PRELIMINARY ECONOMIC ASSESSMENT FOR THE RÖNNBÄCKEN NICKEL PROJECT, SWEDEN

Prepared For

NICKEL MOUNTAIN RESOURCES AB

Report Prepared by

srk consulting

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SE355

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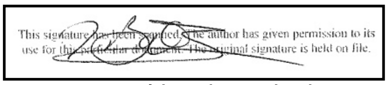
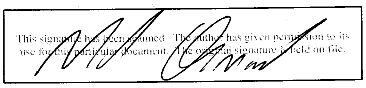
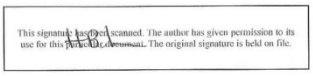
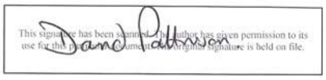
Qualified Person:	 Mr. Johan Bradley, Managing Director, SRK Consulting (Sweden) AB
Qualified Person:	 Dr. Mike Armitage, Chairman, SRK UK Ltd
Qualified Person:	 Mr. Howard Baker, Principal Geologist, SRK UK Ltd
Qualified Person:	 Dr. David Pattinson, Corporate Consultant, SRK UK Ltd

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1 SUMMARY

1.1 Introduction

The Rönnbäcken Nickel Project (Rönnbäcken or the Project), owned by Nickel Mountain AB, is located in the northwest part of Sweden, about 20 km to the south of the village of Tärnaby, Västerbotten County. The Project comprises three discrete sulphide nickel deposits; Rönnbäcksnäset, Vinberget and Sundsberget.

This report comprises a preliminary economic assessment (PEA) of Rönnbäcken and has been prepared by SRK Consulting (Sweden) AB (SRK) on behalf of Nickel Mountain Resources AB (the Company), the parent company of Nickel Mountain AB.

SRK previously authored a PEA on the Project on behalf the Company in April 2011. In addition to a nickel-cobalt sulphide flotation product, this updated PEA considers revenue and cost implications of producing a magnetite concentrate from sulphide flotation tailings.

This Technical Report conforms to National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). SRK most recently visited the property on 7 February 2011.

As part of its work, SRK has prepared independent Mineral Resource estimates for each of the deposits and in addition, has reviewed all other technical work completed on the Project by the Company and its other contractors and consultants to a sufficient level to enable SRK to present its own opinions on the Project and to derive an audited NPV for this.

The Company has initiated a pre-feasibility study on the Project and subject to the results of this, expects to commence a feasibility study in Q1 2013

1.2 Geology

The Project is located in the Swedish Caledonian mountains and is hosted by rocks which formed approximately 400-510 million years ago. The geology in the Rönnbäcken area is dominated by the Köli Nappe consisting of phyllite and felsic to mafic metavolcanics and nickel bearing ultramafic rocks. The ultramafic rocks occur as lenses of various sizes over the Project area together covering an area of roughly 15 km^2.

The nickel-sulphide mineralization which is the target of the proposed mining operation is hosted by serpentines, tectonically displaced from the mantle into the crust, and is considered to be of epigenetic origin and to have formed during the release of nickel from olivine through a process of alteration and serpentinization of the precursor dunite and peridotite rocks..

The Rönnbäcksnäset deposit comprises two separate serpentinized orebodies separated by between 80 m and 140 m of chloritic phyllite. The orebodies dip at approximately 45° west in the north and flatten out into a bowl shaped geometry to a dip of roughly 30° north in the southwest. The deposit has a strike length of roughly 2.4 km and a width of up to 400 m at its widest point.

The Vinberget deposit comprises a single homogeneous serpentinized tabular-shaped orebody (up to 350 m thick, 300 m wide and 700 m long) which dips steeply to the northeast and plunges to the northwest.

The Sundsberget deposit consists of a single serpentinite body that strikes in a north-northeast to south-southwest orientation and dips at roughly 30° to the west northwest. The deposit extends for roughly 1.2 km along strike and is between 500 m and 600 m in width.

1.3 Mineral Resources

Table 1-1 below presents the Mineral Resource Statements for the Project as a whole, combining the SRK statements for Rönnbäcksnäset, Vinberget and Sundsberget. As is typical of ultramafic-hosted disseminated nickel sulphide deposits, nickel is contained both in nickel sulphides and in silicates such as olivine and pyroxene. Conventionally with these deposits the reported nickel grades and recoveries are “Total Ni” which incorporates both the nickel in sulphides and silicates. For Rönnbäcken however, an analysis of the nickel in sulphide has been carried out through the use an analytical technique utilising a weak acid digest. Hence the term Ni-AC. This has enabled evaluation of the project based on the metallurgical performance of the nickel in sulphides only, rather than considering the deportment of nickel in both sulphides and silicates. The rational is that a high proportion of the sulphide nickel is recovered in the flotation process whereas the non-sulphide nickel reports predominantly to tailings. The Ni-AC results here are referred to in this report as “Sulphide Ni” grades and recoveries.

The Resources are presented according to CIM Guidelines for the reporting of Mineral Resources.

Table 1-1: Rönnbäcken Mineral Resource Statement

DEPOSIT	CLASSIFICATION	TONNES (Mt)	Ni-Total %	Sulphide Ni (Ni-AC)	Sulphide Co (Co-AC)	Fe-Total %	Ni-Total ktonnes	Sulphide Ni ktonnes
Rönnbäcksnäset	Measured							
	Indicated	225.4	0.176	0.101	0.003	5.41	397	227
	Measured + Indicated	225.4	0.176	0.101	0.003	5.41	397	227
	Inferred	86.5	0.177	0.100	0.003	5.17	153	86
Vinberget	Measured	28.3	0.188	0.132	0.006	5.19	53	37
	Indicated	23.3	0.183	0.133	0.006	5.14	43	31
	Measured + Indicated	51.5	0.186	0.133	0.006	5.14	96	68
	Inferred	6.8	0.183	0.138	0.007	5.58	12	9
Sundsberget	Measured							
	Indicated	296.9	0.170	0.088	0.003	5.93	505	260
	Measured + Indicated	296.9	0.170	0.088	0.003	5.93	505	260
	Inferred							
TOTAL (Measured & Indicated)	**Measured**	**28.3**	**0.188**	**0.132**	**0.006**	**5.19**	**53**	**37**
	Indicated	**545.6**	**0.173**	**0.095**	**0.003**	**5.68**	**945**	**519**
	Measured + Indicated	**573.9**	**0.174**	**0.097**	**0.003**	**5.66**	**998**	**556**
TOTAL (Inferred)	**Inferred**	**93.2**	**0.177**	**0.103**	**0.003**	**5.55**	**166**	**96**

(1) The effective date of the Mineral Resource Statement for Rönnbäcksnäset and Vinberget is February 25, 2011. Theeffective date of the Mineral Resource Statement for Sundsberget is 28 October 2011.

(2) The Mineral Resource reported for Rönnbäcksnäset, Vinberget and Sundsberget was constrained within a Lerchs-Grossman pit shell defined by a marginal cut-off-grade of 0.031% Ni-AC, a metal price of USD11/lb; slope angles of 50, 48º and 49° respectively; a mining recovery of 95%; a mining dilution of 2.5%; a base mining cost of USD1.35/tonne and an incremental mine operating costs of USD0.07/tonne/10 m below the 450m reference RL and USD0.05/tonne/10m above the 450m reference RL); process operating costs of USD4.96/tonne ore; an effective charge per lb Ni in smelter feed of USD1.14, G&A costs of USD0.40/tonne ore and concentrate transport cost USD0.10/tonne.

1.4 Mine Optimisation, Design and Scheduling

SRK generated pit tonnages and grades for scheduling purposes for each of the three deposits and geotechnical assumptions as presented in the body of this report. The following key parameters were used in the optimisation process.

Table 1-2: Pit optimisation parameters

Description	Unit	Value
Nickel Price	(USD / lb)	9
Base Mining Cost (at Reference Block - RFBK)	(USD / t)	1.35
Mining Recovery	(%)	95
Mining Dilution	(%)	2.5
Ni Processing Recovery	(%)	80
Processing Cost	(USD / t)	4.96

It is currently envisaged that ore production will commence simultaneously at Rönnbäcksnäset and Vinberget, with full production achieved in Year 2. Mining at Sundsberget will then commence in Year 5 and reach full production by the time the Vinberget deposit is depleted. The overall strip ratio is 0.72 (waste:ore).

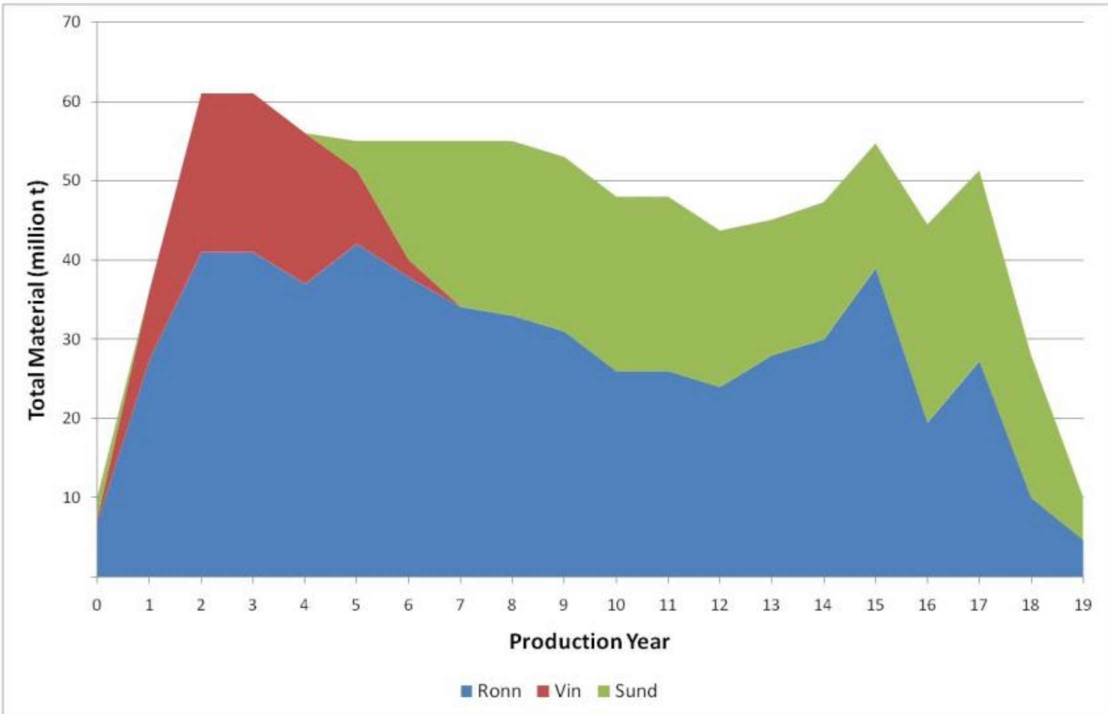

Figure 1-1: Production schedule

The current study assumes contract mining using 700 t hydraulic shovels with 34 m^3 buckets and 225 t haul trucks.

1.5 Metallurgical Testwork

Metallurgical testwork has been undertaken on samples of the nickel sulphide ores from Rönnbäcken to determine the mineralogical, comminution and metallurgical properties of the various mineralised zones within the deposits.

The purpose of the test work program was to develop a process flow sheet that maximises recovery of nickel and cobalt whilst minimising the incorporation of penalty elements (eg magnesia) at the lowest achievable project risk.

Recent test work has established that a nickel feed grade of 0.17% total nickel or 0.1% Sulphide Ni, can produce a concentrate with a grade of 28% at an 80% recovery of Sulphide Ni.

Historical testwork was initially carried out by Boliden during the 1970s where a lab and large scale 4000t pilot program, facilitated by test pit mining, achieved 26 to 34% nickel grade and 67 to 73% sulphide nickel recovery. Typically a primary grind of P_{80} 44µm was required. Autogenous grinding was used, comparing favourably to rod and ball milling.

Phases 1-3 of laboratory based test work conducted by the Company, largely at Outotec Finland's Research Center in Pori, indicated that a closed circuit recovery of 78% was possible yielding a concentrate with a grade of 28% nickel. Composites representing the first few years of production were used and ground to a P_{80} of 50µm. This was demonstrated through minipilot operation which produced a concentrate with a grade of 22% nickel at 80% recovery using the flowsheet in Figure 1-2.

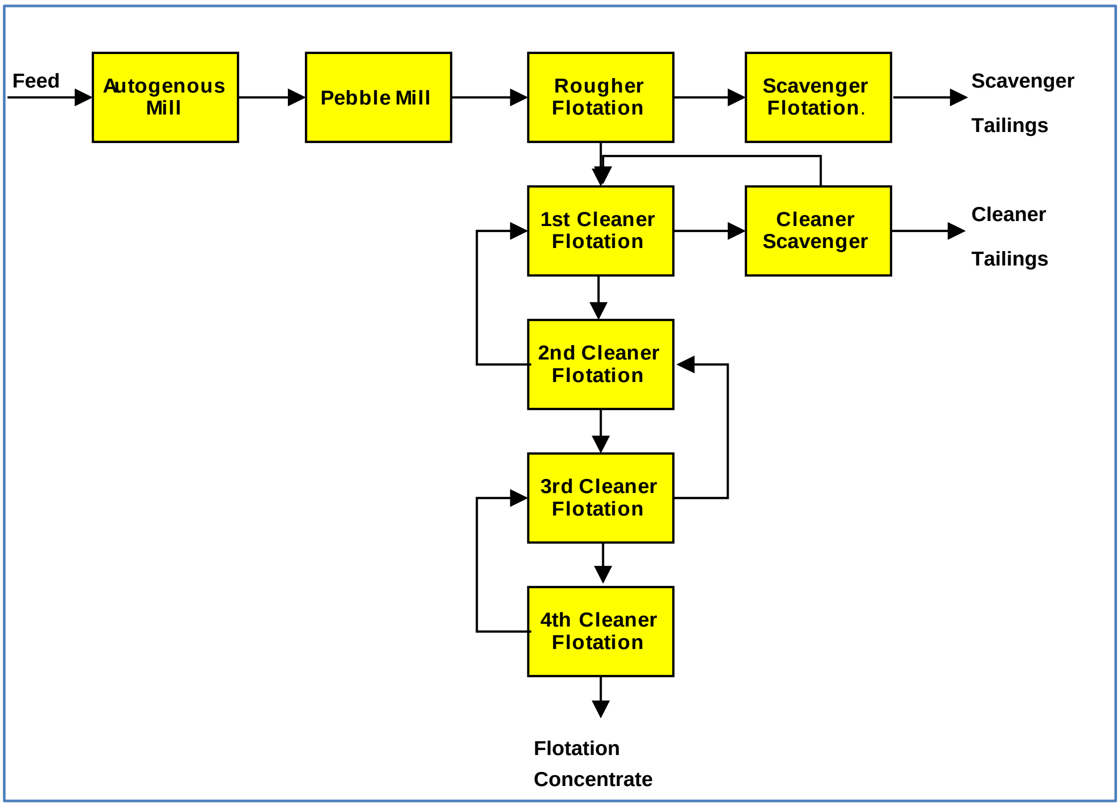

Figure 1-2: Simplified flow sheet

Follow-up laboratory-scale testwork based at Outotec has indicated that concentrates with an increased grade of 28% nickel at 80% sulphide nickel recovery could be produced from the same composite samples used in the minipilot operation. This grade was realised by introducing a new combination of flotation variables, most significantly a new reagent regime, along with slight changes to the flowsheet and, typically, a primary grind of P_{80} 50µm.

Based on the testwork performed, the flowsheet developed for the Rönnbäcken concentrator (Figure 1-3) consists of crushing, grinding, flotation, and dewatering steps, typical of many concentrator operations elsewhere in Sweden and Finland. SRK notes that the engineering company has recently been involved in the development of a similar concentrator in the region and is familiar with the associated costs of such a project. This plant, now operational, and is similar in terms of capital expenditure and process route.

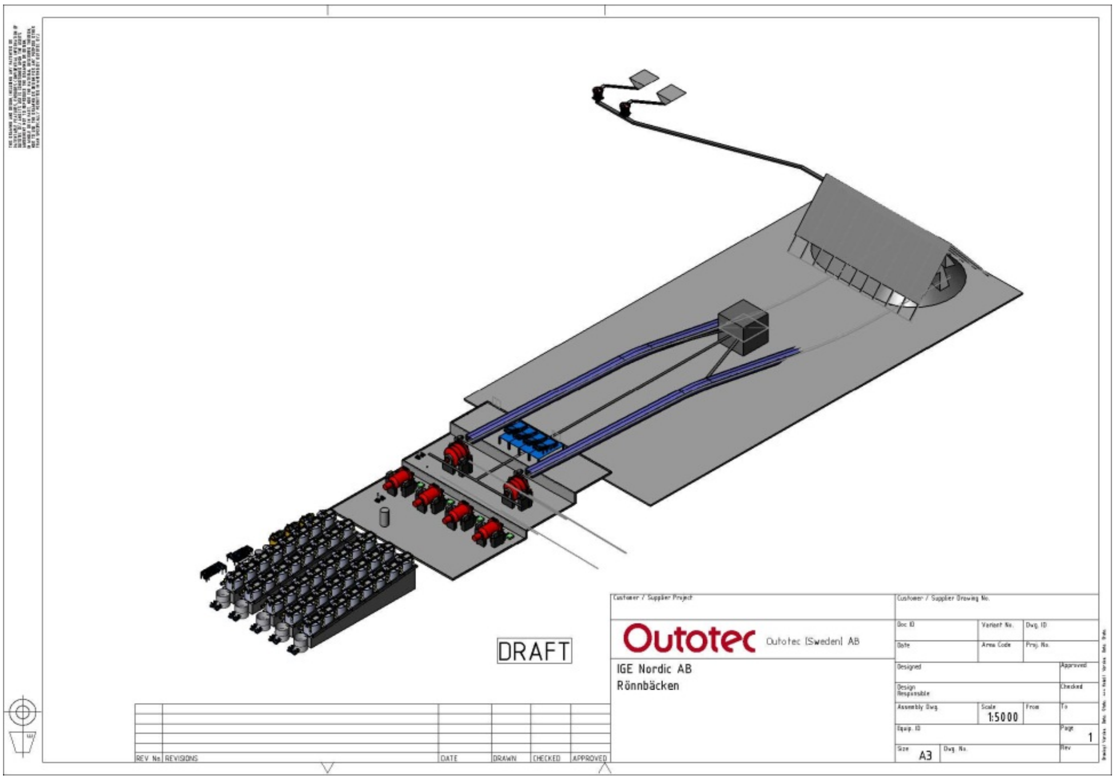

Figure 1-3: Concentrator layout, Oblique View

The conceptual concentrator design is derived from a conventional flowsheet, similar to that in other operations which successfully treat disseminated low-grade nickel sulphides. This design, together with capital and operating cost estimates for the concentrator, has been prepared by Outotec AB (Sweden). The mill will have a capacity of 30 million tonnes per annum (Mtpa) or 3,750 tonnes per hour (tph), and would produce approximately 95,000 tpa of nickel concentrate at 28% Ni.

The processing plant site location has been selected to be in close proximity to the Rönnbäcksnäset deposit and the planned tailings management facility. The plant layout has been chosen to utilize the natural geography and topography of the area (Figure 1-4).

