$$R_{Fe}\ (\%) = \left(98.5 * \left(1 - e^{-0.06038*(Fe\,\% - 6)}\right)\right) * \left(-1.962744 * \frac{S\%}{Fe\%} + 1\right) + 0.25 \quad [4]$$

It should be noted that the iron grade used is inclusive of all forms of iron notably magnetite, pyrite and pyrrhotite.

Kuervitikko

The average recovery of iron to the magnetite concentrate used is 52.1%. There is limited data available for this ore type and consequently this figure is considered to be to pre-feasibility study level. The general recovery figure is acceptable especially as this ore would be processed during the later years of the project.

The recovery figure for Kuervitikko ore is acceptable at this stage, although SRK recommends further metallurgical testwork should be performed to establish a relationship between feed grade and metal recovery to a magnetite concentrate and to allow a more detailed evaluation of the viability of treating this material.

SRK considers that the predicted recoveries for Hannukainen and Kuervitikko used in the financial modelling, based on the applied recovery functions, are a good representation of the performance that should be achieved on the plant.

Copper Recovery Function

The function for copper recovery was based on selected results from both the testing of VOT samples and the 2011 GTK pilot plant. Testing was not optimised and it must be noted that certain samples gave lower recoveries. Based on selected tests, the following recovery function for copper was proposed:

$$Cu\ Rec\% = 15.249 * \ln(Cu\%) + 106.85 \quad [5]$$

In 2012, flash flotation of copper was investigated. In general terms, the results indicated that an additional 3% recovery of copper to the rougher concentrate was possible. Detailed cleaner tests were not performed and, conservatively, an additional 1% recovery has been added to the copper recovery function:

$$Cu\ Rec\% = 15.249 * \ln(Cu\%) + 107.85 \quad [6]$$

Further testwork is required to optimise copper rougher and cleaner flotation and to identify the reasons for the poor flotation response of some ore types.

Gold Recovery

The initial gold recovery function developed by Ardvison based on bench scale testwork was:

Gold Recovery % = 5.89*ln(Au g/t) + 38.9 [7]

Based on typical feed grades of 0.05 to 0.3 g/t the estimated gold recoveries using this finction are 21 to 31 %.

Gold reports to the copper concentrate. There is some evidence that the gold is associated with pyrite and consequently the gold recovery is probably related to the amount of pyrite reporting with the chalcopyrite to the copper concentrate.

Preliminary testwork results were variable and it has not been possible to establish a reliable gold recovery function. Typical gold recoveries for ore from different locations are:

- Laurinoja 76.7% (Pilot plant)
- Lauko 80.0% VOT 20
- Kuervaara-Vuopio 66.0% VOT 10+11
- Kuervittiko >0,14% Cu 28.0% VOT 30
- Kuervittiko <0,14% Cu 50.0% VOT 31

The gold recoveries achieved in the SGS pilot plant in 2011 were 75.9% and 74.7% from a feed containing 0.32% Cu and a gold content of 0.138 g/t.

The mine optimisation was based on the gold recovery function shown in equation [7]. Based on the later testwork and the pilot plant studies it is likely that gold recoveries to the copper concentrate will be higher which would have a positive effect on the mine optimisation and the overall project economics.

Overall the gold recovery testwork is considered to be to pre-feasibility study level.

Comminution Testwork

Comminution testwork has been performed on samples of Hannukainen ore including JKTech Drop Weight Testing (“DWT”), SAG Power Index testing (“SPI”), MacPherson grinding testwork, and Bond ball and rod mill work index determinations. Tests were performed on samples of Hannukainen ore.

In general terms, the material was shown to be suitable for AG or SAG applications.

In initial testing at SGS, the DWT indicated that the sample was relatively soft compared to other materials (A=75.6, b=1.45). The autogenous grinding work index from the MacPherson grindability test was 9.2 kWh/t. Comparative results were determined using the SPI method. Typically the rod mill work index is 7.4 kWh/t and the ball mill work index 18.2 to 19.7 kWh/t. Bond abrasion testing indicated that the samples are abrasive.

High Pressure Grinding Tests were performed but are reported to show no significant benefits from applying this technology.

In 2010, further SMC tests and additional ball mill work index testing was performed on different cluster samples and in general confirmed the findings from the earlier tests.

Pilot scale grinding tests were performed in the 2011 GTK pilot plant campaign.

Testing data has been reviewed by the major mill suppliers and has been incorporated in to the mill circuit design.

Miscellaneous Tests

Regrind jar testing has been performed by Metso on the copper rougher concentrate samples and the results have been used for sizing the Vertimill for copper regrind.

LIMS testwork has been performed on copper rougher tailing samples from the 2011 GTK pilot plant by two separate equipment suppliers. The results have been used for equipment sizing in the HFS.

LIMS tailings and pyrite stream thickener settling testwork has been completed by two separate equipment suppliers. The results have been used for equipment sizing in the HFS.

Magnetite and copper concentrate filtration testing has been undertaken by two equipment suppliers and has been used for equipment sizing in the HFS.

Transportable Moisture Limit

The copper concentrate and magnetite concentrate transportable moisture limit ("TML") were determined by Boliden Minerals AB in 2011 using a standard Proctor/Fagerberg test. The copper concentrate TML was 11.7% w/w. The magnetite concentrate TML was 8.5% w/w.

In both cases, filtration testwork has indicated that the final concentrates should contain moisture well below the measured TML.

Pelletizing Testwork

Pelletizing characteristics of the magnetite concentrate have been studied by COREM and show that the product has excellent properties. The relatively coarse concentrate particle size results in a low Blaine value but testwork has indicated that it can still be used directly for either DR or BF pellets Additional testwork may be required to confirm specific customer requirements.

Recovery Testwork Summary

For the avoidance of doubt, the following recovery functions have been applied to the Hannukainan financial model:

Iron (Hannukainen)

$$R_{Fe}\ (\%) = \left(98.5 * \left(1 - e^{-0.06038*(Fe\,\% - 6)}\right)\right) * \left(-1.962744 * \frac{S\%}{Fe\%} + 1\right) + 0.25 \quad [4]$$

Iron (Kuervitikko)

$$R_{Fe}\ (\%) = 52.1\ \%$$

Copper (All)

$$Cu\ Rec\% = 15.249 * \ln(Cu\%) + 107.85 \quad [6]$$

16.2 Process Plant

16.2.1 Design Basis

The processing facilities have been sized to treat 6.5 Mtpa of ore in a single stream concentrator to produce:

- 2.5 Mtpa of a magnetite concentrate with nominally 68% Fe content and less than 0.05% S; and
- 37,000 to 50,000 dry metric tonnes per year of copper concentrate containing 25% Cu and some gold.

It should be noted that the original plant design was prepared based on 6 Mtpa. Where appropriate, documentation was amended to reflect the increase in throughput to 6.5 Mtpa, equipment sizings were checked and capital and operating costs were recalculated. The engineering drawings were not amended.

The concentrator has been designed to process ore from both open pits, and the typical feed grades are given in Table 16-7.

The main design parameters for the concentrator are:

- Annual Throughput: 6.5 Mtpa
- Feed rate: nominal 825 tph
- Design margin: 15%
- Grinding circuit product: 80% passing 90 microns
- Iron recovery: recovery function equation [2 & 3]
- Copper recovery: recovery functions equation [6]
- Magnetite concentrate grade: > 68% Fe
- Copper concentrate grade: 25% Cu

The target magnetite concentrate parameters are shown in Table 16-8. The target copper concentrate product characteristics are shown in Table 16-9. The radioactivity of samples of both concentrates has been determined and is acceptable in both cases.

Table 16-7: Concentrator feed characteristics

	Hannukainen	Kuervitikko
Total Fe grade (%)	Average 32.92 Range 28.74 to 34.66	Average 25.37 Range 14.93 to 26.39
Copper grade (%)	Average 0.17 Range 0.07 to 0.31	Average 0.18 Range 0.08 to 0.20
Gold grade (g/t)	Average 0.080 Range 0.012 to 0.204	Average 0.214 Range 0.096 to 0.295
S grade (%)	Average 2.35 Range 3.08 to 1.8	Average 2.58 Range 1.35 to 2.74

Table 16-8: Magnetite concentrate characteristics

Parameter	Magnetite concentrate level
Iron grade (%)	Minimum 68%
Sulphur grade (%)	Less than 0.05%
MgO content (% w/w)	Less than 2%
SiO_2 content (% w/w)	Less than 1%
SiO_2 + Al_2O_3 content (% w/w)	Less than 2%
Final moisture content (% w/w)	5
Transportable moisture Limit ("TML) (% w/w)	8.5
Nominal particle size	80% passing 90 microns

Table 16-9: Copper concentrate characteristics

Parameter	Copper concentrate level
Copper grade (% Cu)	nominally 25%
Gold grade (g/tonne)	Not specified but represents approx. 75% gold recovery to copper concentrate
Final moisture content (% w/w)	6
Transportable moisture Limit ("TML) (% w/w)	10 to 11.7
Nominal particle size	80% passing 25 microns

16.2.2 Flowsheet

The selected process route for treatment of the Hannukainen ore is shown in the block flowsheet in Figure 16-1 and includes the following unit processes:

- primary crushing at Hannukainen;
- crushed ore overland conveyor to Rautuvaara:
- fine and coarse stockpile;
- primary AG/SAG milling with screen classification;
- secondary ball mill grinding and classification;
- flash flotation of fast floating copper minerals (located in the ball mill circuit);
- copper flotation (roughing, 3-stage cleaning, cleaner scavenger) including rougher concentrate regrind;
- copper concentrate thickening and filtration;
- copper concentrate truck load out;
- pyrite flotation (copper tailings stream);
- pyrite (copper cleaner tailings and pyrite flotation) concentrate thickening;
- primary magnetic separation (LIMS) of pyrite flotation tailings;
- pyrrhotite flotation on magnetic concentrate;
- secondary magnetic separation (LIMS) of pyrrhotite flotation tailings;

- magnetite concentrate dewatering (LIMS) and filtration;
- final magnetite concentrate storage and rail loadout;
- final copper concentrate storage and loadout;
- high density tailings thickening;
- low sulphur tailings storage; and
- High-S (pyrite and pyrrhotite) tailings storage.

Process Description

RoM ore is crushed by a primary crusher at the Hannukainen mine site. The crushed ore is transferred by a 9 km long overland conveyor to the stockpile at the concentrator in Rautuvaara.

At the Rautuvaara site, the crushed ore is screened into a fine and a coarse fraction. The two fractions are stored separately in the stockpile.

A feed blend of coarse and fine ore is extracted from the stockpile by apron feeders and conveyed to the grinding circuit in the concentrator. The grinding circuit comprises a primary AG/SAG mill operating in closed circuit with +2 mm screens followed by a secondary ball mill operating in closed circuit with hydrocyclones. The +2 mm screen oversize will be recycled to the primary mill screened and minus 2 mm material transferred to the secondary ball mill circuit.

A portion of the cyclone underflow in the ball mill circuit is processed in a flash flotation unit to recover coarse, fast floating chalcopyrite. The hydrocyclone overflow from the grinding circuit passes to the copper rougher flotation section.

The copper rougher concentrate, together with the flash flotation concentrate, is reground in a stirred mill regrind circuit to ensure adequate liberation of copper minerals. The regrind product is upgraded in three stages of copper cleaning. The final copper concentrate is thickened, filtered and stored, ready to be transported to market.

Copper first cleaner tailings pass to a cleaner scavenger stage for recovery of slow floating copper minerals and are recycled to the regrind circuit. The copper cleaner scavenger tailings are pumped to the pyrite concentrate thickener.

Copper rougher tailings are pumped to the pyrite flotation circuit for removal of sulphur rich pyrite. The pyrite flotation concentrate is thickened together with the copper cleaner scavenger tailings.

The non-floating fraction (the high iron fraction) from pyrite flotation is diluted prior to being fed to a number of parallel LIMS for magnetite recovery. Pyrrhotite is also recovered with the magnetite.

The LIMS concentrate is transferred to the head of the pyrrhotite flotation circuit to float the sulphur rich pyrrhotite to generate a final magnetite product with less than 0.05% w/w sulphur. The cleaned magnetite fraction is dewatered in a further magnetic separation stage followed by filtration. The filter cake is loaded onto trains for transportation to port and onward journey to customers.

The non-magnetic tailings streams from the LIMS and dewatering magnetic separation stages are combined and thickened in a high density tailings thickener. The thickened tailings are pumped to the low sulphur TMF.

The high sulphur pyrrhotite flotation concentrate is dewatered by magnetic separator and combined with the pyrite thickener underflow. The combined high sulphur tailings are pumped to the separate High-S deposition area in the TMF.

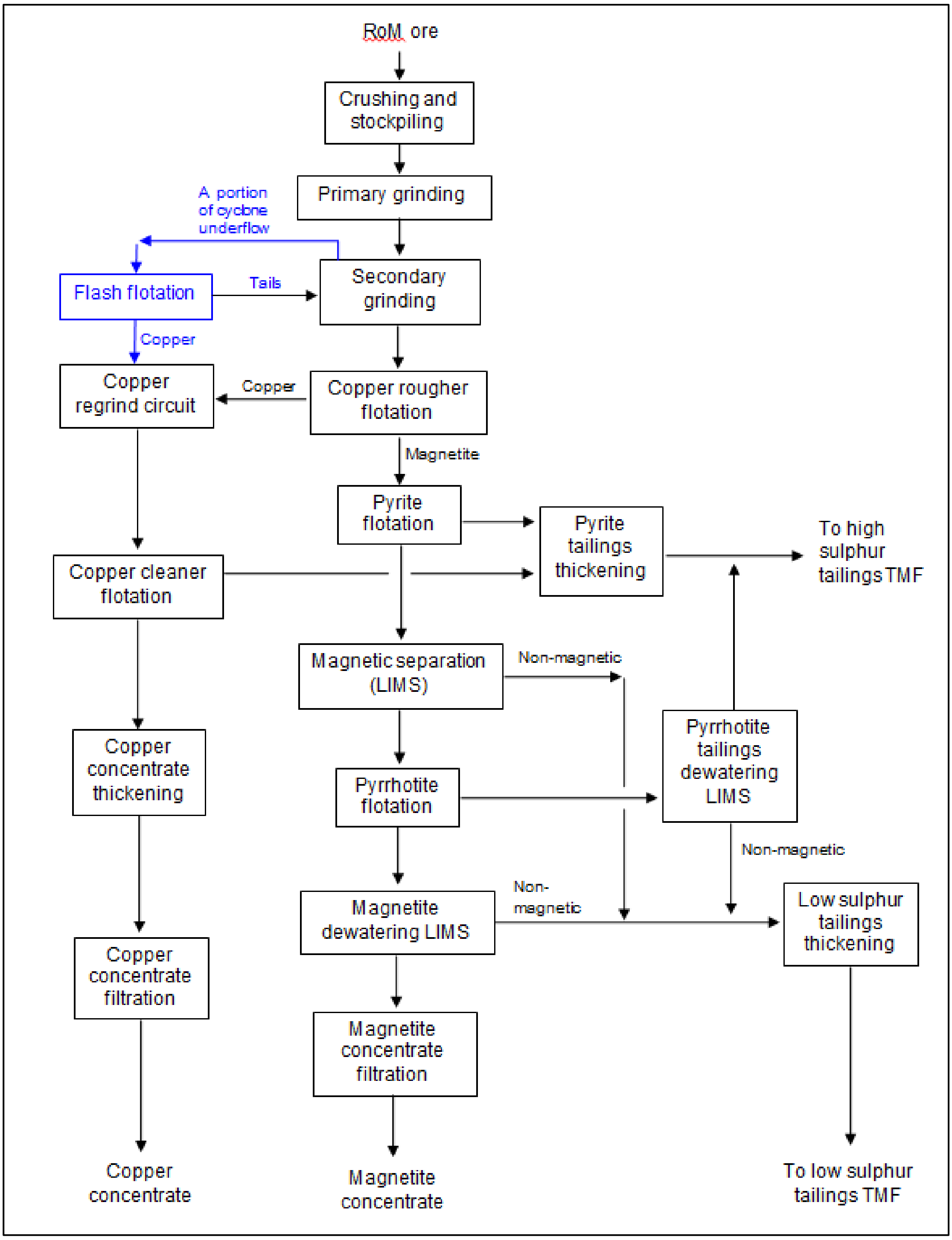


Figure 16-1: Hannukainen Block Flowsheet (courtesy of Jacobs)

The selected process route reflects the metallurgical testwork findings for both the Hannukainen and the Kuervittiko ore types. The design basis for the plant is reasonable and results in a realistic plant design sufficiently flexible to deal with ore variations and to produce acceptable magnetite and copper concentrates.

Process design and equipment sizing incorporates satisfactory design margins to accommodate likely feed variations.

The proposed plant design incorporates adequate process control, metallurgical on-stream analysis and sampling equipment.

16.3 Project Schedule and Production Ramp-up

A detailed schedule for the design, procurement and construction activities has been developed by Jacobs based on the HFS finalised at the end 2013. A summary of the schedule is shown in Figure 16-2.

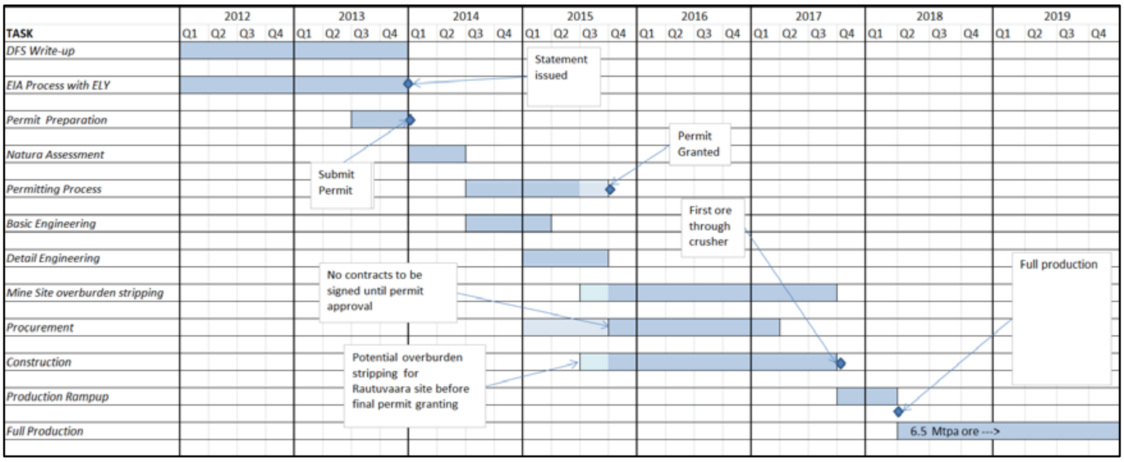


Figure 16-2: Project Schedule (Jacobs)

Engineering of the plant will be prepared in parallel with the permitting process and will commence around mid-2014. Construction activities, including plant commissioning, will commence once the permits are received, for a period of approximately 27 months. An additional 6 month production ramp-up is required. Early erection of the main building will allow installation to be performed throughout the year, irrespective of weather. The effects of any changes in the schedule would need to be assessed to identify potential impacts on site works, especially earthworks and civil activities, due to adverse seasonal weather. Full production is scheduled for the end of the Q1 2018.

16.4 Process Plant (Hannukainen Complex) OPEX

The operating costs have been built up in detail from first principles. The methodology and costs used, and the calculated operating costs have been reviewed and are acceptable. The total operating cost is as presented in Table 16-10. The overall processing operating cost for the Hannukainen complex over the LoM are presented in

The manning schedule and labour costs including overheads have been agreed with Northland and are reasonable.

The power costs have been built up from an assessment of the annual kWh used on the plant and associated infrastructure and services at an energy cost of EUR0.0507/kWh.

Consumables for the plant have been built up based on estimated usages and supplier quotations where appropriate. The costs are acceptable. There are no fixed consumable costs.

Maintenance costs are based on 2.5% of direct merchant account including mechanical items, electrics, instrumentation and piping. This is considered realistic and the figure is acceptable.

Tailings, G&A and Other costs have been reviewed and are acceptable.

Table 16-10: LOM Process Plant Operating Costs

Cost centre	USD over LoM (UAD 1,000)	USD/t ROM
Labour	98,477	0.86
Energy	242,575	2.11
Consumable -variable	326,322	2.84
Consumable - fixed	0	0.00
Operating spares	44,053	0.38
Plant G and A	34,987	0.30
TOTAL PLANT OPERATING COST	**746,415**	**6.49**

16.5 Process Plant (Hannukainen Complex) CAPEX

The capital costs for the concentrator, and filtration plant have been estimated by Jacobs using normal engineering and estimating methods. The earthworks costs, the civil works costs, together with the concentrator building and ancillary building costs have been estimated by Pőyry. The scope includes the crushing at Hannukainen, the crushed ore overland conveying system to Rautavaara, and the crushed ore stockpile, process plant, the filter plant and concentrate rail load-out at Rautavaara. The costs include all EPC costs and an allowance for the initial spares inventory.

The estimated capital cost prepared by Jacobs, excluding any contingency and buying discounts, is USD195.3M. The detail is presented in Table 16-11. The estimate accuracy is stated by Jacobs as ±15%.

It should be noted that the plant duty was increased from 6 to 6.5 Mtpa during the latter stages of the HFS. The effect on the process plant was evaluated and costs adjusted where appropriate to reflect additional or larger equipment and increased bulk quantities.

The additional capital costs developed by Pőyry for buildings, include earthworks, foundations, structures, HVAC, electrical systems and fire alarm systems, is USD75.0M. This includes the crushing station, screen house, stockpile building, process plant and reagent storage buildings, plant workshop, assay laboratory, the concentrate thickeners, pumping stations and the train loading building. The details are given in Table 16-12.

The total estimated capital cost for the processing and associated facilities including the Jacobs and Pőyry estimated costs is USD270.3M (excluding contingency).

Table 16-11: Processing Capital Costs (Jacobs - December 2013)

ELEMENT	No.	DESCRIPTION (Revision 2 - Dec 2013)	Process Plant Area: Crushing & Conveying	Stockpiling	Beneficiation	Tailings	Filtration	Rail Load-Out	Total
			USD	USD	USD	USD	USD	USD	USD
MATERIALS :									
	1	Mechanical Equipment	5,819,270	7,335,447	69,316,035	2,744,696	13,960,603	5,033,558	104,209,608
	2	Discount for Good Buying of Mechanical Equip.	Excluded	Excluded	Excluded	Excluded	Excluded	Excluded	Excluded
	3	Pipework	29,256	29,638	6,367,561	205,559	41,985	-	6,674,000
	4	Electrical - Process Plant	347,294	489,011	9,178,850	646,395	1,190,163	10,428	11,862,142
	5	Electrical - Buildings	264,159	539,338	855,248	-	636,822	-	2,295,568
	6	Instruments	51,820	56,055	3,305,458	631,101	189,750	7,652	4,241,837
	7	Control	-	-	983,530	-	421,513	-	1,405,042
	8	Spares- (Commissioning)	14,548	18,339	173,290	6,862	34,902	12,584	260,524
	9	Spares - (2 Year Operational)	116,385	146,709	1,386,321	54,894	279,212	100,671	2,084,192
	10	Spares - Insurance	Excluded	Excluded	Excluded	Excluded	Excluded	Excluded	Excluded
	11	Escalation (2012 Vs 2013 - Bulk Materials) 2.5%	12,985	20,888	387,950	27,807	46,504	339	496,474
	12	TOTAL MATERIALS	6,655,718	8,635,425	91,954,243	4,317,314	16,801,453	5,165,232	133,529,386
	13								
CONSTRUCTION	14	Site Development	-	-	-	-	-	-	-
SUB-	15	Civil	By Poyry	By Poyry	By Poyry	By Poyry	By Poyry	By Poyry	By Poyry
CONTRACTS :	16	Steelwork (Including Cladding)	By Poyry	By Poyry	By Poyry	By Poyry	By Poyry	By Poyry	By Poyry
	17	Buildings	By Poyry	By Poyry	By Poyry	By Poyry	By Poyry	By Poyry	By Poyry
	18	Building Services	By Poyry	By Poyry	By Poyry	By Poyry	By Poyry	By Poyry	By Poyry
	19	Fire Fighting	Included in Piping	Included in Piping	Included in Piping	Included in Piping	Included in Piping	Included in Piping	Included in Piping
	20	Painting	7,338	7,338	464,701	39,591	19,998	7,338	546,303
	21	Insulation	13,208	13,208	23,896	7,383	-	-	57,694
	22	Mechanical Equipment Installer	272,230	1,274,675	6,845,620	696,472	1,157,262	684,686	10,930,946
	23	Pipework Installer (Fabrication & Erection)	126,852	133,124	8,844,841	847,789	207,109	-	10,159,717
	24	Electrical Installer - Process plant	159,275	185,439	3,359,235	180,155	271,466	8,909	4,164,479
	25	Electrical Installer - Buildings	341,896	694,419	1,093,803	-	663,170	-	2,793,289
	26	Instruments Installer	19,381	29,564	2,293,804	256,860	197,382	9,905	2,806,897
	27	Control Installer	-	-	54,924	-	33,507	-	88,431
	28	Escalation (2012 Vs 2013 - Sub-Contracts) 3.0%	28,205	70,133	689,425	60,848	76,497	21,325	946,433
	29	TOTAL SUB-CONTRACTS	968,386	2,407,900	23,670,250	2,089,098	2,626,392	732,163	32,494,188
	30								
OTHER	31	Scaffold (Balance - excl s/c supplied)	22,273	55,382	544,416	48,049	60,407	16,840	747,366
DIRECT	32	Heavy Lift (Balance - excl s/c supplied)	37,767	93,908	923,140	81,475	102,429	28,554	1,267,273
COSTS :	33	Site Establishment & Main Contractor Provisions	By Poyry	By Poyry	By Poyry	By Poyry	By Poyry	By Poyry	By Poyry
	34	Site Establishment (Contractors Offices Only)	Incl. in Sub-Contr.	Incl. in Sub-Contr.	Incl. in Sub-Contr.	Incl. in Sub-Contr.	Incl. in Sub-Contr.	Incl. in Sub-Contr.	Incl. in Sub-Contr.
	35	Site Establishment (Construction Management Services Offices Only)	19,368	48,158	473,405	41,782	52,528	14,643	649,884
	36	Vendors Assistance	69,831	88,025	831,792	32,936	167,527	60,403	1,250,515
	37	Commissioning (Up to Mechanical Completion) - Artisan Support	58,193	73,354	693,160	27,447	139,606	50,336	1,042,096
	38	Consultants	5,819	7,335	69,316	2,745	13,961	5,034	104,210
	39	Escalation	Excluded	Excluded	Excluded	Excluded	Excluded	Excluded	Excluded
	40	TOTAL OTHER DIRECT COSTS	213,251	366,163	3,535,229	234,434	536,458	175,809	5,061,344
	41								
	42	TOTAL MATERIALS & CONSTRUCTION	7,837,355	11,409,488	119,159,722	6,640,846	19,964,303	6,073,205	171,084,919
	43								
PROJECT &	44	Project & Procurement Services	195,934	285,237	2,978,993	166,021	499,108	151,830	4,277,123
ENGINEERING	45	Engineering Services	391,868	570,474	5,957,986	332,042	998,215	303,660	8,554,246
SERVICES	46	Construction Management	391,868	570,474	5,957,986	332,042	998,215	303,660	8,554,246
	47	Commissioning Support (Up to Mechanical Completion)	78,374	114,095	1,191,597	66,408	199,643	60,732	1,710,849
	48	Expenses	52,902	77,014	804,328	44,826	134,759	40,994	1,154,823
	49	Hot Commissioning	Excluded	Excluded	Excluded	Excluded	Excluded	Excluded	Excluded
	50	TOTAL SERVICES COSTS	1,110,945	1,617,295	16,890,891	941,340	2,829,940	860,877	24,251,287
	51								
	52	Contingency 10.0%	894,830	1,302,678	13,605,061	758,219	2,279,424	693,408	19,533,621
	53								
	54	TOTAL PROJECT ESTIMATE	9,843,131	14,329,461	149,655,674	8,340,404	25,073,667	7,627,489	214,869,826
								without contingency	195,336,206

Table 16-12: Process Buildings and Associated Costs (Pőyry December 2012)

Ore processing buildings	Capital cost (1,000 USD)
Crushing station	11,122
Screen house	368
Stockpile building	9,462
SAG/Ball Mill media storage	1,004
Process plant building	37,915
Plant Workshop	3,826
Assay lab	2,525
Thickener	1,078
Train loading building	3,321
Switch station (Hannukainen)	419
Reagents storage	1,790
Main pumping stations	2,150
TOTAL	**74,980**

The capital cost estimate developed by Jacobs has been built up in Euro, SEK and USD. The Euro cost element represents 88% of the total capital cost with much of the balance in USD. The Pőyry works will be predominantly in Euro.

The cost of the mechanical equipment is USD104.2 M, representing approximately 38% of the total capital cost and has been based on extensive equipment quotations together with costs from the Jacobs in-house cost database.

Equipment costs have been based on multiple quotations from international suppliers. Where appropriate, costs for minor items of equipment have been based on in-house prices.

Detailed material take-offs have been prepared for all bulk items such as concrete, piping and valves, electrics etc and have been costed using market rates obtained by multiple quotations. The unit rates have been reviewed against the contractors in-house database and are considered reasonable.

Similarly installation costs have been based on quotations from multiple in country contractors using the estimated quantities and work scope prepared by the engineer.

EPCM costs have been estimated by the contractor from first principles based on in-house experience.

SRK consider that the level of engineering performed is commensurate with the quoted cost accuracy and that the capital cost developed is realistic provided adequate project contingencies are added to cover unknowns. The overall contingency should be calculated based on an assessment of risks. Jacobs recommend a contingency of 10% or USD19.6 M for the plant should be included.

17 INFRASTRUCTURE

17.1 Proposed Mine Operation Infrastructure

17.1.1 Site Layout

The proposed mine development is split between two key operational areas; Hannukainen and Rautuvaara, with the two areas being separated by approximately 3 km. The Muoniojoki River, flowing northeast to southwest lies just to the south of the proposed Hannukainen area.

The Hannukainen area comprises the following infrastructure assets:

- Hannukainen and Kuervitikko Open Pits;
- Waste Rock Dumps;
- ROM Pad and primary crusher;
- Process Water Pond;
- Water Treatment Plant;
- Explosives Store;
- Truck Workshop;
- Administration/Welfare/Change House; and
- Power Plant.

The Rautuvaara area comprises the following infrastructure assets:

- Process Plant;
- TMF and associated CP; and
- Train Load Out Facility.

17.1.2 Geotechnical Investigation and Bulk Earthworks

The infrastructure design has been based on geotechnical design parameters (Table 17-1) derived from an infrastructure geotechnical site investigation at all development areas.

Table 17-1: HFS Geotechnical Design Parameters

Parameter	Unit	Loose Morraine	Medium Dense Morraine	Crushed Stone Filling
Internal Angle of Friction	Degrees	35	38	42
Unit Weight	kN/m^3	17-20	18-21	22
Effective Unit Weight	kN/m^3	10-12	11-13	12
Compressibility Modulus	MPa	15	25	45

In accordance with Finnish standards, all structural foundations should lie beneath the frost penetration depth of 2.6 m. Heavily loaded and vibrating structures will be founded on bedrock, with all other foundations founded on bedrock or moraine dependant on anticipated loads and allowable settlement. The HFS recommends further targeted geotechnical investigation once asset locations and sizes have been finalised.