Furher metallurgical testwork was performed by Outotec at GTK's facilities in Finland in October 2011 to evaluate the potential for the recovery of a saleable magnetite concentrate from the nickel flotation tailings stream. Fifteen batch tests were performed on the tailings from the mini-pilot plant work performed in March 2010 to evaluate the recovery and grade of magnetite concentrate that can be produced using flowsheets comprising desliming, magnetic separation, concentrate regrinding, flotation and product classification. The testwork demonstrated that a saleable magnetite concentrate could be produced using multiple stages of low intensity magnetic separation. A magnetite recovery of 90.3% at a grade of 66.2% Fe was achieved in open circuit batch tests. The level of chrome impurity in the concentrate was acceptable at around 2.2%. This represents a mass yield of 5 to 6 % or an annual magnetite concentrate tonnage of 1.6 million tonnes from 30 million tonnes of ROM ore. The concentrate produced was very fine, nominally minus 20 µm, and further testwork is required to optimise this parameter and to establish if it will be necessary to pelletize the product.

Adding a magnetite concentrate circuit to the current plant configuration is estimated to cost US$87 million, which together with a US$12 million increase in working capital, raises the start-up capital expenditure for the Project from US$1,161 to US$1,260 million.

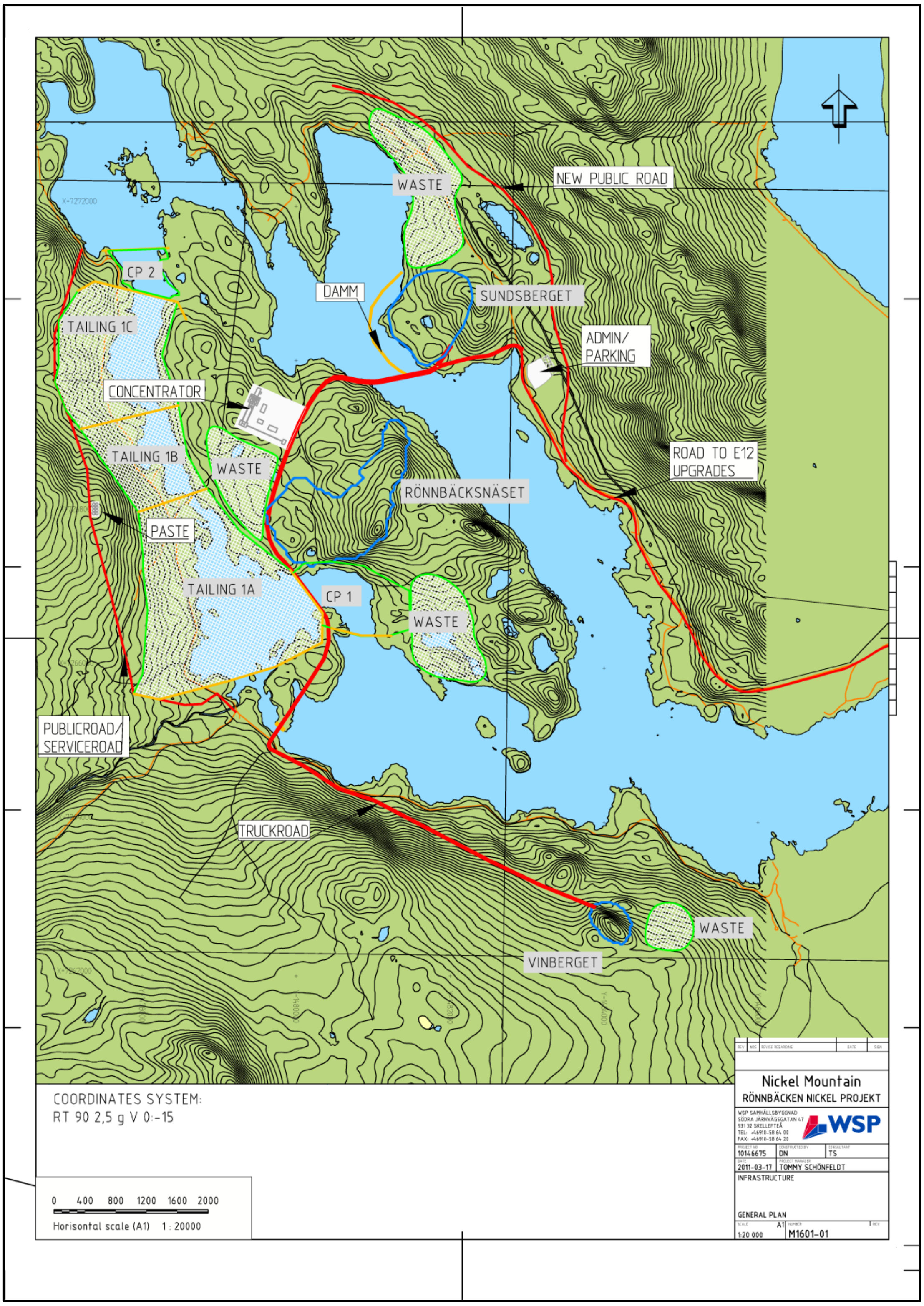

Figure 1-4: Proposed Site Layout

1.6 Project Infrastructure

The key infrastructure required to support the Project as currently envisaged will comprise:

- 14 km of access road upgrading from the E12 highway to the project site. The associated costs will be negotiated with the Swedish Transport Administration;
- 4 km of 144kV power line to tie into the external grid, plus a 160 MW receiving main station 220 V (four transformers), with internal electrical grid and substations to service the process plant, tailings area and mines;
- mine roads connecting to the concentrator, waste dumps and tailings dam, including two causeways across the Gardiken reservoir;
- a 15 ha industrial pad for the process plant site;
- a 2 km coffer dam protecting the Sundsberget pit from the reservoir;
- buildings inclusive of changerooms, offices, restaurants, mechanical and electrical workshops, truck workshops, heated and cold storage facilities; and
- a sewage treatment plant and recycling facilities.

Figure 1-5 illustrates the Project site location relative to local infrastructure with a photograph of the Ajaure hydropower plant as inlay.

Figure 1-5: Rönnbäcken relative to existing infrastructure and a photo of the Ajaure hydropower plant.

1.7 Tailings Management

The proposed design for the tailings management facility (TMF) is to construct a cluster of three cells that will require the construction of four dams, south of the Rönnbäcksnäset pit in Lake Gardiken, a hydro-electric reservoir. Deposition of the tailings will be achieved by spigotting the tailings over the TMF to maximise the storage capacity. Two clarification ponds will be constructed at both ends of the TMF. Tailings will be subject to a thickening process to produce a bulk dry density estimated at 1550 kg/m^3 and a top surface sloping of 4^o, producing a tailings volume of up to 340 Mm^3.

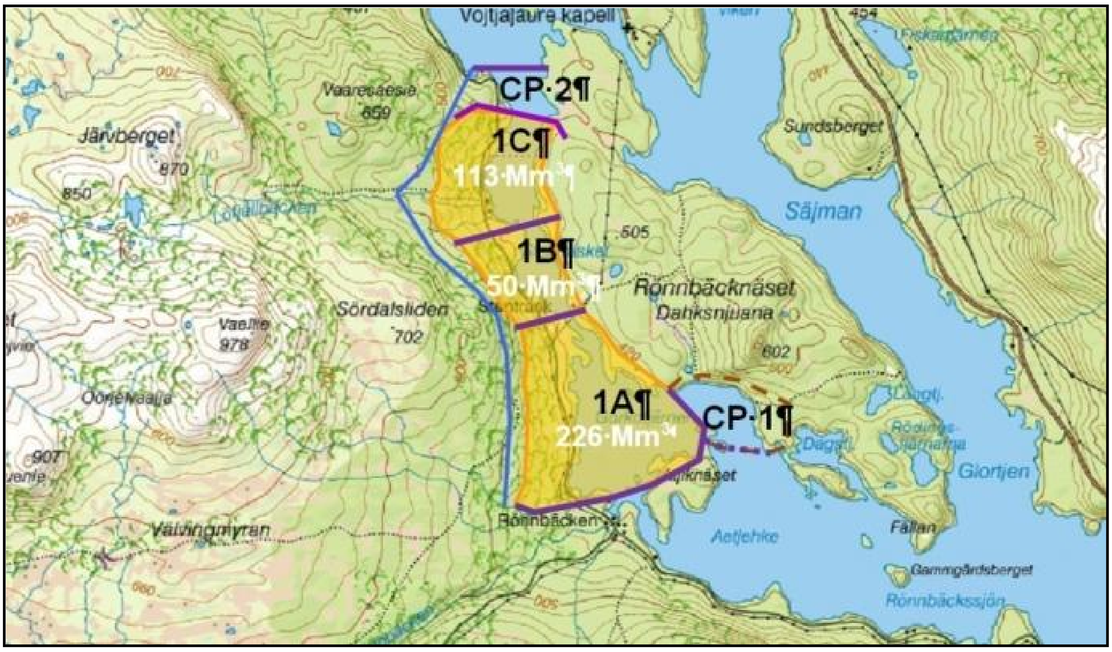

Figure 1-6: Proposed layout of the Project TMF

Preliminary investigations on the waste rock and tailings indicate that there is low potential for development of acid rock drainage, although further testwork is required to confirm this.

1.8 Environmental Studies, Permitting and Social Impacts

There are four types of permits necessary to develop a deposit from the exploration stage to the development stage in Sweden. These are: exploration permits, exploitation concessions, environmental permits and building permits. The Company holds exploration permits for all three deposits. Exploitation concessions have been granted over the Vinberget and Rönnbäcksnäset deposits, supported by environmental impact assessment studies and an application for an exploitation concession over the Sundsberget deposit will be submitted during Q4 2011. Applications for remaining permits, environmental and building permits, will be submitted to the regulatory authorities in Q2 2013 and Q4 2014, respectively, supported by the appropriate studies in each case. SRK notes that final access to land and water areas is a process of negotiation that the Company will need to undertake and must be finalized before filing an application for an environmental permit, which has to be granted before mining operations can begin.

Accepting the level of study and available information, the environmental impacts of the proposed project are not deemed significant. Following cessation of operations, the area is expected to be returned to a prior-to-intervention state except for the presence of pit lakes and new topographic highs from the storage facilities for waste rock and tailings, which shall be rehabilitated.

Social and economic impacts will largely be positive particularly through new job creation, increased economy of the region and increased tax revenue to local authorities. Potential negative impacts stem from loss of land for other uses, e.g. reindeer herding, dwellings, recreational activities, fishing, and hunting. A present, artificial hydro-electric water reservoir will partly be occupied by tailings and embankments resulting in a slight loss of storage capacity, which reduces the ability to keep water volume from summertime to wintertime, with a limited loss of power value (SEK 3M annually) for the producer as power in wintertime is priced higher than in summertime. However, the overall annual power production is not expected to be reduced. Eventual compensation measures will be negotiated with affected parties.

1.9 Capital and Operating Costs

The capital and operating costs estimated as part of this study have been reviewed by SRK and adjusted where appropriate to reflect SRK’s views. These costs total USD 1 668M. SRK notes the following:

- contingencies of between 20% and 25% have been applied to capital cost associated with the process plant and infrastructure (roads, buildings and electrical);
- capital costs have been profiled with roughly 75% of expenditure occurring in pre-production years and the remaining 25% occurring in the first year of production; and
- no capital costs have been assumed for mining equipment.

Figure 20-1 illustrates a breakdown of the envisaged capital expenditure over the life of mine and split between the major cost centres. The total provision for sustaining capital over the LoM is USD 280M.

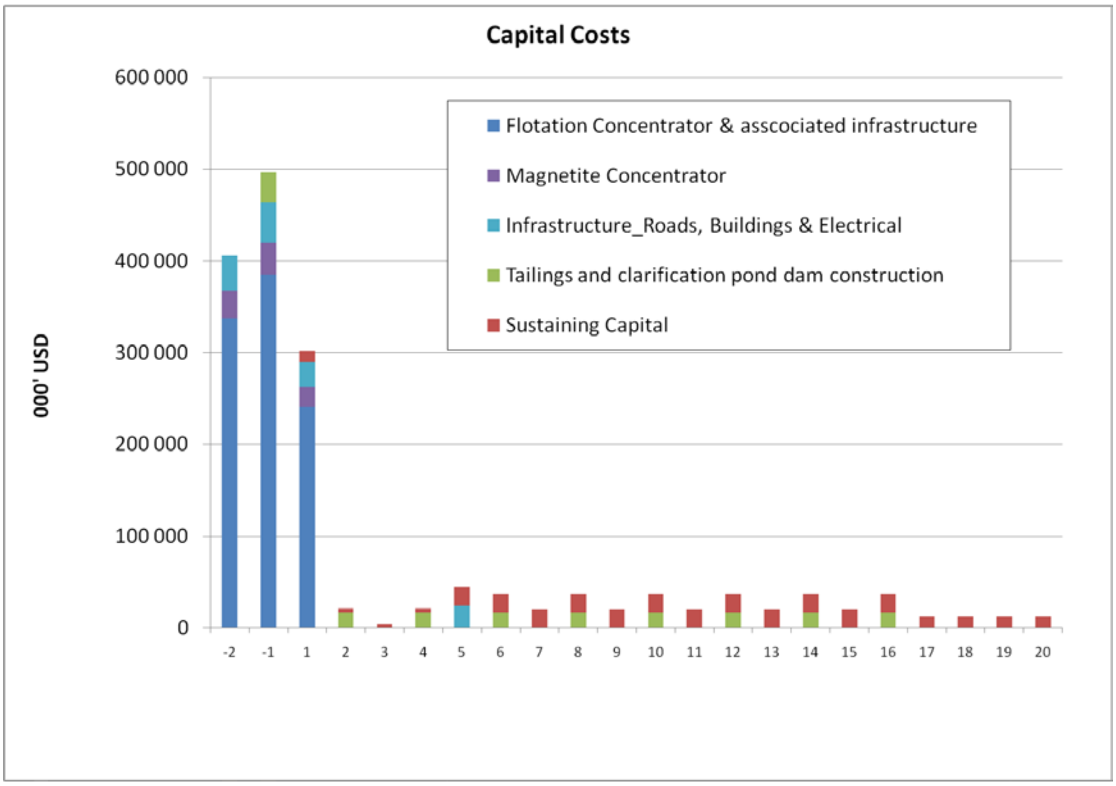

Figure 1-7: Summary of capital cost assumptions by major cost centre

Table 20-2 presents a summary of the capital cost assumptions the Project for start-up capital over Years -2 to 1 and also sustaining and remaining infrastructure capital requird over the remaining LoM.

Table 1-3: Capital cost assumptions

Description	Unit	Value
Flotation Concentrator	(USDM)	962
Magnetite Concentrator	(USDM)	87
Infrastructure	(USDM)	150
Working Capital	(USDM)	59
Start-up Capital[1]	**(USDM)**	**1 258**
Infrastructure (Ongoing)	(USDM)	156
Sustaining	(USDM)	254
Total	**(USDM)**	**1 668**

[1] Includes contingency of 23.5% based on 20% for quoted costs on major equipment items and 25% on general items.

Table 1-4: Operating cost assumptions

	USD/t moved	USD/t milled	USD/lb contained Ni	USD/lb payable Ni
Mining	1.79	3.10	1.61	1.73
Processing	2.89	5.03	2.61	2.80
General & Administration	0.22	0.38	0.20	0.21
On-going rehabilitation	0.07	0.13	0.07	0.07
Operating Cost at Mine Gate[1]	**4.97**	**8.64**	**4.48**	**4.82**
ConcentrateTransport				0.96
TC/RC's				1.20
By-product Credits				-3.42
C1 Cash Cost[2]				**3.55**

[1] Mine Gate operating costs per pound of nickel recovered to concentrate

[2] C1 costs include mining, processing, site admin, transportation, smelting and refining, net of byproduct credits.

The total unit operating costs amount to USD4.97/t of total material mined. The total cash cost is USD3.55/lb Ni, net of both cobalt in the sulphide concentrate and the magnetite concentrate. Net C1 cash costs are illustrated below in Figure 20-2 over the life of mine.

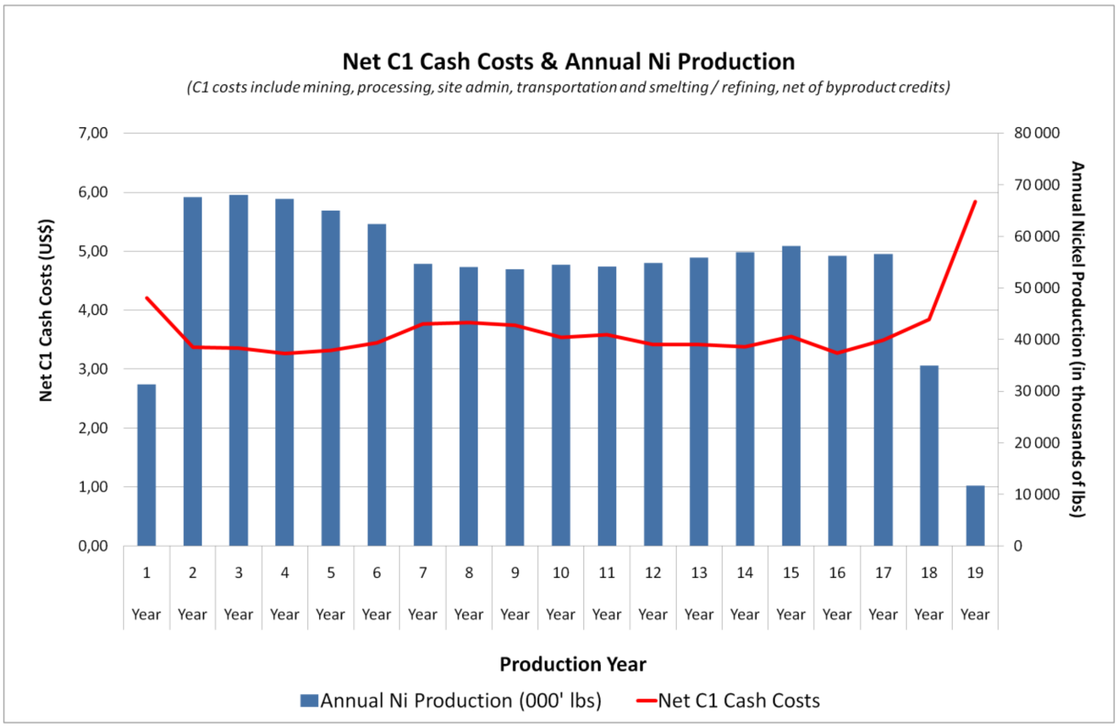

Figure 1-8: Net C1 cash costs over the LoM

1.10 Economic Analysis

SRK has constructed a pre-tax, pre-finance Technical Economic Model (TEM) to derive a Net Present Value (NPV) for the Project. The TEM is based on the technical assumptions developed from work undertaken by SRK and the Company, which SRK has reviewed and adjusted where appropriate.

The economic analysis contained in this report is partially based on Inferred Mineral Resources, and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have mining and economic considerations applied to them and to be categorized as Mineral Reserves. There is no certainty that the production and economic forecasts on which this Preliminary Assessment is based will be realised.

The economic analysis has been undertaken using the US Dollar (USD) as the base currency. Any Swedish Krona (SEK) or Euro (EUR) derived costs have been converted at the exchange rates indicated in the Table 1-5 below, which summarises all of the key financial assumptions made. Table 1-5 to Table 1-7 similarly summarise the technical and cost assumptions made and derived by SRK.

Table 1-5: Economic assumptions. Magnetite Iron Concentrate Prices 65% Fe FOB Mo i Rana (Norway).

Description	Unit	Value
SEK:USD exchange rate	(unit)	8:1
USD:EUR exchange rate	(unit)	1.125:1
Base case discount rate	(%)	8
Base case nickel price	(USD / lb)	9
Base case cobalt price	(USD / lb)	15
Fe price (Year 1)	(USD / t)	110
Fe price (Year 2 & onwards)	(USD / t)	104
LoM	(years)	19

Table 1-6: Physical assumptions

Description	Unit	Value
Total ore mined	(k tonnes)	528 030
Total waste mined	(k tonnes)	379 943
Strip ratio	(w:o)	0.72

Table 1-7: Process, smelting and refining assumptions

Description	Unit	Total
LoM feed tonnage	('000 tonnes)	528 030
Plant through-put per day	(tonnes/day)	90 000
Flotation concentrate (Ni-Co)		
Ni recovery	(%)	80%
Ni concentrate Ni grade	(%)	28%
Co recovery	(%)	70%
Ni concentrate Co grade	(%)	0.90%
LoM Contained Ni	(tonnes)	462 000
LoM Contained Ni	(M lb)	1 018
LoM Contained Co	(tonnes)	13 000
LoM Contained Co	(M lb)	29
LoM Ni concentrate tonnage	(000' tonnes)	1 649
Magnetite concentrate		
Magnetite recovery	(%)	90%
Fe grade	(%)	66%
LoM Magnetite concentrate tonnage	(000' tonnes)	29 000

Figure 1-9 below illustrates Net C1 cash costs over the life of mine.

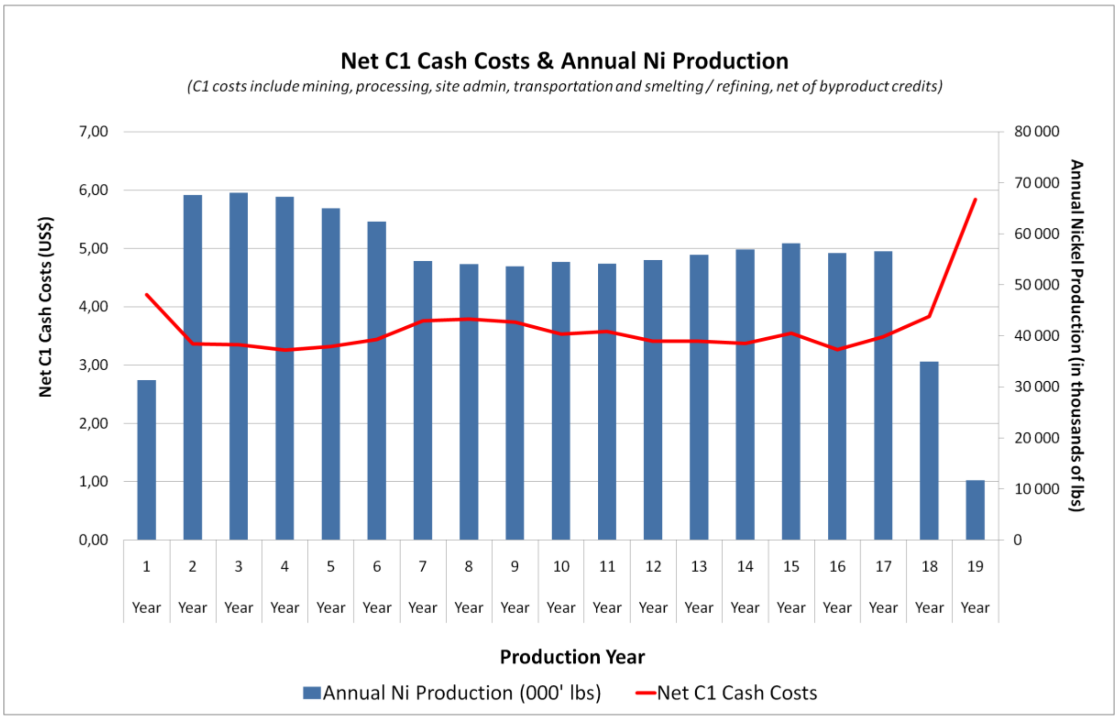

Figure 1-9: Net C1 cash costs over the LoM

SRK's NPV has been derived by the application of Discounted Cash Flow (DCF) techniques to the pre-tax, pre-finance cash flow. In summary, at a Ni price of USD9/lb and an 8% discount rate the Project has an NPV of USD1 045M. A summary of the results of the cash flow modelling and valuation are presented in Table 1-8 and Figure 1-10.

Table 1-8: DCF modelling and valuation

Description	Unit	Value
Ni price	(USD / lb)	9
Net pre-tax cashflow	(USDM)	3 468
Payback period	(Production years)	4.4
Pre-tax pre-finance NPV (8%)	(USDM)	1 045
IRR	(%)	19.9

Figure 1-10 below illustrates the undiscounted net pre-tax cashflow over the LoM. A summary cashflow is also presented in Table 1-9 below.

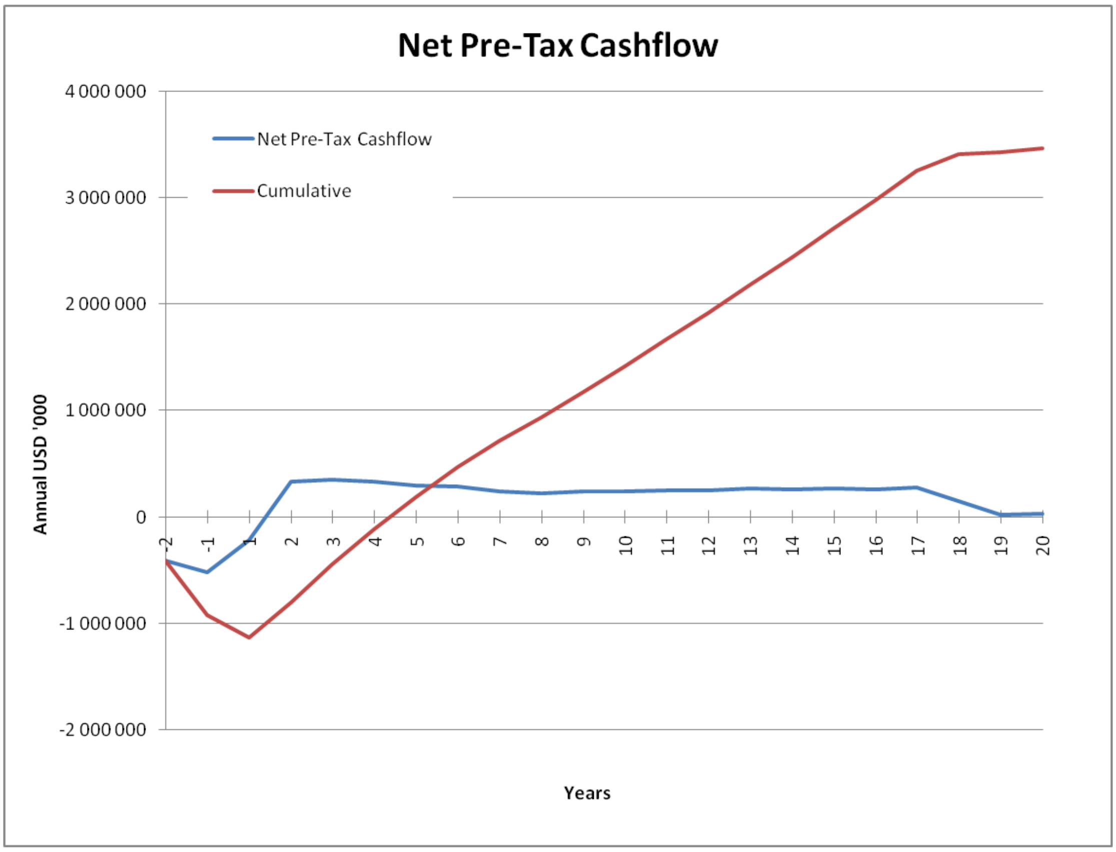

Figure 1-10: Net pre-tax cashflow

Table 1-9: Summary cashflow

SE355 Rönnbäcken TEM	Assumptions	Units	Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20
PHYSICALS																									
Mining																									
Ore tonnes		(000' tonnes)	528 030	0	0	15 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	24 000	9 030	0
Ore grade_Ni AC		(%)	0,109%	0,000%	0,000%	0,119%	0,128%	0,129%	0,127%	0,123%	0,118%	0,103%	0,102%	0,101%	0,103%	0,102%	0,104%	0,106%	0,108%	0,110%	0,106%	0,107%	0,083%	0,074%	0,000%
Ore grade_Co		(%)	0,004%	0,000%	0,000%	0,005%	0,004%	0,005%	0,004%	0,004%	0,003%	0,003%	0,003%	0,003%	0,003%	0,002%	0,002%	0,002%	0,004%	0,004%	0,004%	0,004%	0,004%	0,005%	0,000%
Ore grade_Fe		(%)	4%	0%	0%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	0%
Mass waste		(000' tonnes)	379 943	0	0	21 200	31 000	31 000	26 000	25 000	25 000	25 000	25 000	23 000	18 000	18 000	13 715	15 085	17 300	24 700	14 500	21 293	4 000	1 150	0
Total mass		(000' tonnes)	907 973	0	0	36 200	61 000	61 000	56 000	55 000	55 000	55 000	55 000	53 000	48 000	48 000	43 715	45 085	47 300	54 700	44 500	51 293	28 000	10 180	0
Strip ratio		(w:o)	0,72	0,00	0,00	1,41	1,03	1,03	0,87	0,83	0,83	0,83	0,83	0,77	0,60	0,60	0,46	0,50	0,58	0,82	0,48	0,71	0,17	0,13	0,00
Overburden removed		(000' tonnes)	10 300	0	10 300	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing																									
Ore tonnes		(000' tonnes)	528 030	0	0	15 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	24 000	9 030	0
Ni-Co flotation circuit																									
Ni recovery		(%)	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%
Co recovery		(%)	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%
Ni grade in concentrate		(%)	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%
Co grade in concentrate		(%)	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%
Contained Ni		(000' tonnes)	461,8	0,0	0,0	14,2	30,7	30,8	30,5	29,5	28,3	24,8	24,5	24,3	24,7	24,6	24,9	25,3	25,8	26,4	25,5	25,7	15,9	5,3	0,0
Contained Co		(000' tonnes)	13,1	0,0	0,0	0,5	0,9	1,0	0,9	0,8	0,7	0,6	0,6	0,6	0,6	0,5	0,5	0,5	0,8	0,8	0,9	0,9	0,6	0,3	0,0
Concentrate tonnage		(000' tonnes)	1 649,1	0,0	0,0	50,8	109,5	110,2	108,9	105,3	101,1	88,5	87,6	86,9	88,3	87,8	88,8	90,5	92,2	94,1	91,1	91,7	56,7	19,0	0,0
Magnetite circuit																									
Fe recovery		(%)	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Mass yield		(%)	5,5%	0,0%	0,0%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	0,0%
Concentrate tonnes		(000' tonnes)	28 802	0	0	818	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 309	493	0
Concentrate grade		(%)	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%
REVENUE																									
Commodity price_Ni		(USD/lb)	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9
Commodity price_Co		(USD/lb)	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15
Commodity price_Fe		(USD/t)	105	119	115	110	104	104	104	104	104	104	104	104	104	104	104	104	104	104	104	104	104	104	104
Gross revenue_Ni		(MUSD)	8 521	0	0	262	566	569	563	544	522	457	453	449	456	453	459	468	477	486	471	474	293	98	0
Gross revenue_Co		(MUSD)	239	0	0	9	16	18	16	15	12	11	11	11	11	9	9	9	14	15	17	17	11	5	0
Gross revenue_Fe		(MUSD)	3 000	0	0	90	170	170	170	170	170	170	170	170	170	170	170	170	170	170	170	170	136	51	0
Total gross revenue		(MUSD)	11 760	0	0	361	752	757	749	729	705	639	634	631	638	633	638	647	660	672	658	661	440	155	0
Total charges (TCRC's)		(MUSD)	-1 137	0	0	-35	-76	-76	-75	-73	-69	-61	-60	-60	-61	-60	-61	-62	-64	-65	-63	-64	-40	-14	0
Total royalty		(MUSD)	-24	0	0	-1	-2	-2	-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	0	0
Net Revenue		(MUSD)	10 599	0	0	325	675	680	673	655	634	577	573	570	576	572	576	584	595	605	593	595	400	141	0
OPERATING COSTS																									
Mining		(MUSD)	-1 639	0	-18	-65	-108	-108	-99	-97	-97	-97	-97	-94	-85	-85	-78	-80	-84	-97	-79	-92	-56	-23	0
Processing		(MUSD)	-2 654	0	0	-82	-150	-150	-150	-150	-150	-150	-150	-150	-150	-150	-150	-150	-150	-150	-150	-150	-123	-55	0
Concentrate Transport		(MUSD)	-906	0	0	-26	-53	-53	-53	-52	-52	-51	-51	-51	-51	-51	-51	-51	-51	-52	-51	-51	-40	-15	0
G&A		(MUSD)	-200	0	0	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	0
Closure		(MUSD)	-67	0	0	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-1
Employment grant total		(MUSD)	3	0	1	1	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL OPERATING EXPENDITURE		(MUSD)	-5 463	0	-17	-186	-323	-324	-315	-313	-313	-312	-312	-309	-300	-300	-293	-295	-299	-312	-294	-307	-232	-107	-1
TOTAL CASH COST NET OF BYPRODUCT CREDITS		(USD / lb Ni)	3,55	0,00	0,00	4,21	3,37	3,35	3,26	3,32	3,44	3,76	3,79	3,74	3,53	3,58	3,41	3,42	3,38	3,55	3,27	3,49	3,84	5,84	0,00
CAPITAL COSTS																									
Flotation Circuit (incl 22% contingency)		(MUSD)	-962	-337	-385	-241	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Magnetite Circuit		(MUSD)	-87	-30	-35	-22	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Infrastructure		(MUSD)	-305	-41	-80	-29	-17	0	-17	-24	-17	0	-17	0	-17	0	-17	0	-17	0	-17	0	0	0	0
Sustaining Capital		(MUSD)	-313	0	0	-12	-4	-4	-4	-20	-20	-20	-20	-20	-20	-20	-20	-20	-20	-20	-20	-12	-12	-12	-12
Working Capital		(MUSD)	0	0	0	-47	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	47
TOTAL CAPITAL EXPENDITURE		(MUSD)	-1 668	-408	-499	-350	-21	-4	-21	-44	-37	-20	-37	-20	-37	-20	-37	-20	-37	-20	-37	-12	-12	-12	34
CASHFLOW, PRE-TAX, PRE-FINANCE																									
Net Revenue		(MUSD)	10 599	0	0	325	675	680	673	655	634	577	573	570	576	572	576	584	595	605	593	595	400	141	0
Operating Expenditure		(MUSD)	-5 463	0	-17	-186	-323	-324	-315	-313	-313	-312	-312	-309	-300	-300	-293	-295	-299	-312	-294	-307	-232	-107	-1
Capital Expenditure		(MUSD)	-1 668	-408	-499	-350	-21	-4	-21	-44	-37	-20	-37	-20	-37	-20	-37	-20	-37	-20	-37	-12	-12	-12	34
Net Pre-Tax Cashflow		(MUSD)	3 468	-408	-516	-211	332	352	337	298	285	245	224	241	239	252	247	269	260	273	262	277	155	22	34
NPV	8%	(MUSD)	1 045																						
IRR	8%	(%)	19,9%																						

Figure 1-11 below presents single parameter NPV sensitivities at an 8% discount rate for commodity price, operating costs, capital costs, SEK:USD exchange rate and Ni recovery. In addition, Table 1-10 presents the sensitivity of the NPV to various nickel price assumptions.

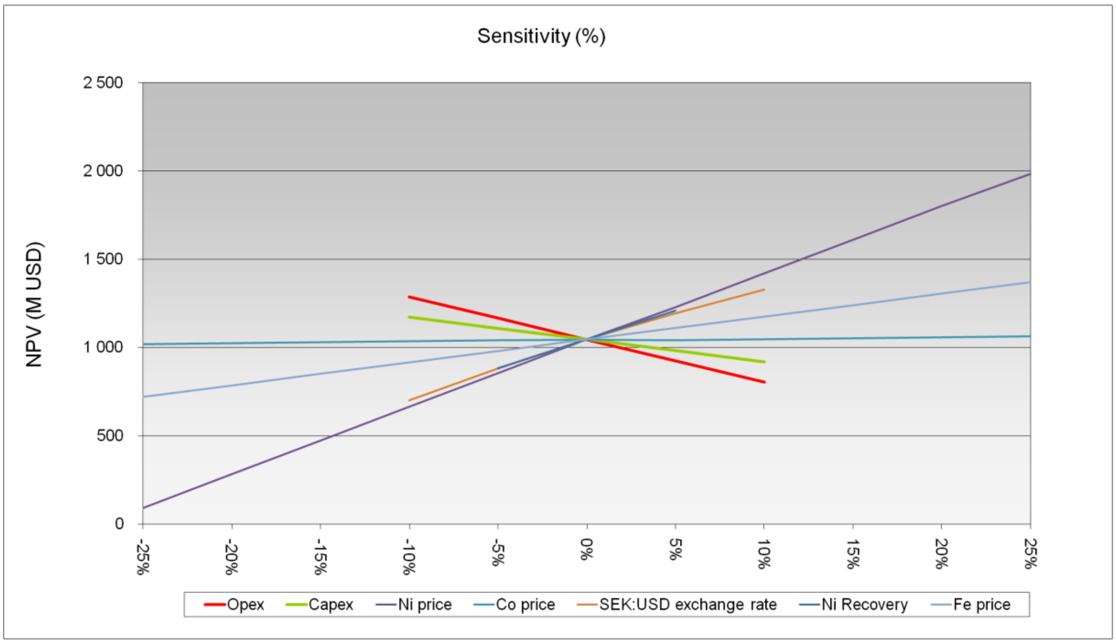

Figure 1-11: NPV sensitivity to multiple variables

Table 1-10: NPV sensitivity under different nickel price scenarios

Nickel Price (USD / lb)								
Description	Unit	7	8	9 base case	10	11	12	13
Net pre-tax cashflow	(MUSD)	1,577	2,522	3,467	4,393	5,338	6,264	7,208
NPV (@ 8% Discount Rate)	(MUSD)	195	620	1,045	1,461	1,885	2,301	2,726
IRR	(%)	10.5	15.4	19.9	24.0	27.9	31.6	35.2
Payback	(Production Years)	7.5	5.4	4.4	3.8	3.3	3.0	2.7

1.11 Interpretation and Conclusions

SRK understands that the Company is proposing to undertake a pre-feasibility study commencing in Q1 2012, with completion expected by late Q1 2013. The budget for the study is some USD8.5M, excluding overheads. Details of these costs are presented in Table 1-11.

Subject to the results of the pre-feasibility study, the Company expects to commence a full feasibility study in Q2 2013 for completion towards the end of 2014.

Table 1-11: PFS Budget (*excluding Company overheads)

	Total PFS Budget*
Planning	$100,000
Geology & Resources	1,440,000
Mining	690,000
Processing	2,300,000
Infrastructure/Services	840,000
Waste Rock/Tailings Mgmt.	380,000
Water Management	430,000
Social/Environment	281,000
Manpower	20,000
Financial	510,000
Contingency	1,503,000
Total Project Budget	**$8,494,000**

1.12 Recommendations

SRK has made several recommendations regarding work that it considers should be undertaken as part of the planned pre-feasibility study. This work is detailed later in this report. SRK is confident that should this work be included then the technical and economic viability of the Project will be properly assessed.

Most notably SRK has recommended that:-

- Further testwork be undertaken to improve and optimise the metallurgical performance of the nickel recovery circuit and to investigate the reduction of the level of impurities, especially MgO.
- Further laboratory scale testwork and mini-pilot plant testing be undertaken to optimise the metallurgical performance of the magnetite recovery circuit and and to investigate the effect of a coarser concentrate regrind size on both recovery and grade of the final magnetite concentrate.

Certainly, in SRK's opinion, the commissioning of a pre-feasibility study is justified by the potential of the Project and the timing and budgets proposed for this by the Company are reasonable given the work planned to be undertaken which incorporates all of the work SRK has recommded.