17.1.3 Heavy Haul Roads

The HFS has proposed 25 m wide non-bituminous haul roads based on accommodating a CAT789D mine truck with a maximum speed of 57.2 kph. The road geometry is based on a 3% crossfall and maximum 10% vertical grade with vertical and horizontal geometric curvature dependent on achieving a minimum 150 m stopping sight distance and to minimise excavation.

The HFS has proposed a variety of sub-formation solutions for the 2.0 m thick road non-bituminous pavement dependent upon in situ ground conditions and soil characteristics. Where peat is encountered (maximum 2 m thickness), it will be replaced with rock fill. Where embankments are anticipated, site won moraine or rock fill is used to achieve the desired formation.

Parallel unlined surface water channels are proposed adjacent to the heavy haul routes where the road is in cutting, low embankment or where surrounding topography falls towards the road. Surface water channels are not proposed on high embankments where surface water can freely discharge to the surrounding environment.

17.1.4 Administration, Welfare and Change House

Separate facilities are proposed at Hannukainen and Rautuvaara, based on an anticipated maximum combined workforce of 333 persons. Both facilities will be constructed from pre-fabricated units to offer a flexible and more cost effective solution than traditional construction methods.

Both facilities include offices, ablutions, welfare and workforce changing rooms. The primary kitchen and restaurant is located at Rautuvaara, whilst a kitchenette is proposed at Hannukainen. A basic triage is located within the administration buildings, with the nearest hospital located 120 km from the Project site.

17.1.5 Truck Workshops

The workshops have been designed to maintain a fleet of 12, 277 t capacity mine trucks and comprises:

- four heavy service bays;
- wash bay;
- three workshops;
- covered tyre storage;
- office for 15 persons;
- welfare; and
- Air-Raid shelter for 120 people.

The structure will be constructed from a steel portal frame with insulated pre-fabricated concrete and corrugated steel cladding. It is proposed to heat the building to achieve +5 to +17ºC in the workshops/wash bay and +20ºC in storage and office areas.

The workshops will have two parallel drive through service bays with roller doors at each end. Each bay will accommodate two trucks and be serviced by a 50 t capacity overhead crane. The crane rail will be located 16.5 m above ground level, compared with the CAT793F overall raised height over 13.88 m.

17.1.6 Laboratory and Core Stores

The laboratory and core storage building will be constructed from a steel portal frame structure with an insulated roof system and wall cladding.

17.1.7 Explosives and Detonator Storage and Management

The explosives store will be located between the concentrating plant and Kuervitikko pit. A 500 m safe distance to all manned workspaces is proposed with stand-alone detonator stores separated from other infrastructure by 10 to 15 m dependent on the quantity of explosives stored. The facility will have a secure perimeter fence line and access gate.

SRK understands that an Explosives Contractor will provide all assets associated with the storage, handling, mixing and deployment of explosives throughout the life of the Project. Therefore, the associated capital costs only include site preparation and provision of utilities to the magazine development platform.

17.1.8 Materials Handling Infrastructure

To facilitate the export of 2.1 Mtpa Fe concentrate and 56.5 ktpa Cu concentrate the materials handling infrastructure will be required to handle 6.5 Mtpa of ROM material, with 30.6 Mtpa of overburden and waste rock stockpiled at Hannukainen.

Mine trucks will transport ROM material from both pits to the crushing station (Figure 17-1) at the ROM pad. Two ground level dump positions will feed the underground primary crusher. A steel frame building covering the crushing station is proposed at ground level, which will contain a ground mounted hydraulic breaker and 40 t overhead crane. Beneath the crusher, a 50 m deep, 14.5 m wide feeder hall is proposed to house a storage silo and conveyor feeder.

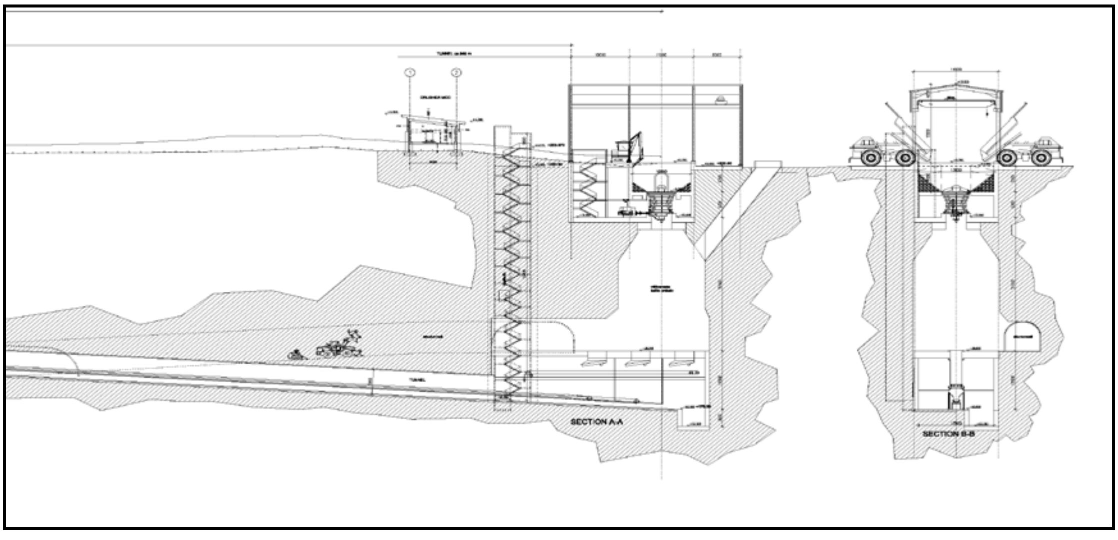


Figure 17-1: Crushing Station

A 700 m long transfer conveyor (Figure 17-2), of which 545 m lies within a proposed 5.3 m wide by 5.55 m high shotcrete tunnel, is proposed to transport crushed ore at an inclination of 3.36° (5.8%) from beneath the storage silo to an elevated transfer tower at ground level.

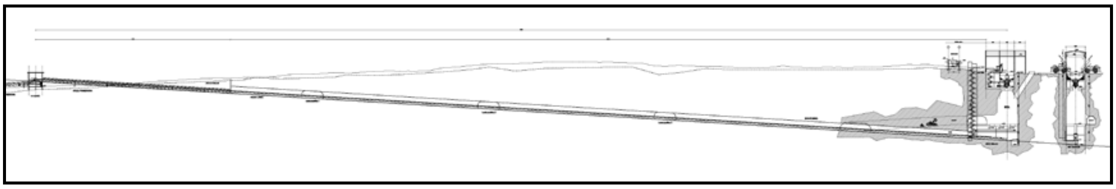


Figure 17-2: Crushing Station Transfer Conveyor

An 8.1 km long covered overland conveyor will then transport the crushed ore from the transfer tower to an ore storage building at the Rautuvaara area.

To facilitate maintenance, a 5 m wide service road will run adjacent to the overland conveyor route.

Where embankments are anticipated, locally imported moraine/sand will be used to achieve the desired formation. Where cut depths are anticipated to exceed 3 m, then earthworks slopes will be protected from erosion by crushed stone revetment. Where bed rock is encountered and cut depths exceed 3 m, the HFS proposes a 1 m berm located 2 m below the interface with overburden material. Blasting along the overland conveyor route is not anticipated.

Four roads cross the proposed overland conveyor route:

- Pakasaivontie Road Crossing – a conveyor bridge and an at grade service road crossing;
- Äkäsjoentie Road Crossing – a 42 m long, corrugated steel panel RUMTEC culvert is proposed to enable the conveyor and service road to pass beneath the existing road; a permit is required for this proposed solution;
- Cottageroad Crossing - a conveyor bridge and an at grade service road crossing are proposed; and
- Yllaksentie Road Crossing - a 62 m long, corrugated steel panel RUMTEC culvert is proposed to enable the conveyor and service road to pass beneath the existing road; a permit is required for this proposed solution.

The HFS proposes the construction of bridges to allow the conveyor to cross the Valkeajoki and Äkäsjoki rivers.

17.1.9 Ore Storage Building

The proposed 28 m high ore storage building of unstated capacity will receive crushed ore from the overland conveyor where a series of tripper conveyors will load the covered storage area. A tunnel reclaimer system will load the mill feed conveyor which will transport crushed ore into the processing plant.

17.1.10 Cu Concentrate Export

The logistics and supporting infrastructure proposed to export Cu concentrate by road trucks is discussed in Section 0.

17.1.11 Fe Concentrate Export

A single storey steel framed building is proposed to accommodate the concentrate warehouse and rail load out facility. Conveyors are proposed to transport and load the Fe concentrate stockpile within the rail load out building where FEL are proposed to load ore trains inside the warehouse.

Further discussion on the logistics and supporting infrastructure proposed to export Fe Concentrate by rail is discussed in Section 0.

17.2 Proposed Mine Support Infrastructure

17.2.1 Light Vehicle Roads

The HFS proposes the following light vehicle roads:

- Rautuvaara and Hannukainen access roads; and
- service roads within the Rautuvaara and Hannukainen areas.

The proposed Rautuvaara and Hannukainen development areas will be connected to the existing highway 940 by a 767 m and 2.848 km long, 7.5 m wide access roads of bituminous construction. The road geometry has been designed based on a 60 kph speed limit, 3% crossfall, maximum 5% longfall and a 75 m stopping sight distance. The road pavement has been designed based on a normal axle load of 120 kN, an exceptional axle load of 160 kN and a maximum 2,500 vehicles per day.

The road drainage proposed is as previously discussed in Section 17.1.3.

17.2.2 Accommodation

SRK understands that an existing unused accommodation camp is located 20 km from the Project site. SRK understands that the camp was constructed in the 1990s and has been acquired by Northland to support the proposed workforce.

17.3 Proposed Mine Services Infrastructure

17.3.1 Communications and Security

Information Communication and Technology (“ICT”) and security systems will be provided that comprise:

- access systems for secure access/egress to buildings and controlled areas;
- CCTV systems;
- building systems;
- WLAN network for mobile communication; and
- digital mobile radio (“DMR”) network for communication between remote operations.

A fibre optic network will link the Hannukainen and Rautuvaara development areas and the existing network running between Kaunisvaara and the Muonio River pumping station.

An Uninterruptable Power Supply ("UPS") is proposed at all ICT locations to provide two hours of backup power in the event of a power failure.

17.3.2 Power

An existing 58.3 km long 110 kV power line from Isoniemi currently extends to the existing Rautuvaara switchyard and 110/45 kV transformer owned by Tornionlaakson Sähkö Oy ("TLS").

The HFS states the maximum anticipated load at Rautuvarra and Hannukainen is 40.1 MW and 5.66 MW respectively (Table 17-2).

Table 17-2: Anticipated Power Load Demand

Asset	Maximum Anticipated Power Demand (MW)
Rautuvaara Area	
Office Area	1.06
Process Area	0.28
Power Plant	0.56
Plant Workshop	0.21
Process Plant Stockpile	2.98
Process Plant Filtration	1.28
Process Plant Tailings	1.59
Process Plant (Others)	8.45
Ball and SAG Mills	19.68
Flotation Fan	0.72
Tailings Pumping	1.54
Water Pumping Stations	0.22
Conveyor Belt Pulling Stations	1.54
Total Rautuvaara Area Demand	*40.10*
Hannukainen Area	
Office Area	0.88
Process Area	0.21
Truck Workshop	0.39
Crushing	0.56
Power Plant	0.25
Water Pumping Stations	0.59
Open Pit	2.78
Total Hannukainen Power Demand	*5.66*
NET Project Power Demand	**45.76**

The location of the proposed TMF will require the diversion of approximately 5 km of 110 kV overhead power line and construction of a new switchyard near Rautuvaara.

Should the Isoniemi switchyard fail and the grid short circuit, capacity will be insufficient to start the mill motors with liquid resistance starter.

A 110 kV incoming switchgear is proposed with a 170 kV bus bar and 1,250 A circuit breaker equipped for local and remote operation.

A 65 MVA power transformer is proposed to enable 10 kV and 20 kV medium voltage distribution throughout the Rautuvaara site.

Throughout the Rautuvaara and Hannukainen areas 20 kV/690 V and 20 kV/400 V distribution oil immersed hermetic type ONAN transformers are proposed, located adjacent to the proposed LV motor control centres (“MCC”).

Indoor 20 kV, 10 kV and 6.6 kV air insulated switchgear are proposed for internal distribution.

Emergency diesel generation is proposed for critical assets such as smoke ventilation, critical motor drives, etc. A separate 690 V emergency diesel generator is proposed to back up the Low Sulphur Tailings Thickener Underflow pump so that the associated pipeline can be flushed during power outage.

A lightening protection system is proposed for all high risk structures and buildings using standard earthing systems.

17.3.3 Heat

Heat generation is required for ventilation, room heating and warm potable water in buildings as heat output from process machinery is anticipated to be insufficient.

The maximum anticipated heat consumption at Hannukainen and Rautuvaara are 10.5 GWh/annum and 29.2 GWh/annum respectively.

Separate solid fuel and oil fired boilers will be installed at both Hannukainen and Rautuvaara (Table 17-3).

Table 17-3: Proposed Heat Generation

Boiler System	Fuel	Hannukainen		Rautuvaara	
		Capacity (MW)	Energy (GWh/annum)	Capacity (MW)	Energy (GWh/annum)
Solid Fuel Fired	Wood chips/peat	1-1.5	8.3	5-6	24.4
Oil	LFO	2-2.3	1.9	8-9	3.9
Total		**3-3.8**	**10.2**	**13-15**	**28.3**

17.3.4 Water Management

Process Water

A process water pond is proposed by constructing a dam across the Kivivuopionoja River. It is proposed that surface water from surrounding areas will be diverted to this pond where it will be pumped to a water treatment plant before reticulation to the process plant.

All excess and treated process water is pumped via a 10.6 km long pipeline to the Muonionjoki River near Kolari, 4.4 km of which follows the route of the existing railway.

Potable and Foul Water

Potable and foul water demands will be provided through new connections to the existing Ylläksen Yhdyskuntatekninen Huolto Oy (“YYTH”) water and sewage water networks.

Surface Water

A ditch system is proposed to capture and convey surface water within the Project site based on a 1 in 20 year event flow of 600 l/s x km² (6 l/s/ha). Surface water will be conveyed via the proposed ditch system to sedimentation basins before discharge to proposed wetlands and eventually existing watercourses. The primary sedimentation basin is the process water pond described above.

To prevent unnecessary flows entering the Project, surface water from adjacent catchments will be intercepted and conveyed to existing nearby watercourses.

All vulnerable earthworks and drainage surfaces will be protected from erosion through placement and compaction of 300 mm thick aggregate cover material with a 100 mm topsize.

Further discussion on the site wide surface water management strategy and proposed infrastructure assets is contained in Section 14.

17.3.5 Vehicle Refuelling

SRK understands that the fuel supplier will be responsible for all storage, handling, distribution and dispensing infrastructure.

17.3.6 Fire Suppression

A fixed fire main system will reticulate fire water to all plant, operations, maintenance and support buildings. Specific fire systems have been specified in accordance with the fire category and planned operations and activities with each building.

17.4 Proposed construction aggregates

A quarry will be opened to generate construction aggregates at Hannukainen, located between the proposed Hannukainen open pit and overburden stockpile. It is proposed that a mobile crushing plant will provide sufficient construction aggregate primarily for road and dam construction.

Moraine borrow pits will be opened to provide sufficient material for the construction of the process water pond dam.

A 40 to 60 m^3/h capacity concrete batching plant is proposed at the Hannukainen area to provide sufficient quantities of concrete for both development areas.

17.5 SRK Comments

17.5.1 Geotechnical Investigation and Bulk Earthworks

It is unclear what terracing, drainage and settlement controls may be required at Hannukainen and Rautuvaara to create the platforms to support the proposed development.

17.5.2 Materials Handling Infrastructure

A subsurface crusher, feed hall and long transfer conveyor decline are proposed. SRK considers conventional above ground solutions may prove more cost effective, improve accessibility and safety during operation/maintenance activities and reduce construction risk/schedule. A typical above ground crushing facility is shown in Figure 17-3.

Weather protection could be achieved by the use of a fabric structure with insulated panels such as RUBB Thermohall System[5] (Figure 17-4).

Figure 17-3: Typical above ground crushing facility

[5] http://www.rubb.co.uk/

Figure 17-4: Typical example of a RUBB Thermohall System

The current status of all required permits for conveyor culvert construction is unknown.

SRK understands that a trade-off study has been undertaken to explore a variety of buried and semi-buried primary crushing solutions. However, SRK understands that the study is not Project specific and does not consider that all potentially viable solutions. Furthermore, SRK does not agree with the method of measurement used to derive the relevant costs for the options considered. SRK considers validation of a conventional above ground primary ore crushing solution a substantial project opportunity which should be considered in further detail during future Project phases.

17.5.3 Truck Workshops and Logistics Storage

The layout of the truck workshops enables drive through access, thereby removing the need for reversing movements. However, the structural layout proposed prohibits drive through movements within the adjacent wash bay. In accordance with international best practice, SRK recommends the layout is modified during the next Project phase.

SRK recommends that a separate building is provided where tyre and rim changing operations can be safely undertaken.

SRK considers that insufficient storage is proposed for spares/consumables/tyres associated with the maintenance of heavy mobile equipment. In addition an outside compound is not proposed for the secure storage of large non-palletised items.

SRK would anticipate separate stores for operation and maintenance consumables in close proximity to workshops, with secure vehicular access sufficient lay down areas for assembly and storage of large components.

However, SRK notes that the above recommendations will not have a material impact on overall project costs.

17.5.4 Workforce Accommodation

Given that the existing accommodation camp was constructed over 20 years ago, SRK recommends that the condition of these assets is assessed during the next project phase. Further capital investment may therefore be required should their condition not be considered suitable when the project is implemented.

17.5.5 Fire Suppression

Requirements for fire suppression in the pit and on overburden/waste dumps does not appear to be have been defined. Typically fire suppression would be managed by mobile water bowsers. However, SRK considers that this will not have a material impact on overall project costs.

17.5.6 Mine Site Waste Management

It is unclear what infrastructure and environmental protection measures are proposed to support the safe management and disposal of non-industrial, industrial (including waste hydrocarbons) and domestic waste. SRK would anticipate a waste management facility where waste streams can be separated for appropriate disposal. Typically, waste hydrocarbons are stored in a bunded facility before either on-site incineration or offsite disposal. A lay down area will allow industrial waste items (such as engine filters) to be secured and packaged before transport to specialist offsite waste management providers. However, SRK considers that this will not have a material impact on overall Project costs.

17.5.7 Power

Although the HFS presents the anticipated power demand, it does not define the anticipated power consumption which will be required to define operating costs.

17.5.8 Capital Cost Adjustments

SRK considers that the above recommendations will not have a material impact on overall capital costs.

17.5.9 Operating Costs

SRK understands that 10% of all operating costs within the Industrial Area are associated with the rental of furniture and equipment within all Project site buildings. Over the life of the Project, this equates to approximately USD8.8M. SRK considers this a potential opportunity to be validated at a later Project phase.

17.5.10 Contingency

A detailed project level qualitative and quantitative risk assessment has been developed for both capital and operating costs. Based on this assessment a 10 % contingency has been applied to capital costs and 5% to operating costs.

17.5.11 Project Delivery Schedule

SRK considers the overall construction schedule reasonable. However, SRK notes the schedule does not appear to present activities concerning the planning and improvement to existing rail and port infrastructure by third parties.

17.6 Concentrate Transport Logistics

17.6.1 Introduction

Various options have been investigated by Northland for the transportation of Cu-Au and Fe concentrate from the Project to market.

The HFS undertaken by Northland has considered two different transportation modes for the concentrate with:

- Cu-Au concentrate being transported by road; and
- Fe concentrate being transported by rail and ship.

The studies undertaken reflect the point of sale (“POS”) of each type of concentrate and reflect the volumes anticipated from the beneficiation process for each type of concentrate produced.

The Cu-Au concentrate study anticipates that logistics for the concentrate will be provided by the customer with the POS being the northland Rautuvaara facility.

The Fe concentrate study anticipates that logistics will be provided by Northland at a POS at a customer port in either Europe, the Middle East, or Asia. As such, this study has evaluated the complete logistics chain to the POS including transportation for the base case port (Kokkola, Finland) to the customers’ port. Transportation costs after the POS, including unloading of the concentrate at the customers’ port are not considered within the HFS.

17.6.2 Copper Gold Concentrate

Approximately 66,500 tpa of Cu-Au concentrate will be produced at Hannukainen and this will be transported using conventional 60 t heavy goods vehicles with a 40 t payload, to avoid the requirements for special permitting by the Finnish and Swedish roads authorities and the HFS proposes that up to four trucks per day will be required. Several potential destinations exist for this concentrate within Sweden and Finland, including facilities at Gällivare (Sweden); Skellefteå (Sweden) or Harjavalta (Finland).

Cu-Au concentrate will be received from the dewatering stage of the beneficiation plant and will be stored in a covered storage area of the Cu-Au filtration building at the Industrial Area. The storage area within the filtration building has an approximate capacity of 1,000 t of filtered concentrate equivalent to approximately 1 week’s production.

It is anticipated that the Cu-Au concentrate will be sold ex-works at Hannukainen and that the customer will collect the concentrate at Rautuvaara and transport to either the Boliden facility at Gällivare where it will be either processed or consigned to rail for shipment to other smelters, or the concentrate will be transported directly from Rautuvaara to the smelters.

It is anticipated that truck will weigh in at the Rautuvaara facility, enter the filtration building storage area, be loaded by FEL supplied by the customer, who will then cover their load to limit load exposure and dust and then exit the facility.

17.6.3 Iron Ore Concentrate

The Project will produce approximately 2.1 Mtpa of Fe concentrate, equating to approximately 45,700 t per week. It is anticipated that this volume of concentrate will require a suitable bulk handling solution and that this would entail rail transportation to a suitable port for onward shipment to market.

Historically, studies for the Project have considered export routes via:

- road and rail to Narvik in Norway; and
- concentrate slurry pipeline and rail to Kemi in Finland with onward barge transport to a "hub port" in either the:
  - Southern Baltic; or
  - North Sea.

These options together with direct transport via rail to ports in the Gulf of Bothnia have been considered in the HFS for concentrate export and a base case has been derived by evaluating a number of environmental, social, economic and operational parameters.

Discussions have been undertaken by Northland and its Consultants with suppliers and operators for the significant elements of the export logistics chain for the base case adopted and for several other variations for export via the Gulf of Bothnia.

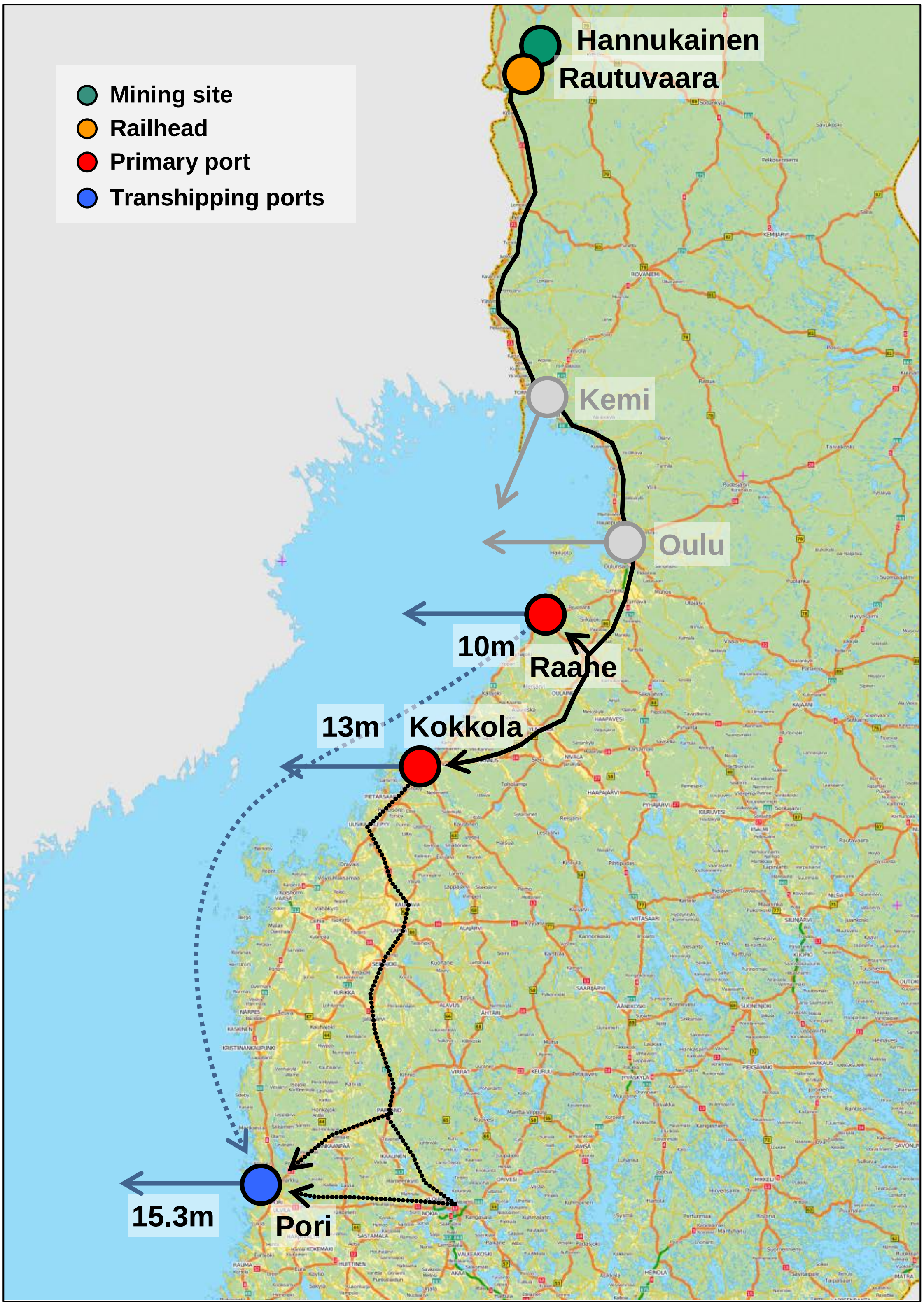


Figure 17-5: Potential port location in the Finnish Gulf of Bothnia (Source: Northland)

Adopted Base Case

The HFS base case for Fe concentrate transportation is from the Rautuvaara rail head to the port of Kokkola (approximately 520 km south of Rautuvaara) by train. The system developed comprises several discrete components:

- warehouse / stockpiles at Rautuvaara;
- reinstated Rautuvaara - Kolari rail line;
- existing Kolair-Kokkola rail Line;
- port of Kokkola; and
- Ocean Going Vessels for shipment to customers.

Facilities at Rautuvaara

The main facilities provided at Rautuvaara include the rail loading siding and the concentrate warehouse.

The rail siding and associated rail infrastructure (track, switches, signalling, communications etc) will be provided by the rail operator as part of the re-establishment and refurbishment of the Rautuvaara to Kolari rail line.

The concentrate warehouse is provided by Northland and is a conventional steel portal frame structure that has been sized to accommodate approximately three days' Fe concentrate production (19,800 t). "Open air" areas adjacent to the warehouse have been designated as temporary stockpiles in case of extraordinary events to provide additional stockpile capacity at Rautuvaara.

Train loading will be achieved by FEL reclaiming from the stockpile and dumping into rail cars within the warehouse building. Loading and warehouse operations are anticipated to be outsourced to the contractor responsible for managing the rail operations and including the managing of the stockpiles and the supply and operation of the FEL for train loading.

Rail – Rautuvaara to Kokkola

Historically, the Company concluded the negotiation of an agreement for the co-funding of transport corridors with the Finnish rail authority for export of Fe concentrate from Pajala in Sweden to the port of Kemi. The underlying principle being that the Company was to be responsible for the costs resulting from the mining transport; however, this option has never been used.

For the Project, the Company proposes to transport concentrate from the Rautuvaara rail head to the port of Kokkola on the Gulf of Bothnia.

A new rail head will be constructed at Rautuvaara by the Finnish Transportation Agency ("FTA") which will connect to the existing (currently disused) track which runs to Kemi.

The disused section of track between Rautuvaara and Kolari will be refurbished by FTA. At present, the line allows for train consist lengths of up to 550 m and the train length proposed by the Company is approximately 750 m therefore passing loops on this section of track as well as those between Torino and Kolari (Ylitornio and Pello) will require extension.

Additional upgrading works such as railway crossings, vibration protection and access roads may be required and FTA is currently analysing the required needs based on the Company's proposed train consist length and proposed timetable, it is anticipated that this improvement will principally be required in the Sieppijärvi area.

The section of line between Kolari and Kemi has sufficient capacity to meet the Company's requirements after the improvements suggested by FTA have been enacted, however the studies note that the capacity of this section of line is very sensitive to train speeds and buffer times and thus prone to delays. The Company's consultant suggest the additional passing loop proposed by the FTA would dramatically increase the capacity of the track and reduce potential perturbation.

Between Kolari and Kokkola, the track is currently in service and is maintained by FTA, this section of track is not anticipated to require significant upgrades or modifications; however, it is noted that this section of track, whilst having a high capacity, is heavily utilised by existing rail companies and the studies recommend that the Company secure agreement with FTA for train slot allocation.

The modelling undertaken suggests that the capacity of the complete rail system between Rautuvaara and Kokkola is between 2.0 and 3.3 Mtpa for a two train consist operation increasing to between 4.0 and 5.0 Mtpa for a three train consist operation.

Facilities at Kokkola

The port of Kokkola has been selected as a base case port for Fe concentrate export based on discussions held by the Company with various potential ports and an initial trade-off study undertaken by the Company in 2011.

At present, the Port of Kokkola can accommodate 72,000 deadweight tonnage ("DWT") ore carriers directly loading at the berth with a potential availability of 110,000 DWT ore carriers should "topping off" be used in the Gulf of Bothnia.

Figure 17-6 shows the proposed indicative layout at the Port of Kokkola.