2 INTRODUCTION

This report comprises a preliminary economic assessment (PEA) of the Rönnbäcken Nickel Project (Rönnbäcken or the Project) by SRK Consulting (Sweden) AB (SRK) and has been prepared on behalf of Nickel Mountain Resources AB (Nickel Mountain or the Company), the parent company of Nickel Mountain AB which is the project owner. The Company's primary focus is development of the Project, which comprises three nickel deposits, Rönnbäcksnäset, Vinberget and Sundsberget, all located in the northwest part of Sweden.

SRK has prepared independent Mineral Resource estimates for Rönnbäcksnäset, Vinberget and Sundsberget. In addition SRK has reviewed all other technical work completed on the Project by the Company and its other contractors and consultants to a sufficient level to enable SRK to present its own opinions on the Project and to derive an audited NPV for this.

The Project is at a conceptual stage but it is currently envisaged that it will comprise three open pit mines feeding a single processing operation producing both a nickel sulphide concentrate and a magnetite concentrate through conventional flotation and magnetic separation. This report is an update of the previous PEA prepared by SRK in April 2011.

The work undertaken by SRK in compiling this report has been managed by Mr Johan Bradley (CGeol FGS, EurGeol) and reviewed by Dr Mike Armitage (CGeol FGS, CEng MIoM3). Both Mr Johan Bradley and Dr Armitage are Qualified Persons (QP) as defined in National Instrument 43-101 of the Canadian Securities Administrators (NI 43-101).

The details of the various contributing authors and their respective areas of technical responsibility are presented in Table 2-1 below.

Table 2-1: Contibuting authors and respective area of technical responsibility

Contributing Author	Area of technical responsibility	Sections of this report
Johan Bradley (QP)	Geology and Technical Economic Model	1 to 11, 21 & 22
Howard Baker (QP)	Resource Estimation	12 & 14
Chris Reardon	Mine Optimisation, Design and Scheduling	16
Neil Marshall	Geotechnical assumptions	16.2
Dr David Pattinson (QP)	Process Metallurgy, Infrastructure, Markets and Concentrate Transport	17 & 18
Michel Noël	Tailings Dam Design and Waste Rock Dumps	18.1
Matt Dey	Acid Rock Drainage and Metal Leaching	18.2
James Bellin	Hydrology and Water Management	18.3
Fiona Cessford	Environmental, Permitting and Social Impacts	20
Dr Mike Armitage (QP)	Reviewer	All sections

For the purposes of this report, the following persons act as QP: Johan Bradley, Howard

Baker, Dr David Pattinson, and Dr Mike Armitage. Appropriate QP certificates for these individuals accompany this report.

As part of this work, SRK has undertaken site visits and made first hand observations of the core, collection and core logging procedures employed and reviewed all the Project data available. The most recent site visit was undertaken by Mr Bradley on 7 February, 2011.

3 RELIANCE ON OTHER EXPERTS

Sections 4 to 6 of this report are to some degree extracts from the Company's report entitled "Technical Report On The Preliminary Assessment Of Rönnbäcken Nickel Project, Sweden, dated 6th November 2009" (the Scott Wilson PA Report), which was prepared by Jason J. Cox, Wayne W. Valliant, and Kevin C. Scott of Scott Wilson Roscoe Postle Associates Inc. on behalf of IGE Nordic AB (IGE), owners of the Project at the time and parent company of Nickel Mountain AB.

The opinions and conclusions presented in the Scott Wilson PA Report are based largely on information and technical reports provided to the authors by the Company and its consultants. The additional information reviewed in preparing this report has also largely been provided directly by the Company and its associated consultants, contractors and business partners.

Notwithstanding the above comment, SRK has conducted face to face meetings with those consultants responsible for certain technical aspects of the Project. This included the exploration data capture undertaken by Company staff, mining and infrastructure aspects undertaken by Rolf Ritzén of Ritzén Consult, environmental and social impact assessments undertaken by Per Broman of Per B Consult and tailings management facility design undertaken by Tom Lundgren of Ambiental.

SRK has also confirmed that the mineral resources reported herein are within the exploration permit boundaries given below and that the exploration permits and exploitation concessions as presented by the Company reflect the publicly available information at the Mining Inspectorate of Sweden. SRK has not, however, conducted any legal due diligence on the ownership of the exploration permits or exploitation concessions themselves.

4 PROPERTY DESCRIPTION AND LOCATION

The Rönnbäcken Nickel Project is located 40 km by road south-southeast of Tärnaby, Storuman Municipality, Västerbotten County, as illustrated in Figure 4-1 and Figure 4-2. The Rönnbäcken K nr 1 exploitation concession is on Vinberget on the mainland south of Lake Gardiken. The Rönnbäcken K nr 2 exploitation concession is located on what now is an island, Rönnbäcksnäset, in Lake Gardiken. The island was created in 1963 when a hydro power station was built and raised the water levels. SRK understands that the Company intends to submit an application for an additional exploitation concession to cover the Sundsberget deposit on the mainland, directly to the north of Rönnbäcksnäset, on a hill called Sundsberget. The properties are centred at approximately:

RT 90 2.5 gon v; 148200E, 726600N

SWEREF 99 lat long (WGS84); north latitude 65°29’43”; west longitude 15°24’58”

4.1 Property Description

The Project currently comprises three discrete deposits: Rönnbäcksnäset, Vinberget and Sundsberget, located within and surrounded by two exploitation concessions and 7 contiguous exploration permits, see Figure 4-2 below. The Vinberget deposit is located within the Rönnbäcksjön nr 1 exploration permit and is covered by the Rönnbäcken K nr 1 exploitation concession. The Rönnbäcksnäset deposit is located within the Rönnbäcksjön nr 8 exploration permit and covered by the Rönnbäcken K nr 2 exploitation concession. The Sundsberget deposit is located within the Rönnbäcksjön nr 7 exploration permit.

Notably, the Geological Survey of Sweden (SGU) classified the Rönnbäcken nickel deposits as “an Area of National Interest for Mineral Extraction” on 25 August 2010. Deposits of national interest are assessed and selected by SGU with reference to certain criteria relating to, for example, community development and emergency supply preparedness. Chapter 3, Section 7, paragraph 2, of the Environmental Code states that for such areas, the extraction interest shall be protected against measures that may be prejudicial to extraction.

Figure 4-1: Rönnbäcken property location in Scandinavia

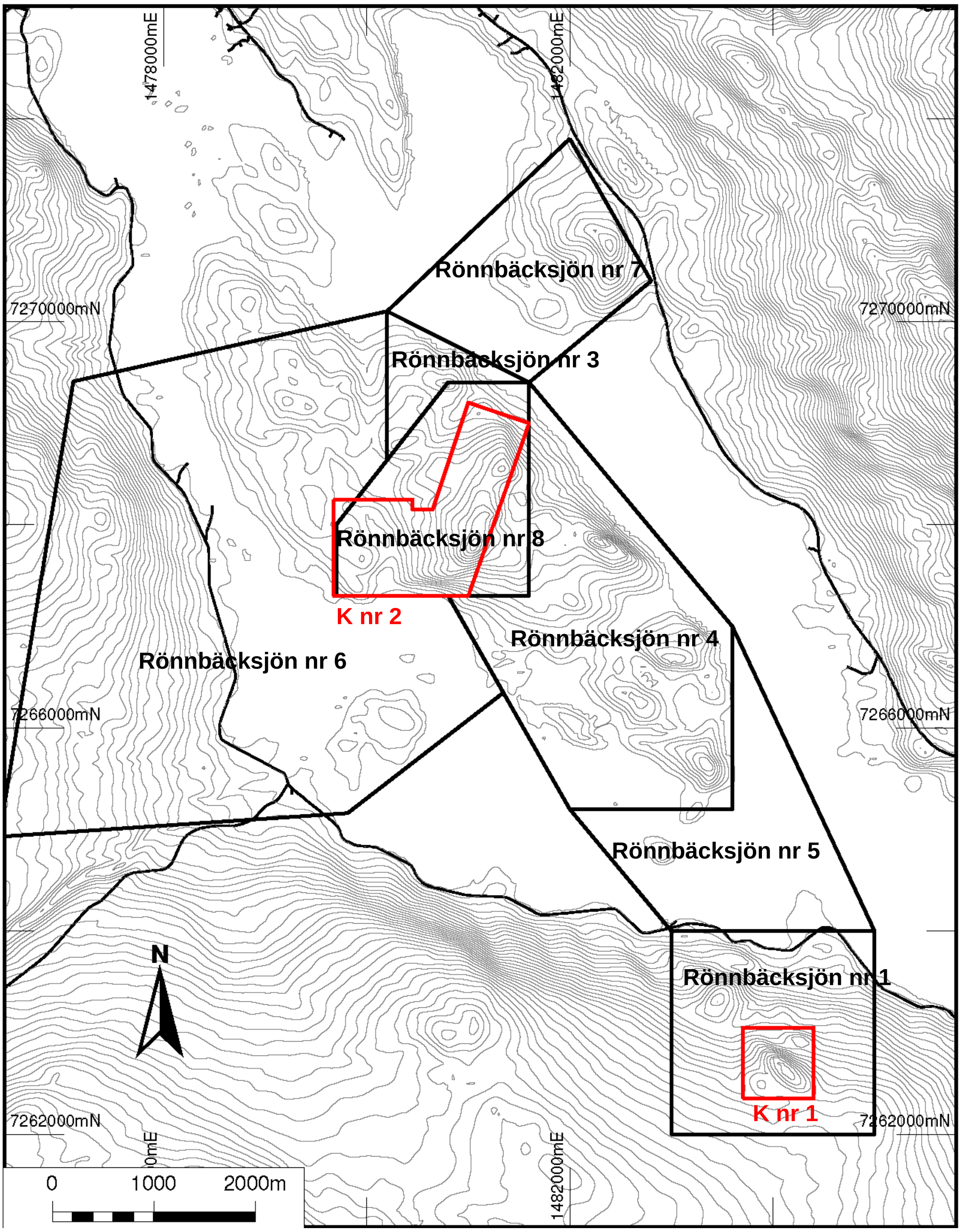

Figure 4-2: Rönnbäcken exploitation concessions (red polygons) and exploration permits (black polygons).

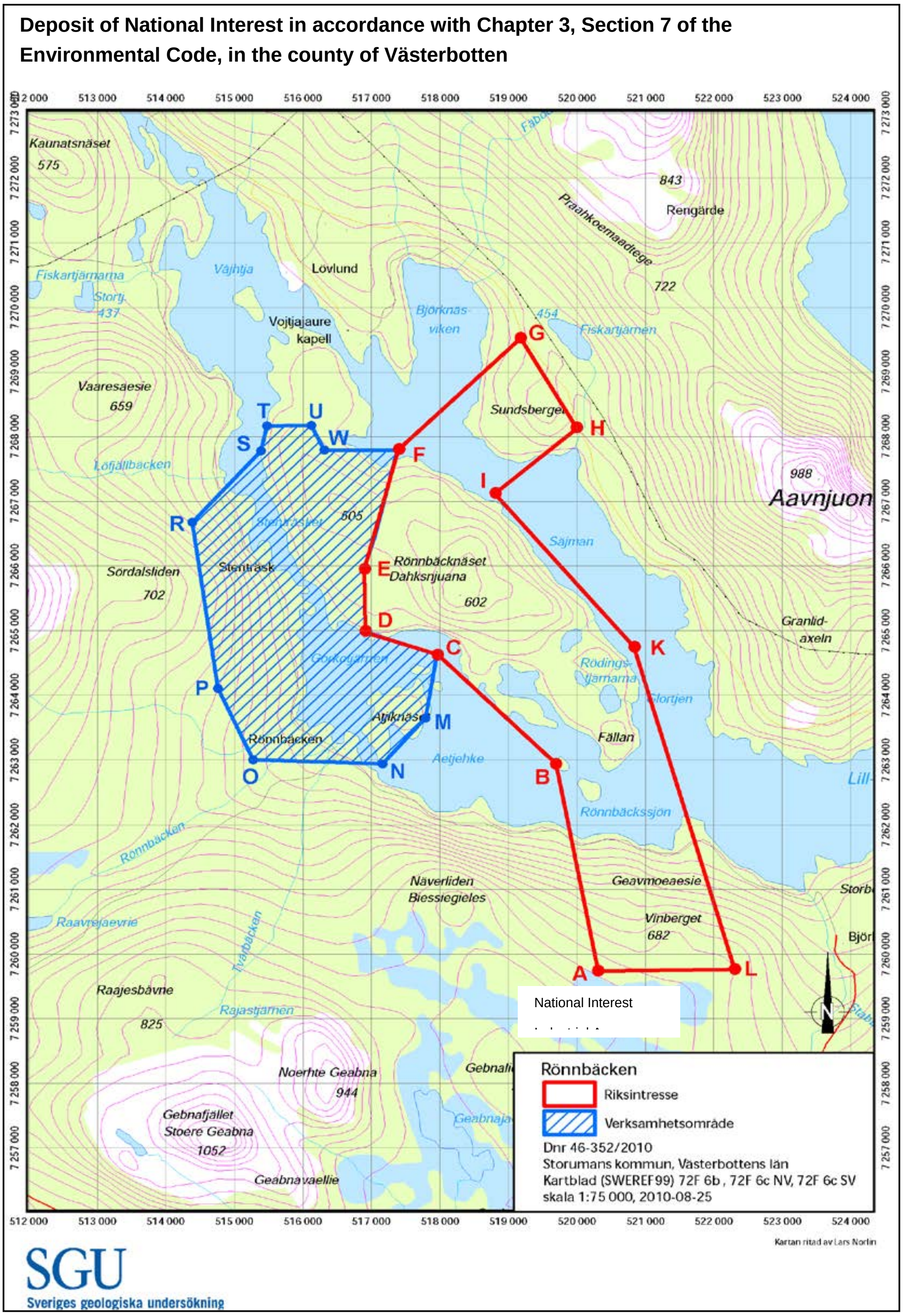

Figure 4-3: Deposit of National Interest in accordance with Chapter 3, Section 7 of the Environmental Code, in the county of Västerbotten

4.2 Property Ownership

There are four types of permits necessary to develop a deposit from the exploration stage to the development stage in Sweden. These are: exploration permits, exploitation concessions, environmental permits, and building permits. For the purpose of this report, the exploration permits and exploitation concessions are all that are required to provide the Company with exclusive mineral rights to the properties in question. Notwithstanding this, SRK notes that final access to land and water areas is a process of negotiation that the Company will need to undertake and must be finalized before filing an application for an environmental permit. No estimate of the cost of this has been included in the TEM presented later in this report. However, given the low population density and current land use in the Project area, SRK does not anticipate these costs to be material, although further study will be necessary, particularly with regard to reindeer husbandry (see also Section 20.1.8)

On June 2, 2010, the Company and IGE Resources AB entered into an agreement with Mitchell River Group Pty Ltd. ("MRG") of Australia (the "MRG Agreement") to form a strategic partnership with MRG of Australia. MRG has an experienced technical team that has conducted the work of resource evaluation, metallurgy, permitting, feasibility studies and project management for the developments of the Sally Malay nickel project in Western Australia, the Munali nickel project in Zambia and the Santa Rita nickel project in Brazil.

Pursuant to the MRG Agreement, MRG agreed to provide experienced personnel, systems and technical resources for the development of the Rönnbäcken Nickel Project for an initial term of 18 months, commencing June 2010 and in return MRG received an option to acquire a 10% interest in the Project (the "Rönnbäcken Option") for an agreed upon cash payment. Pursuant to the MRG Agreement, MRG was entitled to accrue costs incurred during the term of the MRG Agreement and offset such costs against the cash payment. Such costs not paid by the Company would accrue as a loan to the Company to a maximum amount of US$500,000 and would either be offset against the cash payment to exercise the option or be repaid by the Company. To secure this loan, the Company agreed to grant MRG a fixed and floating charge over all assets of the Company (subsequently amended on March 21, 2011 to 10% of the assets of Nickel Mountain AB). Pursuant to the terms of the MRG Agreement, a steering committee was established comprised of experienced personnel from MRG and an equal number of representatives of the Company, and in consultation with the Company supervises the scope of work and provides strategic and technical advice to the Company. MRG is a resource investment group that specialises in the evaluation and development of base metal projects. MRG provides funding and management for early stage resource projects, and has a strong value development track record in nickel projects. Overall control of the Rönnbäcken Nickel Project remains with the Company.

The Rönnbäcken property consists of seven granted exploration permits (Rönnbäcksjön nr 1 and Rönnbäcksjön nr 3 to nr 8), totalling 3,718 ha. Exploration permits are granted initially for three years, with possible extensions of up to 15 years. Annual fees for the first three year period are SEK4, SEK6, and SEK10/ha in each successive year. Table 4-1 summarizes the status of the Project exploration permits.

Table 4-1: Exploration permit summary table

Exploration Permit Number	Permit Name	Grant Date	Expiry Date	Area (ha)
2005:134	Rönnbäcksjön nr 1	2005-08-01	2014-08-01	351
2007:339	Rönnbäcksjön nr 3	2007-12-11	2013-12-11	72
2007:340	Rönnbäcksjön nr 4	2007-12-11	2013-12-11	642
2009:104	Rönnbäcksjön nr 5	2009-06-11	2012-06-11	342
2009:126	Rönnbäcksjön nr 6	2009-06-25	2012-06-25	1683
2009:161	Rönnbäcksjön nr 7	2009-10-01	2012-10-01	306
2010:163	Rönnbäcksjön nr 8	2010-11-04	2013-11-04	322

Applications for exploitation concessions for Vinberget (Rönnbäcken K nr 1) and Rönnbäcksnäset (Rönnbäcken K nr 2) were granted by the Swedish Mining Inspector on 23 June 2010, see Figure 4-2 above. An exploitation concession (Bearbetningskoncession) gives the holder the right to exploit a proven, extractable mineral deposit for a period of 25 years, which may be extended. The exploitation concession is the next step in mine permitting after the granting of an exploration permit.

Table 4-2: Exploitation concession summary details

Exploitation concession	Application Date	Status	Area (ha)
Rönnbäcken K nr 1	2010-02-12	Granted 2010-06-23	49.0
Rönnbäcken K nr 2	2010-02-12	Granted 2010-06-23	195.75

There is no requirement to legally survey the boundaries of exploitation concessions in Sweden; instead boundaries are assigned Swedish RT90 coordinates by the Inspector of Mines on granting. The coordinates, in the Swedish RT90 system, of the exploitation concessions are presented in Table 4-3 below.

Table 4-3: Exploitation concession vertices, 2010-02-12(Projection RT 90 2.5 gon v)

Exploitation Concession Name	Vertex	Northing	Easting
Rönnbäcken K nr 1	1	7262350	1483700
	2	7262350	1484400
	3	7263050	1484400
	4	7263050	1483700
Rönnbäcken K nr 2	1	7268250	1479675
	2	7268250	1480450
	3	7268150	1480450
	4	7268150	1480650
	5	7269200	1481000
	6	7269000	1481600
	7	7267300	1481000
	8	7267300	1479675

Figure 4-4 shows the exploration permits discussed above in relation to the optimised pits generated to constrain the mineral resource presented later in this report. Figure 4-4 clearly demonstrates that the modelled mineralisation lies within the permit boundaries.

Figure 4-4: Rönnbäcken Project pit shells and boundaries for exploration permits (red polygons) and exploitation concession (yellow polygons)

4.3 Additional Permits and Payments

SRK is not aware of any special royalties in addition to the 0.20% royalties prescribed by the Swedish Mining Act, back-in-rights, payments or any other agreements associated with the Rönnbäcken Project.

SRK is not aware of any environmental liabilities associated with the Rönnbäcken project.