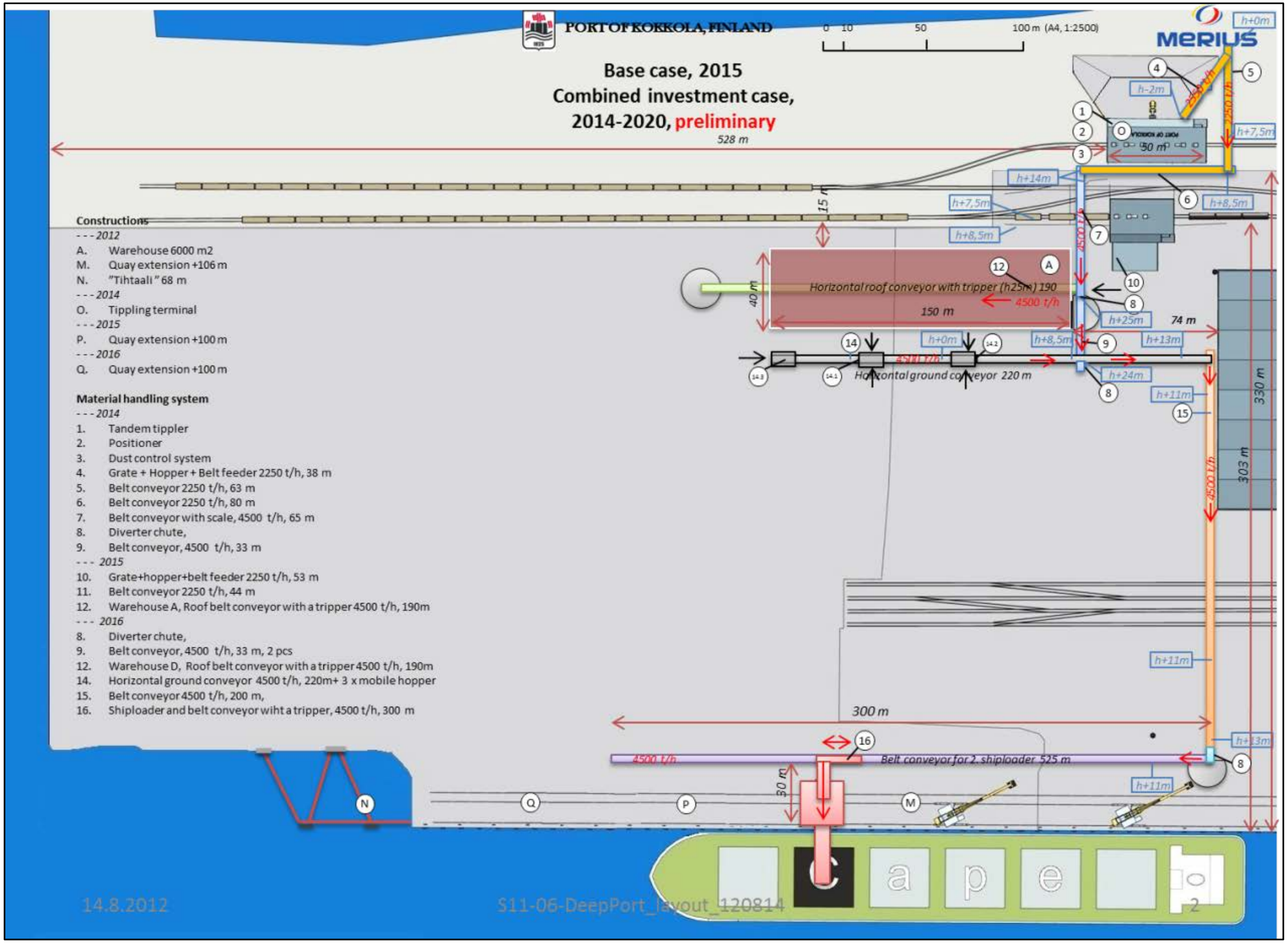


Figure 17-6: Proposed indicative layout Port of Kokkola

The Port of Kokkola will develop the port infrastructure to allow unloading of railway wagons, transport of concentrate from unloading station to warehouse, transport of concentrate from warehouse to the quay and ship loading.

Ocean Shipment

The Company has developed a shipping regime covering potential customers in Europe; the Middle East and Asia. The base case utilises the Port of Kokkola with ship sizes up to 72,000 DWT being accommodated at the berth during open water conditions and up to 45,000 DWT when ice class vessels are required (Figure 17-7).

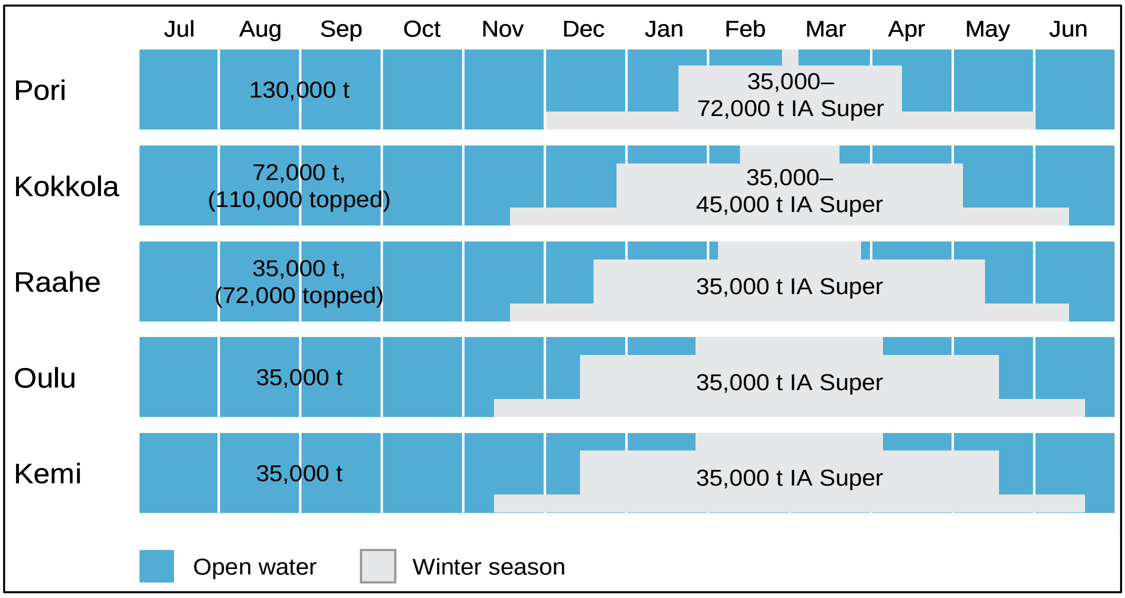


Figure 17-7: Ice Condition at Port in the Gulf of Bothnia

Additionally "topping off" of larger ships can be undertaken in the Gulf of Bothnia where larger vessels are loaded at the berth to their draft limits, then anchored offshore to allow additional concentrate to be loaded from smaller vessels until their capacity is reached. Whilst this may increases the total time in berth of the vessel, the additional amounts of ore shipped generally offset this cost.

17.6.4 Overall Export Logistics Capacity

Whilst each of the elements of the export logistics system proposed are consistent and offer a reasonable base case for the Project the system as a whole need to be interrogated to ensure that it is feasible for the volumes and types of operations proposed.

In undertaking this review, the Company and its consultants have recognised certain areas where limitations to throughput may occur.

Rail Operations

Several capacity restrictions exist within the existing and proposed rail network. Notably between Kemi and Kolari the capacity is limited due to the distance between signalling system equipment and the restricted passing stations. Between Ylivieska and Kokkola, limitations also occur because of heavy existing traffic on the line and refurbishment works that will continue until 2017.

The modelling undertaken indicates that for two train consists per day, the maximum capacity is approximately 3.3 Mtpa using ore wagons reducing to 2.6 Mtpa if a containerised solution is adopted.

The system has then been modelled with a third daily train giving capacities of 5.0 Mtpa and 4.0 Mtpa respectively as shown in Table 17-4.

Table 17-4: Simulation of railway capacity between Kolari and Kemi

Wagon type	Two trains per day		Three trains per day	
	length 585 m	length 750 m	length 585 m	length 750 m
Northland type	2.45 Mt/y	3.18 Mt/y	3.67 Mt/y	4.77 Mt/y
VR Pieksamäki wagon	2.56 Mt/y	3.32 Mt/y	3.84 Mt/y	4.98 Mt/y
Innofreight container	2.05 Mt/y	2.66 Mt/y	3.07 Mt/y	3.99 Mt/y
VR container. option 1	1.98 Mt/y	2.56 Mt/y	2.97 Mt/y	3.85 Mt/y
VR container. option 2	1.81 Mt/y	2.35 Mt/y	2.72 Mt/y	3.52 Mt/y

From information received from existing operators, the Company has assumed that currently there are six daily round-trip trains between Kolari and Kemi.

The simulation undertaken estimated the maximum additional capacity assuming that an additional passing loop is installed approximately 20 km north from Tornio.

Table 17-5 shows the export sensitivity to train consist speed.

Table 17-5: Export Sensitivity to Train Consist Speed.

Train speed	One new passing loop			Two new passing loop		
	0 min	5 min	10 min	0 min	5 min	10 min
60 km/h	3.25 Mt/y	3.25 Mt/y	1.62 Mt/y	8.12 Mt/y	6.50 Mt/y	6.50 Mt/y
70 km/h	4.87 Mt/y	4.87 Mt/y	3.25 Mt/y	8.12 Mt/y	8.12 Mt/y	6.50 Mt/y
80 km/h	8.12 Mt/y	6.50 Mt/y	4.87 Mt/y	12.99 Mt/y	11.37 Mt/y	9.74 Mt/y

Additionally studies have been undertaken to model the total export capacity against the number of wagons required and the anticipated train lengths (Figure 17-8). This indicates that for the chosen base case option of two 750 m long train consists per day a capacity of in excess of 3.0 Mtpa can be achieved.

The results of the analysis show that the proposed route is sensitive to both train speeds and buffer times at passing points, which indicates that whilst the capacity of the track can be increased to meet the Company's requirements, perturbation is a possibility and that whilst installation of additional loops mitigates this perturbation, it does not remove it.

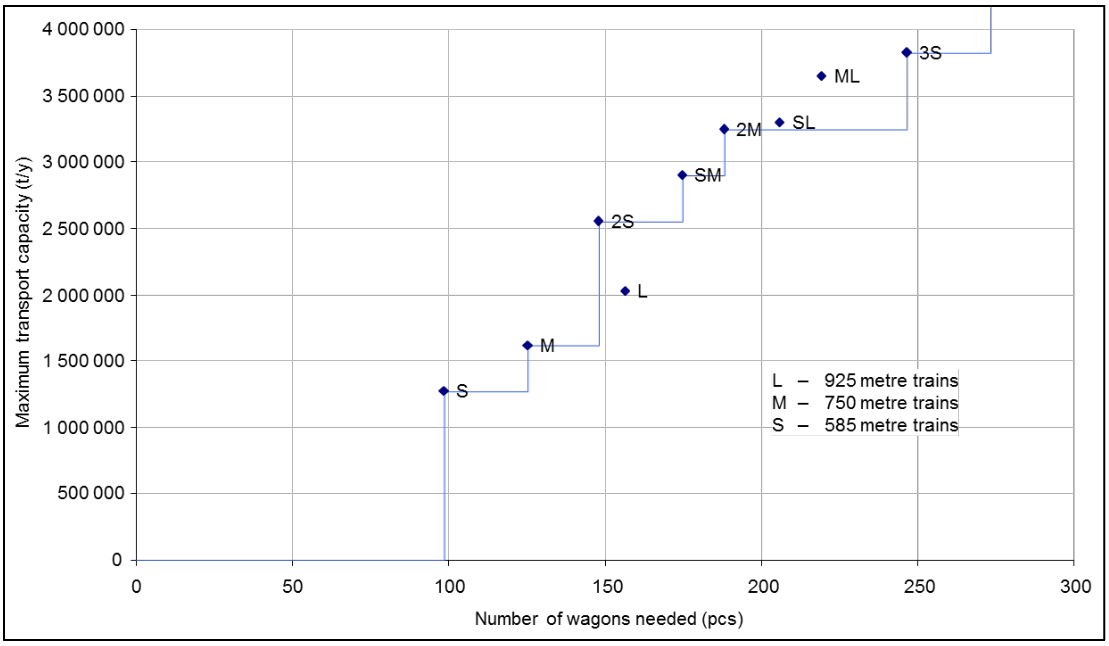


Figure 17-8: Train Length and Wagon Numbers vs total Transport Capacity

Port and Shipping Operations

Shipping operations have been considered at a number of ports within the Finnish Gulf of Bothnia to validate the base case choice of the Port of Kokkola.

The analysis undertaken considered factors such as berth length and draught restrictions due to ice, ship loader capacity, ore carrier size and “topping off”, capital and operating costs.

Table 17-6 shows the Finnish Gulf of Bothnia Port characteristics.

Table 17-6: Port Characteristics in Finnish Gulf of Bothnia

Port	Deep quay length	Deep quay draught	Ship loader capacity	Baby cape size (110 kt)	Panamax (72 kt)	Handy size (35 kt)
Kemi	178 m	10.0 m	na	–	–	na
Oulu	150 m	10.0 m	na	–	–	na
Raahe	355 m	10.0 m	2250 t/h	–	–	1.18 d
Kokkola	425 m	13.0 m	4500 t/h	–	1.21 d	0.59 d
Pori	450 m	15.3 m	2000 t/h	5.57 d	3.64 d	1.77 d

Restrictions due to ice present a particular problem to the bulk export of materials and the regularity with which vessels can access berths.

Table 17-7 summarizes the lengths of restriction period to normal navigation and the period of IA ice class in 2002 to 2011. Large Cape size and Panamax vessels have normally not been ice classed and they have historically not been assisted by ice breakers due their size and the difficulty in navigation.

Table 17-7: Ice Restrictions 2002 to 2011

Port	Restriction period (average)	Restriction period (limits)	IA restriction period (average)	IA restriction period (limits)
Pori	2.7 months	0–6.0 months	0.2 months	0–2.1 months
Kokkola	4.3 months	1.2–6.9 months	2.7 months	0–4.8 months
Raahe	4.8 months	1.8–6.9 months	3.4 months	0.7–5.5 months
Oulu	5.1 months	2.3–7.2 months	3.8 months	1.4–5.8 months
Kemi	5.1 months	2.3–7.2 months	3.8 months	1.4–5.8 months

The ports on gulf of Bothnia forms three groups:

- mild ice situation – Pori;
- moderate ice situation - Oulu, Kemi, Kokkola & Raahe; and
- severe ice situation - Tornio.

It should be noted that Tornio has been discounted due to its low availability and high ice severity.

During the winter time, Cape size and Panamax vessels cannot be used between a 10 to 20 week period depending on the severity of the winter and the port selected.

Results of System Analysis

The results of the rail analysis show that the proposed route is sensitive to both train speeds and buffer times at passing points, which indicates that whilst the capacity of the track can be increased to meet the Company's requirements, perturbation is a possibility and that whilst installation of additional loops mitigates this perturbation it does not remove it.

The results of the shipping analysis indicate that no port has clear advantages in terms of operational efficiency, CAPEX or OPEX when the system is considered as a whole.

Whilst Pori has a significantly less restricted ice period and can accommodate larger vessels at berth, the additional rail haul offsets these advantages.

17.6.5 SRK Comments

Cu-Au Concentrate

The proposals presented for the Cu-Au concentrate logistics are considered to be appropriate.

General

SRK notes that a "whole system" model has not been undertaken and that the logistics chain has been modelled using discrete stages (plant; rail; ship etc).

It is typical to model the entire logistics chain using a dynamic simulation model to ensure that capacities are maintained and that perturbation within the system is accounted for.

In general SRK would anticipate that a dynamic simulation model of the logistics system would include such a model as the practical throughput capacity of any export system directly relates to how each of the system components collectively interacts. Dynamic simulation can therefore be used to inform the design and optimise export infrastructure assets and equipment selection.

However, SRK notes that several steps within the logistics chain are assigned to contractors or third party service providers who will be responsible for providing and maintain elements of the logistics system such as the warehousing at the port.

Providing contracts are established that clearly specify the Company's minimum requirements, SRK does not foresee the lack of a "whole system" model to be a flaw to the logistics analysis undertaken.

Warehouse / Stockpiles at Rautuvaara

SRK notes that the sizing of the warehouse is for three days of Fe concentrate (19,800 t) and that the assumption that this material is delivered regularly over the "interval of hours". It is likely that significant variations can be anticipated form the beneficiation plant and that Fe concentrate delivery will vary considerably over the "interval of hours".

It is normal to model warehouse capacity based upon the anticipated availability of the beneficiation plant to allow adequate storage capacity to be retained so as to not impede the flow of Fe concentrate along the railway system.

Rail Transportation Rautuvaara to Kokkola

At present, the Company proposes to use conventional ore wagons as part of the base case for Fe Concentrate transport; however, SRK notes that several other options have been considered including the use of containers and flat wagons for transportation. SRK understands that these options may be considered during the next stage of Project development; however, this is not considered significant in the overall logistics chain.

Port of Kokkola

The proposed layout for the Port of Kokkola appears to be in line with standard industry practice and would appear to offer an efficient means of unloading rail cars and ship loading.

Whilst the Company has engaged with the contractor and port authority, the contractor and port remain responsible for the efficient and economic transhipment of Fe concentrate from the rail line to the OGV.

SRK notes that additional ports have been investigated outside the "base case" and that whilst these offer certain advantages and disadvantages due to ice conditions, available berths and rail haul distances to all ports considered provide similar outcomes and offer no clear advantage over the base case proposed.

Shipment to customers via Ocean Going Vessels

SRK notes that the shipping cost estimate provided does not include:

- variability of supply and demand within the shipping market;
- availability of empty vessels in different ports; and
- restrictions due to winter conditions.

17.6.6 Concentrate Logistics Summary

It is SRK's opinion, the proposed method and route for the export of concentrates from Hannukainen has been established in principle and in sufficient detail to determine the feasibility of the selected route and associated capital and operating costs.

The options selected have been developed to a reasonable degree of detail enabling an estimate to be produced that has a degree of confidence that is suitable for the HFS. The capital costs required for the infrastructure and the transportation of concentrates have been assessed in detail and where the designs are conceptual, reasonable allowances have been made.

17.7 Fe Concentrate Transport Operating Costs

OPEX has been derived for the proposed base case and other options considered on the basis of quotations from suitably qualified contractors or operators for the proposed infrastructure. SRK notes that OPEX associated with Shipping, Port Fees and Fairway Dues are included within the price calculations and are therefore not included in the transport OPEX summarised in Table 17-8 and Table 17-9.

Table 17-8: Proposed OPEX Costs

WBS	Item	OPEX Costs (USD/t)	Total LOM OPEX (USD'M)
NH_LG_02_03	**Rail Transport**	**11.20**	**390.82**
	Wagon Loading (Rautuvaara)	0.19	
	Rail Transport	8.19	
	Wagon Rental	2.82	
NH_LG_03_00	**Port Operations**	**6.44**	**226.13**
	Port Operations in Kokkola	6.47	
NH_PS_01_01	**Project Management**	**0.08**	**2.67**
		0.08	
	Total	**17.75**	**619.62**

Table 17-9: Base Case % Split Onshore Activities OPEX

Cost item	Total share	Driver	Driver share
Wagon loading	1%	Fuel[e]	50 %
		Labour[e]	30 %
		Capital[e]	20 %
Wagon rental	14%	Capital	100 %
Rail transport	42%	Rail fees	30 %
		Fuel	25 %
		Capital	18 %
		Other	27 %
Port operations	43%	Fuel[e]	30 %
		Labour[e]	30 %
		Capital	40 %
Total	100%		

Shipping costs presented have been established by the Company's consultant on the basis that a definite export location is unknown and is likely to be split between several geographic markets with the POS being either in:

- Europe;
- Middle East; or
- Asia.

The shipping costs are included as a mix based on the assumption that 60% of the concentrate is shipped to Europe and 40% to the Middle East. It was assumed that Rotterdam would be the European POS and an average of the ports of Alexandria, Doha and Mumbai would represent the Middle East markets POS.

The OPEX costs for ice free conditions have been derived based on direct shipping with Panamax vessels being the optimal solution for European and Middle Eastern destinations and shipping to Asian POS using topped 110 kt Babycape vessels from the Port of Kokkola

These costs from Kokkola to the POS have been converted into a "shipping triangle" (Figure 17-9) in that it presents the total transportation costs from the Hannukainen mine to target markets for both unrestricted and ice restricted scenarios.

The shares of transport to key target markets are assigned and the triangle includes relevant linear combinations of the costs. Depending upon what mix of market share is adopted and what ice restriction is applied, the range of total costs vary significantly (Figure 17-10).

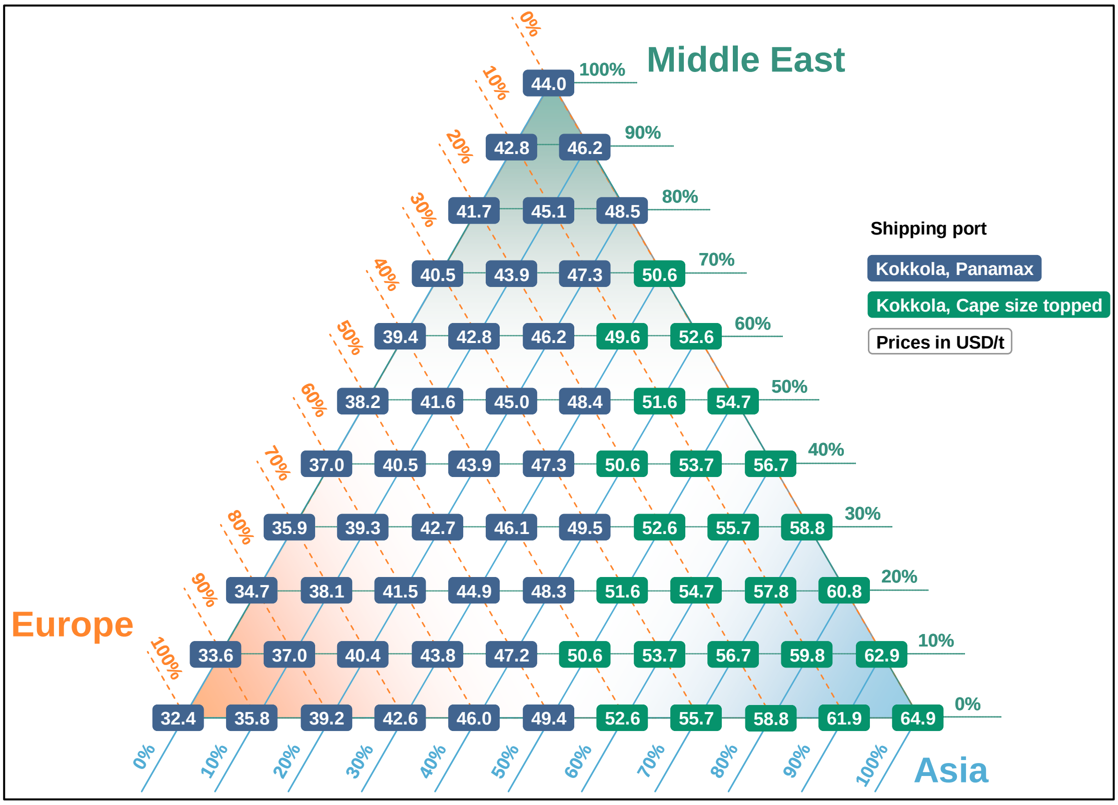


Figure 17-9: Logistics cost mix to different target markets during summer season)

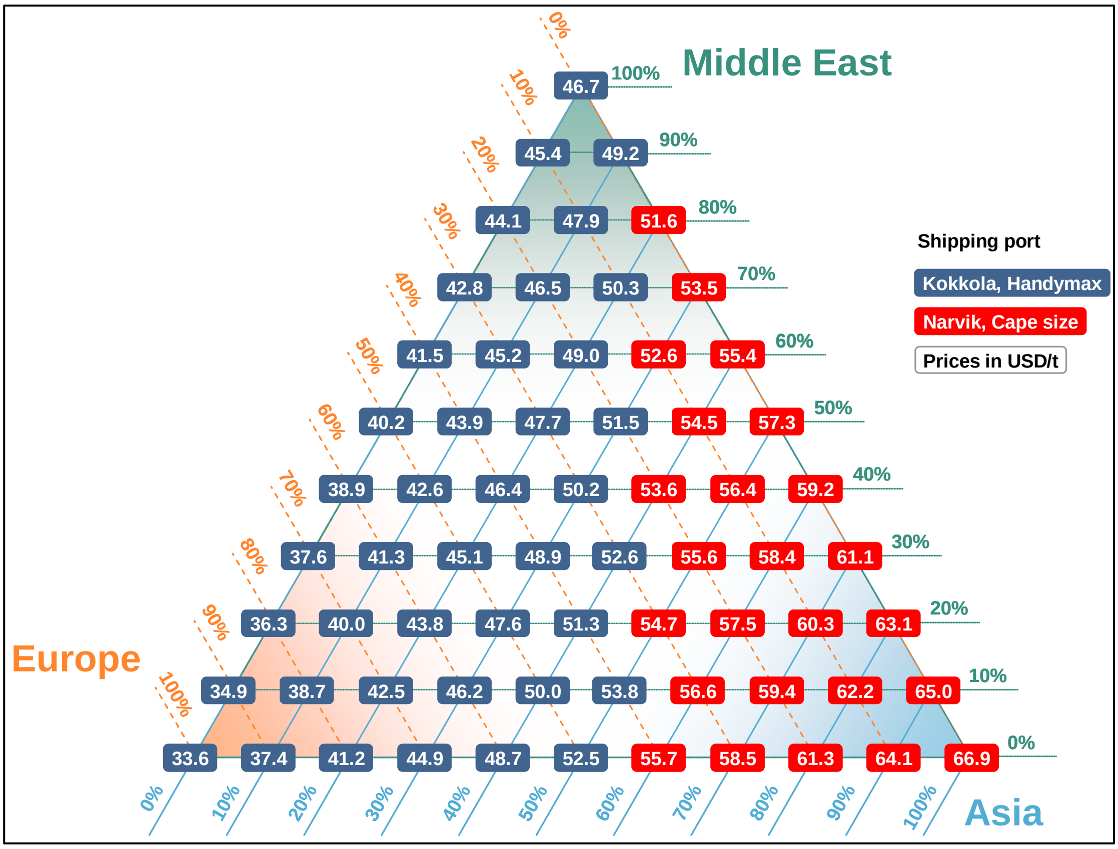


Figure 17-10: Logistics cost mix to different target markets during winter season

17.7.1 SRK Comments – OPEX Concentrate Transport

The OPEX derived for the Project combines conventional OPEX and third party costs from suppliers and contractors.

Other capital investments, from other third parties, such as the Port of Kokkola have no Company contribution and it is therefore appropriate that no allowance be made for provision of funding by the Company as CAPEX and that the contractor/operator will recover their CAPEX as part of and OPEX fee.

The Company has undertaken a sensitivity analysis based upon a large number of export scenarios and these are considered to be adequate for the purposes of informing on potential destinations for Fe concentrate. The results of this anaylsis indicate that the project is not sensitive to export location within Finland as the savings achieved by utilising larger vessels is generally offset by the increased rail transportation costs for the same POS.

It should be noted that all OPEX has been assumed to be a variable component of the mine production on a per tonne basis and that no fixed or semi variable cost components have been identified.

17.8 Infrastructure Operating Costs

Table 17-10 presents the total Operating Costs over the life of mine associated with Work Package E (Infrastructure) comprising the Industrial Area (IA) and Process Plant (PP). All other operating costs are specifically excluded.

Table 17-10: Infrastructure Operating Cost Summary

Asset	Total Operating Cost over the Life of Mine		
	(EUR'M)	(USD'M)	%
Industrial Area (IA)			
Pre Development (01)			
Site preparation in Rautuvaara	1.84	2.36	2%
Site preparation in Hannukainen	0.68	0.87	1%
Facilities for const. period	0.56	0.71	1%
Access Road	0.25	0.32	0%
Bulidings (03)			
Heat Procurement	45.33	58.02	55%
Admin & Welfare Building	7.97	10.20	10%
Truck workshop	2.02	2.59	2%
Plant workshop	0.67	0.85	1%
Assay Lab	0.17	0.22	0%
Reagent storage	0.15	0.19	0%
Gate Building	0.09	0.11	0%
Switch station	0.02	0.02	0%
Roads and infrastructure (04)			
Internal Roads	0.71	0.91	1%
Car Parking	0.30	0.39	0%
External Site Lighting	0.23	0.29	0%
Aerial drainage	0.08	0.10	0%
Utilities & Systems (05)			
Water pumps & piping	7.78	9.96	9%
Utilities and Systems	3.01	3.86	4%
ICT System	1.96	2.51	2%
Sewage Treatment System	0.51	0.65	1%
Process Water Ponds	0.06	0.08	0%
Equipment (07)			
Workshop Equipment	0.04	0.05	0%
Total Industrial Area (IA)	**74.44**	**95.28**	**90%**
Process Plant (PP)			
Buildings (02)			
Processing Plant Buildings	1.17	1.50	1%
Stockpile Building	0.38	0.48	0%
Crushing Station	0.20	0.25	0%
Process (03)			
Crushing & Conveying	6.27	8.02	8%
Total Process Plant (PP)	**8.01**	**10.25**	**10%**
NET Total Operating Costs	**82.45**	**105.53**	**100%**

17.9 Mine Site Infrastructure Capital Costs

Table 17-11 presents a summary of capital costs associated with the Hannukainen Mine, Industrial Area, Rautuvaara Process Plant and Project Support Services.

Table 17-11: Mine Site Infrastructure Capital Costs

Asset	Capital Cost (EUR'M)	Capital Cost (USD'M)
Hannukainen Mine (HM)		
Explosives Store	0.15	0.19
Total Hannukainen Mine (HM)	**0.15**	**0.19**
Industrial Area (IA)		
Pre Development (01)		
Site preparation in Rautuvaara	4.52	5.78
Site preparation in Hannukainen	2.55	3.27
Access Road	1.58	2.02
Construction Facilities (02)		
Construction Power Supply	1.30	1.66
Fencing & Access	0.50	0.64
Buildings (03)		
Truck workshop	7.25	9.27
Plant workshop	3.02	3.87
Assay Lab	1.99	2.55
Switch station	1.49	1.91
Reagent Storage	1.41	1.81
Admin & Welfare Building	1.24	1.59
Weighbridge	0.23	0.29
Power plant	0.05	0.06
Gate Building	0.03	0.04
Roads and infrastructure (04)		
Internal Roads	9.69	12.40
Aerial drainage	1.42	1.82
External Site Lighting	0.83	1.06
Car Parking	0.38	0.48
Utilities & Systems (05)		
Water pumps & piping	18.80	24.06
Elect. Distribution in Rautuvaara	4.76	6.09
Electrical Power Supply	4.03	5.16
ICT System	2.35	3.01
Electrical distribution	2.15	2.76
Process Water Ponds	2.10	2.69
Sewage Treatment System	2.07	2.65
Fuel Supply System	0.01	0.02
Tailings Mgmt Facility (06)		
Power plant	0.14	0.18
Total Industrial Area (IA)	**75.89**	**97.13**
Rautuvaara Process Plant (PP)		
Buildings (02)		
Processing Plant Buildings	30.83	39.47
Crushing Station	8.87	11.35
Stockpile Building	8.04	10.29
Conveyor Foundations	3.49	4.47
Rail Load Out building	2.62	3.35

Asset	Capital Cost (EUR'M)	Capital Cost (USD'M)
Thickener Foundations	0.85	1.09
Process (03)		
Crushing & Conveying	23.38	29.93
Tailings Thickener & pumps	0.57	0.73
Beneficiation	0.36	0.46
Total Process Plant (PP)	**79.01**	**101.12**
Project Support Services (PS)		
Final design engineering	9.28	11.88
Project Management	5.44	9.96
Site Security	0.42	0.54
Janitorial services	0.42	0.54
Survey Control	0.41	0.53
Third Party Vendor Inspection	0.19	0.24
Third Party NDT	0.17	0.22
Medical Services	0.11	0.13
Site catering	0.02	0.03
Total Project Support Services (PS)	**16.46**	**21.07**
NET Total Capital Costs	**171.50**	**219.51**

17.10 Concentrate Transport Capital Costs

The Company has prepared estimates for Capital expenditure for its own operations and has held discussions with various third part organisations such as FTA, the ports in the Gulf of Bothnia and train operating companies to establish total capital expenditure (CAPEX) for the logistics element of the Project.

The capital expenditure of the warehouse has been assessed by the Company as part of the mine infrastructure package and is therefore commented on in that section of their report.

The capital expenditure for the rail head installation and the refurbishment of the government owned rail line between Rautuvaara and Kokkola has been established by the FTA and is presented in Table 17-12. SRK notes that a capital investment of USD 14.7M for Logistics is presented in the financial model with the remainng USD 0.7M Planning Fee presented under Project Support.

Table 17-12: Concentrate Rail Transport Capital Costs

Item	Project Capital Investment by the Company	
	%	USD'M
New Rail Yard at Rautuvaara	100	2.6
Refurbishment of rail line Niesa to Rautuvaara	100	6.4
Niesa to Kolari Upgrade	50	3.8
Kolari to Kemi Upgrade	50	1.9
Planning Fee	50	0.7
Total		**15.4**

The only CAPEX costs attributable to the Company is the USD15.4M associated with the construction of the new rail yard at Rautuvaara and the refurbishment of the rail line to Kemi where the burden is shared with the FTA.

During the Company's negotiations with the Port of Kokkola, the port authority has committed to build adequate port infrastructure to serve Northland's requirements. This investment is estimate at USD64,700,000 for the Port of Kokkola which does not require the Company to provide capital for this investment. This cost has been included within the Port of Kokkola's offer on a per tonne basis of concentrate exported over the LoM.

17.10.1 SRK Comments - CAPEX

The proposal outline for the Project is based upon historic agreements with FTA that have not been executed. The Project has progressed on the basis that the cost sharing between the Company and FTA will be a 50:50 split of capital costs as a "worst case" scenario for the Company and the expectation is that through negotiation with FTA the Company's contribution can be reduced below the 50% burden.

SRK remains unsighted to the agreements with FTA and it is unclear if this method of funding is acceptable to FTA or if the Company will be required to provide their proportion of the CAPEX over the period of construction.

18 MARKET STUDIES AND CONTRACTS

18.1 Introduction

Northland engaged Raw Materials Group ("RMG") to undertake a study into the iron ore market with forecasts of iron ore prices and as a by-product, a copper concentrate containing gold. On the iron ore market, a forecasts of iron ore prices and a net back calculation for the Hannukainen iron ore concentrate has been made and for the copper concentrate an in-depth market study including a high level copper price forecast has been made.

The following comprises a summary of the RMG market report dated October 2013 that has been included in the Hannukainen FS document, the outcomes of which are used in the economical analysis of the Project and reflect additional assumptions by Northland.

Northland will produce about 2 Mtpa of Fe concentrate and about 40,000 tpa of copper concentrate over the LoM. The quantity of copper concentrate produced varies each year due to the large variation in the copper grade of the ore mined. In the last year of operation, only about 12,500 t of concentrate are produced. This section of the report looks at the potential markets for the products from Hannukainen, the pricing of these and their net value.

18.2 Iron Concentrate

18.2.1 Introduction

The Hannukainen mining and process operations are located in Northern Finland, with concentrate being transported by rail, around 450 km, to the proposed Port of Kokkola, where it is discharged, stored and then reclaimed for loading to ships.

Metallurgical testing has shown that processing the Hannukainen ore requires fine grinding to liberate the valuable magnetite, copper and gold, which allows the unwanted waste materials to be discarded. The final products are; a very high quality magnetite iron ore concentrate (70% Fe); and a copper concentrate (>25% Cu) with gold credits (7 g/t of Cu-concentrate).

The properties of this Fe concentrate makes it most suitable for the production of iron ore pellets, both pellets for use in blast furnaces ("BF-pellets") and for use in DRI production ("DR-pellets"). Direct reduced iron ("DRI") is primarily then used in electric arc furnaces ("EAF"). It will also be most suitable for sinter production due to its high Fe content and its low level of impurities.

Table 18-1 shows the expected product quality, based on pilot scale tests conducted by SGS Canada Inc at its test facilities in Lakefield, Ontario, Canada.

Table 18-1: Hannukainen Product Quality Certification

Fe	S	SiO_2	Al_2O_3	MnO	CaO	MgO	P_2O_5	TiO_2
70.0	<0.05	<2.5	<0.4	<0.2	<0.6	<0.6	<0.01	<0.1

Source: SGS – Certificate of Analysis

18.2.2 Pricing methodology

Iron prices are settled on spot basis or tied to an index describing the spot market. There are currently three generally available price indices, published by Metal Bulletin, The Steel Index and Platts. They vary slightly in construction but, by and large, reflect the markets for low grade (58% Fe), medium grade (62% Fe) and high grade (65-66% Fe) iron ore. There is a premium paid for higher grade ore. The premium for high grade ore (typically 66 % Fe or above) is not expected to be proportional to the premiums achieved by medium grade above low grade but may be progressively increasing, reflecting an increased value to consumers. However, as demand for productivity follows the changes in demand for steel products, this may come to vary over time and thus also be progressively lower than presently assumed.

As can be seen from Figure 18-1, in spot market pricing, the different iron content standards are priced independently creating a premium for 62% Fe that is not proportional to the 58% Fe price; that is, if the 58% Fe price is USD100/t, the 62% Fe price is not necessarily USD107/t (100*62/58), indicating a 7/4 = USD1.75/% Fe price. On the contrary, during the period of the present pricing scheme, there has been a premium paid for the higher material ranging from 2 to USD13/% Fe.

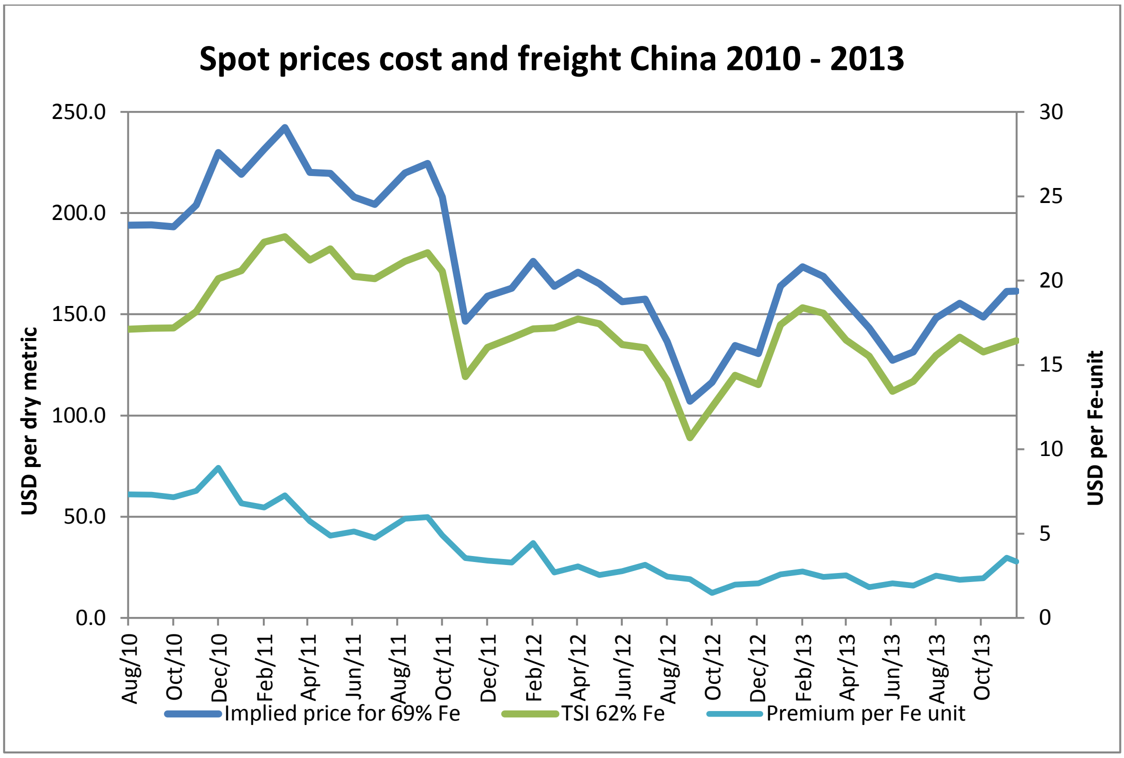


Figure 18-1: Prices in USD/DMT CFR China 2010-2013

Source: The Steel Index

Pricing of iron ore has historically been done in US cents per Fe-unit; that is, a producer of iron ore has been paid equal amount in US cents for every Fe-unit delivered. When the benchmark price was determined for a 62% Fe product at for example 150 c/dmtu (US cents per dry metric tonne unit Fe) that producer would get 62 x 150 cents = 93 USD per DMT. A producer of a high grade product of 69% Fe would get 69 x 150 = 103.50 USD per DMT.

However, since the breakdown of the benchmark pricing system there has been a premium paid for high grade ores and concentrates over and above what the c/dmtu gave. This premium has varied between 2.50 and almost USD10 per Fe-unit indicating that from time to time high grade ores are paid substantially more not only in absolute terms, but also in cents per Fe-unit. In times of increasing and strong steel demand and hence a need for increased production and productivity in the blast furnace the premium will rise and approach its maximum (so far USD12.6/% Fe) as the steel company is trying to maximise production and productivity.

In such times, the steel producers are willing to pay a premium for ores of high grade. On the other hand, in times of low steel demand the blast furnace operator will try to decrease production/productivity without having to close down the blast furnace, because this is a difficult, time consuming and costly procedure. The blast furnace is further a highly, capital intensive unit and will carry considerable capital costs which need to be covered irrespective of the blast furnace is operating or not. In such difficult times the steel company does not want to close down the furnace, but only decrease its production/productivity by feeding lower grade material, which not only reduces costs (which is, of course, of prime importance when steel demand and hence steel prices are low), but also reduces output. Northland has, however, taken a conservative approach and in the price assumptions only calculated with straight cents per Fe-units, but added a small so called Value-In-Use premium of USD3 flat.

The Value-In-Use premium is motivated by, for example, it being a magnetite which saves up to 60% of energy consumption in the pelletising process; the very fine particle size also saves energy for the buyer; and the low level of impurities contribute to better steel quality, as well as better productivity in the iron and steel making processes.

The prices for Hannukainen concentrate are calculated according to the following formula:

Reference price TSI 62% CFR China + Fe-premium per Fe-unit above 62% x (Actual Fe – 62%) + Value-In-Use premium for added value to the Buyer - Freight cost differential vs competing product

Competing product for Hannukainen concentrate is primarily Brazilian concentrate where the freight cost differential is calculated by deducting freight cost from Brazil to China, adding freight cost from Brazil to relevant market and subtracting freight cost from Kokkola to the same market.

Table 18-2 and Table 18-3 show the price calculations for shipments to Europe and the Middle East. If volumes are to be shipped to the Middle East, those customers would have to compete for the volumes with the European steel mills and pay the same price.

Table 18-2: Price calculation for Hannukainen Concentrate to Europe

Shipments to Europe	**2015 to 2019**	**2020 to 2024**	**2025 to 2029**	**2030**
Reference product, TSI 62% Fe, CFR China (USD/DMT)	130.00	120.00	126.00	133.00
Vale pellet feed CFR China	136.92	126.39	132.71	140.08
Freight cost Tubarao - China	22.86	22.86	22.86	22.86
FOB price Tubarao for Vale pellet feed shipments to Europe	**114.06**	**103.53**	**109.85**	**117.22**
Freight cost Tubarao - Europe	13.71	13.71	13.71	13.71
Implied CFR price Europe for Vale pellet feed	**127.78**	**117.24**	**123.56**	**130.94**
Hannukainen conc. CFR Middle East	136.97	125.68	132.46	140.36
Freight cost Kokkola - Europe	10.64	10.64	10.64	10.64
FOB price Kokkola for shipments to Europe, excl. of VIU premium	***126.33***	***115.04***	***121.82***	***129.72***
VIU premium	3.00	3.00	3.00	3.00
FOB price Kokkola for shipments to Europe, incl. of VIU premium	***129.33***	***118.04***	***124.82***	***132.72***

** For reference only as no shipments have yet started.*

Table 18-3: Price calculation for Hannukainen Concentrate to Middle East (Egypt)

Shipments to Middle East (Egypt)	**2015 to 2019**	**2020 to 2024**	**2025 to 2029**	**2030**
Reference product, TSI 62% Fe, CFR China (USD/DMT)	130.00	120.00	126.00	133.00
Vale pellet feed CFR China	136.92	126.39	132.71	140.08
Freight cost Tubarao - China	22.86	22.86	22.86	22.86
FOB price Tubarao for Vale pellet feed shipments to Middle East	**114.06**	**103.53**	**109.85**	**117.22**
Freight cost Tubarao - Middle East	22.86	22.86	22.86	22.86
Implied CFR price Middle East for Vale pellet feed	**136.92**	**126.39**	**132.71**	**140.08**
Hannukainen conc. CFR Middle East	146.77	135.48	142.26	150.16
Freight cost Kokkola - Middle East	26.60	26.60	26.60	26.60
FOB price Kokkola for shipments to Middle East, excl. of VIU premium	***120.18***	***108.89***	***115.66***	***123.57***
VIU premium	5.00	5.00	5.00	5.00
FOB price Kokkola for shipments to Middle East, incl. of VIU premium	***125.18***	***113.89***	***120.66***	***128.57***

** For reference only as no shipments have yet started.*

18.3 Copper Concentrate

The Hannukainen deposit is an IOCG system. In addition to producing an iron concentrate, there is an option to also produce a copper concentrate. The value of this copper concentrate could give an important support to the total value created in the Project.

The copper concentrate produced in the Project is expected to have the characteristics set out in Table 18-4.

Table 18-4: Hannukainen Copper concentrate characteristics.

	Cu	**Au**	**Moisture**
Hannukainen Cu concentrate	25%	7.1g/t	10%

Source: SGS - Certificate of Analysis

In addition, there has been some preliminary analysis made on the content of elements that could possibly render a penalty. These results indicate that the Hannukainen concentrate would be considered “clean” or possibly even “very clean” and hence not likely to be subject to any penalties. There will also be some gold credits paid and it is also possible that there could be further silver credits for the concentrate depending on the silver content. Silver credits have, however, not been included in the financial analysis.

18.3.1 Pricing Methodology

The standard price of copper concentrate per tonne is based on the calculation of minimum deduction of 1 unit (%Cu in concentrate) multiplied by 96.65% (metal recovery) and the copper price. The costs of smelting and refining the concentrate are charged to the miner as treatment and refining charges (“TC/RC”).

The copper concentrate also includes 7.1 g/t gold which would add value to the product. Gold is forecast by RMG to move towards a long term price of USD2,000/oz.

Using the above copper concentrate data and applying a TC of USD75 /t and an RC of 7.5 USc/lb to the copper concentrate and a refining charge of USD6 /oz Au, then it is possible to calculate a net smelter return (“NSR”) per tonne concentrate at set metal prices. Further in this calculation, transport handling and insurance have been set at zero, assuming the concentrate will be bought at mine gate moving the costs to the smelter company. It is assumed these costs will be covered by the smelting company.

These factors and assumptions result in the following equation to calculate an NSR per tonne concentrate.

1. Concentrate x grade x metal deduction x metal recovery = Payable Metal
2. Payable metal x metal price = Gross Value of Metal
3. Gross Value of metal – Total Charges = Net Value of Metal

Table 18-5 shows the calculated NSR for each tonne of concentrate by adding each metal calculation. In the following example, the NSR is USD1,743/t (that is USD1,499/t for Cu + USD244/t for Au).

Table 18-5: Calculation example using USD6,950 /t Cu and USD1,250 /oz Au

	Copper	Unit	Gold	
Tonnes of Conc	35,000	t		
Metal in Conc	8,750	t	7,989	oz
Metal per Tonne of Conc	0.25	t	0.23	oz
	551	lb	7.1	g
Metal Deduction	0.01	t	1	g
Metal Recovery	96.65 %			
Payable Metals	0.23	t	6.1	g
Metal Price	6,950	USD/t	1,250	USD/oz
Gross Value of Metal	1,612	USD/t conc	245	USD/t conc
Treatment Charge	75	USD/t conc		
Penalties (As, Sb, Bi, Hg)	0	USD/t conc		
Price Participation	0	USD/t conc		
Refining Charges	0.075	USD/lb	6	USD/oz
	38.35	USD/t conc	1	USD/t conc
Transport	0	USD/t conc		
Handling	0	USD/t conc		
Insurance	0	USD/t conc		
Total Charges	113	USD/t conc	1	USD/t conc
Net Value of Metal	**1,499**	**USD/t conc**	**244**	**USD/t conc**

18.4 Off Take Contracts

SRK has been informed by Northland that it has not entered into any off take contracts relating to concentrate production from Hannukainen.

19 ENVIRONMENT AND SOCIAL ASSESSMENT, PERMITTING AND MANAGEMENT

An Environmental and Social Impact ("ESI") study is being compiled by Ramboll and Pöyry. ERM is preparing the ESI chapter for the HFS, which is being compiled by Jacobs. SRK's comment on the status of environmental and social issues associated with the Project is given based on a review of the HFS, ESI chapter and discussions with Northland personnel. An indication of whether the issues are considered material to the Project and how the issues will be managed going forward is also provided.

A number of studies have been undertaken by various organizations to design the Project, characterize and understand the Project's environmental and social setting and evaluate the impacts likely to arise from its activities. Key studies include:

- Project Description for the Project;
- Hannukainen Iron, Copper and Gold Mine Project – ESI (draft sections);
- Hannukainen Preliminary Economic Assessment Project Study Report;
- Hannukainen FS Waste Rock Geochemical Acid Rock Drainage Metals Leaching ("ARDML") Characterization Report, SRK (2011);
- Technical Report on the Mineral Resource Estimates and Preliminary Assessment of the Project; and
- Hannukainen Project – Social Impact Assessment (final sections).

19.1 Project Description

The following Project components are located in two areas: the Hannukainen area ('mining area'); and Rautuvaara area ('industrial/process area'). Figure 19-1 shows the layout of the proposed mine.

Hannukainen area (approximately 1800 ha):

- Hannukainen and Kuervitikko open pits;
- East, West and North Waste Rock Management Facilities ("WRMF");
- primary ore crusher;
- clarification pond;
- workshop area;
- access roads; and
- 1 km buffer zone (safety zone).

Transport corridor between Hannukainen and Rautuvaara areas (approximately 9 km, covering 45 ha):

- Conveyor transporting ore, water pipeline in which Hannukainen effluent is pumped to Rautuvaara clarification ponds, and access road.

Rautuvaara area (approximately 580 ha):

- process plant including filtration plant;
- TMF with lined High-S and unlined low sulphur sections;
- pipeline from the plant to TMF;
- north and south clarification ponds containing effluent including; 1) Hannukainen and Kuervitikko pit water, 2) runoff from the high sulphur WRMF; and 3) tailings return water; and the ponds supply make up water to the plant;
- pump station at Niesajoki; and
- rail load-out station.

Water pipeline from Rautuvaara to Muonionjoki (9.3 km):

- to discharge excess water from the clarification ponds to Muonionjoki.

Railroad between Rautuvaara railway load-out station and the port at either Raahe, Kemi or Kokkola:

- the existing disused rail corridor between Rautuvaara and Kolari will be upgraded and restored by the Finnish Transport Authority.

The fhistorical mine components and industries currently within Hannukainen and Rautuvaara areas are shown in Table 19-1 and Figure 19-1.

Table 19-1: Historical mine components within the project area

Hannukainen area	Rautuvaara area
Laurinoja and Keurvaara open pits	Two open pits
Waste rock dump	North and south clarification ponds
Aggregates quarry and crushing facility	Waste rock dump
	Concentrator plant site
	Tailings deposits
	Ylläs operational wastewater treatment facility

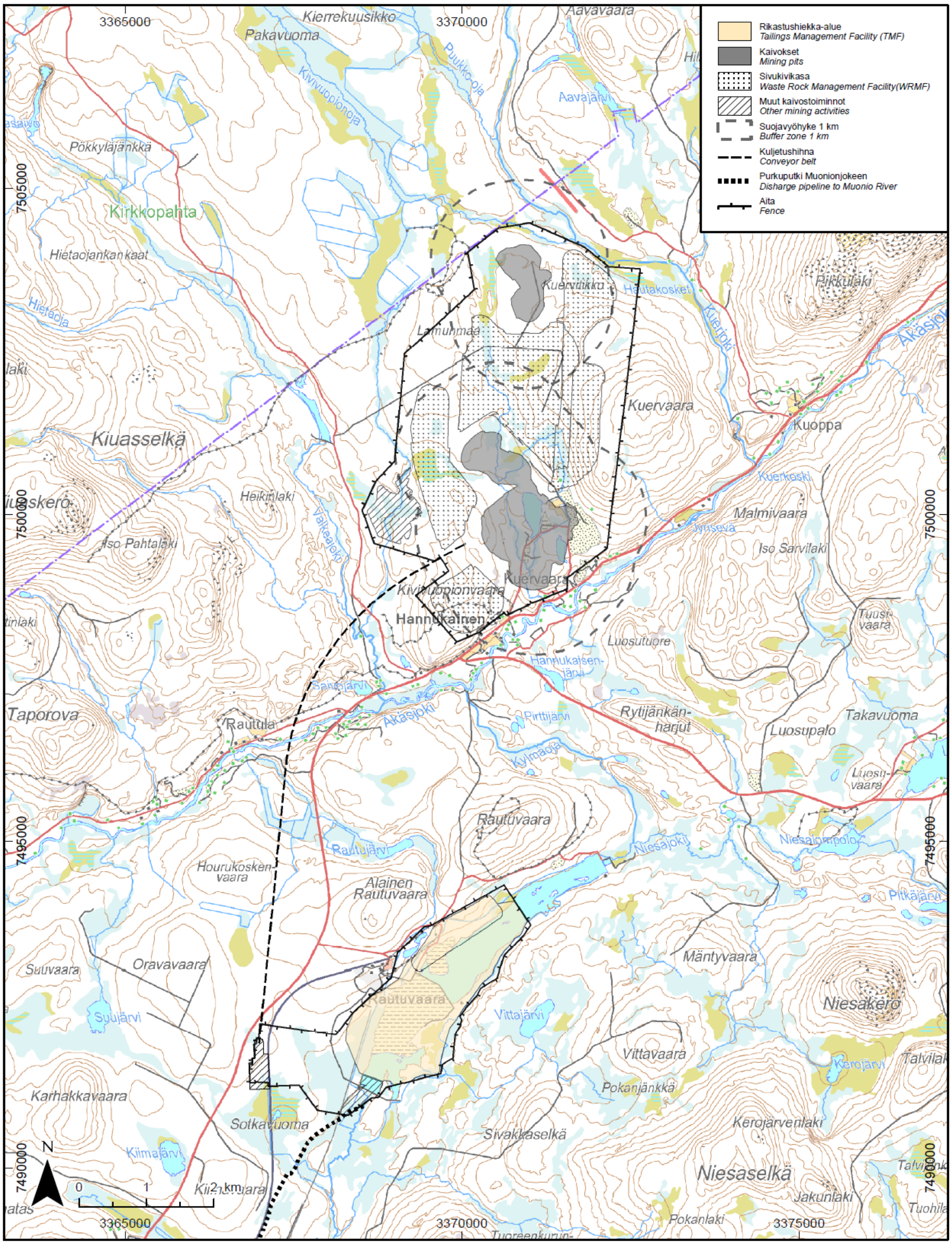


Figure 19-1: Proposed Hannukainen mine layout (after the HFS ESI chapter)

19.2 Project setting

The pre-disturbance environment has been well characterized. Northland has undertaken a complete baseline description incorporating air, water, traffic and noise monitoring data; some data collection has taken place over several years. Geochemical, hydrological and hydrogeological studies including numerical modelling has been carried out and included in the HIA study completed by Northland. Social data has been collected as part of a social impact assessment. Based on a collation of this information provided in the HFS, a summary of the bio-physical and socio-economic setting of the Project is summarised below.

19.2.1 Bio-physical

Hannukainen's mining components will be located between a group of small hills to the west and the Kuervaara hill to the east. Waste rock will be placed on the eastern and northern ridges of this hill. At Rautuvaara, industrial/process facilities will be located in a river basin (Niesa Basin) between groups of hills. Climate data was recorded at local weather stations from 1971 onwards. The average annual precipitation is 580 to 610 mm. Wettest and driest months are between July and August and February and March, respectively. Snow cover usually lasts from October to May. The average annual temperature is -0.1ºC. Maximum monthly averages are +15ºC in July and 13ºC in January. Wind speeds are predominantly from the south, northwest and southeast and are between 3.8 and 4.5 m/s over 90% of the time. Air quality is generally good; annual average dust, PM_{10} (respiratory particles) and metal concentrations are below thresholds. Certain plants indicate pollution from historical mining at Rautuvaara.

Figure 19-2 shows the hydrological setting of the Project. The following surface water courses are classified as either Class 1 - 'Excellent' or Class 2 - 'Good', with respect to their ecological and chemical status. The Project covers four river basins in the greater Torne– Muonionjoki Basin in Finland ('joki' means river in Finnish). The (1) Kuerjoki and (2) Valkeajoki are immediately east and west of the Hannukainen area respectively. These rivers are tributaries of the (3) Äkäsjoki, which is located directly on the southern boundary of the Hannukainen pit. This river flows westwards into the Muonionjoki. The TMF at Rautuvaara will be located in the Niesa Basin in the headwaters between the (4) Niesajoki and Kylmaoja Stream, a tributary of the Äkäsjoki. The Niesajoki flows from the Niesa Basin into the Muonionjoki. The Niesajoki was diverted upstream into the Kylmaoja Stream due to historical mining. The Muonionjoki forms the national boundary between Finland and Sweden along part of its length.

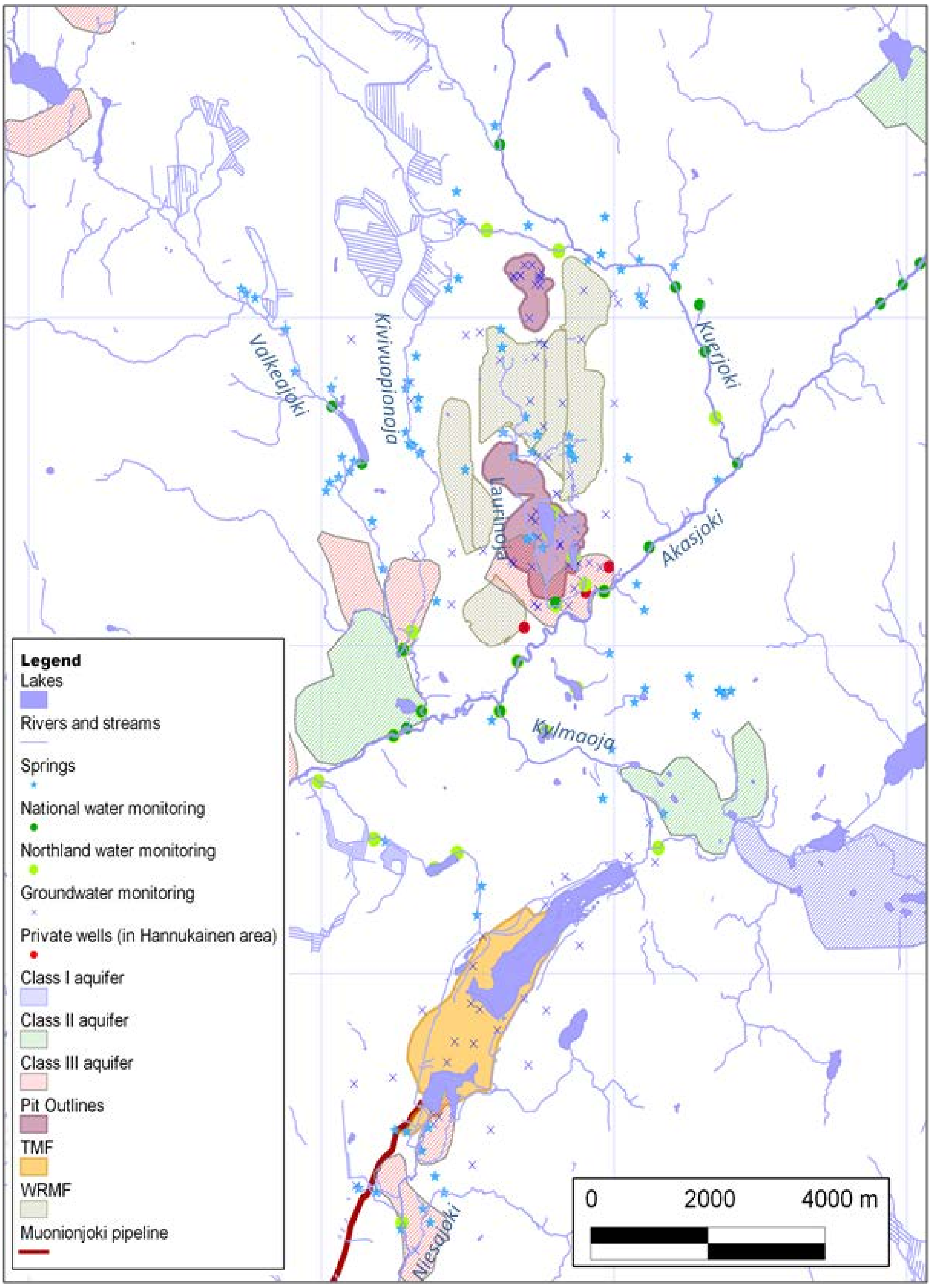


Figure 19-2: Hydrological setting of the Project

The Kivivuopionoja and Laurinoja Streams in the Hannukainen area are tributaries of Äkäsjoki; Laurinoja flows through the Hannukainen pit.

There are three aquifers in the Project areas confined to glacial moraine (mainly sand and gravel deposits) overlying fractured bedrock comprising meta-sediments and meta-volcanics. At Hannukainen, groundwater levels generally mirror topography and are between 1.5 and >10 m below ground level, with the upper overburden often unsaturated. Most groundwater drains to the Äkäsjoki with minor flow to the north. Artisanal groundwater is present at Hannukainen. At Rautuvaara, groundwater levels in the Niesajoki Basin follow topography, with flow directions towards the Niesajoki. Groundwater generally meets local standards with exception of electrical conductivity, sulphate, aluminium and iron, which are elevated due to sulphide mineralisation in the historical Kuervaara and Laurinoja deposits.

Project areas are characterised by barren forest heaths, open areas, seeding stands eutrophic fens and wetlands. Project areas generally have high conservation value. The riverine habitats are of critical importance. Habitats, fauna and flora species with conservation importance/protection status in Project areas also follow.

- The Torne-Muonionjoki Natura 2000 river protection site (FI301912, SCI) and two of the rivers' tributaries in Project areas, namely the Äkäsjoki and Niesajoki are protected Natura 2000 rivers. The Natura 2000 rivers extend into Sweden, where it is known as the Torne-Kalix Natura 2000 river (SE0820430, SCI). The Torne- Muonionjoki system is one of few currently not exploited for hydroelectricity. Only the water systems are protected, no land is included.
- Niesaselkä Natura 2000 site (FI1300706, SCI), a nature conservation area about 2 km southeast of Rautuvaara. Niesaselkä is also protected as an old-growth forest.
- Eutrophic fens at Hannukainen classified as threatened habitat.
- Numerous natural state springs protected under water legislation, which may contain unique species of flora.
- Certain forest habitats totalling 1 ha are protected under the Forest Act but may be used for other purposes.
- The Äkäsjoki (including the river's Valkeajoki and Kuerjoki tributaries), Muonionjoki and Laurinoja Stream are breeding rivers for the vulnerable Atlantic Salmon and critically endangered Sea Trout. The Äkäsjoki is one of the most important sea trout breeding grounds in the Torne-Muonionjoki system.
- Protected otters occur in the Äkäsjoki and its tributaries and in the Niesajoki.
- Certain bogs in the vicinity of the Niesa Basin (including Sotkavuoma, south-west of Rautuvaara) and old forests have diverse breeding bird populations. Nine threatened species breed in the vicinity of Rautuvaara, which are of regional importance. Bird populations are relatively barren at Hannukainen.
- Three vulnerable large birds of prey feed in Project areas but their nesting sites are outside the areas.