4.4 Surface Rights

For the purposes of this report, all surface rights are covered by the Rönnbäcksjön exploration permits and Rönnbäcken exploitation concessions as detailed in Table 4-1 and Table 4-3 above. Additional permitting is required prior to commencement of mining operations and a discussion of these is presented in Section 20, below. In addition, a discussion of potential tailings storage area, potential waste storage area and potential processing plant sites are included in Section 18 below.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Property Access

5.1.1 International Access

The nearest airport to the Project is Hemavan Tärnaby Airport in Hemavan, 15 km northwest of Tärnaby and roughly 40 km from the project area. The airport has daily flights to and from Stockholm depending on the season (Figure 5-1).

Figure 5-1: Hemavan-Tärnaby Airport

5.1.2 Regional and Local Access

The Project can be accessed from both north and south from highway E12. From the north via the town Tärnaby (E12), it is necessary to travel west for 9 km and then on gravel roads approximately 31 km passing the community of Ängesdal on the way to the project site. From the E12 in the south, over the Ajaure hydro dam, it is approximately 14 km of gravel road to the Project site.

5.2 Physiography and Climate

5.2.1 Physiography

The elevation within the exploration permits ranges from 395 metres above sea level (masl) to 666 masl; that is, a difference of about 270 m from the lowest to the highest point. The exploration permits are in low mountain terrain which for the most part is covered by coniferous forest, principally spruce and pine trees, except for some higher areas in which birch trees predominate.

Lake Gardiken surrounds the island of Rönnbäcksnäset and is controlled by Gardiken hydro power station which is located at the Lake Gardiken outlet in Umeälven approximately 300 km from the river mouth. Water levels throughout the year may vary by as much as 20 m.

The continental glaciation movement direction in the area is from the southeast. The till cover in the exploration permit is generally thin, but can be up to 20 m thick in some places. The most frequent type of exposed lithology in the area is ultramafic rock, as this rock type has been more resistant to glacial erosion compared to the surrounding phyllites.

Figure 5-2: The island of Rönnbäcksnäset, looking north, viewed from the top of Vinberget

5.2.2 Climate

Northern Sweden belongs to the temperate coniferous-mixed forest zone (Köppen classification) 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 precipitation is, on average, moderate in all seasons.

Between the years 1961 and 1990, the average annual temperature in Hemavan (about 15 km northwest of Tärnaby) was -0.5°C, with an average precipitation of 745 mm/year. Annual precipitation in the Lappland Mountains ranges between 1,000 mm and 1,500 mm. The mean winter temperature (December-January) in Tärnaby is -11.5°C, with occasional low temperatures of -40°C. Bogs, lakes and rivers are typically frozen for four to five months of the year.

Exploration work can also be conducted during the winter by taking advantage of the frozen ground, which minimises environmental impact during access. Notwithstanding this, should the Project be put into operation, it should be able to operate throughout the entire year.

Northern Sweden has aspects of both maritime and continental climate depending on the direction of airflow. When westerly winds from the Atlantic Gulf Stream prevail, the weather is warm and clear. When airflow is from the east, the Asian continental airflow prevails resulting in severe cold in winter and dry heat in summer. The mean temperature in northern Sweden is several degrees higher than that of other areas in these latitudes such as Siberia and southern Greenland due to the moderating effect of the Atlantic Ocean and the Baltic Sea.

5.3 Local Resources and Regional Infrastructure

The Ajaure hydro power plant, rated for 85 MW, is located upstream of Lake Gardiken, approximately 12 km from the Project site by gravel road. European route E12 is 14 km from the Project site, running in a southeast-northwest direction connecting Storuman to the port of Mo i Rana in Norway. This port is 166 km distant and is the closest of three within 500 km. The nearest rail access is at the town of Storuman, 107 km to the southeast. Water is plentiful around the site, but permission must be obtained to use it.

Figure 5-2: Rönnbäcken relative to existing infrastructure and photo of the Ajaure hydropower plant.

6 HISTORY

The chromite exploration campaign undertaken during the Second World War resulted in a discovery of nickel rich sulphides in ultramafic rocks collected in the Björkvattnet-Seimajaure region. Some extraction test work for nickel was undertaken without success.

The Boliden Mining Company (Boliden) first staked exploration permits in two areas in 1942. Metallurgical tests to recover nickel were performed in the 1960s with promising results. Nickel metal prices were rising at the time and a number of companies began to explore in the mountain chain and investigated assay techniques for nickel.

In the 1970s, Professor P. G. Kihlstedt at the Royal Institute of Technology (KTH) in Stockholm conducted research studies on the extraction of nickel from the peridotites and serpentinites from the Caledonian mountain chain. The work was funded by the Board for Technical Development (STU, today NUTEC), The Northland Fund (Norrlandsfonden), and a private company which was part of the Johnson Group. Three diamond drillholes were drilled at the Murfjället, Graipisvare, and Rotiken properties funded by the Northland Fund. The cores were used for metallurgical test work. Boliden drilled one core hole in 1972 along the road below the Vinberget deposit. The core intersected 125 m of serpentinite and was used for metallurgical tests at KTH in Stockholm and Boliden. The surveys were supplemented by studies of the possible by-products, including magnesite and brucite, for extraction of magnesium.

Boliden performed extensive studies during the 1970s on the sulphide nickel-bearing ultramafic rocks along the Caledonian mountain chain. In Rönnbäcken, grab samples were taken by blasting of exposed outcrops (68 samples in total). The samples were distributed over the exposed outcrops on Vinberget and on parts of Rönnbäcksnäset. The samples were analyzed for sulphur, total nickel, and bromine-methanol-soluble nickel. The latter was intended to determine the proportion of nickel present in nickel sulphides. Metallurgical tests were carried out on some of the samples.

Boliden drilled a total of 21 holes in the area. Apart from the hole below Vinberget Hill, the company drilled 20 core holes on the Rönnbäcksnäset Island. The holes on Rönnbäcksnäset consisted mainly of short vertical holes of approximately 10 m, one vertical hole down to 50 m, and one inclined hole (50°) to 81.4 m. Analysis was conducted on sulphur, total nickel, and bromine-methanol-soluble nickel. The boreholes were not drilled for the purpose of producing a resource estimate but rather just to highlight the vertical distribution of nickel sulphides. Analysis was made in intervals of 10 cm to 5 m. No significant leaching of sulphide nickel was detected at surface. The leaching of nickel in sulphides was to a depth of less than 0.5 cm to 1.0 cm which correlates to the weathering that also could visually be seen in the colour, brown to greyish, of the surface.

Pilot mining of 4,000 t in an open pit was conducted by Boliden in 1974 adjacent to the road below Vinberget. The average grade of the bulk sample was 0.21% Ni, 0.11% Ni in sulphide, and 0.07% S. The sample was used for metallurgical test work in Boliden's pilot plant in Boliden which produced nickel concentrates grading 26% Ni to 34% Ni, 1.5% Co, 5 g/t Au, and 2 g/t combined PGM at a sulphide nickel recovery of 67% to 73%. SRK notes that the grades of this sample are not representative of the current resource at the Rönnbäcken project.

The investigations in outcrops, core drilling, and beneficiation experiments were compiled and used for an application of exploitation concessions (called “Utmål” at the time, later replaced by the new term “Bearbetningskoncession”) submitted in 1976 for an area on Rönnbäcksnäset and one area on Vinberget.

An exploitation concession was only granted to those restricted areas where the drillholes and pilot mine were located, and not the parts that were sampled in outcrops. The exploitation concessions Rönnbäck nr 26 and nr 59 were granted to Boliden in 1982 following the application in 1976. In 1990-1993, Boliden held an exploration permit in connection with the exploitation concessions, but no exploration was carried out. The exploitation concessions were released in 2003 by a notification of withdrawal from Boliden.

IGE Resources AB was granted the Rönnbäcksjön nr 1 exploration permit in the area around Vinberget in 2005, which was later transferred to its subsidiary Nickel Mountain Resources AB in 2007. The remaining exploration permits were granted to the Company in 2007, 2009 and 2010 and the exploitation concessions in June 2010.

The Company carried out ground magnetic surveys and core drilling on Vinberget and Rönnbäcksnäset in the spring of 2008. In addition, metallurgical testwork was carried out on drill core material and material from Boliden's historic test mining pit. A first NI 43-101 compliant report inclusive of a Mineral Resource estimate was prepared by Scott Wilson Roscoe Postle Associates Inc. and was published in April 2009.

Geological mapping, geophysical surveys and outcrop sampling around the Project area was conducted in the summer of 2009 and the Scott Wilson PA Report was then completed in November 2009. Three new exploration permits were also granted, Rönnbäckssjön nr 5-7, during 2009.

From the mapping and sampling carried out during the summer of 2009, several potential drill targets were identified. Drilling commenced at Sundsberget at the end of 2009 and following this, on several other targets in the Project area. A reconnaissance ground magnetic survey was then conducted in 2010, covering a large part of the Project area.

On 12 February 2010 two exploitation concession applications were submitted to the Mining Inspectorate of Sweden (Bergsstaten), namely Rönnbäcken K nr 1 (Vinberget) and Rönnbäcken K nr 2 (Rönnbäcksnäset). These exploitation concessions were granted on 23 June 2010, and took legal effect on 22 October 2010, due to an appeals process. On granting of the exploitation concessions, an extension to the exploration permit Rönnbäcksjön nr 2 was also granted, now called Rönnbäcksjön nr 8.

On 5 October 2010, the Rönnbäcken exploration permits and exploitation concessions were transferred to Nickel Mountain AB.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Project is located in the Swedish Caledonian mountain chain which formed approximately 400-510 million years ago with the closure of the Iapetus Ocean, previously formed during the late Precambrian off the continent of Baltica. It is generally believed that the ocean crust moved downward along a subduction zone, with simultaneous build-up of sediment-filled basins linked to island arcs along the marginal zones of the ocean. The closure of the Iapetus Ocean and eventual collision between the two continents Baltica and Laurentia, created an extensive rock complex that was then thrusted over the Fennoscandian shield. These units are termed allochthons, subdivided into nappe and nappe complexes, and may have been transported several hundreds of kilometres to the east or southeast over the shield. The top nappe is usually associated with the longest transport distance, while the lower units tend to be more local. Alpine-type ultramafic rocks are tectonically displaced from the mantle into the crust. They occur along nappe boundaries in the Scandinavian Caledonides and most frequently in the Upper Allochtonous which host the Seve and Köli nappes. The regional geology is illustrated in Figure 7-1 and Figure 7-2 below.

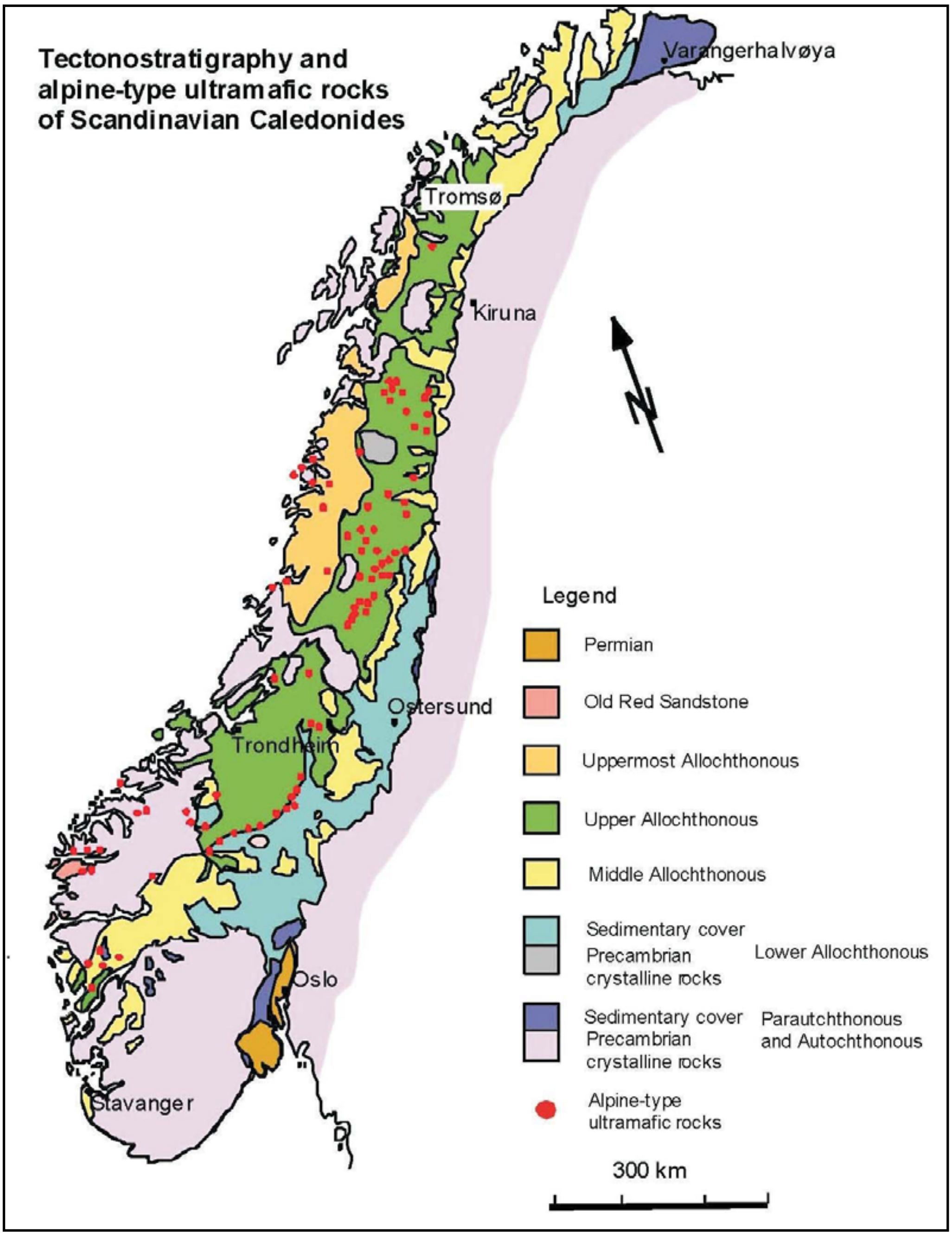

Figure 7-1: Tectonostratigraphy and alpine-type ultramafic rocks of the Scandinavian Caledonides

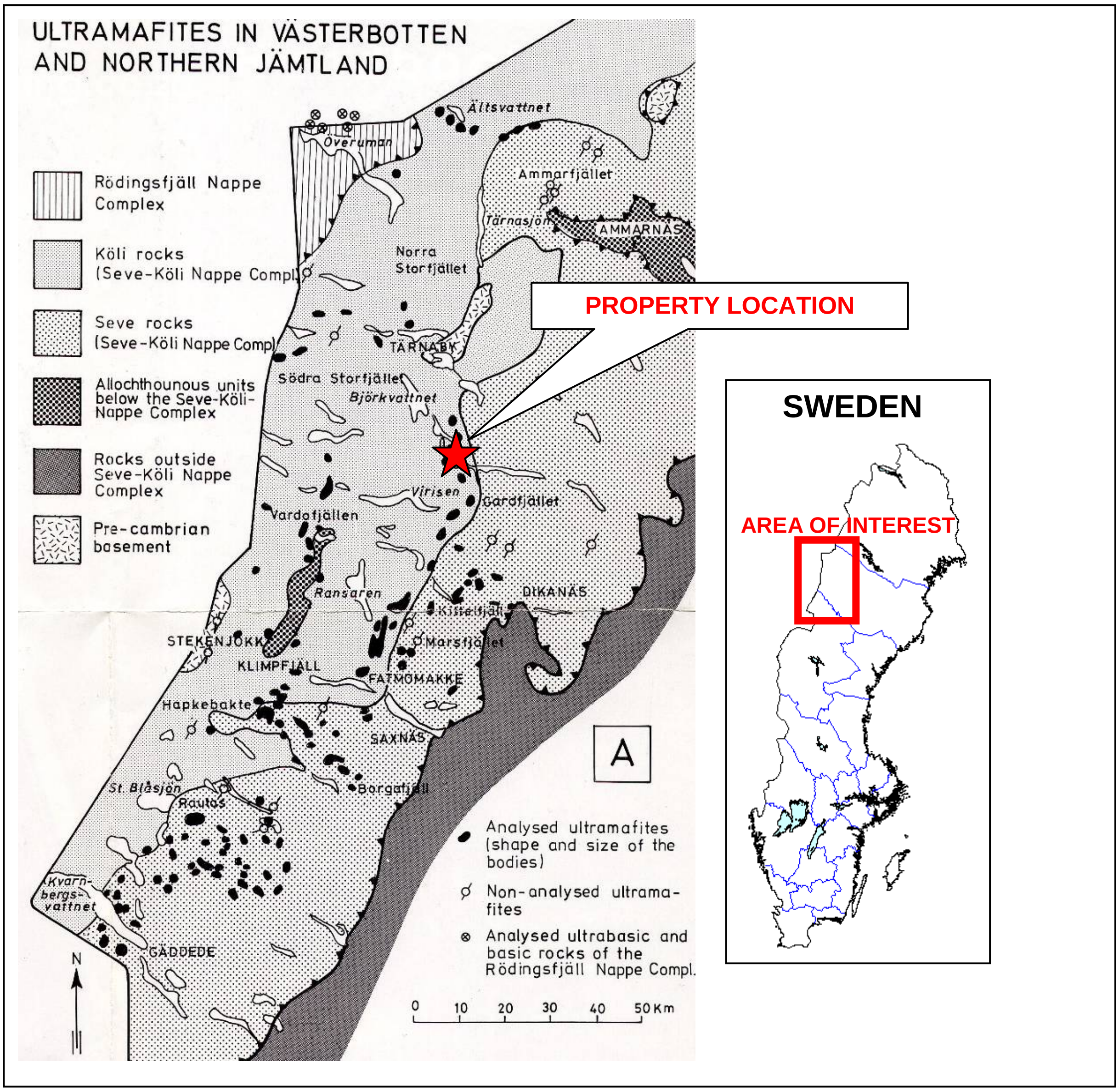

Figure 7-2: Location of the Rönnbäcken ultramafics and other ultramafics in Västerbotten and Northern Jämtland County

7.2 Local Geology

The geology in the Rönnbäcken area is dominated by the Köli Nappe which is situated near the border to the Seve Nappe in the east. The Köli Nappe includes rocks of greenschist metamorphic facies and the Seve Nappe rocks, which are of higher metamorphic facies, mainly amphibolite facies. The rocks in the Köli Nappe include the Tjopasi Group which in the Rönnbäcken area consist primarily of phyllite and felsic to mafic metavolcanics and nickel bearing ultramafic rocks. The ultramafic rocks occur as lenses of various sizes over an area of approximately 15 km^2. The complex folding has resulted in local variations in strike and dip. The ultramafic rocks are serpentinized, which is seen in the colour of the weathering surface. The most serpentinized rock is often grey, while more olivine and pyroxene rich rocks have a more brownish colour. The rocks vary from massive lenses to compositional layered rocks to erosion products such as serpentinite conglomerates and sandstones. In general, the ultramafic rocks are more serpentinized in the Köli Nappe, while the Seve Nappe consists of rocks that are more olivine and pyroxene rich and also contain less nickel in sulphides. Figure 7-3 illustrates the local geology.

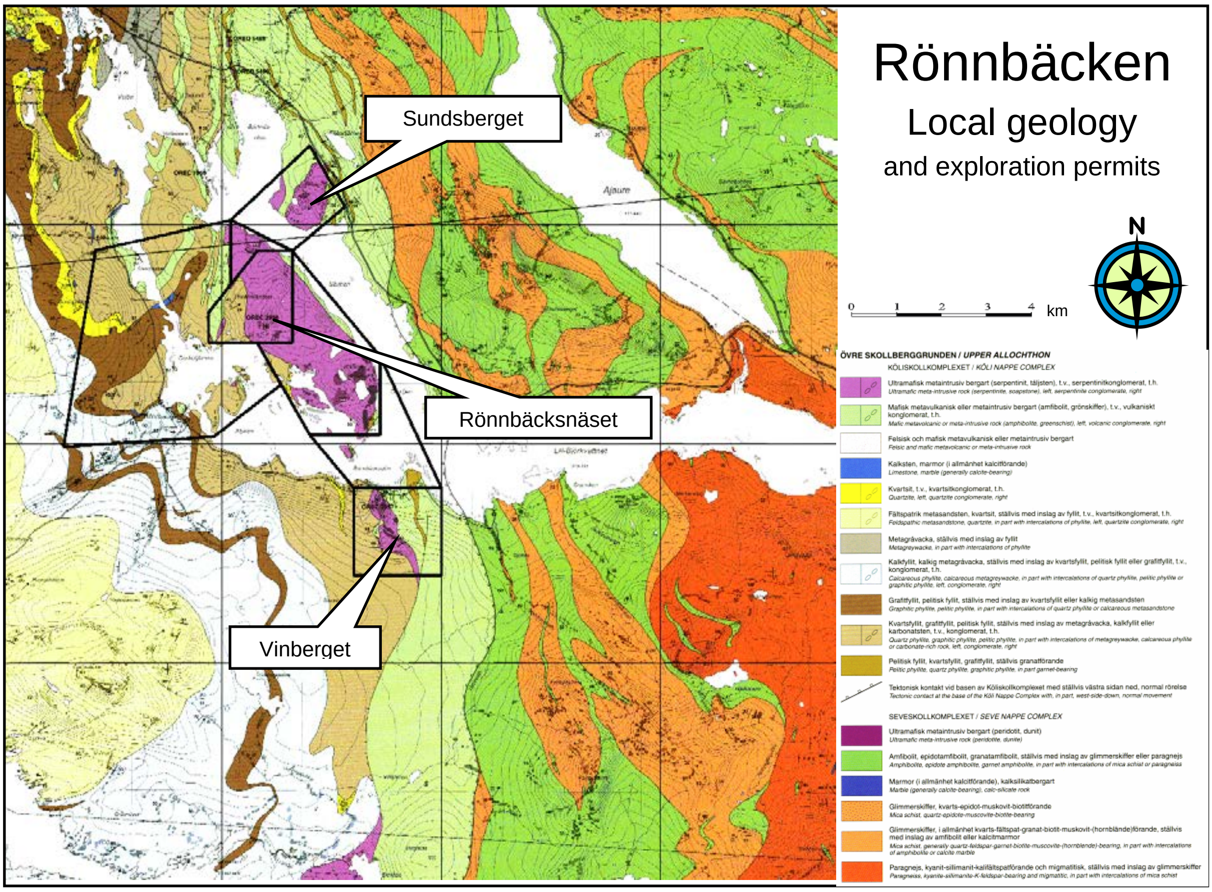

Figure 7-3: Local geology

7.3 Property Geology

The geology in the immediate Project area comprises highly serpentinized rocks which have been the target for the exploration of sulphide nickel mineralization. Some of the ultramafic lenses known in the area are less serpentinized and are therefore of less interest for nickel sulphide mineralization.

The Vinberget deposit comprises a homogeneous serpentinized tabular-shaped deposit up to 350 m thick, 300 m wide and 700 m long. The deposit is steeply dipping to the northeast and plunges to the northwest and is primarily hosted within a graphite bearing pelitic phyllite with intense quartz veining. A zone of soapstone between 1 to 5 m wide has been intersected at the contact between the mineralisation and the phyllite. The foliation of the phyllite follows the contact zone.

The Rönnbäcksnäset deposit comprises two seprentenite horizons separated by 80 m to 140 m of chloritic phyllite. The horizons dip approximately 45° west in the north and flatten out into a bowl shaped geometry to a dip of roughly 30° north in the southwest. The deposit has a strike length of roughly 2.4 km and a width of up to 400 m at its widest point. The upper horizon is thin and of less economic interest and is most likely not present in the southwestern area. This is overlain by pelitic phyllites, while chlorite dominates altered phyllite between the upper and lower horizons. The lower serpentinite horizon that is of economic interest is divided into the following four units:

1. Upper serpentinite unit.
2. Lower serpentinite unit.
3. Mafic intrusion unit (pyroxenite).
4. Low sulphur unit.

The mafic intrusion is found mainly between the upper and lower serpentinite unit throughout the area. The low sulphur unit is found in the two western sections in the Rönnbäcksnäset south area. The lower serpentinite horizon is underlaion by pelitic phyllites though near the contact with the mineralisation these contain a series of minor quartz conglomerate horizons.

The Sundsberget deposit consists of a single serpentinite body that strikes in a north-northeast to south-southwest orientation and dips roughly 30° to the north-northwest. The deposit measures roughly 1.2 km along strike and 500 to 600 m in width. The serpentinite is overlain mainly by chloritic phyllite and pelitic phyllites. Below the serpentinite, in the footwall pelitic phyllites dominates but as at Rönnbäcksnäset there are also quartz conglomerate horizons. There are mafic intrusions within the serpentine unit, but the geometry of these is not yet certain.

Talc alteration zones are a common feature at the contacts zones between serpentinite and country rock in all three deposits.

7.4 Mineralization

To date, the majority of work to characterise the nature of mineralisation in the Project area has been undertaken on samples collected from the Rönnbäcksnäset and Vinberget deposits. As such, the following sections of this report focus on these deposits. The initial indication from visual inspection of drill core, multi-element ICP analysis and metallurgical testwork is that the nickel sulphide mineralisation at Sundsberget is of a similar character to that at Rönnbäcksnäset and Vinberget.

The nickel sulphide mineralization in the Project area is hosted in serpentinized ultramafic rocks, which were altered from dunites and peridotites. The gangue mineralogy is dominated by antigorite, olivine, pyroxene, chlorite, carbonates (mainly calcite and dolomite), magnetite, and chromite.

The dominating nickel-rich sulphides in the deposits Rönnbäcksnäset, Sundsberget and Vinberget are heazlewoodite (Ni_3S_2), pentlandite $(Fe,Ni,Co)_9S_8$, often containing more than 40% Ni and various amounts of Co, and to a lesser extent millerite (NiS). Other minerals found are cobaltite (CoAsS) and maucherite ($Ni_{11}As_8$), which probably are the most frequent arsenic bearing minerals. The dominating cobalt bearing minerals are pentlandite, millerite, and cobaltite. Only traces of pyrrhotite and pyrite are present. Nickel is also found in various amounts in olivine, serpentine, magnetite, and brucite.

In Vinberget, pentlandite dominates as the most frequent nickel rich sulphide. In Rönnbäcksnäset and Sundsberget, however, the mineralization is more variable, both in terms of grade, nickel sulphide species and host rock type. In some parts, heazlewoodite dominates and in other areas pentlandite occurs as the most frequent nickel sulphide.

The elements arsenic, gold, and sulphur are not unique to any of the lithologies and may have been introduced later or have been remobilized. Overall, the nickel sulphides are fine grained (often about 25 µm) and occur as individual grains in serpentine or oxides or as mineral aggregates together with other nickel sulphides or magnetite.

7.4.1 Specific Mineralogical Studies

Various mineralogical investigations have been carried out by Ekström Mineral AB (Ekström), Xstrata Process Support (Xstrata), Outotec Research Oy (Outotec research centre, or ORC), Finland, Qumex Material Teknik AB (Qumex), and more recently by the Geological Survey of Finland (GTK). The results of this work are summarised below.

EKSTRÖM - OPTICAL MICROSCOPY

Eleven samples from six drill cores, two from Rönnbäcksnäset and four from Vinberget, were sent to Ekström for basic mineralogical thin section study and for Scanning Electron Microscope/Energy Dispersive Spectroscopy (SEM/EDS) analysis of the sulphides.

Rock forming minerals

All of the samples were found to be dominated by serpentine, except one sample from Rönnbäcksnäset (RON 5801) which was found to be dominated by chlorite with lesser pyroxene. Carbonate was found to be common in samples from Vinberget. Chrysotile asbestos was identified in three of the samples.

Opaque oxides

Magnetite and chromite dominate the opaque minerals, with magnetite formed as an alteration product from chromite during the serpentinization process.

Sulphides

Pentlandite and heazlewoodite were identified in all samples except RON 5801, with pentlandite as the dominant sulphide phase. Accessory and rare sulphide phases are detailed below. The results of the study are illustrated in Table 7-1 to Table 7-3.

Table 7-1: Thin section sample summary

Hole	Sample Number	Sample Depth	SEM	Rock Type
RON57	RON5702	50.4		Serpentinite
RON58	RON5801	31.0		Pyroxenite
RON58	RON5802	47.2	X	Serpentinite
VIN26	VIN2601	20.0		Serpentinite
VIN26	VIN2604	80.0	X	Serpentinite
VIN27	VIN2702	40.0	X	Serpentinite
VIN30	VIN3001	40.0	X	Serpentinite
VIN30	VIN3003	120.0		Serpentinite
VIN30	VIN3005	200.0		Serpentinite
VIN31	VIN3101	10.0		Serpentinite
VIN31	VIN3103	90.0		Serpentinite

Table 7-2: Mineral composition and relative frequency of samples from Rönnbäcksnäset

Drill core number Sampled at	RON 5801 31.0 m	RON 5802 47.2 m	RON 5702 50.4 m
serpentine (antigorite)		xxxx	xxxx
carbonate	x		x
chlorite	xxxx	x	
amphibole	xx		x
olivine			xx
pyroxene	xxx		xx
phlogopite	xx	xx	
chrysotile asbestos		xx	
bolingwite			xx
iddingsite			x
brucite		xx	xx+
epidote	xx		
magnetite		xxx	xxx
chromite		xx	x
pentlandite		xx	xx
heazlewoodite		x	x
maucherite		xx	r
millerite		x	r
violarite,mackinawite		x	
chalcopyrite	x		
pyrrhotite			
ilmenite	r		

Relative frequency: xxxx = high, xxx = intermediate, xx = little, x = acc, r = rare

Table 7-3: Mineral composition and relative frequency of samples from Vinberget

Drill core number Sampled at	Vin2601 20.0 m	Vin2604 80.0 m	Vin2702 40.0 m	Vin3001 40.0 m	Vin3003 120.0 m	Vin3005 200.0 m	Vin3101 10.0 m	Vin3103 90.0 m
serpentine antigorite	xxxx	xxxx	xxxx	xxxx	xxxx	xxxx	xxxx	xxxx
carbonate	xxxx	xxx	xxx	xxx		xx	xxx	xxx
chlorite								x
amphibole								xx
olivine	xx					pseu	xx	x
pyroxene	x				xx	pseu		xx
phlogopite							x	
chrysotile asbestos		xxx					x	
epidote								
iddingsite	x						x	
brucite	x	x	r	x		xx	x	
magnetite	xxx	x	xxxx	xxx	xxx	xx	xxx	xxx
chromite	xx	x	xxx	xx	xxx	xx	xx	x
pentlandite	xx	xx	xx	xx	xx	xx	xx	xx
heazlewoodite	x	x		xx	xx	x	x	x
maucherite				x	r			
millerite	r							
violarite mackinawite	r		x					
chalcopyrite		x						
pyrrhotite		x						
awaruite					r			r
cobaltite						xx	x	x
pyrite		x						

Relative frequency: xxxx =high, xxx = intermediate, xx= little, x= acc, r =rare

Table 7-4: SEM-EDS Analysis (RON58 & 57, VIN26, 30 & 31)

Mineral	Formula	Range % Ni	Range % As	Range % Co	Comment
Pentlandite	(Ni,Fe,Co)9S8	39.8-44.2		1.5-2.8	
Millerite	NiS	69.1-69.8			
Heazlewoodite	Ni3S2	71.5-76.3			
Maucherite	Ni11As8	50.8-51.4	44.6-45.85	0.2-0.4	1.5-1.7%Sb
Cobaltite	CoAsS			23	
Pyrrhotite		1.6-2.3			
Chromite					>5% Mn

EKSTRÖM - QUALITATIVE FIBRE MEASUREMENT

Two samples of diamond drill core from VIN30 at 219.2 m and RON58 at 52.3 m were selected for qualitative analysis of fibres, and examined with optical microscopy by Ekström for light refraction, anisotropy, angle of extinction, elongation, and pleochroism. Both samples showed the same optical properties as chrysotile.

XSTRATA - QEMSCAN AND EPMA

Mineralogical studies were performed by Xstrata using Quantitative Elevation of Materials by Scanning Electron Microscope (QEMSCAM) and Electron Probe Micro Analysis (EPMA) on four composite samples. The samples were composed of a quarter of the core and crushed and successively sieved to avoid the finest fraction.

The objective of the study was to characterise the Ni bearing species in each composite and to produce quantitative measurements of Ni deportment as a basis for comparison to total nickel and sulphide nickel chemical assays. The assays were completed at ALS Chemex in

Vancouver Canada and at Labtium in Finland. Modal mineralogy, grain size distributions, and mineral composition data was also presented as part of the study.

The modal abundance of gangue and sulphide minerals present in each sample is presented inFigure 7-4and Figure 7-5

Table 7-5: Xstrata QEMSCAN samples

Hole	Interval (m)
VIN30	20.0-26.0
VIN30	192.0-198.0
RON53	52.0-58.0
RON53	76.0-82.0

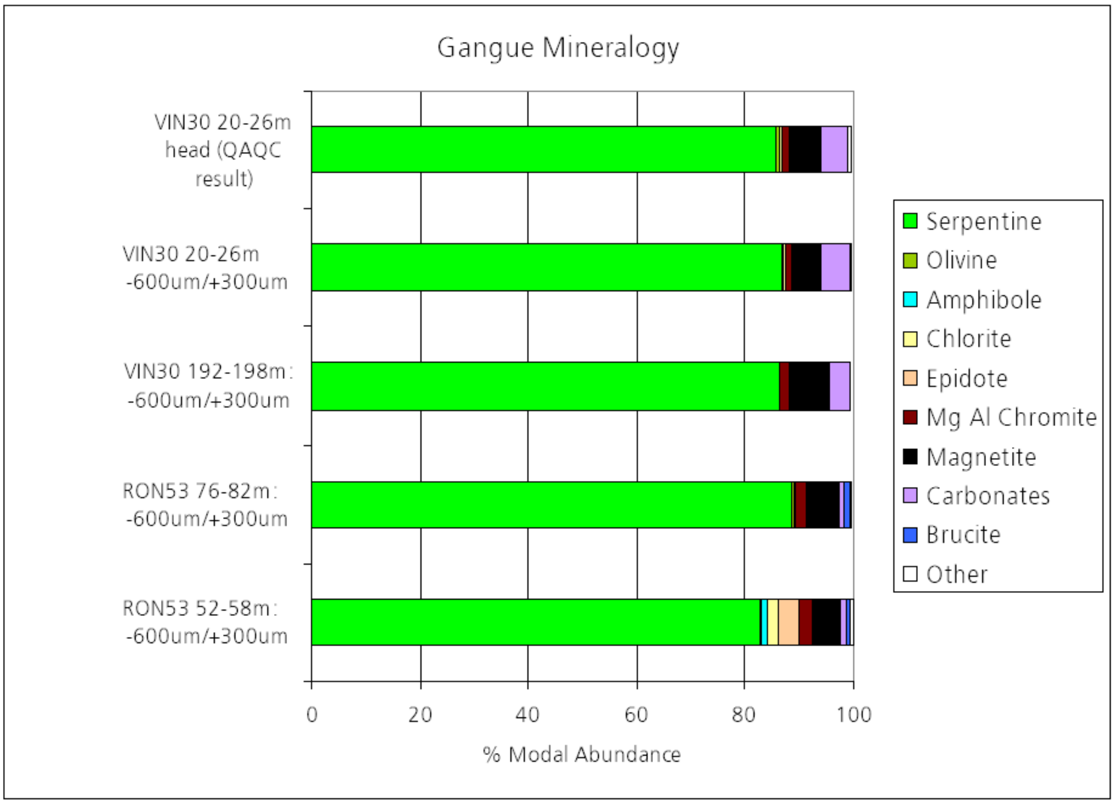

Figure 7-4: Gangue Mineralogy in selected samples

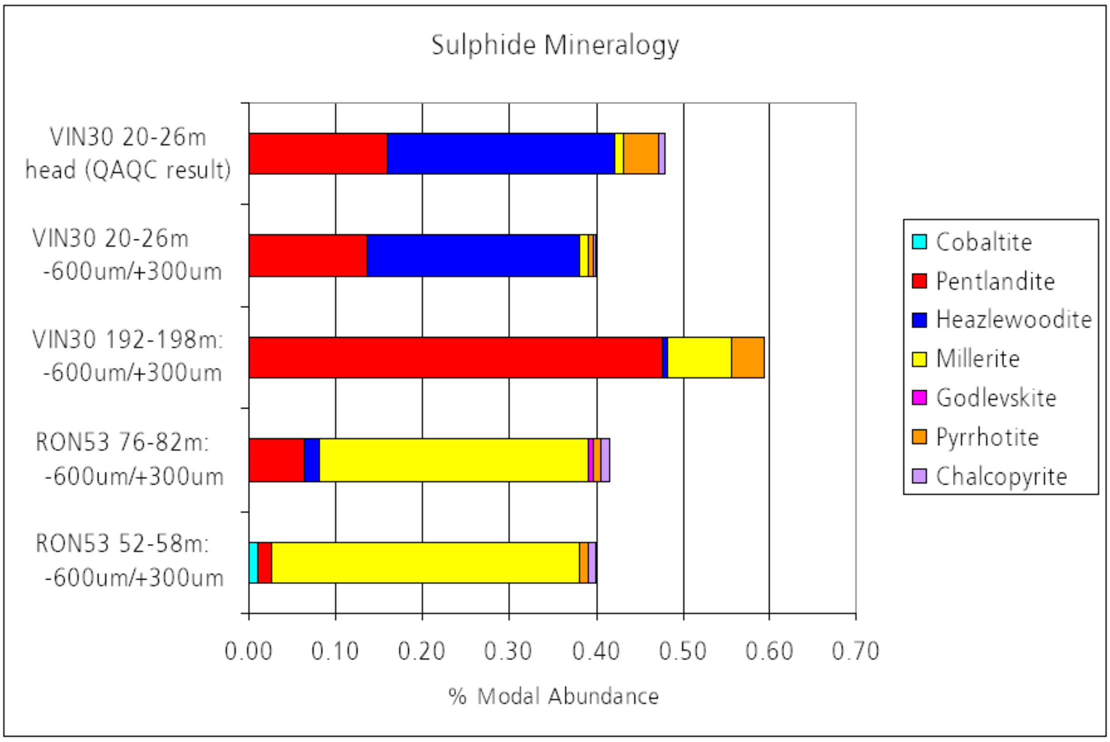

Figure 7-5: Sulphide Mineralogy in selected samples

Modal analysis highlighted minor mineralogical differences between the RON53 and VIN30 samples, including a higher percentage of carbonates in the VIN samples and the presence of brucite in the RON samples. The samples were also found to vary with respect to the proportion and type of Ni (Fe) sulphide as indicated above. Cobalt was found to occur in solid solution in pentlandite and millerite. A few cobaltite grains were also found.

The following observations were made with respect to these Ni deportment calculations:

Gangue minerals (oxide + silicate) contributed approximately 30% of the total Ni in the VIN30 samples, and 40% of the total Ni in the RON53 samples.

The major Ni bearing sulphide in the two RON53 samples was millerite. A minor amount of the nickel was contributed from pentlandite and heazlewoodite.

The two VIN30 samples had different proportions of Ni-bearing sulphide species. VIN20 20-16 m contained much more heazlewoodite, compared to VIN30 192-198 m, which was dominated by pentlandite.