- The Hudson Bay Sedge and Fragrant Orchid (IUCN Red List plant species classified as vulnerable), were identified at Rautuvaara. Near-threatened IUCN Red List species at Hannukainen include the Common Moonwort and Weak Sedge. The Hudson Bay Sedge, northern Lapland Marsh Orchid (*Dactylorhiza lapponica*) and Fragrant Orchid are vulnerable IUCN Red List species at Hannukainen. A list of other vascular and moss species also classified as either vulnerable or near threatened IUCN Red List species in Project areas, is given in the EIS chapter of the HFS.

A habitat survey of the area disturbed by the pipeline to the Muonionjoki (the 'Ristimellanjänkkä sub region) has been carried out. The region comprises habitats typical of Northern Finland, such as bogs, coniferous forest and small lakes and ponds. The forests are generally in commercial use and the area appears to not have significant conservation value.

19.2.2 Socio-economic

The Project is located within Kolari and Muonio Municipalities in Finnish Lapland. The Hannukainen area is 1 km from Hannukainen Village (Figure 19-1). Rautuvaara is 3 km from Hannukainen Village and is not expected to impact on properties there. Ninety percent of Laplanders are Finnish, whilst ethnic minorities are from Sweden, Russia and include Sámi (two people registered their first language as Sámi in Kolari). Kolari and Muonio populations are 3,839 and 2,401 respectively (Statistics Finland). The Kolari population has stabilised from a historical decline due to rapid growth in the tourist industry, mainly at Ylläs ski resort (10 km east of Hannukainen) and in neighbouring Kittilä Municipality. Land within the Hannukainen area is used for urbanisation – 1) private households and cabins; 2) forestry; 3) industry; 4) private properties with no buildings; and 5) reindeer husbandry. Additional comment on tourism and reindeer husbandry follows.

Tourism is the largest source of direct income in Kolari, contributing EUR64M in 2008. Tourists are attracted to the area's natural beauty and the Pallas-Yllästunturi National Park, some 15 km east of the Hannukainen area. Recreational activities include horse riding, hunting, fishing, skiing, canoeing, hiking, berry and mushroom picking. Water courses in the vicinity of Project areas are used for recreational purposes, particularly fishing.

Project areas fall within the herding range of the Muonio Reindeer Herding Cooperative (2,674 km^2), mainly with autumn and early winter pastures. Although husbandry appears to be moderately healthy in the cooperative, the industry is marginal and sensitive.

There are a number of archaeological artefacts in Project areas, none of which are reported to have national conservation value. There are three private wells in Project areas.

The Project will utilise the existing road and rail network. Part of the rail network will be upgraded. Roads in the vicinity of Hannukainen include Regional Road 940, which runs along the southern boundary of Hannukainen and connects with Road 21 to the west, and Road 9404. These roads have strong seasonal variation in traffic volume due to the high proportion of tourist movement during peak seasons.

19.3 Regulatory requirements

A summary of the Finnish regulatory requirements and the status of Northland's primary authorisations follow.

An environmental impact assessment ("EIA[6]) process is initially administered by one authority and then the permitting process starts with the EIA used to support applications to other authorities for the various permissions needed to construct and operate the Project, including the environmental permit, water permits and building permits. In the case of the environmental and water permits, they are granted following a joint application submission in accordance with the Environmental Protection Act (EPA 86/00), Environmental Protection Decree (169/00) and the Finnish Water Act (264/1961) by the Aluehallintovirasto ("AVI")/Regional State Administrative Agency.

19.3.1 Finnish EIA procedure

The Act on Environmental Impact Assessment Procedure ("EIA") (468/94) and the EIA Decree (713/06) defines requirements for the EIA procedure. Mining is a listed activity under the EIA Decree and the Project is therefore subject to approval of an EIA. The EIA procedure is a phased approach. The Project proponent initially compiles an EIA programme and thereafter an EIA report. Both the programme and report are submitted to Einkeino, Liikenne- ja Ympäristöministeriö (ELY)/Centre for Economic Development, Transport and the Environment . ELY publicises the programme and report and issues a statement on their adequacy.

Northland has prepared the EIA Programme and plans to complete the EIA report and procedure end 2013. Northland is anticipating ELY will issue its statement on the EIA report end 2013.

Northland has based its consultations with stakeholders on the communication plan outlined in the EIA programme. Stakeholders included various local, regional, national and international authorities and the Swedish government in terms of the Espoo Convention. Five focus groups were involved in the EIA consultation process including tourist groups, reindeer herders and local residents. Northland's draft HFS ESI chapter notes that various changes were made to the Project in response to comments during the consultation process.

Northland does not expect that an EIA or permit is required for an upgrade to the rail corridor between Rautuvaara and Kolari. However, Northland has integrated an assessment of the impacts associated with the additional rail movements required on the network to service the Project into the draft HFS EIA chapter.

[6] The Finnish regulatory system drops the 'social' from the title and uses the term EIA though this implicitly includes impacts to the socio-economic environment as well as the bio-physical environment.

19.3.2 Environmental permit process

Northland plans to submit its environmental permit application report (being prepared in parallel to the HFS ESI chapter and EIA report) end 2013 assuming ELY require no material changes to the EIA report. Based on Northland's written response to ELY's Statement and public complaints and opinions on the EIA report, AVI processes the permit application and issues a record of decision on granting an environmental permit. Northland anticipates AVI will issue a record of decision on its permit application at the end of the third quarter of 2015.

19.3.3 Water and building permits

Water permits are required for activities impacting on waters and/or the water supply (including groundwater) in terms of environmental protection and water legislation (Water Act 587/2011 and Decree 1560/2011),. Northland must apply to AVI for a water permit, which is issued together with the environmental permit. Water act derogation permits are also required to allow the Project proponent to affect certain water features (such as springs and creeks). These permits are approved by ELY and are required before a building permit can be approved by Kolarin ja Muonion Kunnat/Municipalities (Kolari and Muonio). The permit applications should be supported by the EIA Statement. Northland plans to submit its water and derogation permit application jointly with the environmental permit application end 2013.

19.3.4 Nature Conservation Act ("NCA") derogation permit

Derogation permits may also be required in terms of the NCA (1096/1996, Chapters 48 and 49), Forest Act (1093/1996, Chapters 10 and 11) or Law of Cultural Heritage (295/1963) if the development impacts on certain plant species, habitats and forests of conservation importance. NCA derogation permits are administered by ELY. Northland will a require a derogation permit for disturbing certain plant species at Hannukainen (Lapland Marsh Orchid - *Dactylorhiza lapponica*, The Hudson Bay Sedge, northern Lapland Marsh Orchid - *Dactylorhiza lapponica* and Fragrant Orchid). This permit application will be submitted together with the environmental permit application.

19.3.5 Natura assessment

If a Project development is likely to have significant adverse effect on the ecological value of a Natura 2000 site, the Project proponent must conduct an assessment of its impact. The Äkäsjoki and Niesajoki are protected Natura 2000 rivers and therefore Northland must carry out the assessment. Northland plans to complete a separate Natura assessment at the beginning of 2014 and anticipates ELY will finalise its review of the assessment at the end of the second quarter of 2014.

19.3.6 Land use amendments and building permit

Regional land use plans exist for all areas in Finland. In many cases there are also local land use plans (zoning and town plans) to guide construction and other land use changes in areas where land is used intensively or in sensitive areas. Land use and building planning is regulated by the Land Use and Building Act (132/1999). Concerning Northland, local land use amendments and building plans and must be approved by the Lapland Council and Kolari and Muonio Municipalities. (No variation to the regional land use plan is required.) The EIA Statement is required for the amendments to be approved. Following approval of the amendments to local detailed plans, building permits are required from the Municipalities prior to commencing construction. Northland's application for a building permit must demonstrate right of access to land in the mining concession.

19.3.7 Mining concession

Northland must submit an application to Tukes (Finnish Safety and Chemicals Agency) for a mining concession to exploit minerals within certain areas prior to construction of the Project. Before final approval of the concession application is granted, the EIA should be provided to ELY and the authority must have issued a statement on the report. The application for a mining concession must also demonstrate right of access to land required for the Project.

19.3.8 Reindeer husbandry

Legislation governing reindeer husbandry is the Reindeer Husbandry Act (1990 with amendments in 2000). Reindeer husbandry is managed by the Reindeer Herders' Association and ELY, which fall under the Ministry of Agriculture and Forestry. With certain restrictions, reindeer may pasture freely in reindeer husbandry areas, regardless of land ownership and management rights. The Act states that land shall not be used in a way that causes significant harm to reindeer husbandry[7].

19.3.9 International and Swedish Regulatory Requirements

Because Finland is a member of the European Union ("EU"), Finnish legislation incorporates requirements of certain EU directives summarised in the HFS chapter. Key requirements relate to the EU 1992 Habitats Directive, which establishes a network of protected Natura 2000 sites. Article 6 of the Habitats Directive states Projects potentially impacting on Natura 2000 sites must be subject to appropriate assessment. The sites are either designated under the 1992 Habitats Directive as a Site of Community Importance ("SCI") or under the 1979 Birds Directive as a Special Protection Area ("SPA").

The Project areas are in close proximity to the Sweden/Finland border (about 20 km east of the border), which is demarcated by the Muonio-Tornio River System, a Natura 2000 site. Because the Project will impact directly on this Natura 2000 river, Sweden's Environmental Code (1998:808) - Chapter 7, which relates to Natura 2000 sites is applicable to the Project, as is the Espoo Convention on EIA in a Trans-boundary Context.

[7] *In Finland, as opposed to Sweden, reindeer husbandry can be a commercial business not associated with indigenous people's rights.This is the case at Hannukainen.*

Finland has been granted a derogation regarding requirements in Annex II species for the designation of special areas of conservation. This means that salmon and otter (amongst other species) are not protected in the Finnish Natura 2000 areas, although the otter is classified to be near threatened and is protected under the Finnish Hunting Act. Similarly, Arctic Salmon and Sea Trout are protected under Finnish environmental legislation. Moreover, these fish are protected under Chapter 7 of Sweden's Environmental Code (see above paragraph).

19.3.10 International financial standards

The Equator Principles (updated in July 2006) is an assessment standard used by financial institutions when evaluating Projects for potential investment. Although Northland has not decided to seek funding from a listed Equator Principle Finance Institution, it has reportedly considered these Principles when undertaking its EIA.

SRK firstly considers there is a significant risk ELY will require Northland to revise its EIA report and this could affect the subsequent joint application for the environmental and water permits. It is also possible the environmental permit process will be subject to delays due to the significant public interest and sensitivity of the water and conservation issues (see below). There is a risk ELY will require revision of the Natura assessment, which could delay processing of the environmental permit by AVI; technically AVI starts processing the permit after completion of the Natura assessment. In addition, both the EIA and the permit application process may be subject to appeals (by either Northland or members of the public), further delaying the approval process.

19.4 Status of land access rights

The HFS chapter discusses legislation governing land acquisition and expropriation, including the 'old' Mining Act (503/1965) and 'new' Mining Act (621/2011). Northland will attempt to purchase land for the TMF, process plant and private cabins and plots just south of Hannukainen through voluntary purchases in terms of the aforementioned legislation, i.e. 'willing buyer, willing seller scenario'. The costs for purchasing land and compensating small businesses in the land take are provided in Section 21. Northland plans to rent all other land for the life of mine; rental costs are also provided in Section 21.

If required, legal mechanisms are in place to expropriate land for the mining concession but not for land in the 1 km buffer zone around Project areas. The Mining Act (503/1965) states that if a property is expropriated, the landowner will receive a compensation of 1 x fair value, which is determined by comparison of realised purchase prices in sales of similar properties. The new Mining Act (621/2011) gives landowners the right to demand their property be expropriated, conditions for which are given in the HFS chapter. In such cases compensation is 1.5 times the redemption value of the area and no excavation fee is provided.

The Company has defined the Project's land take and current land uses. Approximately 3,300 ha of state and privately owned land are required including 1800 ha at Hannukainen, 580 ha at Rautuvaara, 60 ha for the transport corridor and 45 ha for the discharge pipeline. The HFS chapter sets out the commercial terms and basis for estimating the purchase price of the following:

- 11 permanently occupied private households;
- 35 temporarily occupied private cabins, mainly bordering Regional Road 940 along the southern boundary of the area;
- 6 private properties supporting businesses;
- forestry areas from Metsähallitus, the State Forest Authority; and
- 42 private properties with no buildings.

Northland has consulted the landowners concerned during the initial EIA process and will continue discussions to reach agreements based on monetary compensation, with terms giving property owners incentives to find new land/property on their own initiative. Northland does not intend to assume responsibility for relocating the population; however, the Company will act proactively, on a case by case basis, to assist property owners. Whilst this is acceptable under Finnish requirements, it is not fully aligned with IFC standard Performance Standard 5, Land Acquisition and Involuntary Resettlement.

In line with the 2011 Mining Act, Northland will pay a separate non-recurring payment (50%) to compensate the uncertainty and harm caused due to the commencement of mining activities, in addition to the sale price. This relates to potential impacts to the seller's living, business or recreational activities, as well as to costs associated with the reorganisation of housing or business and removal costs.

There is a moderate risk that the scale and significance of acquisitions may lead to delays in obtaining both regulatory approval to commence construction, as well as the Projects 'social licence to operate' from the community (discussed further below). This risk is higher for land in the buffer zone, which may not be expropriated.

19.5 SRK Summary

Based on the review of the HFS, the principal substantive environmental and social issues and or liabilities relating to the asset/s are listed below.

19.5.1 Water management

Extensive hydrological, hydrogeological and geochemical studies have been undertaken to characterize the baseline environment and predict impacts on water resources as discussed in the HIA of the HFS. Northland has based its impact assessment on site specific water quality objectives, which the company has defined and which it considers reasonable. Finnish authorities may specify further or different water quality standards and the applicable mixing zones in the conditions to the environmental or water permit.

Substantive water management issues follow.

- Dewatering Hannukainen and Kuervitikko pits will lower groundwater levels in a cone of depression around the pits. Nearby water courses are protected Natura 2000 rivers and breeding grounds for protected fish species (Section 19.3). The highly sensitive Äkäsjoki is immediately (200 to 250 m) south of the Hannukainen pit and there is a potential for a direct impact on base flow and indirect impact on riverine ecology. Numerical modelling studies show there is no major threat to base flow of the Äkäsjoki, Valkeajoki and Kuerjoki. Impacts indicate dewatering will result in no greater than a 2% decrease in monthly flows in Äkäsjoki in an average year, with up to a 5% decrease in a dry year with negligible changes in water levels and surface flow area. Concerning the Valkeajoki (about 2 km west of the Hannukainen pit) and Kuerjoki (about 2.5 km east of the Kuervitikko pit) predicted dewater impacts are negligible (0.01 m change in water levels). Impacts will be temporary as flows will eventually be restored with cessation of de-watering at closure. Full natural recovery of groundwater levels is predicted to take about 70 years for Hannukainen and closer to 200 years at Kuervitikko. Northland will monitor river flow. If results determine the aforementioned impacts are greater than predicted, additional mitigation and costs may be required to maintain the flow regime in these water courses of high conservation value.
- Effluent from clarification ponds at Rautuvaara will be discharged into the Muonionjoki, a Natura 2000 river. Northland's predicated water quality objectives will be exceeded at the point of discharge (22 metals and salts) which, according to Northland, may be permissible under Finnish law as long as the objectives are met outside of a certain mixing zone, which authorities have not yet defined. Several parameters will be exceeded after dilution up to 2 km downstream. Additional dilution further than 2 km downstream will return river quality to baseline conditions. If discharged water does not meet water quality objectives specified by authorities within a legislated mixing zone, then Northland will have to consider alternative mitigation such as adding lime to high sulfur tailings to neutralise seepage, increasing storage capacity and optimising water balance to limit discharge rates during critical periods of low flows, costs for which have not been not been allowed for in the financial model.
- Geochemical tests conducted by SRK indicate waste rock has net acid generating potential due to the general lack of carbonate minerals and available buffering capacity. Some geochemical tests (humidity cell test work) are ongoing. During operation, there is a risk groundwater beneath the WRMF at Hannukainen will be contaminated and impact on base flow. Impacts have been numerically modelled and pre-mitigation results indicate significant impacts on aquifer water quality in the immediate vicinity of the Hannukainen pit. Relatively minor and seasonal exceedances of some metals are predicted for the Äkäsjoki during operation as groundwater flow is mainly towards this river. No impacts are predicted on the quality of the Valkeajoki and the Kuerjoki. Northland plans to separate the high sulphur material, locating and designing the dump in such a way as to capture acid runoff in the Hannukainen pit. Alternative mitigation includes neutralising waste rock with lime and covering it with overburden; costs for these have not been allowed for in the financial model.

- On-going infiltration of water through WRMF and decanting of the pits after closure may impact on the Äkäsjoki via groundwater base flow. Studies predict there will be an initial build-up of acid in the pit lakes due to seepage and runoff from the pit walls. As a result, the un-mitigated impact evaluation indicates water quality objectives may be exceeded in Äkäsjoki after closure. A large magnitude impact is also observed for the Kuerjoki in this regard. Northland, proposes covering the waste rock to reduce water and oxygen infiltration ingress. Other mitigation which has not been costed in the financial model includes diversion of groundwater seepage from pits through wetlands (facilitating passive treatment) or installation of a permeable reactive barrier for in-situ passive treatment. It has also been suggested Northland artificially flood the pits (sulphide weathering will cease in anaerobic conditions); preliminary calculations indicate both pits can be flooded within 20 years. Mitigation will be studied further during the life of mine and alternative measures implemented if required.
- The efficacy of Northland's mitigation in the two bullets above will be determined from results of the groundwater monitoring programme. There is risk exceedance of metals will be higher than predicted during operation and after closure in surface water courses and more expensive mitigation may be required. The closure plan may need to be revised and closure costs (Section 20) may be significantly higher than predicted.
- The historical Kuervaara pit lake waters are acidic (pH as low as 3.14 at surface) with high metal concentrations indicative of weathering of sulphide minerals, or ARDML. This water will be mixed with pH neutral (6-7) from Laurinoja pit and discharged to a clarification pond at Hannukainen prior to commencement of mining. Work is ongoing to evaluate the need for water treatment to minimise impacts. Mixing may be sufficient to comply with expected permit conditions (i.e. additional treatment not necessary). If this is not the case, alternative mitigation may be required at higher cost.
- There is a risk of uncontrolled discharge to the Niesajoki due to overtopping of the clarification ponds at Rautuvaara during operation and after closure. Northland will consider appropriate design and freeboards, as well as additional pumping capacity to allow temporary storage in pits in the case of high water levels in ponds. With the commitments made by Northland, this impact is not expected to pose a significant risk to the Project.
- Approximately 4% of the total Project area comprises wetland. Of this wetland area, 14% will be impacted by mining (either directly in the footprint of Project infrastructure or by lowering of the groundwater table). The wetlands are important as breeding areas and migratory stops for bids.

19.5.2 Tailings management

There will be a lined high sulfur section and an unlined low sulfur section at the TMF at Rautuvaara. Seepage from the high sulfur section will be collected and treated at a treatment plant before discharge into the clarification ponds where it will mix with process water from the low sulfur section. Water quality in the ponds is predicted to worsen during operation with concentrations of some metals up to 50 times higher than the baseline. Potential impacts associated with this contamination on the following receptors have been numerically modelled: 1) aquifers beneath the TMF and ponds; 2) water courses receiving the groundwater as base flow; and 3) at the headwaters of the Niesajoki (Natura 2000 river) after closure. Water quality in Niesajoki is predicted to improve during operation compared to baseline because the present day discharge from the historical TMF's south pond will be stopped. However, without mitigation, water quality objectives are predicted to be exceeded in the Niesajoki after closure. Northland has committed to monitoring surface and groundwater quality in the vicinity of the TMF during operation and after closure. Northland has proposed additional mitigation including adding lime to the high sulfur tailings and lining the low sulfur section and ponds with peat (natural adsorption properties) to reduce seepage; costs for this mitigation have not been included in the financial model. Mitigation strategies have not been finalised and more costly alternatives are possible, including alternative closure options (maintaining the pipeline to the Mounionjoki and ongoing passive treatment via wetlands).

19.5.3 Nature conservation and biodiversity

Given there are predicted changes in the water quality of Muonionjoki system including Äkäsjoki and Niesajoki Natura 2000 rivers, there is a risk of impacts on the biodiversity of riverine habitats. These habitats are breeding areas for the Arctic Salmon and Sea Trout. Northland has proposed mitigation measures (refer to water management), which theoretically reduces the likelihood of impacts. However, given the high conservation value of water courses near the Project, authorities may impose stricter conditions on water quality than currently designed for, which Northland will have to adhere to. More expensive mitigation may be required than that envisioned to mitigate impacts on protected species. Northland may have to revise its closure plans and costs to meet water quality objectives for post closure protection of these waters.

There will be unavoidable loss of individual threatened plants (for example, the Lapland Marsh-orchid) within the area of direct impact at Rautuvaara. Northland will seek the requisite derogation permit and may have to relocate the plants or compensate; such as support another protected area where the orchids occur.

19.5.4 Changes to the socio-economic setting

The Project will bring economic benefits to the area, including direct and indirect employment opportunities, taxes and revenue for the public sector (ideally resulting in improved services), increased availability of goods and services, and help alleviate the reversal of the demographic decline and trend for young people to move away in search of employment. These benefits will be felt mainly by the Kolari and Muonio municipalities, and local people and businesses. However, these benefits are partially offset by a number of negative impacts, which will be felt predominantly by owners of properties in the Hannukainen buffer zone who will be displaced. It is also predicted there will be additional pressure on education and day care services at Hannukainen villages; impacts on the social cohesion of this community are predicted to be high (unmitigated).

Stakeholder consultation showed that communities' attachment to the land as a place of natural beauty and source of recreation is strong. Communities are aware of and protective of the importance of conserving the area for Arctic Salmon and Sea Trout. Owners of recreational cottages were concerned about disturbance (noise, dust, traffic). There was particular concern about impacts on tourism due to visual changes in landscape and deterioration of the image of the area as a tourist destination and liabilities following closure. Studies predict the impact on the tourism is expected to be small as the industry is robust and forecasts predict continued growth. Northland has recognized these negative impacts need to be managed sensitively to ensure it receives its 'social license to operate' and it is proposing a number of measures to minimize the negative impacts and optimize the positive impacts, notably including:

- Northland will purchase properties within buffer zone (Section 19.4);
- communication, grievance and land acquisition plans are already in place and being utilized;
- dedicated environmental and social teams have been appointed to manage identified impacts and communicate with local communities; and
- identifying responsible and appropriate corporate social investment opportunities to bolster local social activities and networking opportunities.

19.5.5 Reindeer husbandry

Impacts on reindeer husbandry have been assessed. The Project is located in a specially designated reindeer herding area belonging to the Finnish Muonio Cooperative. The Project will affect the central and southern zones of the cooperative's pastures with 4,000 and 500 reindeer respectively. Northland is impacting on a relatively small area. Predicted impacts include: 1) loss of 1% of the cooperative's pasture; 2) minor disturbance from accessing roundup structures, for example, Lamumaa corral; 3) grazing land avoidance behaviour and enforcement of five summer and winter migration route changes between pastures; 4) herders and the cooperative realising increased costs for winter feeding and fuels; 5) increased reindeer loss from traffic accidents; 6) barrier effect of the transport corridor between Hannukainen and Rautuvaara; and 7) herders working for the mine, i.e. cultural changes. Northland will consult reindeer herders and the corporative to agree compensation and facilitate monitoring reindeer; the value allowed for in the financial model accounts for costs of moving reindeer fences and other infrastructure. Annual (OPEX) compensation costs have also been included in the financial model for the additional work reindeer herders may have to undertake once the mine is operational. The actual payments will be determined through collaboration with the reindeer herders and monitoring. Northland will also construct cross over points along the conveyor between Hannukainen and Rautuvaara (CAPEX costs provided for in the financial model). There is potential for additional compensation, but it is not anticipated this will have a long term material impact on the Project.

19.5.6 Transport and pipelines

The Project will utilise the existing road network during construction and operation. Ore will be conveyed overland from Hannukainen to Rautuvaara. Concentrate will be transported by rail to an existing port (Northlands EIA assessment is based on Kokkola Port as this port is the furthest viable option, although there are alternatives). Part of the existing rail network will be upgraded by the Finnish Transport Authority who will be responsible for environmental permitting of this activity. The port authorities will also be responsible for undertaking any necessary upgrades, development plans and permit applications to accommodate Northland's concentrate.

With the implementation of the mitigation measures outlined in the HFS chapter, overall rail transport impacts are predicted to be minor. However, Northland will have to rely on owners of the port and railway for the upgrades and permits – these will be third party contracts with likely penalty clauses in Northland's favour if the parties do not deliver on their obligations. In the case of the port, other authorisations including an environmental permit may be required.

19.5.7 Cumulative impacts

The HFS chapter has considered cumulative impacts of the following industrial/anthromorphological influences:

- Northland's Kaunisvaara project in Sweden is approximately 30 km west of the Project;
- expansion development at the Yllästunturi Tourist Centre; and
- Nordkalk Limestone Quarry, Ruonaoja, Kolari.

The discharge of effluent into the Muonionjoki by the Kaunisvaara and the Project is discussed here. The predicted level of dilution within Muonio will result in potential cumulative impacts on water being of minor significance. Northland is aware both Projects will be required to comply with permissible emission standards under environmental permits and potential material impacts of this are discussed under water management.

19.6 Closure requirements

Legal requirements for closure are set out Finland's mining law. This requires Northland to have a closure fund and closure plan, which should be kept up to date. Some requirements are in Finland's Best Practice Guideline for mining.

A closure plan has been prepared by Ramboll for Hannukainen and Rautuvaara. The plan indicates: the general closure objectives and guidelines relevant to the Project; the natural setting and pre-mining conditions; the alternative closure options for each type of facility; and the preferred closure strategy for each facility. Risk and uncertainties have also been identified, along with a schedule for closure implementation.

SRK considers the closure plan is appropriate for HFS. Section 19.5 identifies a number of risks to surface and groundwater after closure. Northland has proposed a number of mitigation measures including:

- continue treating runoff from Rautuvaara until water quality meets agreed standards;
- continue piping excess water from the south clarification pond to Muonionjoki until water quality meets agreed standards;

- assessment of the feasibility of using lime or peat to neutralise or attenuate seepage from TMF; and
- attenuation of tailings seepage through the establishment of a wetland.

The main closure uncertainty relates to closure objectives and water quality standards, which must be agreed with the authorities. Authorities will likely emphasise protection of water courses of critical ecological importance and impose strict standards in this regard. As a result, the closure plan will need to be reviewed as the Project is implemented and additional monitoring information becomes available and additional mitigation may be required to that accounted for in the financial model.

19.7 Mine Closure Costs

Closure costs are given in Figure 19-3 and have been estimated for the mine components in Section 20. Provision has been made for water treatment 5 years after closure and monitoring for 25 years after closure. The HFS chapter outlines assumptions and the basis on which the costs were determined as well as risks associated with the estimation.

A security bond is used to cover remediation cost in case of business failure or other unforeseen events. It has to be set according to Finnish environmental and mining legislation. The draft HFS ESI chapter outlines the broad mechanisms of the bond, which has yet to be posted by Northland.

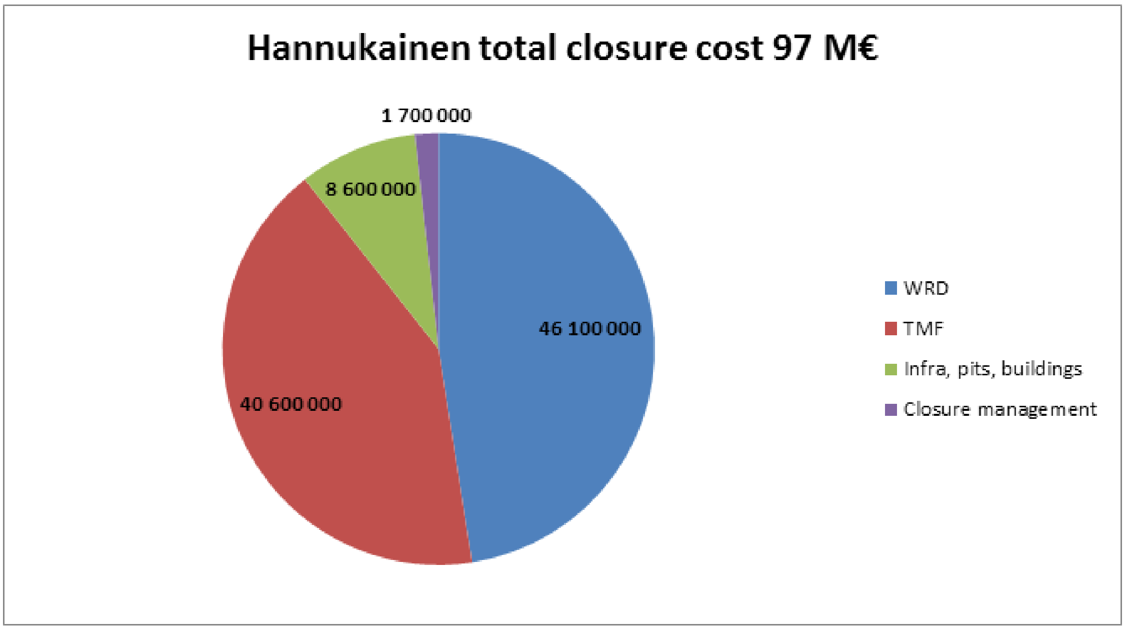


Figure 19-3: Summary of estimated closure costs for the Project (after draft HFS ESI chapter)

SRK notes that the financial model assumes 91.6M€ in closure related costs, which apparently contradicts the total closure cost sum as presented above by 5.4M€. It has been suggested that this difference may relate to rehabilitation costs for the high sulphur tailings. SRK has assumed that these costs are included elsewhere in Northland's operating costs, although this remains unclear at this stage and should be investigated further, with any necessary adjustments made to the financial model.

20 CAPITAL AND OPERATING COSTS

20.1 Operating Costs Summary

The operating costs estimated as part of the HFS have been incorporated into SRK's financial model with no material adjustments. SRK has reviewed these costs and considers them to be reasonable for the Project. Figure 20-1 illustrates an overall breakdown of the operating expenditure over the life of mine, split between the major cost centers and excluding contingency. These are also summarized in Table 20-1. An overall contingency of 5% has been assumed for operating costs.

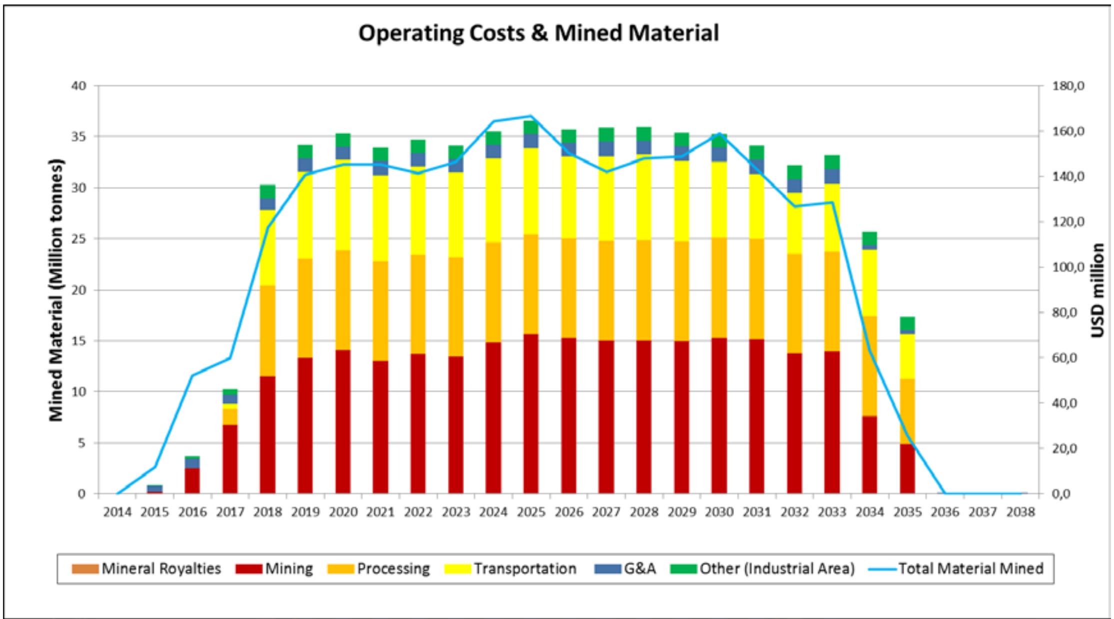


Figure 20-1: Summary of operating costs over the LoM, by major cost centre.