One of the key objectives of the mineralogical study performed by Xstrata was to assess the reliability of assays, both in terms of total Ni and sulphide Ni. Analyses was completed on nine size fractions from VIN30 (20 to 26 m) plus the coarse fraction (-600/-300 µm) from the remaining composite.

Reconciliation between total nickel chemical assays and calculated total nickel assays from the mineralogical analyses was found to be, in general, very good. The analyses were completed on the 9 size fractions from VIN30 20-26 m, plus the coarse fraction (-600/+300 µm) from the remaining three composites. The difference between measured and calculated total nickel was found to be less than 0.02% in nine of the twelve measurements. In one of the twelve comparisons the difference was marginally greater than 0.03%Ni.

A reconciliation between sulphide nickel content as determined from chemical assays and sulphide nickel content calculated from mineralogy was completed on the same twelve samples. Seven of the twelve measurements compared very well (within 0.01% Ni), while the chemical assay and calculated assay differed by 0.03-0.06% Ni in the remaining five samples. These results are summarised in Figure 7-6 below.

SRK notes that comparison of calculated sulphide Ni determined from the mineralogical measurements made against sulphide Ni assays performed at ALS Chemex were done using method ME-OG62. A comparison against sulphide Ni assays performed at Labtium using ammonium citrate hydrogen peroxide leach with ICP-AES finish (method code 240P), may have been more appropriate given that this is the Company's principal laboratory and assay method used for determination of sulphide nickel Rönnbäcken.

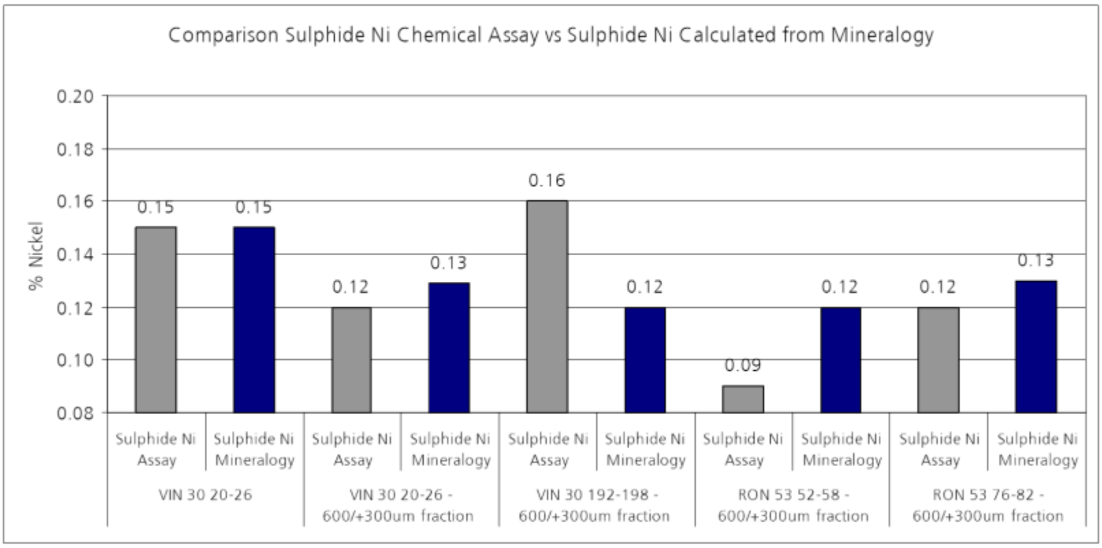

Figure 7-6: Sulphide Ni assays performed at ALS Chemex vs calculated sulphide Ni determined from mineralogical measurements

Nickel sulphide grain size distribution was assessed as part of the Xstrata study. The size fraction chosen for measurement was -600/+300µm, coarser than the liberation state, so as to ensure all textures and original grain sizes were maintained. QEMSCAN measurements of Ni-bearing sulphides were isolated and plotted as a distribution. All nickel-bearing sulphide species were combined and are referred to as Ni (Fe) Sulphide. A total of 11,121 Ni (Fe) Sulphide particles were included in this analysis. The grain size distributions of Ni (Fe) sulphide for each of the four samples given in Table 7-5 are presented in Figure 7-7 below.

The results of the work indicated that the majority of nickel-bearing sulphides fall within the range of 15 to 50 µm with averages closer to 25 µm.

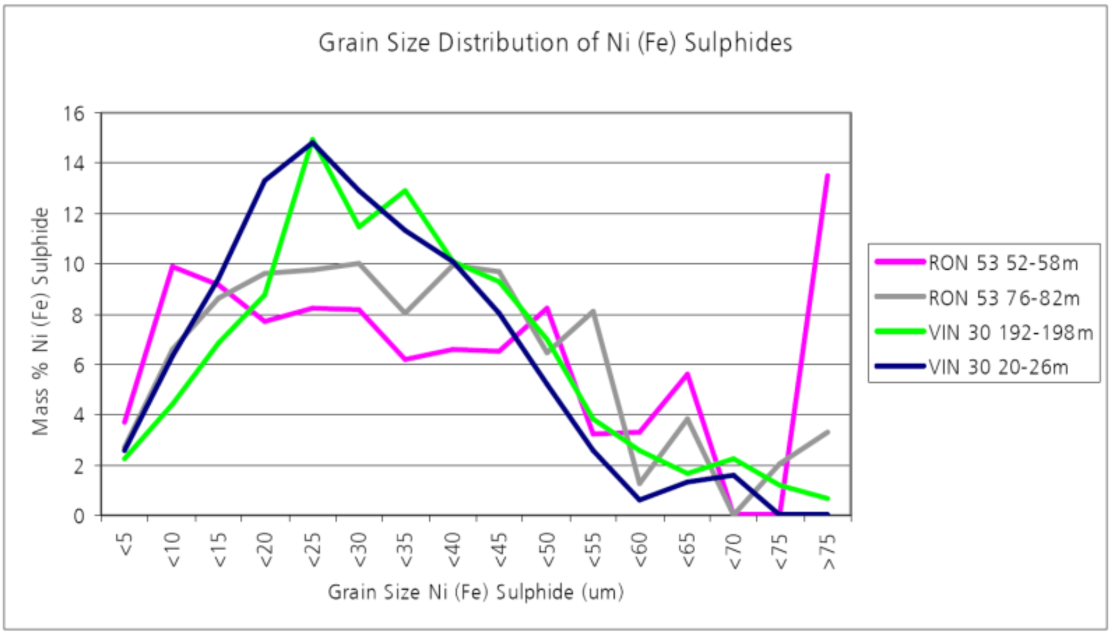

Figure 7-7: Grain Size distributions of Ni (Fe) Sulphides from -600µm/+300µm fraction

QUMEX - QUANTITATIVE FIBRE MEASUREMENT

Samples were collected from particularly fibrous-rich parts in the cores and sent to Qumex for quantitative analysis of fibres. The samples were prepared, crushed, and pulverized for the standard intervals, for assaying of two metre core length. The samples were evaluated using an electron microscope with a magnification of 250 times, with 25 fields per sample evaluated regarding fibre content (volume units).

Table 7-6: Fibrous volume in samples

Hole	Section	Fibrous Volume	Report	Date
VIN27	20.0 – 22.0	0.1	4360-01-08	2008-06-03
RON53	92.0 – 94.0	0.5	4431-01-08	2008-11-19

Geological Survey of Finland (Gtk) Modal Mineralogy Study

A selection of samples including 48 thin section samples from Rönnbäcksnäset and Vinberget, and 32 drill core samples from Sundsberget were sent to GTK for modal mineralogy study by Mineral Liberation Analyser and measurement by XMOD-std. One thin section was prepared by for each of the drill core samples submitted from each deposit. The results are illustrated below.

While serpentine is the dominant gangue mineral in all three deposits, there are slight differences in gangue mineralogy, with more pyroxene and chlorite at Rönnbäcksnäset and Sundsberget than at Vinberget. Talc occurs at the footwall contacts.

The modal abundance of magnetite is relatively constant in all three deposits, with slightly higher levels at Sundsberget. Pentlandite dominates the sulphide mineralogy at Vinberget while Heazlewoodite is more prevalent at Rönnbäcksnäset and Sundsberget.

The thin sections are regarded as indicative of modal abundance and SRK understands that more definitive mineralogy work is planned for work programmes going forward.

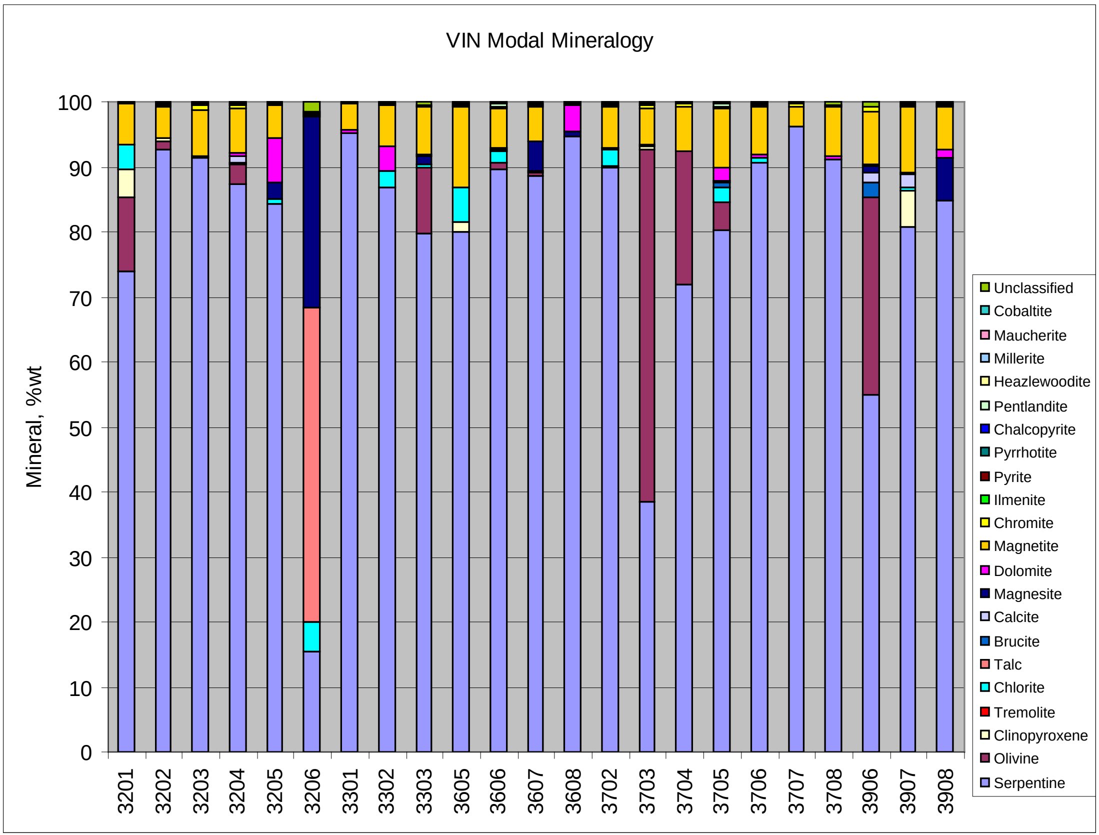

Figure 7-8: Modal mineralogy in 23 thin section samples from Vinberget

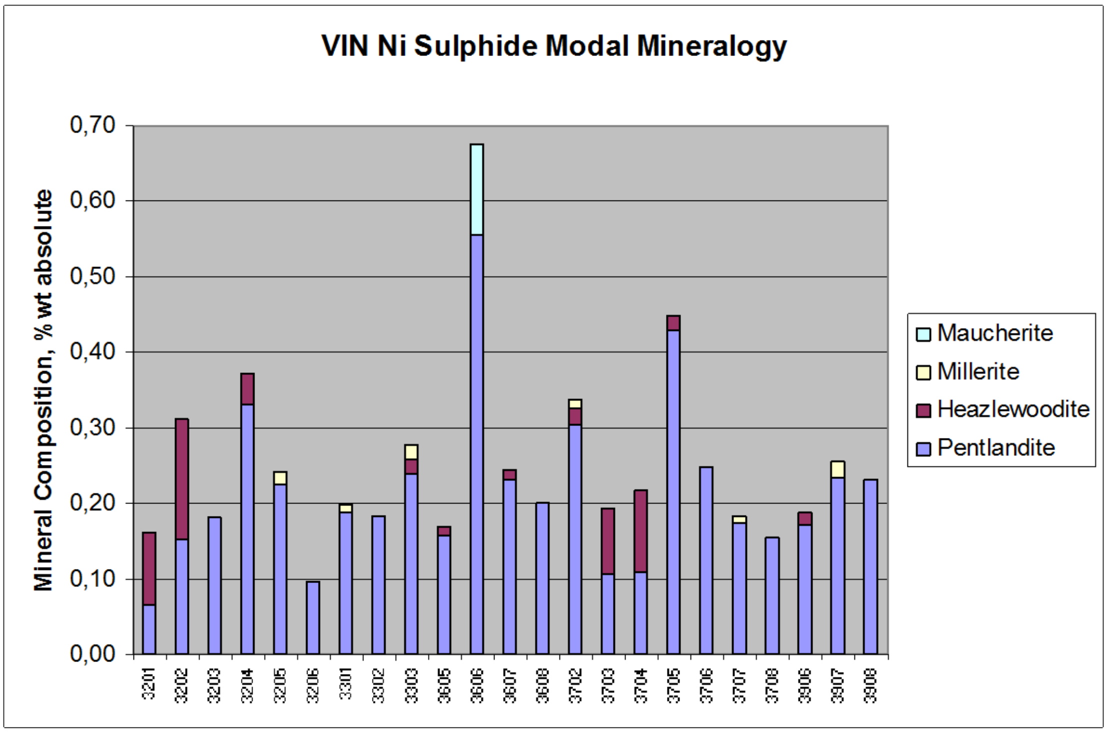

Figure 7-9: Nickel sulphide modal mineralogy in 23 thin section samples from Vinberget

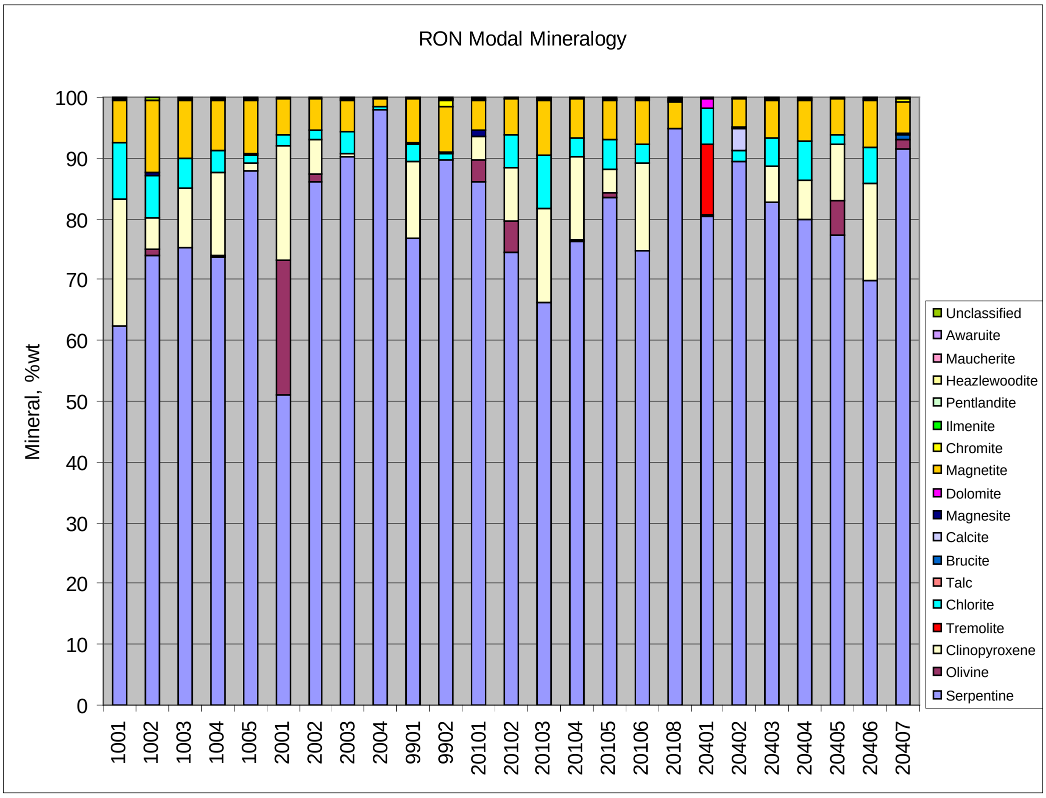

Figure 7-10: Modal mineralogy in 25 thin section samples from Rönnbäcksnäset

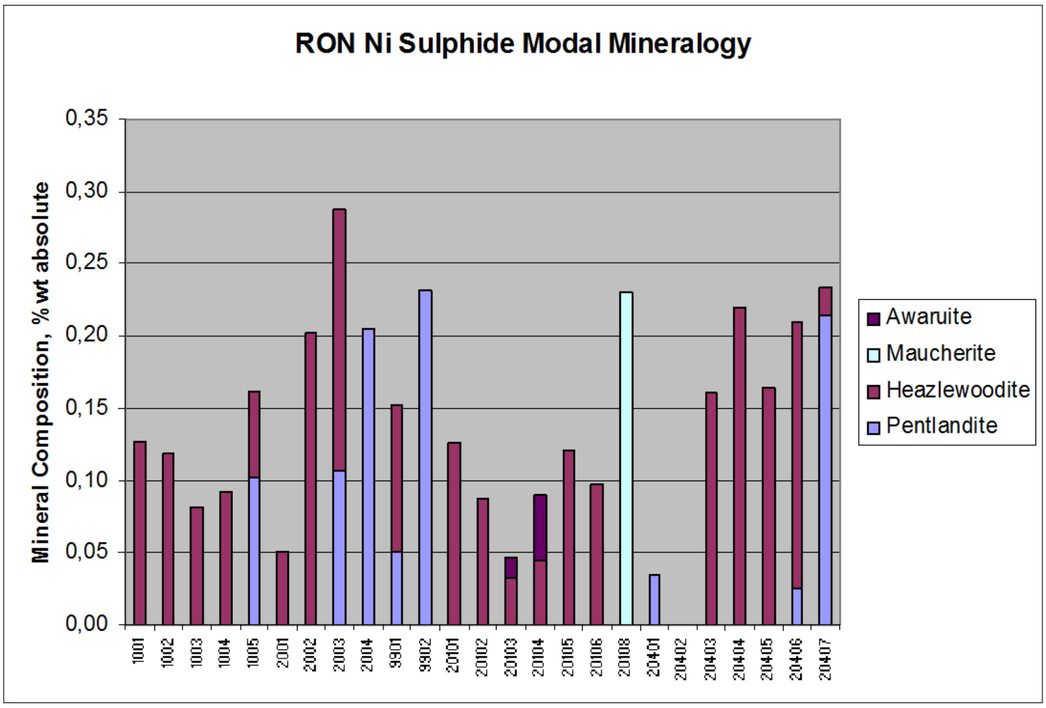

Figure 7-11: Nickel sulphide modal mineralogy in 25 thin section samples from Rönnbäcksnäset