Table 20-1: Summary of unit operating costs

	Unit Operating Costs per tonne total material (USD / tonne)	Unit Operating Costs per tonne milled (USD / tonne)	Unit Operating Costs in US cents per dmtu for concentrate sold
Mining	2.00	9.80	49.3
Processing	1.39	6.79	34.2
G&A	0.20	0.99	5.0
Other (Industrial Area)	0.20	0.98	4.9
Transportation	1.10	5.37	27.0
Mineral Royalties	0.01	0.07	0.4
Total Operating Expenditure (pre-contingency)	4.91	24.02	120.8
Total Operating Expenditure (incl 5% contingency)			126.9
TCRC's (Cu/Au Concentrate)			3.6
By-product credits (Cu/Au Concentrate)			-58.7
C1 Cash Costs*			71.8

** C1 costs include mining, processing, site admin, transportation, smelting and refining, net of byproduct credits*

Operating costs were converted to USD for the purposes of financial modelling at exchange rates presented below. The reader is referred to Section 21.8 (Sensitivity Analysis) where the sensitivity of the Project valuation to exchange rate (between EUR and USD) is assessed.

20.2 Capital Cost Summary

The capital costs estimated as part of the HFS have been incorporated in to SRK's financial model with no material adjustments. These costs total USD736M, pre-contingency. A contingency of 10% is applied. Overall, SRK considers these costs to be reasonable for the Project.

Figure 20-2 below gives an overview of the envisaged capital expenditure over the life of mine, excluding contingency. Figure 20-3, Figure 20-4 and Table 20-2 present a breakdown of initial and sustaining capital between the major cost centres.

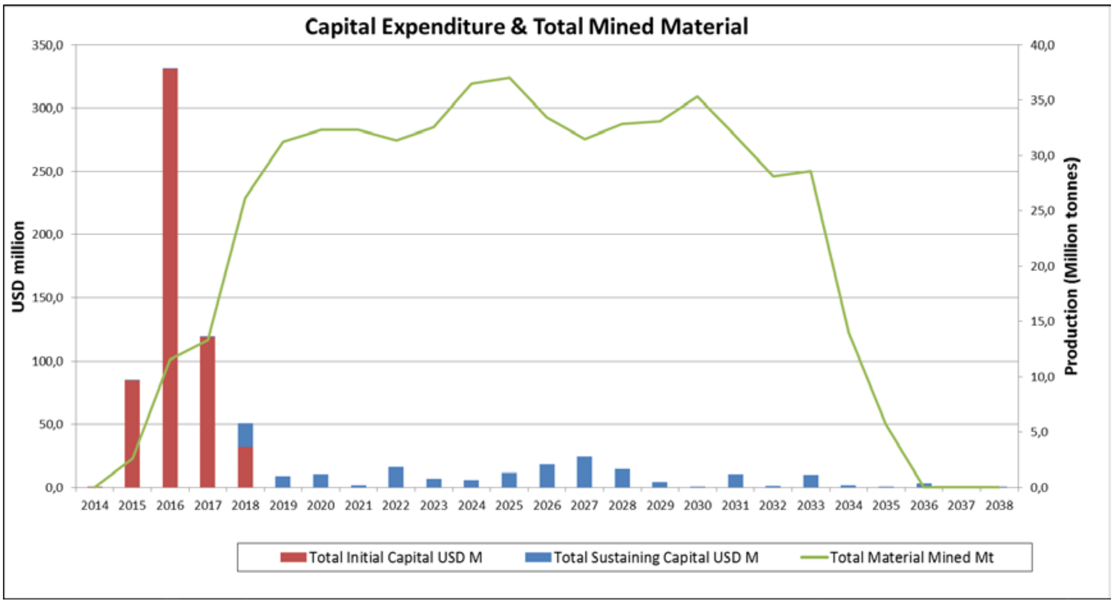


Figure 20-2: Forecast capital expenditure against total material movement

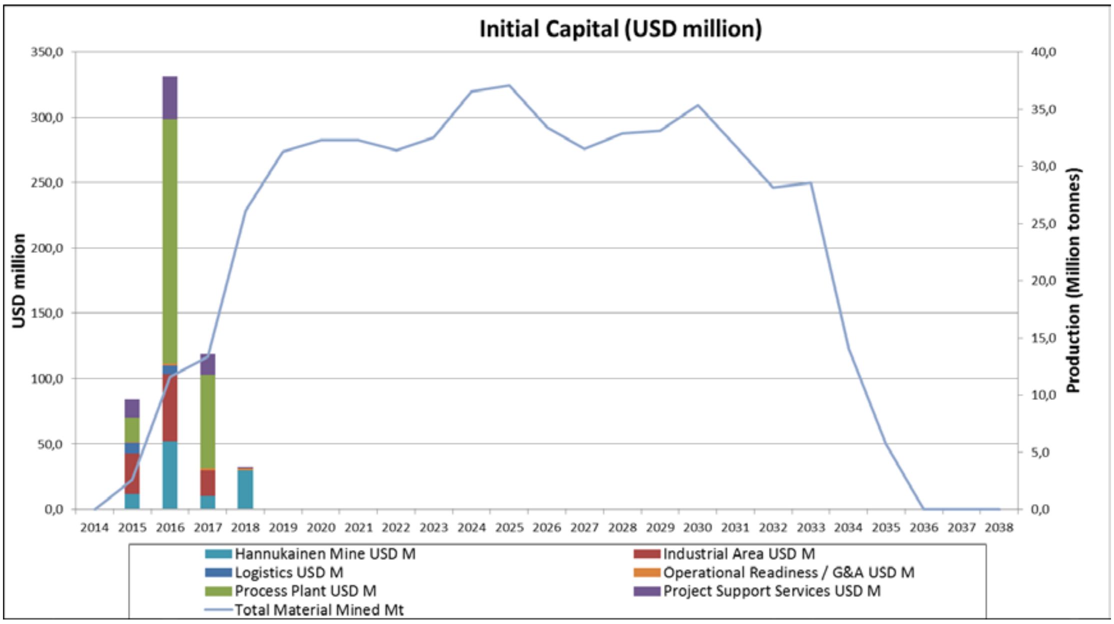


Figure 20-3: Initial capital expenditure by major cost centre and total material movement

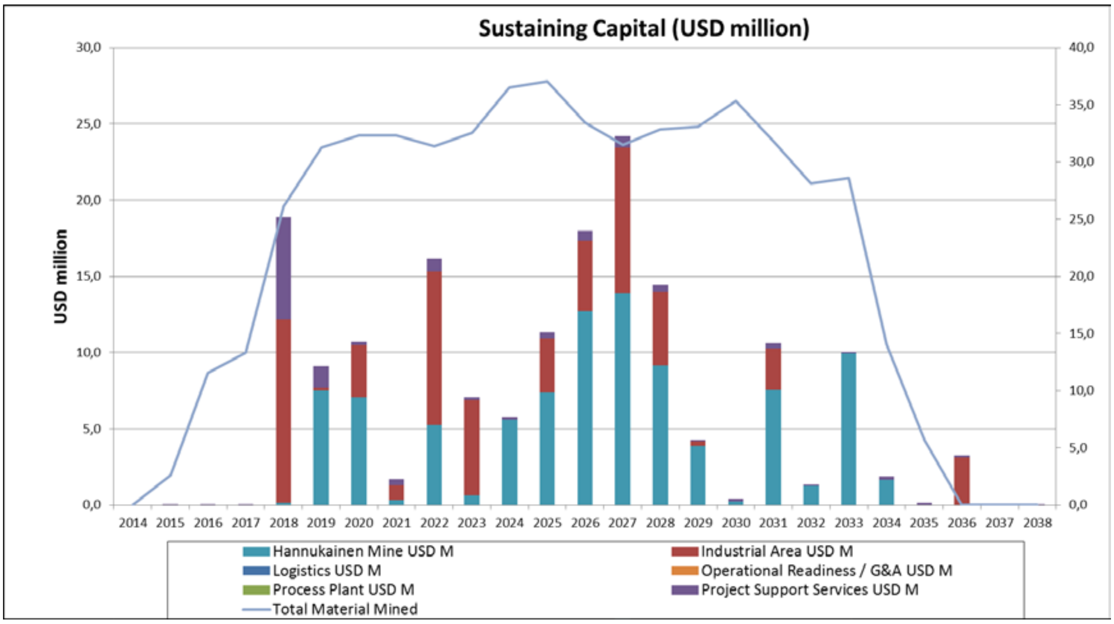


Figure 20-4: Sustaininig capital expenditure by major cost centre and total material movement

Table 20-2: Breakdown of initial and sustaining capital expenditure by major cost centre

Initial Capital	USD million	Sustaining Capital	USD million
Hannukainen Mine	104	Hannukainen Mine	94
Industrial Area	102	Industrial Area	62
Logistics	15	Logistics	0
Operational Readiness / G&A	6	Operational Readiness / G&A	0
Process Plant	277	Process Plant	0
Project Support Services	64	Project Support Services	14
Total Initial Capital	567	Total Sustaining Capital	169
Total Capital Expenditure (pre-contingency)	736		
Capital Contingency (10%)	74		
Total Capital Expenditure	**810**		

Capital costs were converted to USD for the purposes of financial modelling at exchange rates presented below. The reader is referred to Section 21.8 (Sensitivity Analysis) where the sensitivity of the Project valuation to exchange rate (between EUR and USD) is assessed.

21 ECONOMIC ANALYSIS

21.1 Introduction

As part of this technical report, SRK's role has been to construct a financial model in order to derive a post-tax, pre-finance Net Present Value ("NPV") for the Project and independently verify (or otherwise) the valuation derived by Northland for the HFS. SRK has constructed its independent financial model using the cost data from Northland's cost templates, as well as extracting the underlying technical assumptions, macro-economic assumptions and life of mine plan from the HFS financial model itself.

The figures presented and discussed in this section of the report correspond to the inputs and outputs (resulting forecast cashflows and subsequent valuation) from SRK's financial model. Whilst there are slight differences in forecast cashflows between the two models, these are considered to be non-material.

Northland's financial model has been constructed using a process flow, which is summarised in Figure 21-1.

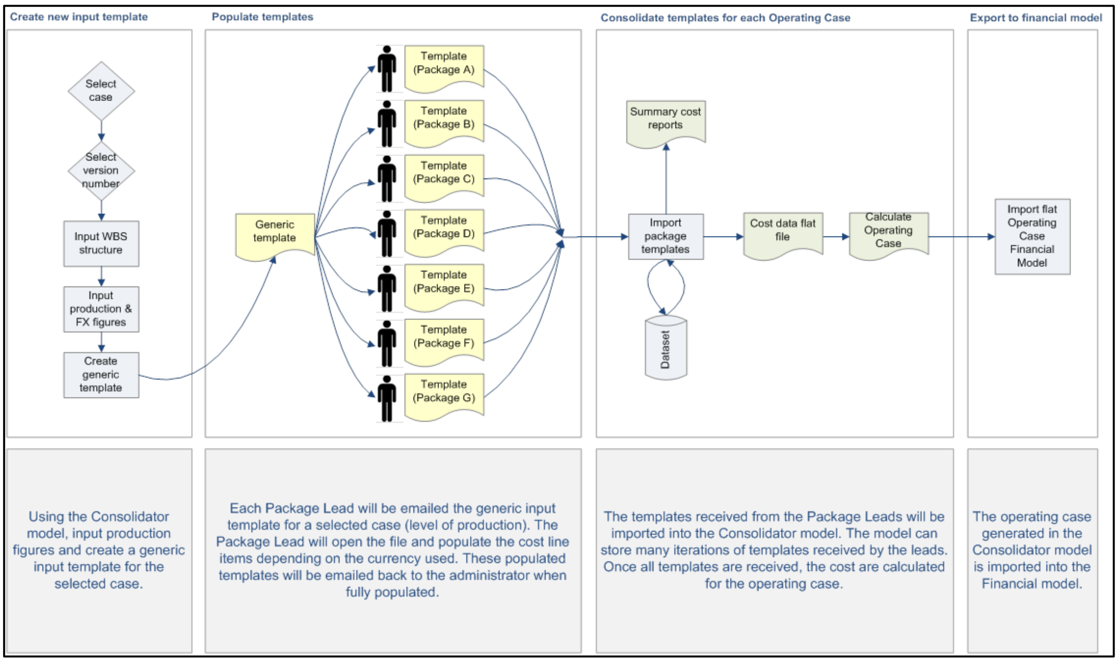


Figure 21-1: Process flow for the Northland

A generic MS Excel cost template was developed by Northland for the purposes of standardising the format of cost inputs and in order to expedite the import of this data to their financial model. These templates were then populated with hardcoded costs by the respective work package managers for their respective areas of technical responsibility, with separate templates for each currency depending on the denomination of the cost item (EUR, SEK or USD). Northland used a series of macros to automate the import of this cost data into the financial model.

21.2 Valuation Process

21.2.1 General Assumptions

The model is based on production from two open pit mines (Hannukainen and Kuervitikko), feeding a single process stream with a combined maximum annual throughput of 6.5 Mtpa and housed within a single processing plant. The plant produces a magnetite concentrate of 70% Fe, and a copper-gold concentrate of 25% Cu and 7.1 g/t Au. These concentrates are planned to be transported by rail from site to the port of Kokkola, Finland.

The valuation currency is USD, with any EUR or SEK derived costs being converted at the following rates, which SRK note are assumed to be consistent over the life of mine:

- SEK:USD exchange rate of 6.9:1; and
- EUR:USD exchange rate of 0.7813:1

SRK notes that for operating expenditures, some 87% is denominated in EUR and 2% in SEK, with the remaining 11% denominated in USD. For capital expenditures some 94% is denominated in EUR and 1% in SEK, with the remaining 5% denominated in USD.

SRK also notes that the spot closing exchange rates as at 19 December 2013 are: SEK:USD exchange rate of 6.59:1 and EUR:USD exchange rate of 0.73:1. Single parameter sensitivities of the Project valuation to variations in the EUR:USD exchange rates is presented below in Section 21.8.

NPV as presented in this Technical Report are on a post-tax and pre-finance basis and assume a base case discount factor of 8%. All figures are presented in real terms.

Working capital assumptions are as follows:

- Debtor days = 30
- Creditor days = 30
- Inventory days = 8

21.2.2 Taxes and Mineral Royalty

Corporation Tax

Straight-line depreciation has been applied to the sum total of capital expenditures over the LOM to derive profits before tax. A useful economic life of 5 years is assumed along with zero salvage value. A corporate income tax rate of 24.5% is applied to pre-tax profits to arrive at a post-tax cashflow.

SRK has taken Northland's depreciation and corporate tax workings at face value and has not sought qualified independent advice to confirm that these assumptions are reasonable in the context of (a) Finnish tax law and (b) standard accounting practices in Finland.

Social Tax

Labour rates provided to SRK by the Client and incorporated into the financial model are inclusive of on-costs. These on-costs include employers social contributions which SRK understand may range between 26.5% and 34.2%.

A summary of tax assumptions and mineral royalties are presented in Table 21-1 below:

Table 21-1: Summary of tax and mineral royalty assumptions

Type	Value
Mineral Royalty	0.15%
Corporation Tax	24.50%
Depreciation	5 years
Employers Social Contributions	Between 26.5% and 34.2%

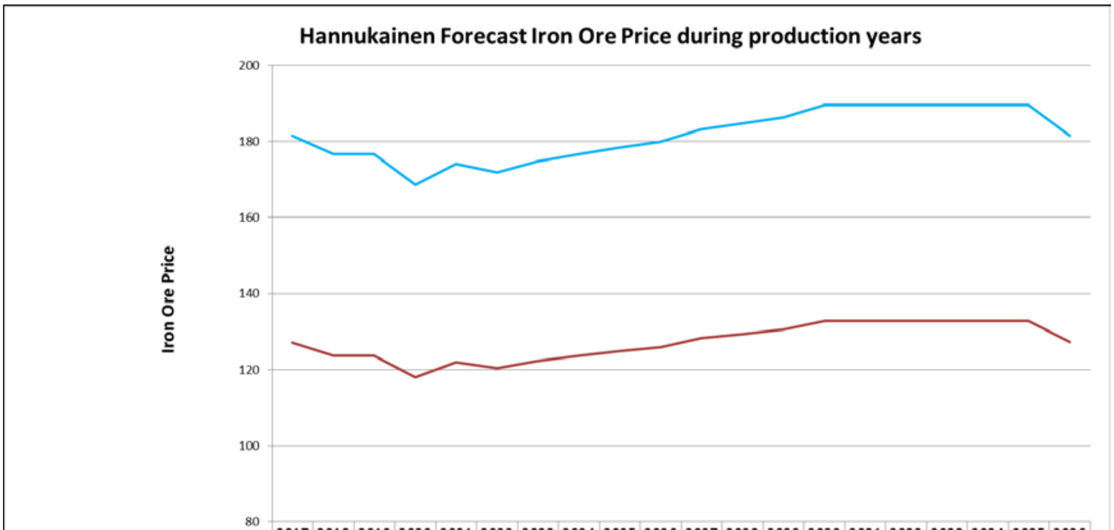
21.3 Commodity Price Assumptions

21.3.1 Forecast iron ore price

Commodity price forecast data was provided to SRK by Northland. Northland developed this forecast internally, based primarily on an independent third party report by RMG, dated October 2013. RMG developed the base case iron ore price forecast model, as presented in Table 18-2.

The base case price forecast was subsequently adjusted by Northland in consideration of a Value-In-Use premium of USD3 dmt.

Figure 21-2 below presents the iron ore price forecast for the LoM as incorporated in SRK's financial model.



	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Commodity Price Fe US c / dmtu	181,5	176,7	176,7	168,6	174,2	171,9	174,7	176,7	178,3	179,9	183,2	184,8	186,4	189,6	189,6	189,6	189,6	189,6	189,6	181,5
Commodity Price Fe USD / t	127,1	123,7	123,7	118,0	121,9	120,3	122,3	123,7	124,8	126,0	128,2	129,3	130,5	132,7	132,7	132,7	132,7	132,7	132,7	127,3

Figure 21-2: Northland forecast iron ore prices for Hannukainen product during production years

21.3.2 Consensus Market Forecast ("CMF") for Iron Ore

Commodity prices are influenced, inter alia, by commodity demand-supply balances for iron ore and steel production and the cost of transportation all of which are influenced by global economic growth and industrial production. The denominator in the determination of the unit price is based on a dry metric tonne unit ("dmtu").

The CMF as generated by SRK, is the median of brokers' equity research forecasts and are reported in real terms, updated quaterly. The CMF database generally indicates price forecasts for the next three calendar years and a long term price ("LTP") corresponding to all periods beyond a five year period.

It is also important to note that the CMF exists within a range, which can be wide, accordingly the CMF is focused on the median and not the average and is importantly stated in 'real' money terms. When comparing with other forecasts it is always important to ensure that comparisons are on a like for like basis.

SRK are not specialists in metal price forecasting and rely on the CMF when considering expected trends in metal prices. The majority of CMF prices considered for comparative purposes, show an expected decrease in price until 2019.

21.3.3 Forecast copper and gold prices

Northland has assumed the following forecast metal prices for copper and gold, which for the purposes of the model, are assumed to be consistent during the entire life of mine:

- Copper: 6,950 USD / tonne; and
- Gold: 1,350 USD / troy ounce.

SRK notes that Northlands price assumptions fall within the range of available CMF data for Q4 2013 and that these lie at the upper end of this range.

21.3.4 Commodity Price Summary

For the avoidance of doubt, the following commodity prices have been utilised throughout the HFS.

Table 21-2: HFS Commidity price summary

Commodity	Unit	Resource Reporting	Reserve Reporting	Pit Selection	Economic Analysis
Fe	USD/dmtu	1.5	1.4	1.25	Price profile used – see Figure 21-2
Cu	USD/t	7,385	6,305	5,620	6,950
Au	USD/oz	1,375	1,250	1,116	1,350

21.4 Mining Physical Assumptions

A summary of the combined mass movement of material is presented in Table 21-3 below. Figure 21-3 illustrates combined ore and waste tonnages mined and Fe% grade over the life of mine. Figure 21-4 illustartes the LOM Cu and Au grades.

Table 21-3: Hannukainen & Kuervitikko combined RoM Ore and Waste movement

Description	Units	Life Of Mine Totals
Ore tonnes	(Mt)	114.8
Ore grade	(% Fe)	30.5
Ore grade	(% Cu)	0.19
Ore grade	(Au g/t)	0.11
Mass waste	(Mt)	446.8
Total Material Mined	(Mt)	561.6
Strip ratio	(W:O)	3.9
Overburden Volume	(Mt)	74.8

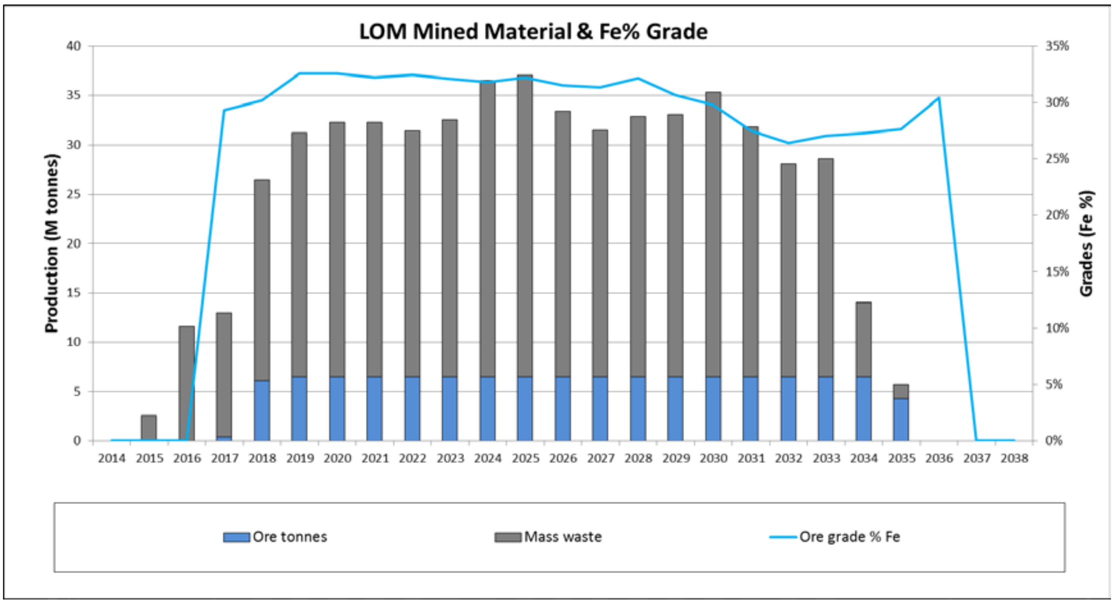


Figure 21-3: Hannukainen & Kuervitikko combined RoM Ore and Waste movement with Fe% grade.

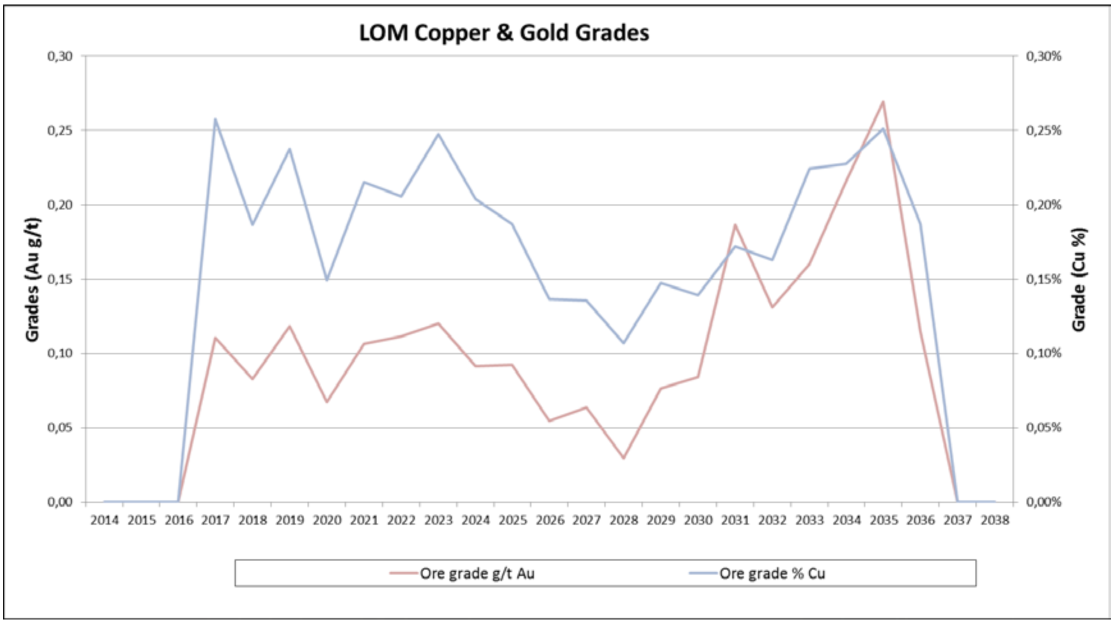


Figure 21-4: Hannukainen & Kuervitikko combined LOM copper and gold grade

21.5 Process Physical Assumptions

Table 21-4 summarises combined recoveries, concentrate grades and concentrate tonnages over the LOM. Figure 21-5 and Figure 21-6 below illustrate concentrate production and plant performance for magnetite and copper-gold concentrate products respectively.

Table 21-4: LOM Process Physical Assumptions

Description	Units	Hannukainen (combined)
Magnetite Concentrate		
Contained recoverable Fe	(Mt)	35.0
Iron recovery	(%)	65%
Mass yield	(%)	29%
Grade of final magnetite concentrate	(% Fe)	70%
Concentrate tonnage (dry)	(Mt)	32.8
Copper-gold Concentrate		
Contained Cu	(Mt)	0.2
Contained Au	(Million troy ounce)	0.4
Copper recovery	(%)	84%
Gold recovery	(%)	26%
Mass yield	(%)	0.6%
Grade of final Cu/Au concentrate	(% Cu)	25
Grade of final Cu/Au concentrate	(g/t Au)	7.1
Concentrate tonnage (dry)	(Mt)	0.72

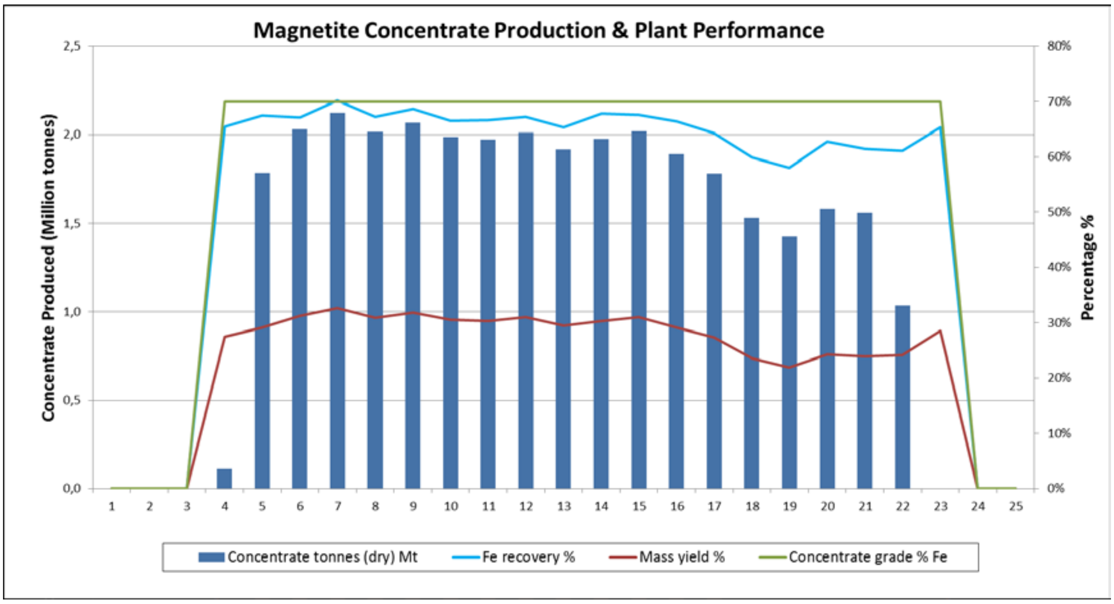


Figure 21-5: Magnetite concentrate production and plant performance (Fe recovery % and mass yield %)

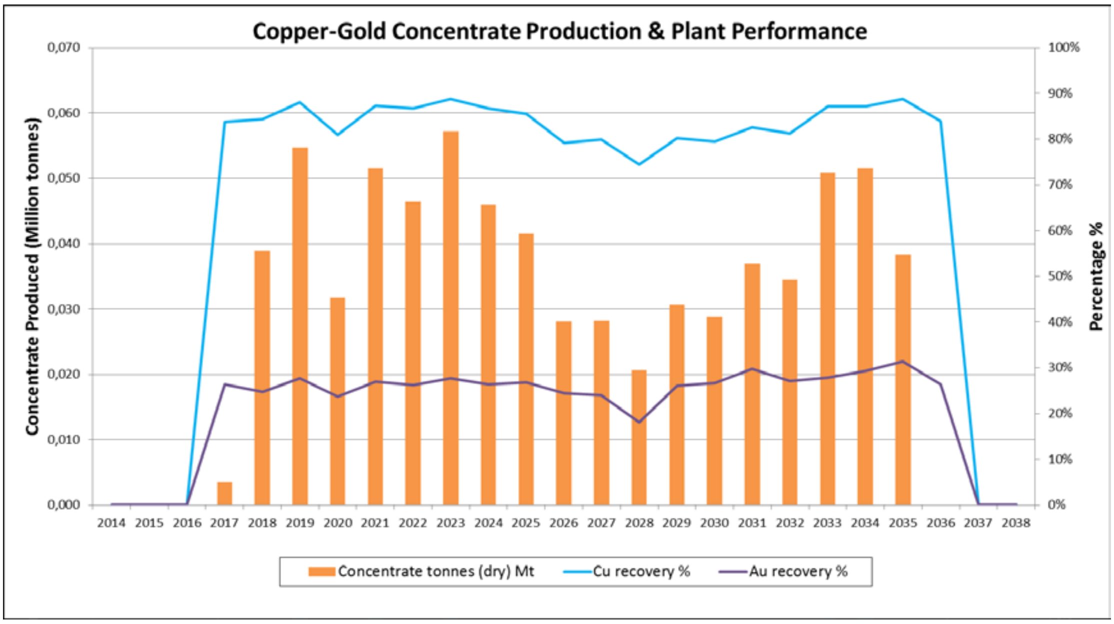


Figure 21-6: Copper-gold concentrate production and plant performance (Cu recovery % and Au recovery %)

21.5.1 Handling losses

Table 21-5 below summarises assumed handling losses.

Table 21-5: Assumed handling losses

Description	Unit	Assumed Handling Losses over LOM
Magnetite Concentrate		
Processing	(%)	0.02%
Product to railcars	(%)	0.23%
Product to ship	(%)	0.23%
Ship unloading	(%)	0.23%
Total losses	(%)	0.71%
Total losses	(Mt)	0.23
Concentrate available for sale	(Mt)	32.6
Copper-gold Concentrate		
Total losses	(%)	1.0
Total losses	(t)	7 202
Concentrate available for sale	(Mt)	0.71

21.6 Revenue

Table 21-6 below summarises gross revenues, deductions, treatment charges and refining costs (“TCRC’s”) and resulting net revenues by concentrate product.

Table 21-6: Gross Revenues, deductions TCRC's and resulting net revenues by concentrate product.

Description	Unit	Value
Magnetite Concentrate		
Concentrate available for sale	(Mt)	32.6
Gross Revenue	(USD million)	4,120
Copper-Gold Concentrate		
Concentrate available for sale	(Mt)	0.71
Unit deduction copper (4%)	(t)	7,130
Cu metal recovery from concentrate	(%)	96.65%
Payable copper	(t)	165,380
Payable copper	(lb)	364,601,000
Cu gross revenue	(USD million)	1,149
Unit deduction gold (1 g/t)	(kilogram)	713
Payable gold	(kilogram)	4,349
Payable gold	(troy ounce)	139,827
Gross revenue gold	(USD million)	189
Gross revenue copper-gold concentrate	(USD million)	1,338
Total TCRC's	(USD million)	82
Net revenue copper-gold concentrate	(USD million)	1,257
Total net revenue	**(USD million)**	**5,377**

Figure 21-7 below presents annual contribution to gross revenue over the LOM, by concentrate product.

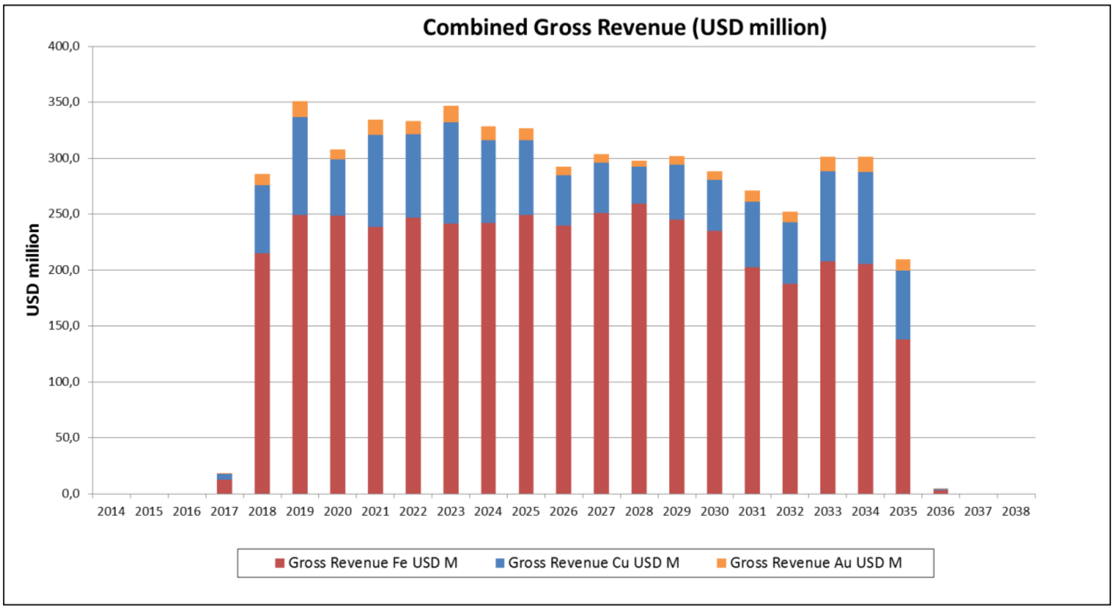


Figure 21-7: Annual contribution to gross revenue over the LOM, by concentrate product

21.7 Cash Flow Projections

A valuation of the Project has been derived based on the application of Discounted Cash Flow ("DCF") techniques to the post-tax, pre-finance cash flow developed for the HFS.

The resulting post-tax, pre-finance real terms (1 December 2013) NPV derived by SRK is USD 248 million, assuming an 8% discount rate. SRK notes that Northland report a NPV of USD 251 million (post-tax, pre-finance at 8% discount) in the HFS. This difference is not considered to be material.

A summary of the results of the cash flow modelling and valuation are presented below in Table 21-7 and Figure 21-8. In addition, Table 21-8 presents summary annualised forecast cash flow for the Project.

Table 21-7: Summary results of cash flow modelling

Description	Units	Total (USDm)
Net Revenue	(USD million)	5,377
Total Operating Expenditure	(USD million)	-2,895
Total Capital Expenditure	(USD million)	-810
Other Expenses (Environmental Bond Payments)	(USD million)	-117
Net pre-tax, pre-finance cashflow	(USD million)	1,555
Corporation Tax (24.5%)	(USD million)	-379
Interest & Fees (Fees on Environmental Bond)	(USD million)	-22
Net post-tax, pre-finance cashflow	(USD million)	1,155
Payback period	(years)	9
NPV 8% (post-tax, pre-finance)*	(USD million)	248
IRR	(%)	14.0%

**Northland HFS estimate an NPV 8% (post-tax, pre-finance) of USD 251 million*

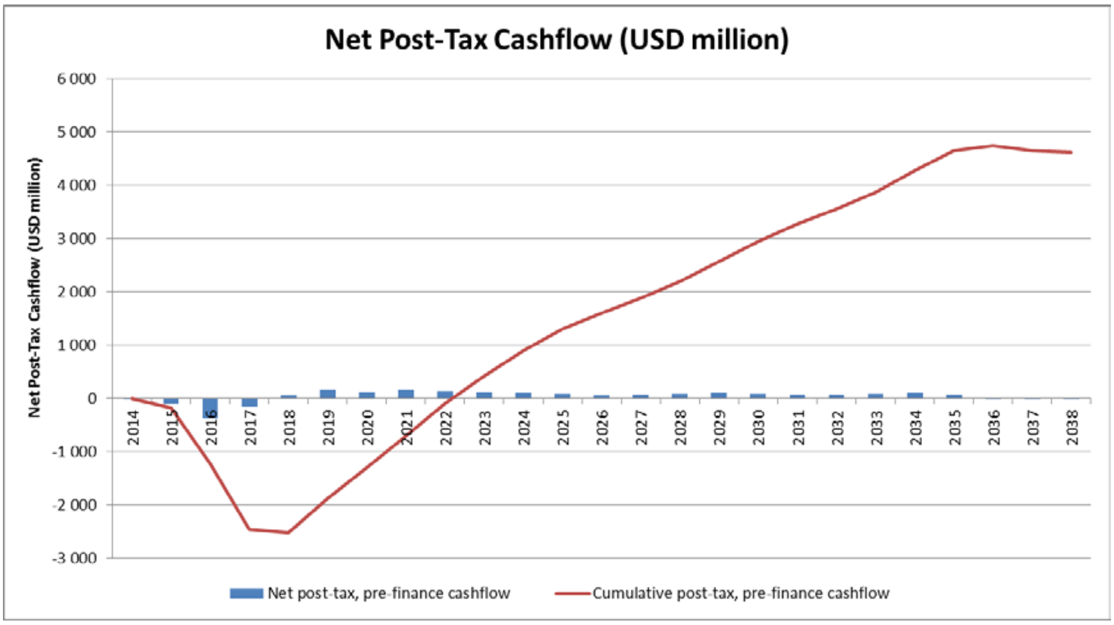


Figure 21-8: Annual net post-tax, pre-finance cashflow

Table 21-8: Summary annualised forecast cash flow

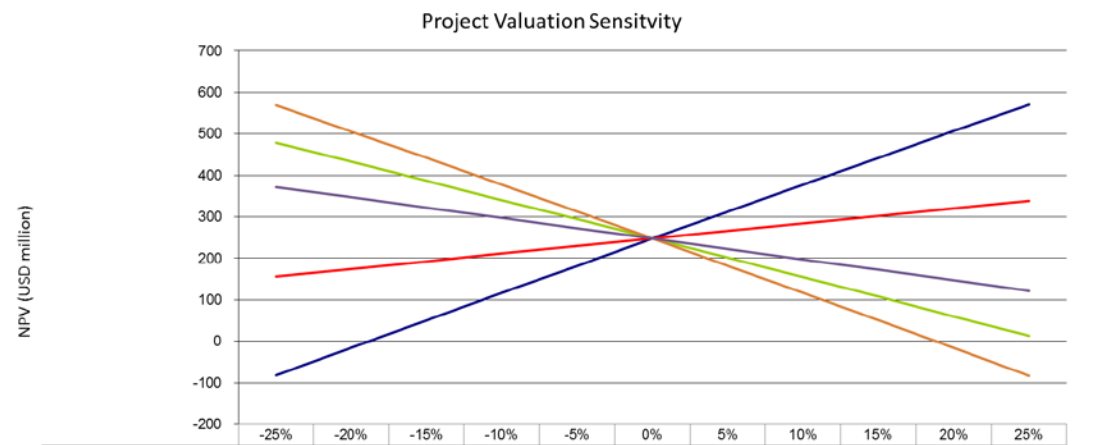
SE353_U4985 Hannukainen Cashflow			End of period	31-12-2014	31-12-2015	31-12-2016	31-12-2017	31-12-2018	31-12-2019	31-12-2020	31-12-2021	31-12-2022	31-12-2023	31-12-2024	31-12-2025	31-12-2026	31-12-2027	31-12-2028	31-12-2029	31-12-2030	31-12-2031	31-12-2032	31-12-2033	31-12-2034	31-12-2035	31-12-2036	31-12-2037	31-12-2038
Cashflow Annual			Year	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
	Assumption	Unit	Total																									
CASHFLOW																												
Mining																												
Ore tonnes		Mt	114,8	0,0	0,0	0,0	0,4	6,1	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	4,3	0,0	0,0	0,0
Ore grade		% Fe	30,50%	0,00%	0,00%	0,00%	29,35%	30,20%	32,58%	32,59%	32,23%	32,44%	32,11%	31,85%	32,24%	31,56%	31,35%	32,19%	30,65%	29,80%	27,55%	26,40%	27,09%	27,33%	27,67%	30,44%	0,00%	0,00%
Ore grade		% Cu	0,19%	0,00%	0,00%	0,00%	0,26%	0,19%	0,24%	0,15%	0,22%	0,21%	0,25%	0,20%	0,19%	0,14%	0,14%	0,11%	0,15%	0,14%	0,17%	0,16%	0,22%	0,23%	0,25%	0,19%	0,00%	0,00%
Ore grade		g/t Au	0,11	0,00	0,00	0,00	0,11	0,08	0,12	0,07	0,11	0,11	0,12	0,09	0,09	0,05	0,06	0,03	0,08	0,08	0,19	0,13	0,16	0,22	0,27	0,11	0,00	0,00
Mass waste		Mt	446,8	0,0	2,6	11,6	12,6	20,4	24,8	25,8	25,8	24,9	26,0	30,0	30,6	26,9	25,0	26,4	26,6	28,8	25,3	21,6	22,1	7,6	1,4	0,0	0,0	0,0
Total Material Mined		Mt	561,6	0,0	2,6	11,6	13,3	26,1	31,3	32,3	32,3	31,4	32,5	36,5	37,1	33,4	31,5	32,9	33,1	35,3	31,8	28,1	28,6	14,1	5,7	0,0	0,0	0,0
Strip ratio		W:O	3,9	0,0	0,0	0,0	31,0	3,3	3,8	4,0	4,0	3,8	4,0	4,6	4,7	4,1	3,8	4,1	4,1	4,4	3,9	3,3	3,4	1,2	0,3	0,0	0,0	0,0
Processing																												
Ore tonnes		Mt	114,8	0,0	0,0	0,0	0,4	6,1	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	6,5	4,3	0,0	0,0	0,0
Magnetite Concentrate																												
Fe recovery		%	65%	0%	0%	0%	65%	68%	67%	70%	67%	69%	67%	67%	67%	65%	68%	68%	66%	64%	60%	58%	63%	61%	61%	65%	0%	0%
Mass yield		%	29%	0%	0%	0%	27%	29%	31%	33%	31%	32%	31%	30%	31%	29%	30%	31%	29%	27%	24%	22%	24%	24%	24%	28%	0%	0%
Concentrate tonnes (dry)		Mt	32,8	0,0	0,0	0,0	0,1	1,8	2,0	2,1	2,0	2,1	2,0	2,0	2,0	1,9	2,0	2,0	1,9	1,8	1,5	1,4	1,6	1,6	1,0	0,0	0,0	0,0
Concentrate grade		% Fe	70%	0%	0%	0%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	0%	0%
Concentrate Available for sale (after losses & Inventory)		Mt	32,6	0,0	0,0	0,0	0,1	1,7	2,0	2,1	2,0	2,1	2,0	2,0	2,0	1,9	2,0	2,0	1,9	1,8	1,5	1,4	1,6	1,5	1,0	0,0	0,0	0,0
Copper-Gold Concentrate																												
Cu recovery		%	84%	0%	0%	0%	84%	84%	88%	81%	87%	87%	89%	87%	85%	79%	80%	74%	80%	80%	83%	81%	87%	87%	89%	84%	0%	0%
Au recovery		%	26%	0%	0%	0%	26%	25%	28%	24%	27%	26%	28%	26%	27%	24%	24%	18%	26%	27%	30%	27%	28%	29%	31%	26%	0%	0%
Mass yield		%	0,6%	0,0%	0,0%	0,0%	0,9%	0,6%	0,8%	0,5%	0,8%	0,7%	0,9%	0,7%	0,6%	0,4%	0,4%	0,3%	0,5%	0,4%	0,6%	0,5%	0,8%	0,8%	0,9%	0,6%	0,0%	0,0%
Concentrate tonnes (dry)		Mt	0,7	0,0	0,0	0,0	0,0	0,0	0,1	0,0	0,1	0,0	0,1	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,1	0,1	0,0	0,0	0,0	0,0
Concentrate grade		% Cu	25,0%	0,0%	0,0%	0,0%	25,0%	25,0%	25,0%	25,0%	25,0%	25,0%	25,0%	25,0%	25,0%	25,0%	25,0%	25,0%	25,0%	25,0%	25,0%	25,0%	25,0%	25,0%	25,0%	25,0%	0,0%	0,0%
Concentrate grade		g/t Au	7,1	0,0	0,0	0,0	7,1	7,1	7,1	7,1	7,1	7,1	7,1	7,1	7,1	7,1	7,1	7,1	7,1	7,1	7,1	7,1	7,1	7,1	7,1	7,1	7,1	7,1
Concentrate Available for sale (after losses & Inventory)		Mt	0,7	0,0	0,0	0,0	0,0	0,0	0,1	0,0	0,1	0,0	0,1	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,1	0,1	0,0	0,0	0,0	0,0
Gross Revenue																												
Magnetite Concentrate																												
Gross Revenue Fe		USD M	4 120,4	0,0	0,0	0,0	12,8	215,2	249,2	248,6	238,9	247,2	241,3	242,4	249,5	239,8	251,2	259,2	245,3	234,9	202,7	187,9	207,7	205,6	138,0	3,0	0,0	0,0
Copper-Gold Concentrate																												
Gross Revenue Au		USD M	188,8	0,0	0,0	0,0	0,8	10,0	14,4	8,3	13,5	12,2	14,9	12,1	10,9	7,4	7,4	5,5	8,0	7,6	9,6	9,1	13,2	13,5	10,1	0,2	0,0	0,0
Gross Revenue Cu/Au		USD M	1 338,2	0,0	0,0	0,0	5,9	70,6	102,1	59,0	95,4	86,4	105,8	86,0	77,4	52,7	52,4	38,7	56,7	53,6	68,2	64,2	93,9	95,8	71,7	1,5	0,0	0,0
Total Gross Revenue (Magnetite & Cu/Au Concentrate)		USD M	5 458,5	0,0	0,0	0,0	18,8	285,9	351,3	307,6	334,3	333,6	347,1	328,4	326,9	292,5	303,6	297,9	302,0	288,5	270,9	252,1	301,6	301,4	209,7	4,5	0,0	0,0
Net Revenue																												
Magnetite Concentrate																												
Net Revenue		USD M	4 120,4	0,0	0,0	0,0	12,8	215,2	249,2	248,6	238,9	247,2	241,3	242,4	249,5	239,8	251,2	259,2	245,3	234,9	202,7	187,9	207,7	205,6	138,0	3,0	0,0	0,0
Copper-Gold Concentrate																												
Cu/Au Concentrate Net Revenue		USD M	1 256,5	0,0	0,0	0,0	5,6	66,3	95,8	55,4	89,6	81,1	99,3	80,8	72,7	49,5	49,2	36,3	53,2	50,3	64,1	60,2	88,2	90,0	67,3	1,5	0,0	0,0
Total Net Revenue		USD M	5 376,9	0,0	0,0	0,0	18,4	281,5	345,0	304,0	328,5	328,3	340,7	323,1	322,2	289,3	300,4	295,5	298,6	285,2	266,7	248,1	295,9	295,5	205,4	4,4	0,0	0,0
Operating Expenditure																												
Mining		USD M	1 125,1	0,0	0,7	11,1	30,4	51,9	60,0	63,5	58,8	61,6	60,5	66,7	70,5	68,8	67,5	67,7	67,3	68,9	68,3	62,0	62,8	34,3	21,9	0,0	0,0	0,0
Processing		USD M	779,7	0,0	0,0	0,1	7,1	39,9	43,9	43,9	43,9	43,9	43,9	44,1	44,1	44,1	44,1	44,1	44,1	44,0	44,0	44,0	44,0	44,0	28,8	0,0	0,0	0,0
G&A		USD M	114,0	0,3	2,9	4,1	4,2	5,0	6,0	5,7	6,3	5,8	6,1	5,9	6,1	5,9	6,6	6,0	6,4	6,2	6,4	6,1	6,6	2,2	2,0	0,4	0,4	0,4
Other (Industrial Area)		USD M	112,9	0,0	0,2	1,1	2,3	6,0	5,8	6,0	5,9	6,0	6,0	6,1	6,1	6,1	6,3	6,2	6,2	6,2	6,2	6,3	6,3	5,8	5,9	0,0	0,0	0,0
Transportation		USD M	617,0	0,0	0,0	0,0	2,1	33,5	38,2	39,9	37,9	38,9	37,3	37,1	37,9	36,0	37,1	38,0	35,5	33,5	28,8	26,8	29,7	29,3	19,5	0,0	0,0	0,0
Mineral Royalties		USD M	8,2	0,0	0,0	0,0	0,0	0,4	0,5	0,5	0,5	0,5	0,5	0,5	0,5	0,4	0,5	0,4	0,5	0,4	0,4	0,4	0,5	0,5	0,3	0,0	0,0	0,0
Operating Contingency	5%	USD M	137,8	0,0	0,2	0,8	2,3	6,8	7,7	8,0	7,7	7,8	7,7	8,0	8,3	8,1	8,1	8,1	8,0	8,0	7,7	7,3	7,5	5,8	3,9	0,0	0,0	0,0
Total Operating Expenditure		USD M	2 894,5	0,3	4,0	17,4	48,4	143,5	162,1	167,4	161,0	164,5	162,0	168,5	173,4	169,3	170,1	170,4	167,9	167,2	161,8	152,7	157,3	121,9	82,4	0,4	0,4	0,4
Initial Capital																												
Hannukainen Mine		USD M	103,6	0,0	11,6	52,1	10,3	29,7	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Industrial Area		USD M	101,7	0,0	31,1	51,3	19,3	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Logistics		USD M	14,7	0,0	8,1	6,6	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Operational Readiness / G&A		USD M	5,9	0,0	0,8	1,7	1,7	1,7	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Process Plant		USD M	276,8	0,0	18,5	187,0	71,3	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Project Support Services		USD M	64,3	0,3	14,3	32,4	16,5	0,7	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Total Initial Capital		USD M	567,0	0,3	84,4	331,2	119,0	32,1	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Sustaining Capital																												
Hannukainen Mine		USD M	94,3	0,0	0,0	0,0	0,0	0,2	7,5	7,1	0,3	5,2	0,6	5,6	7,4	12,7	13,9	9,2	3,9	0,3	7,5	1,3	10,0	1,7	0,0	0,0	0,0	0,0
Industrial Area		USD M	61,5	0,0	0,0	0,0	0,0	12,0	0,2	3,4	0,9	10,1	6,3	0,0	3,5	4,6	9,5	4,8	0,3	0,0	2,7	0,0	0,0	0,0	0,0	3,1	0,0	0,0
Logistics		USD M	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Operational Readiness / G&A		USD M	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Process Plant		USD M	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Project Support Services		USD M	13,6	0,0	0,0	0,0	0,0	6,7	1,4	0,2	0,4	0,8	0,1	0,2	0,4	0,6	0,8	0,5	0,1	0,2	0,4	0,1	0,1	0,2	0,1	0,2	0,0	0,0
Total Sustaining Capital		USD M	169,4	0,0	0,0	0,0	0,0	18,9	9,1	10,7	1,7	16,2	7,0	5,8	11,3	18,0	24,2	14,5	4,3	0,4	10,6	1,3	10,0	1,9	0,2	3,3	0,0	0,0
Capital Contingency	10%	USD M	73,6	0,0	8,4	33,1	11,9	5,1	0,9	1,1	0,2	1,6	0,7	0,6	1,1	1,8	2,4	1,4	0,4	0,0	1,1	0,1	1,0	0,2	0,0	0,3	0,0	0,0
Total Capital Expenditure		USD M	810,1	0,4	92,8	364,3	130,9	56,1	10,0	11,8	1,9	17,8	7,8	6,4	12,5	19,8	26,6	15,9	4,7	0,5	11,7	1,5	11,0	2,1	0,2	3,6	0,0	0,0
Cashflow																												
Net Revenue		USD M	5 376,9	0,0	0,0	0,0	18,4	281,5	345,0	304,0	328,5	328,3	340,7	323,1	322,2	289,3	300,4	295,5	298,6	285,2	266,7	248,1	295,9	295,5	205,4	4,4	0,0	0,0
Total Operating Expenditure		USD M	-2 894,5	-0,3	-4,0	-17,4	-48,4	-143,5	-162,1	-167,4	-161,0	-164,5	-162,0	-168,5	-173,4	-169,3	-170,1	-170,4	-167,9	-167,2	-161,8	-152,7	-157,3	-121,9	-82,4	-0,4	-0,4	-0,4
Total Capital Expenditure		USD M	-810,1	-0,4	-92,8	-364,3	-130,9	-56,1	-10,0	-11,8	-1,9	-17,8	-7,8	-6,4	-12,5	-19,8	-26,6	-15,9	-4,7	-0,5	-11,7	-1,5	-11,0	-2,1	-0,2	-3,6	0,0	0,0
Other Expenses (X7_Reclamation / Bond payments)		USD M	-117,3	0,0	-5,0	-6,6	-6,6	-6,6	-6,6	-6,6	-6,6	-6,6	-6,6	-6,6	-6,6	-6,6	-6,6	-6,6	-6,6	-6,6	-6,6	-6,6	0,0	0,0	0,0	0,0	0,0	0,0
Working Capital		USD M	0,0	0,0	5,2	8,3	3,3	-8,5	7,9	8,0	4,9	6,9	4,0	5,9	-0,7	-4,7	2,0	7,1	6,1	7,7	7,7	7,4	-10,6	-18,6	-18,3	-10,2	-20,6	-0,1
Net pre-tax, pre-finance cashflow		USD M	1 555,0	-0,6	-96,6	-380,0	-164,3	66,9	174,3	126,3	164,0	146,3	168,3	147,6	129,0	88,9	99,1	109,7	125,5	118,6	94,3	94,8	116,9	152,9	104,5	-9,8	-21,1	-0,6
Corporation Tax		USD M	-378,6	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	-5,4	-39,4	-33,7	-32,3	-24,7	-26,6	-24,9	-26,2	-23,4	-20,7	-19,2	-32,0	-40,8	-28,6	-0,7	0,0	0,0
Interest & Fees (Fees on Environmental Bond)		USD M	-21,8	0,0	0,0	-0,2	-0,3	-0,4	-0,6	-0,7	-0,8	-1,0	-1,1	-1,2	-1,2	-1,1	-1,1	-1,2	-1,3	-1,4	-1,6	-1,7	-1,7	-1,4	-1,0	-0,6	-0,2	0,0
Net post-tax, pre-finance cashflow		USD M	1 154,5	-0,6	-96,6	-380,2	-164,6	66,4	173,7	125,6	163,1	139,9	127,8	112,7	95,6	63,1	71,4	83,6	98,1	93,8	72,1	73,9	83,2	110,7	74,9	-11,1	-21,3	-0,6

21.8 Project Sensitivities

For illustrative purposes the following analysis presents the sensitivity of the Project valuation (post-tax and pre-finance) for various capital costs, operating costs, commodity price, exchange rate and discount rate scenarios.

21.8.1 Single Parameter

Figure 21-9 shows the Project valuation for varying single parameter sensitivities at an 8% discount rate for commodity price, operating costs, capital costs and EUR:USD exchange rate.



	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
Fe Price	(82)	(15)	51	117	182	248	313	377	442	506	571
Cu Price	156	175	193	211	229	248	266	284	302	320	338
Operating Expenditure	479	433	387	340	294	248	201	154	108	61	13
EUR:USD Exchange rate	569	506	442	377	313	248	182	117	51	(16)	(83)
Capital Expenditure	372	347	322	298	273	248	223	197	172	147	121

Figure 21-9: NPV (8%) single parameter sensitivities

Assuming all other assumptions remained unchanged, SRK notes that the Project would be roughly break-even should the Fe ore price fall by around 20%.

The Project appears less sensitive to variations in capital costs, with a 25% increase in capital expenditure resulting in the valuation falling by roughly half, from USD 248 million to USD 121 million.

21.8.2 Twin Parameter

At a fixed basecase discount rate of 8%, Table 21-9 below shows the sensitivity of the Project valuation (USD million), to simultaneous changes in two parameters for; operating costs and Fe price, capital costs and Fe price, and operating costs and capital costs respectively. SRK notes that the Project is roughly break-even at:

- A decrease in Fe-ore price of 10% and simultaneous increase in operating costs of around 12%; and
- A decrease in Fe-ore price of 15% and simultaneous increase in capital costs of 10%.

At a variable discount rate, Table 21-9 below shows the sensitivity of the Project valuation (USD million), to simultaneous changes in two parameters for; operating costs and Fe price, capital costs and Fe price, and operating costs and capital costs respectively.

Table 21-9: Twin Parameter Project Sensitivities in USD million - Fixed Discount Rate (8%)

REVENUE V OPEX SENSITIVITY

		Fe Price										
	247,6	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
Operating Expenditure	-25%	154	220	285	350	414	479	543	607	672	736	799
	-20%	108	173	238	303	368	433	497	562	626	690	754
	-15%	61	126	192	257	322	387	451	516	580	644	708
	-10%	13	79	145	210	276	340	405	470	534	598	662
	-5%	-34	32	98	164	229	294	359	424	488	552	617
	0%	-82	-15	51	117	182	248	313	377	442	506	571
	5%	-131	-63	4	70	136	201	266	331	396	460	525
	10%	-182	-111	-44	23	89	154	220	285	350	414	479
	15%	-234	-161	-92	-25	42	108	173	238	303	368	433
	20%	-290	-213	-141	-72	-6	61	126	192	257	322	386
	25%	-350	-266	-192	-121	-53	13	79	145	210	275	340

REVENUE V CAPEX SENSITIVITY

		Fe Price										
	247,6	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
Capital Expenditure	-25%	47	112	178	243	307	372	436	501	565	628	692
	-20%	21	87	153	218	283	347	412	476	540	604	668
	-15%	-4	62	127	193	258	322	387	451	516	580	644
	-10%	-30	36	102	167	233	298	362	427	491	555	620
	-5%	-56	11	77	142	208	273	337	402	467	531	595
	0%	-82	-15	51	117	182	248	313	377	442	506	571
	5%	-108	-41	26	92	157	223	288	352	417	482	546
	10%	-135	-67	0	66	132	197	263	328	392	457	522
	15%	-162	-93	-26	41	107	172	237	303	367	432	497
	20%	-188	-119	-52	15	81	147	212	278	343	407	472
	25%	-216	-146	-78	-11	56	121	187	252	318	382	447

OPEX V CAPEX SENSITIVITY

		Operating Expenditure										
	247,6	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
Capital Expenditure	-25%	601	555	510	464	418	372	326	280	233	187	140
	-20%	577	531	485	439	393	347	301	255	208	162	115
	-15%	552	507	461	415	369	322	276	230	183	137	90
	-10%	528	482	436	390	344	298	251	205	158	111	64
	-5%	503	457	411	365	319	273	226	180	133	86	39
	0%	479	433	387	340	294	248	201	154	108	61	13
	5%	454	408	362	315	269	223	176	129	82	35	-12
	10%	429	383	337	290	244	197	151	104	57	9	-38
	15%	405	358	312	265	219	172	125	78	31	-16	-64
	20%	380	333	287	240	194	147	100	53	5	-42	-90
	25%	355	308	262	215	168	121	74	27	-20	-68	-116

Table 21-10: Twin Parameter Project Sensitivities in USD million - Variable Discount ` Rate

DISCOUNT FACTORS V IRON PRICE

NPV Discount Factors	Fe Price											
	1 154,5	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
	0%	378,1	533,3	688,6	843,9	999,2	1 154,5	1 309,8	1 465,1	1 620,4	1 775,7	1 931,0
	2%	204,7	328,0	450,8	573,4	695,9	818,2	940,5	1 062,6	1 184,8	1 306,8	1 428,8
	4%	78,4	177,7	276,3	374,6	472,7	570,5	668,2	765,7	863,2	960,5	1 057,8
	6%	(14,1)	67,0	147,3	227,2	306,8	386,1	465,3	544,3	623,1	701,8	780,4
	8%	(82,2)	(15,1)	51,2	117,0	182,4	247,6	312,6	377,3	442,0	506,4	570,7
	10%	(132,3)	(76,1)	(20,8)	34,0	88,5	142,7	196,7	250,4	304,0	357,4	410,7
	12%	(169,0)	(121,6)	(74,9)	(28,7)	17,1	62,7	108,0	153,1	198,0	242,8	287,3
	14%	(195,8)	(155,4)	(115,6)	(76,3)	(37,4)	1,3	39,7	78,0	116,0	153,9	191,6
	16%	(215,2)	(180,3)	(146,2)	(112,4)	(79,0)	(45,9)	(13,0)	19,6	52,2	84,5	116,7
	18%	(228,8)	(198,6)	(169,0)	(139,8)	(111,0)	(82,3)	(54,0)	(25,8)	2,2	30,1	57,7
	20%	(238,1)	(211,7)	(185,9)	(160,5)	(135,3)	(110,4)	(85,8)	(61,3)	(37,0)	(12,8)	11,2

DISCOUNT FACTORS V COPPER PRICE

NPV Discount Factors	Cu Price											
	1 154,5	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
	0%	937,9	981,3	1 024,6	1 067,9	1 111,2	1 154,5	1 197,9	1 241,2	1 284,5	1 327,8	1 371,2
	2%	647,8	681,9	716,0	750,1	784,2	818,2	852,3	886,3	920,4	954,4	988,5
	4%	434,1	461,4	488,7	516,0	543,3	570,5	597,7	625,0	652,2	679,4	706,5
	6%	275,3	297,5	319,7	341,9	364,0	386,1	408,2	430,3	452,4	474,5	496,5
	8%	156,3	174,6	192,9	211,2	229,4	247,6	265,8	284,0	302,1	320,3	338,4
	10%	66,5	81,8	97,1	112,3	127,5	142,7	157,9	173,0	188,1	203,2	218,3
	12%	(1,6)	11,3	24,2	37,0	49,9	62,7	75,5	88,2	101,0	113,7	126,4
	14%	(53,5)	(42,5)	(31,5)	(20,6)	(9,6)	1,3	12,2	23,1	33,9	44,8	55,6
	16%	(93,1)	(83,6)	(74,2)	(64,7)	(55,3)	(45,9)	(36,6)	(27,2)	(17,9)	(8,6)	0,7
	18%	(123,3)	(115,0)	(106,8)	(98,7)	(90,5)	(82,3)	(74,2)	(66,1)	(58,0)	(50,0)	(41,9)
	20%	(146,2)	(139,0)	(131,8)	(124,7)	(117,5)	(110,4)	(103,4)	(96,3)	(89,2)	(82,2)	(75,2)

DISCOUNT FACTORS V OPERATING EXPENDITURE

NPV Discount Factors	Operating Expenditure											
	1 154,5	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
	0%	1 701,0	1 591,7	1 482,4	1 373,1	1 263,8	1 154,5	1 045,3	936,0	826,7	717,4	608,1
	2%	1 249,3	1 163,2	1 077,0	990,8	904,5	818,2	731,9	645,5	559,0	472,4	385,8
	4%	915,9	846,9	777,9	708,9	639,7	570,5	501,2	431,8	362,3	292,7	222,9
	6%	666,9	610,9	554,8	498,7	442,5	386,1	329,7	273,2	216,5	159,7	102,7
	8%	478,9	432,8	386,6	340,4	294,1	247,6	201,0	154,4	107,5	60,5	13,3
	10%	335,5	297,2	258,7	220,1	181,5	142,7	103,8	64,7	25,6	(13,8)	(53,4)
	12%	225,3	193,0	160,6	128,0	95,4	62,7	29,8	(3,2)	(36,4)	(69,7)	(103,3)
	14%	140,0	112,4	84,8	57,1	29,3	1,3	(26,8)	(55,0)	(83,3)	(111,8)	(140,6)
	16%	73,4	49,7	26,0	2,1	(21,8)	(45,9)	(70,1)	(94,4)	(118,9)	(143,5)	(168,4)
	18%	21,2	0,7	(19,9)	(40,6)	(61,4)	(82,3)	(103,4)	(124,5)	(145,8)	(167,3)	(188,9)
	20%	(19,8)	(37,8)	(55,8)	(73,9)	(92,1)	(110,4)	(128,9)	(147,4)	(166,1)	(184,9)	(203,9)

DISCOUNT FACTORS V CAPITAL EXPENDITURE

NPV Discount Factors	Capital Expenditure											
	1 154,5	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
	0%	1 308,1	1 277,4	1 246,7	1 216,0	1 185,3	1 154,5	1 123,8	1 093,1	1 062,4	1 031,7	1 001,0
	2%	963,0	934,1	905,2	876,2	847,2	818,2	789,2	760,1	731,0	701,9	672,7
	4%	707,6	680,3	652,9	625,5	598,0	570,5	542,9	515,3	487,6	459,9	432,1
	6%	516,5	490,6	464,6	438,5	412,3	386,1	359,9	333,5	307,1	280,7	254,1
	8%	372,0	347,2	322,5	297,6	272,6	247,6	222,5	197,4	172,1	146,8	121,5
	10%	261,5	237,9	214,2	190,5	166,6	142,7	118,7	94,6	70,4	46,2	21,9
	12%	176,4	153,9	131,2	108,4	85,6	62,7	39,7	16,6	(6,6)	(29,8)	(53,1)
	14%	110,4	88,8	67,0	45,2	23,3	1,3	(20,8)	(42,9)	(65,1)	(87,4)	(109,7)
	16%	58,8	38,0	17,2	(3,8)	(24,8)	(45,9)	(67,1)	(88,4)	(109,7)	(131,1)	(152,5)
	18%	18,2	(1,7)	(21,7)	(41,9)	(62,1)	(82,3)	(102,7)	(123,1)	(143,6)	(164,1)	(184,7)
	20%	(13,7)	(32,9)	(52,2)	(71,5)	(90,9)	(110,4)	(130,0)	(149,6)	(169,3)	(189,1)	(208,9)

DISCOUNT FACTORS V EUR:USD EXCHANGE RATE SENSITIVITY

NPV Discount Factors	EUR:USD Exchange rate											
	1 154,5	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
	0%	1 796,9	1 668,3	1 540,0	1 411,4	1 283,0	1 154,5	1 026,1	897,7	769,5	640,8	512,4
	2%	1 343,9	1 238,9	1 134,1	1 028,8	923,6	818,2	712,7	607,1	501,4	395,2	288,8
	4%	1 009,2	921,8	834,4	746,6	658,6	570,5	482,1	393,5	304,7	215,3	125,4
	6%	758,8	684,8	610,6	536,0	461,2	386,1	310,8	235,1	159,2	82,6	5,5
	8%	569,2	505,5	441,6	377,2	312,5	247,6	182,3	116,7	50,8	(15,8)	(83,0)
	10%	424,2	368,6	312,7	256,3	199,7	142,7	85,4	27,7	(30,3)	(89,0)	(148,3)
	12%	312,2	263,0	213,5	163,5	113,3	62,7	11,7	(39,6)	(91,3)	(143,6)	(196,5)
	14%	224,8	180,9	136,5	91,8	46,7	1,3	(44,5)	(90,6)	(137,0)	(184,1)	(231,7)
	16%	156,2	116,5	76,5	36,0	(4,8)	(45,9)	(87,4)	(129,2)	(171,4)	(214,0)	(257,3)
	18%	101,9	65,8	29,3	(7,6)	(44,8)	(82,3)	(120,2)	(158,4)	(196,9)	(236,0)	(275,5)
	20%	58,7	25,6	(7,9)	(41,8)	(76,0)	(110,4)	(145,3)	(180,4)	(215,8)	(251,7)	(288,0)

22 ADJACENT PROPERTIES

22.1 Northland's Pajala Projects

Northland also own three advanced-staged projects near to Pajala, in Norbotten, Sweden. The Sahavaara, Tapuli and Pellivuoma projects are all magnetite skarn deposits with no economic Cu present. They also lie on the Pajala Shear Zone fault system, which extends across Northern Sweden and Finland.

Northland is currently operating the Tapuli Project and producing a magnetite concentrate.

22.2 Other Significant Fe and Cu deposits in Northern Fennoscandia

The economically most important iron deposits in Northern Fennoscandia are the Kiruna and Malmberget magnetite-apatite deposits, both currently operated by LKAB.

The Kiruna deposit, located 150 km northwest from the Pajala area, was discovered in 1696 and has been mined on a regular basis since 1900. The ore is currently mined underground.

The Kiruna iron deposit consists of a 5 km long, up to 100 m thick, steeply dipping mineralisation with the sole mineral of economic interest being magnetite.

Another currently operating iron mine is the Malmberget magnetite-apatite deposit located at Gällivare in Sweden, about 100 km northwest from Pajala. The Malmberget deposit consists of some 20 ore bodies over an underground area about 5 x 2.5 km. The mineralisation type is the same as found at Kiruna.

The largest copper deposit currently in production in Europe, Aitik, is located some 18 km away from Malmberget. Aitik was discovered in 1930 and has been in production since the 1960s. The deposit has been interpreted as a metamorphosed porphyry deposit (for example, Wanhainen *et. al.,* 2003).

23 INTERPRETATION AND CONCLUSIONS

23.1 Mineral Resource Estimate

It is the opinion of SRK that the quantity and quality of available data is sufficient to generate Measured, Indicated and Inferred Mineral Resources and that the Mineral Resource Statement has been classified by Howard Baker (MAusIMM(CP)), who is a Qualified Person, in accordance with the CIM guidelines and definitions. It has an effective date of 24 October 2012 and incorporates all drilling undertaken to date.

Historical data provided included technical reports collected up to the present day, which have been reviewed and found to contain information that has been collated and interpreted in a professional manner and provide support to the electronic database for the Project.

SRK has relied heavily upon the information provided by Northland and in particular that all of the information available has been provided and none held back; however, SRK has, where possible, verified data provided independently during the site visit.

The drillhole spacing, varying between 50 to 250 m, has allowed a robust geological model to be created.

Sampling, sample preparation and analysis has been undertaken using standard and appropriate methodologies with reasonable QAQC procedures followed.

SRK verified the 2006 to 2012 Northland drilling programme database by comparing geological logs and laboratory assaying sheets with drill core on site.

The metallurgical and processing properties of the mineralisation have been investigated thoroughly in several different testwork programmes.

SRK has constructed a wireframe geological model for the Project based upon a combination of logged lithologies and analytical and magnetic susceptibility results. This has allowed the splitting of the deposit into geological domains comprising low grade Fe clinopyroxene-rich skarn, high grade Fe magnetite skarn and high grade Cu skarn. The mineralisation forms gently undulating tabular bodies with a shallow (0-30º) dip.

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

SRK has not undertaken a legal review of the licences and assume that all the required licences are in place. It should be noted that all exploration claims covering the Hannukainen deposit have now expired, however, the mining concession application is on-going, which allows for the site to be continue to be explored and investigated by Northland.

The Mineral Resource Statement generated by SRK is split into two categories: open pit and underground. The open pit Mineral Resource has been restricted to all classified material falling within a Whittle Shell representing metal prices of USD1.50/dmtu for magnetite concentrate USD3.35/lb for copper and USD1,375/oz for gold, and through the application of reasonable mining and processing costs and recoveries. The underground Mineral Resource has been restricted to all material outside of the Whittle shell, above an Fe equivalent value of 35.6%, where:

Fe equivalent=(FE/100+(CU_PPM/1000000*82.8833)+(AU_PPB/1000*0.13237512))*100

This represents the material which SRK considers has reasonable prospect for eventual economic extraction potential.

In total, the Project has a joint underground and open pit combined Measured and Indicated Mineral Resource of 187 Mt grading 30.04% Fe Total, 0.18% Cu and 0.114 g/t Au. In addition, there is a total of 63 Mt grading 32.05% Fe Total, 0.15% Cu and 0.047 g/t Au in the Inferred category.

It is the opinion of SRK the geological block model generated is suitable for detailed mine planning activities and suitable for use in the HFS being undertaken by Northland.

23.2 Mineral Reserve Estimate

23.2.1 Mining Block Model

The following mining model has been used for the optimisation, mine design and production planning studies:

- a 6.25 mX x 6.25 mY x 5 mZ sized regularised mining model has been selected for the Project; and
- the average mining recovery and dilution for the regularised block model evaluated at a 13% CoG are 97.7% and 6.5%, respectively.

23.2.2 Pit Optimisation

On the basis of a 17 year mine life minimum, the 0.89 revenue factor pit shell (relating to an iron ore price of USD1.25/dmtu) was selected as the basis for the pit designs. The pit shell contains 117.6 Mt of ore and 416.0 Mt of waste.

The following conclusions can be drawn from the pit optimisation results:

- the pit optimisation is relatively insensitive to changes in Cu concentrate transport and selling costs, Au selling price, and slope angles; and
- the pit optimisation is most sensitive to the Fe selling price followed by the Fe processing recovery.

23.2.3 Pit Design

The resulting mineable tonnages from the engineered pit design are 114.8 Mt of ore with 446.8 Mt of waste at a variable CoG depending on the processing recovery. The engineered pit designs resulted in a 2.9 Mt reduction in ore and 31.3 Mt increase in waste compared with the optimised pit shell.

23.2.4 Waste Dump Design

The results for the waste dump design are summarised below:

- three waste dumps have been designed: East, West and West Overburden;
- the waste dumps have been designed based on the waste inventory from the pit designs; and
- the dump capacity has been limited by the groundwater catchment area for the PAF material and to a lesser extent the NAF material.

23.2.5 Mine Dewatering

The main conclusions and outcomes from the study are as follows:

- There are currently two pits located at the Hannukainen site which will need to be dewatered before operation mining can commence and will be completed in approximately 6 months.
- Pre-operational pit dewatering requirements have been incorporated into the operational dewatering design such that equipment, including both pumps and pipe-work, can be utilised during both phases of the mine operation.
- The predicted transient state inflow from groundwater to the pit based on the numerical model was 30 to 300 m3/hr at Hannukainen and 110 to 320 m^3/h at Kuervitikko. These values represent flow throughout the lifetime of the mine and illustrate the increase in flow rate as the pits expand. It should be noted that these values are considered the most likely given the estimates of hydraulic properties available at the time of study but pit inflows could be higher or lower depending on the properties of the rock mass. This is highlighted as both a potential risk and opportunity.
- surface water pumping requirements have been assessed on the basis of two key factors; firstly, average precipitation and the seasonal effects of snow-melt, and secondly, the impact of large rainfall events. The maximum monthly rainfall plus snowmelt volume occurs in May when a capacity of < 410 m^3/h may be required at Hannukainen and <110 m^3/h at Kuervitikko (about four times that produced by precipitation within other months). Significant additional volumes of water can be produced from 24-hour rainfall events.
- These flow rates are illustrative of the transient state inflows that may be expected in the pit and form the basis on which a flexible dewatering scheme has been designed and cost estimated at a conceptual level.
- Pump requirements have been assessed and cost estimated based on the inflow results from the numerical modelling. Flexible systems have been outlined and example electrical sump and in-line booster pump options presented.
- An engineered surface water diversion system should be put in place comprising of bunds and ditches around the perimeter of the pit and will provide additional protection from surface water inflows.
- If properly managed, the predicted groundwater inflows are unlikely to cause significant operational problems. Surface water inflow (including direct precipitation) will be significant and will need careful management to ensure that the inflow volumes associated with the spring thaw do not affect mine operations.

23.2.6 Mine Production Schedule

The results of the mining schedule are summarised below:

- a two year pre-stripping period (including 350 kt of ore) is scheduled to develop the initial stages down to large ore areas;
- the HC region is targeted in the early parts of the schedule to access high Fe grades, it is blended with HS and HN ore to lower the strip ratio in the early years;
- the KU region is left until the end of the schedule due to the low Fe grades and is mined with the final HC stage to increase the Fe grades;
- Fe grades ramps up to approximately 32% to 34% for the first 10 years and then slowly decrease with the introduction of lower quality ore, especially when KU ore starts to be fed in 2029;
- S grades fluctuate between 2.0% to 2.8% depending on the ore source, with the lowest S grades in HC region;
- Cu and Au grades follow the same trend ranging from 0.11% to 0.26% and 0.03 to 0.27 g/t, respectively;
- the Fe concentrate ranges between 1.9 to 2.2 Mtpa for the initial 13 years and then drops as ore feed quality decreases;
- the Cu concentrate fluctuates with the Cu grade in the ore feed;
- the product recoveries follow the trends of the ore feed grades;
- a topsoil clearing schedule has been split into three occasions beginning in 2015 Q3, 2019 Q1 and 2020;
- contractors will be used for all material movement until 2016 Q3 and all topsoil clearing;
- 26 m^3 face shovels will mine waste material, a maximum of two units will be required, to be purchased in 2016 Q4 and 2018 Q1;
- a 19 m^3 FEL will supply ore feed and mine excess waste material which will be required from 2017 Q4;
- it has been assumed that all Run of Mine re-handle will be handled by a small FEL;
- a maximum of 12 haul trucks are required (year 2025);
- the haulage fleet follows the general trend of the total material movement profile, with an increase in trucks required towards the end of the schedule due to the deepening of the pit and the increasing height of the waste dumps; and
- mine labour requirements for the mine operations, mine technical services and mine maintenance groups are 155, 14, and 38 employees, respectively, for a total of 207 employees at maximum production.

23.2.7 Operating Strategy

The findings of the operating strategy are summarised below:

- the mining loading fleet consists of 26 m^3 face shovels and a 19 m^3 FEL;
- the haulage fleet comprises 227 t capacity haul trucks;

- waste will be mined in full 10 m benches, while the ore is flitch mined in 5 m benches;
- a mine dispatch system is required to manage the fleet as well as recording and reporting purposes;
- overburden material will be mined by free digging, with all other material requiring drilling and blasting; and
- a mixed fleet of 229 mm and 172 mm blasthole drills will be used for production, trim and in-fill drilling.

23.2.8 Cost Estimate

The results from the cost estimate are summarised below:

- an average LoM mine operating cost of USD2.00/tmoved;
- a total LoM mine operating cost of USD1,125M;
- a total LoM mine capital cost of USD198M, including sustaining capital costs;
- haulage costs, maintenance costs, diesel prices and labour rates are the largest contributors to the operating cost; and
- the cost estimate is most sensitive to changes in the haulage costs.

23.3 Recovery methods

Metallurgical testwork has been performed in a number of campaigns using a total of 36 metallurgical samples including two pilot plant tests and has demonstrated that high quality iron and copper concentrate can be produced in Hannukainen with the developed flowsheet.

A geometallurgical model has been established for the Hannukainen ore. The model has been used in selecting samples for variability and metallurgical testing and for pit optimisation. Variability testing included more than 200 samples. The ore bodies have been classified in terms of metal recovery with respect to iron, copper and sulphur content. There is sufficient understanding of the main ore types to be able to define the expected metallurgy and to predict the metallurgical recoveries of iron and copper in to saleable magnetite and copper concentrates.

Sufficient testwork has been performed to define the metallurgy and the selected process flowsheet is considered to be appropriate for the different ore types at Hannukainen.

Comminution testing has been completed in laboratory and pilot scale testing. The results have been incorporated in to the grinding circuit design.

The selected flowsheet includes two stage grinding using a SAG – ball mill configuration with flash flotation of fast floating chalcopyrite. Hydrocyclone overflow, nominally 80% minus 90 microns passes to copper rougher flotation followed by pyrite flotation. The tailings from pyrite flotation are treated by LIMS to produce a combined magnetic concentrate containing magnetite and pyrrhotite. This is cleaned to remove the sulphur bearing pyrrhotite by flotation. Copper concentrate is cleaned in three stages of flotation incorporating a regrind circuit. Final tailings are stored as high and low sulphide tailings in two separate impoundment areas.

The magnetite and copper concentrates contain acceptable levels of impurities.

Pelletizing characteristics of the magnetite concentrate have been studied by COREM and show that the product has excellent properties.

Satisfactory recovery functions have been developed for iron in terms of % iron and % sulphur levels in the feed and for copper recovery in terms of copper in the feed. Recovery functions for gold require further work if considered significant in terms of the project economics. These recovery functions represent the projected performance of the latest flowsheet.

Sufficient engineering has been performed to establish the plant capital cost.

The process operating costs assumed for the HFS were estimated from first principles and SRK considers the underlying assumptions and overall costs to be reasonable.

The implementation schedule and the plant ramp up time for the Project are considered realistic. The potential effect of weather windows on the overall schedule should be reassessed if the Project start date changes significantly.

23.4 Environment

Northland is in the process of completing an EIA for the mine site at Hannukainen and the Rautuvaara processing site and tailings storage facility. A number of issues exist that will need to be evaluated proactively to ensure additional material costs are not incurred, in particular:

- the acid rock drainage potential associated with some of the mine waste rock and tailings and the potential to contaminate aquifers and water courses via contact with base flow;
- dewatering of aquifers and impacts on base flow of water courses;
- the impacts on ecologically protected areas and protected plant and animal species;
- land acquisition;
- the relationships with the local reindeer husbandry cooperative and its members; and
- other parties who are responsible for undertaking the necessary assessments and gaining permits for the railway and port upgrade.

Some water courses in Project areas are Natura 2000 sites and breeding habitats for the protected Arctic Salmon and Sea Trout of critical ecological importance. Water quality standards and mixing zones have yet to be agreed with authorities, who may impose stricter standards than those used for modelling impacts in the EIA. As a consequence, costlier mitigation may be required than that proposed in the EIA.

The necessary permits have still to be obtained but there is a strategy in place to obtain these. SRK considers there is a significant risk ELY will require Northland to revise its EIA report and this could affect the subsequent joint application for the environmental and water permits. It is also possible the permit process will also be subject to delays due to the significant public interest and sensitivity of the water issues (see above). In addition, both the EIA and the permit process may be subject to appeals (by either Northland or members of the public), further delaying the approval process.

23.5 Economic Analysis

As part of this technical report, SRK's role has been to construct a financial model in order to derive a post-tax, pre-finance Net Present Value ("NPV") for the Project and independently verify (or otherwise) the valuation derived by Northland for the HFS. SRK has constructed its independent financial model using the cost data from Northland's cost templates, as well as extracting the underlying technical assumptions, macro-economic assumptions and life of mine plan from the HFS financial model itself.

The figures presented and discussed in this section of the report correspond to the inputs and outputs (resulting forecast cashflows and subsequent valuation) from SRK's financial model. Whilst there are slight differences in forecast cashflows between the two models, these are considered to be non-material.

A valuation of the Project has been derived based on the application of Discounted Cash Flow ("DCF) techniques to the post-tax, pre-finance cash flow developed for the HFS.

The resulting post-tax, pre-finance real terms (1 December 2013) NPV derived by SRK is USD 248 million, assuming an 8% discount rate. SRK notes that Northland report a NPV of USD 251 million (post-tax, pre-finance at 8% discount) in the HFS. This difference is not considered to be material.

24 RECOMMENDATIONS

24.1 Mineral Resource Estimate

Based on the results of this MRE, with the majority of the material being classified as a Measured Mineral Resource, SRK does not recommend any further Mineral Resource definition drilling. Should additional geophysical survey data suggest potential extensions to mineralisation, being below the current open pit optimised Mineral Resource, then scout drilling may be warranted to define additional potential underground mineral Mineral Resources. Furthermore, additional drilling may be required for any on-going metallurgical testwork being conducted on the various Hannukainen ore types.

24.2 Mineral Reserve Estimate

24.2.1 Pit Optimisation

SRK recommends that the following work be undertaken following this study:

- update the metallurgical recovery parameters for Fe and Cu based on testwork results, also verify the metallurgical parameters for Au which have remained unchanged since March 2011; and
- undertake another round of pit optimisation using the updated costs and input parameters from the HFS findings to check for any significant changes.

24.2.2 Pit Design

SRK recommends that the following work should be undertaken during the detailed engineering phase of the Project:

- the pits and cutbacks have been designed for 181 t class trucks, a re-evaluation of the mine schedule has determined that larger trucks (227 t) are more economical for this Project and these have been used for this HFS; therefore, a re-design of the pits will be required to ensure the ramp width is suited to the 227 t class trucks; that said, the current design of 27 m ramp width is within the standard tolerance limits of 3 to 3.5 times the truck width should a 227 t truck be used; and
- the footwall slopes should be laid back following the ore contact to decrease ore loss.

24.2.3 Waste Dump Design

SRK recommends that the following work should be undertaken during the detailed engineering phase of the Project:

- the waste dumps have been designed for 181 t class trucks, a re-evaluation of the mine schedule has determined that larger trucks (227 t) are more economical for this Project, therefore a re-design of the waste dumps will be required to ensure the ramp width is suited to the 227 t class trucks;
- evaluate the potential to store PAF and NAF material outside of the groundwater catchment area, which would decrease haulage cycle times; and
- the potential for in-pit dumping should be re-evaluated to decrease haulage distances.

24.2.4 Mine Dewatering

SRK recommends that the following work should be undertaken during the detailed engineering stage of the Project:

- conduct a cost benefit analysis of various sump pumps;
- estimates of hydraulic properties of the area can be improved by long duration pumping tests, which will enable the predicted pit inflows to be better constrained;
- additional field testing could be undertaken in the exploration wells to identify zones of high permeability and map the variability of hydraulic conductivity, particularly at Kuervitikko where the pit inflow predictions are sensitive to the estimation of the Hannukainen Thrust Zone hydraulic conductivity;
- investigate the potential for advanced dewatering at Kuervitikko pit;
- additional groundwater monitoring during advanced dewatering to confirm the hydraulic properties of the fault zone and constrain the likelihood of connection between the pit and the Äkäsjoki River;
- during operations a flexible pump dewatering system enables overall pump capacity to be varied according to demand by adding (or removing) pumps to the system if required;
- the mine operators should consider horizontal drain investigations into fault zones prior to push backs in order to identify zones of potentially high pit inflows;
- spring melt water peak may be higher than that predicted or coincide with a storm event, resulting in greater pumping time required, pumping costs and possible disruption to mining schedules; however, an engineered surface water diversion system will ensure that all ex-pit surface water runoff is prevented from entering the pit and therefore minimise this risk; and
- use of flexible pump system allows variation in pumping capability as required. This could allow prioritisation of north (or south) sump pumping at Hannukainen or Kuervitikko if required.

24.2.5 Mine Production Schedule

SRK recommends that the following work should be undertaken for the production schedule during the detailed engineering phase of the Project:

- review the potential for ore stockpiling for extended periods to enable high quality ore feed to be targeted in the early years while stockpiling the lower grade ore.

24.2.6 Operating Strategy

SRK recommends that the following work be undertaken during the detailed engineering phase of the Project:

- it has been assumed that the FEL will mine all ore, due to the inflexibility of the face shovels; further investigation should be undertaken to determine whether the FEL can achieve selectivity to the SMU size;
- benchmark operating time estimates with production data for similar operations;

- benchmark operating delays due to weather with operations in similar climates;
- benchmark the estimated drill production rates with supplier information; and
- benchmark the blasting parameters with an explosives supplier.

24.2.7 Cost Estimate

SRK recommends that the following work should be undertaken during the detailed engineering phase of the Project:

- evaluate the sensitivity of the currency exchange rates; and
- verify the availability of the truck and FEL tyres from the manufacturers; and
- benchmark truck and FEL tyre life with similar operations using the same tyre manufacturers.

24.2.8 Mineral Reserves Risks and Opportunities

Risks

The key risks for the Project are summarised below:

- the ore must be mined to a selectivity of the SMU block size to achieve the mining recovery and dilution predicted; if the SMU size cannot be achieved the mining dilution and ore loss will increase;
- the pit optimisation is most sensitive to Fe selling price, a decrease would result in ore loss and a decrease in Project value;
- the pit optimisation and scheduling are based on preliminary metallurgical parameters from March 2011 for Cu and Au recovery, changes to these could affect the pit size and have an impact on the Cu concentrate product;
- there is potential for some ore loss and waste gain in the redesigns required for the pit and cutback designs due to the change in truck size;
- the cost estimate is most sensitive to changes in the haulage costs, increases to these costs will significantly impact the operating costs;
- equipment manufacturers have highlighted the potential decrease in availability of truck and FEL tyres in the coming years, costs may escalate should this occur;
- SRK has estimated three non-production days in the schedule due to adverse weather, a change in non-production days will impact productivity and operating costs;
- there is currently a global shortage of skilled workers available in the mining industry, a shortage of skilled workers will decrease productivity and possibly delay the start up of the Project;
- the global mining industry is experiencing an increase in mine start up activity and lead times for mining equipment is increasing as build spots are being booked over 12 months in advance. This creates a risk for the start up of the operation if mining equipment cannot be purchased and delivered in a timely manner.

Opportunities

The key opportunities for the Project are summarised below:

- there is potential to lay back the footwall slopes to follow the ore contact directly to decrease ore loss and excess waste mining;
- evaluate the potential to store PAF and NAF material outside to the groundwater catchment area, which would decrease haulage cycle times;
- the potential for in-pit dumping should be re-evaluated to decrease haulage distances;
- review the potential for ore stockpiling for extended periods to enable high quality ore feed to be targeted in the early years while stockpiling the lower grade ore, this would increase the value of the Project with high grade ore being produced earlier.

24.3 Recovery Methods

Further metallurgical testwork is required prior to the final design to:

- verify the improved magnetite recovery at the coarser grind by further pilot scale testing;
- further testing to improve the magnetite recovery by coarsening the grinding and/or optional reagents in pyrrhotite flotation;
- copper cleaning flotation optimisation in a pilot scale;
- testing of very low copper grade samples;
- gold deportment studies and optimisation of the gold recovery; and
- further laboratory testing was performed in 2013 and indicated that a coarser grind may be feasible without significantly affecting grade and recovery. This should be investigated prior to final plant design.

24.4 Mine Site Infrastructure

24.4.1 Geotechnical Investigation and Bulk Earthworks

SRK recommends that the terracing required at Hannukainen and Rautuvaara to create the platforms required to support the proposed development are defined and bulk earthworks quantities assessed.

24.4.2 Explosives and Detonator Storage and Management

SRK recommends that appropriate safety zones (dependent upon the mass of explosives stored), in accordance with accepted international ANFO & explosives handling and storage regulations, are used to separate bulk explosives and detonators from one another, other mining assets and public roads. These standards should also be used to define the structural form, internal fit out and services to safely store explosive components.

24.4.3 Materials Handling and Load Out Infrastructure

SRK considers validation of a conventional above ground primary ore crushing solution a substantial Project opportunity which should be considered in further detail during future project phases.

24.4.4 Mine Site Waste Management

SRK recommends a waste management strategy is clearly defined in accordance with accepted international environmental practice to inform the definition of appropriate supporting infrastructure.

24.4.5 Power

SRK recommends that the anticipated power consumption for all assets is developed in order to facilitate negotiation with the power authority and to determine annual operating costs.

24.5 Water Management

SRK is currently developing a Waste Rock Geochemical Characterisation Report and an updated Hydrological Impact Assessment on behalf of the Company. Therefore, SRK recommends that surface water management pollution control infrastructure should be defined based on the recommendations made within these reports.

24.5.1 Project Delivery Schedule

SRK recommends the Project delivery schedule is developed to include, but not be limited to:

- completion of all outstanding technical studies to support definition of the HFS;
- sufficient period to raise the required capital investment;
- application and approval periods for any required permits and licenses;
- procurement of construction and service contracts;
- engagement with statutory bodies and national infrastructure operators;
- mobilisation;
- third paty activities (e.g. FTA and Port of Kokkola);
- commissioning; and
- Commencement of mining operations.

24.6 Concentrate Transport Logistics

24.6.1 Fe Concentrate

SRK recommends that:

- a whole system dynamic simulation model be undertaken as part of the next stage of project development to ensure that warehousing and logistics pathways are adequate for the anticipated production tonnages;
- contracts / agreements are finalised with the third parties to ensure surety of costs within the economic model; and
- marketing input is used to establish the % destination of the concentrate to the POS to allow finalisation of the economic model.

24.6.2 Cu-Au Concentrate

For the next development of the Project, off-take agreements should be established confirming the POS.

24.7 Economic Analysis

SRK recommends updating the financial model following any material changes to the underlying technical and cost assumptions for the project. This would include (but not be limited to) any update of the mining schedule to incorporate the latest metallurgical testwork results.

25 REFERENCES

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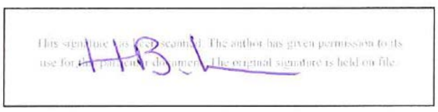
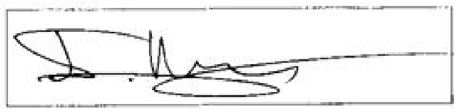
For and on behalf of SRK Consulting (UK) Limited





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