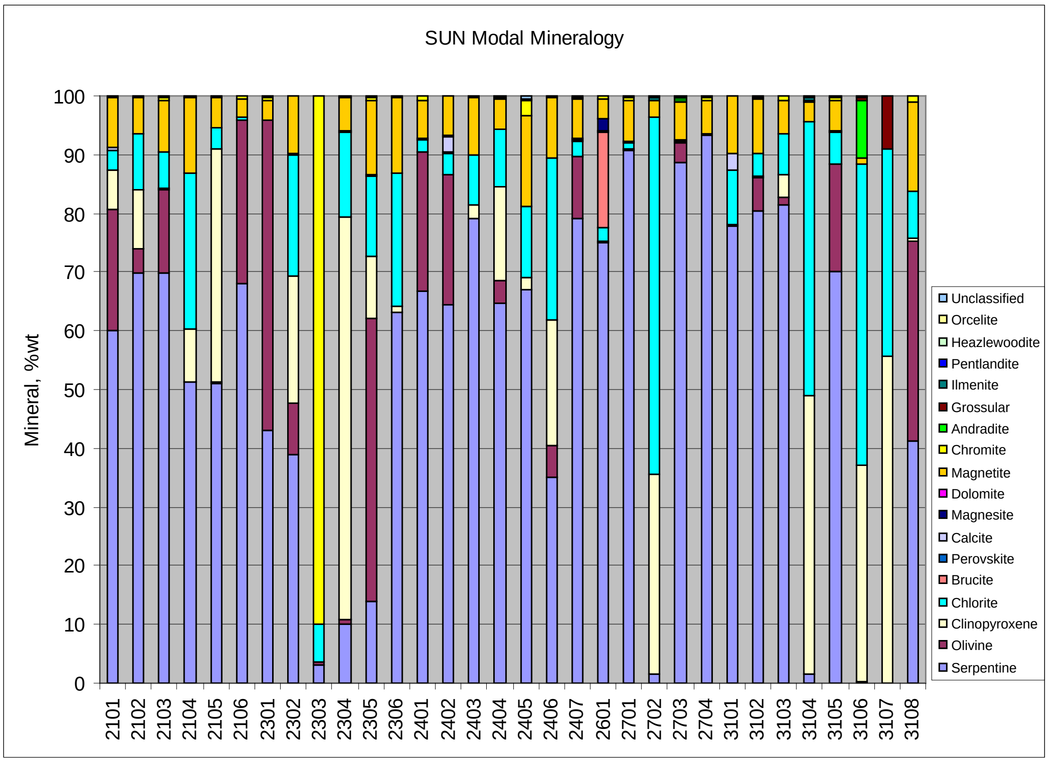

Figure 7-12: Modal mineralogy in 32 thin section samples from Sundsberget

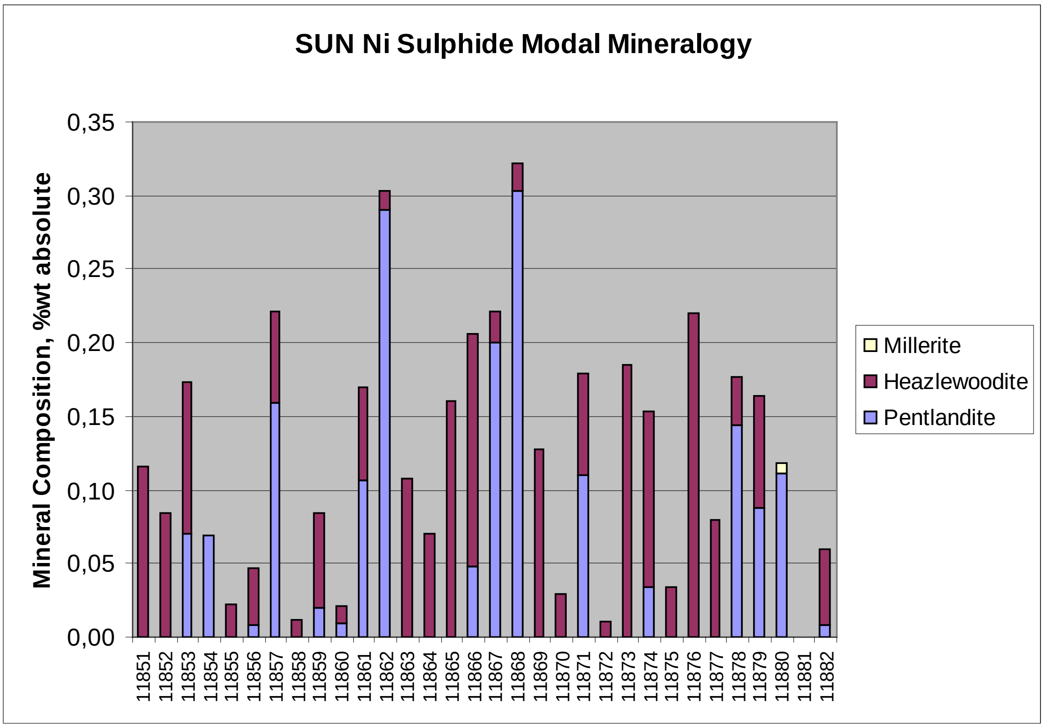

Figure 7-13: Nickel sulphide modal mineralogy in 32 thin section samples from Sundsberget

8 DEPOSIT TYPE

Mineralization in the project area is hosted by serpentine and is mainly of an epigenetic, nickel-sulphide type, with minor magmatic nickel sulphides. Nickel was originally located mainly in the olivine lattice in the ultramafic rocks, such as dunites and peridotites.

Due to serpentinization of the olivine, the nickel in the olivine was released and nickel bearing sulphides were formed depending on sulphur availability. Olivine in the ultramafic rocks is magnesium-dominant and contains up to 0.5% NiO. Serpentinization of ultramafic rocks and the olivine occurs through the supply of water, S, and CO_2. The reaction can be summarised as follows:

Olivine (Ni) + H_2O + S_2 + CO_2 → Serpentinite + Brucite + Carbonates + Fe_3O_4 + Ni_xS_y* + H_2
* Ni-rich sulphides

Serpentinization of the ultramafics within the three deposits (Vinberget, Rönnbäcksnäset and Sundsberget) is pervasive. As a consequence, both nickel sulphide and magnetite are widespread and of relatively consistent grade throughout.

9 EXPLORATION

9.1 Introduction

Exploration programmes carried out to date at the Project have comprised geological mapping, outcrop sampling, ground magnetic surveys, magnetic susceptibility surveys and drilling programmes.

9.2 Geological Mapping and Sampling

The Company sampled serpentinite outcrops in the Rönnbäcken area for the first time in the summer of 2005 within the framework of a regional sampling programme. The programme included tests on several exploration permit along the borders of the Caledonian mountains with the objective of testing the serpentinites for potential nickel, platinum, and palladium.

The Klumpliklumpen, Rotiken and Fjelkaområdet areas were tested in addition to Rönnbäcken. In total, approximately 70 samples were taken of which five were from Rönnbäcksnäset, four from the Rönnbäcksjön nr. 1 exploration permit, and one sample from the Rönnbäcksjön nr 4 exploration permit. In 2007, an additional 30 samples were collected by the Company, the emphasis this time being the serpentinite outcrops within the exploration permits Rönnbäcksjön nr 3 and nr 4.

In the summer of 2009, the Company mapped approximately 15 km^2 and collected 117 samples for analyses by the ammonium citrate method for Ni, Co, Cu, and S in an attempt to identify ultramafic rocks suitable for future drill targets. Twenty-three of the samples returned values greater than 1,000 ppm (0.1%) Ni as determined by ammonium citrate method (Ni-AC). In addition to analysis of nickel in sulphides, analysis of major elements, trace elements and precious metals were performed as well as surveys of specific gravity and magnetic susceptibility.

In total, the Company has now collected 157 rock samples from within the Rönnbäcken permits.

9.3 Geophysics

9.3.1 Magnetic susceptibility and specific gravity surveys

Much of the magnetite in the project area is secondary, having formed during the process of serpentinization, and as such, has been used by the Company’s geologists as an exploration tool in the field as well as during the core logging to identify areas of serpentinization and possible nickel sulphide mineralisation.

Magnetic susceptibility measurements on core were initially taken routinely every metre, on every bag of coarse rejects, as well as on outcrops during the geological mapping programme.

Measurements were taken with an SM-20 Magnetic Susceptibility Meter from GF Instruments, a hand-held instrument with a sensitivity of 1x10-6 SI units. The measurement gives relative readings and no corrections have been made for geometry or volume of the sample bags of coarse reject from the sample preparation of drill cores or rocks (see also Section 9.13 below).

Field susceptibility surveys were carried out at Vinberget to identify the presence and extent of serpentinization of ultramafics in the project area. Measurements were taken at ten metre intervals on 20 m section lines. At each surveyed point, two measurements were taken separated by a distance of 10 to 20 cm on a flat surface of the outcrop. Sections were surveyed between 390N to 2140N in the Vinberget local grid. These surveys were carried out during July and August 2008 and a total of approximately 450 measurements collected from an area of approximately 17 ha were taken.

In 2009, magnetic susceptibility surveys were taken on all outcrops mapped on the Rönnbäcken project. A total of 2,287 readings were taken.

9.3.2 Ground magnetic surveys

Between 2008 and 2010, ground magnetic surveys were performed with a GEM system, GSM-19T proton magnetometer. Measurements were taken at ten metre intervals along sections some 100 m or 200 m apart. For the reconnaissance survey of the Project area, measurements were taken at twenty metre intervals on 500 m sections. The results of the ground magnetic surveys for each deposit are illustrated in Figure 9-1, Figure 9-2 and Figure 9-3 respectively. Low magnetic areas, blue in Figure 9-2, are caused by rough topography and a consequent absence of data.

Vinberget

The first ground magnetic surveys at Vinberget were carried out in the spring of 2008 on the southwest side of the deposit. The survey covered sections 0 to 1200N, with additional more detailed measurements taken from sections 550N to 950N to better understand the horseshoe-shaped serpentinite outcrop south of Vinberget. During the same period in the summer of 2008, surveys were expanded to also cover the north side of Vinberget.

In 2009, additional surveys were carried out to cover the remainder of the ultramafic rocks in the Rönnbäcksjön nr 1 exploration permit. In total, approximately 25 line-km have now been surveyed at Vinberget.

Rönnbäcksnäset

During the spring of 2008, ground magnetic surveys at Rönnbäcksnäset were carried out over sections 0 to 500E in the south area and 0 to 1300N in the north area. During the summer of 2008, the survey was expanded to cover sections 100W to 600W in the south area. Towards the end of the drill programme, the survey was extended further south to assist locating “satellite holes” RON205 and RON206. After completion of the drilling programme, the survey was extended in several areas to facilitate planning of the next phase of diamond drilling. In 2009, additional surveys were carried out to cover the remainder of the ultramafic rocks in the Rönnbäcksnäset area. A total of approximately 64 line-km have been surveyed at Rönnbäcksnäset.

Sundsberget

A ground magnetic survey was carried out at Sundsberget in 2009 over an area of roughly 3 km^2 and using 10 m station spacing on 100 m spaced sections, totalling approximately 21 line-km.

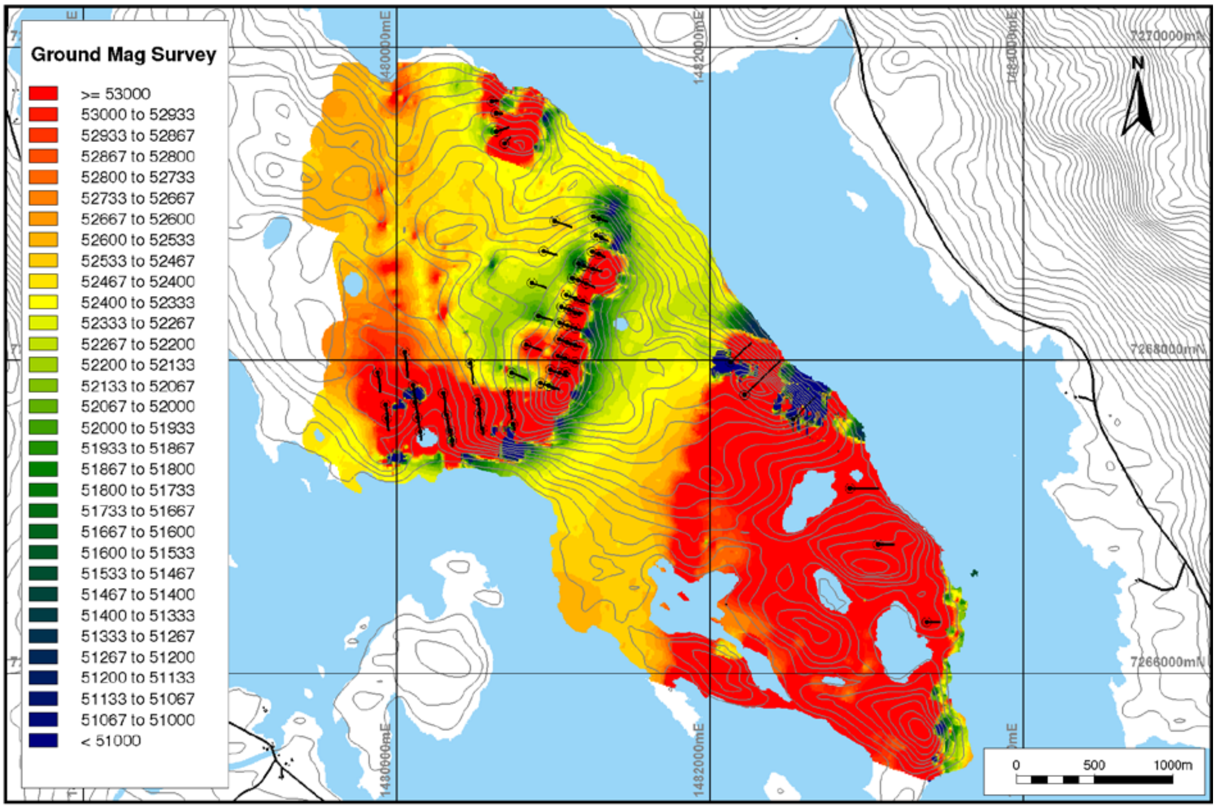

Figure 9-1: **Ground magnetic survey grid (TMI), Rönnbäcksnäset**

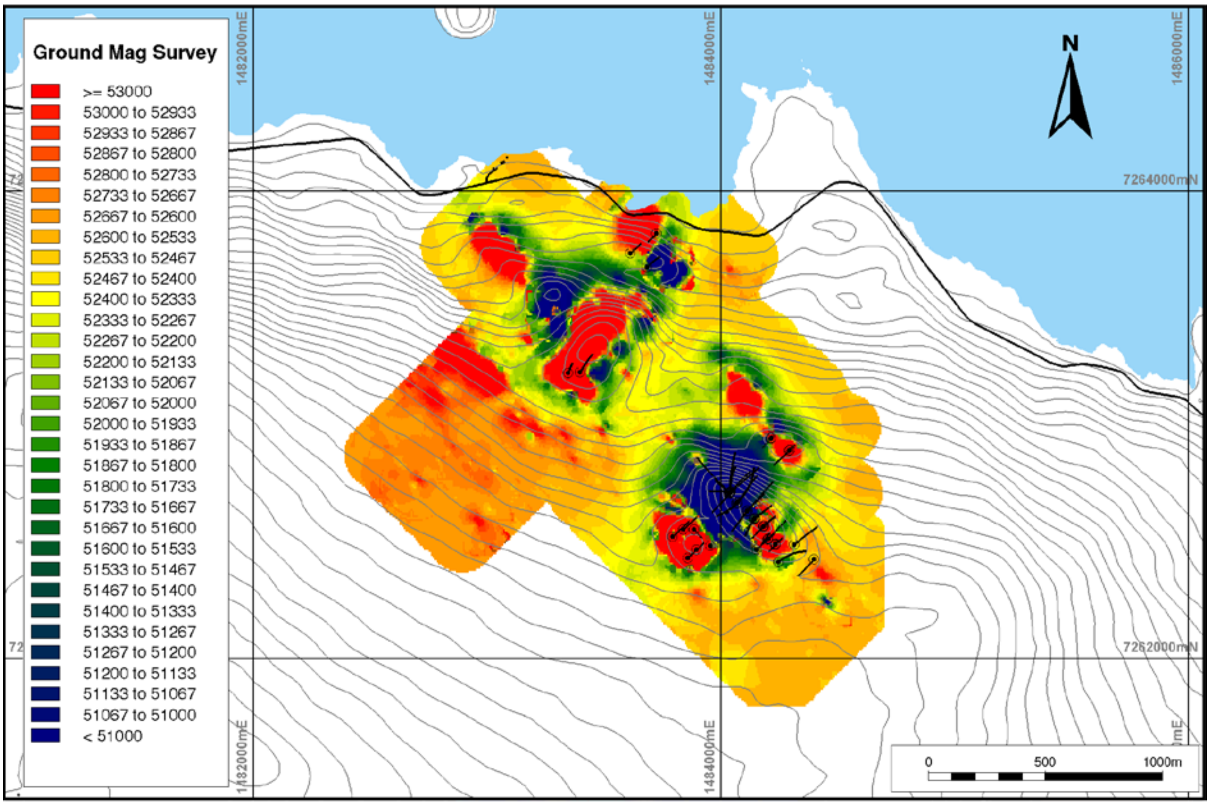

Figure 9-2: **Ground magnetic survey grid (TMI), Vinberget**

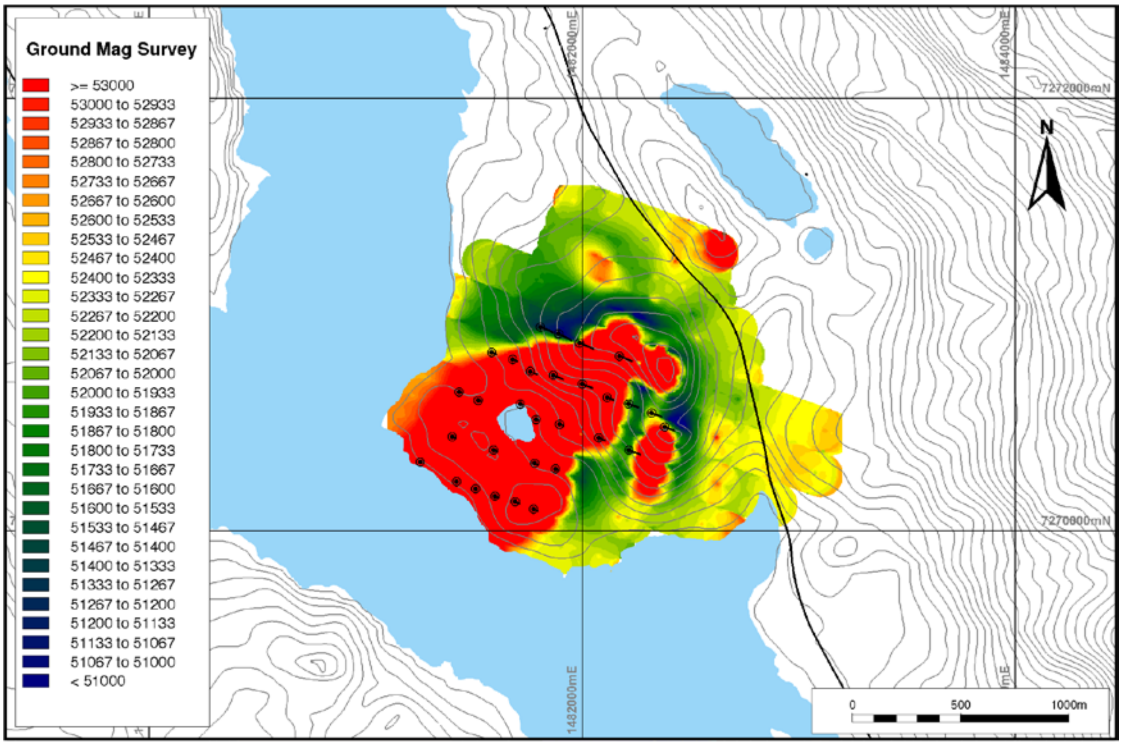

Figure 9-3: Sundsberget ground magnetic survey grid (TMI), with drillhole traces and surface mapping stations.

9.4 Geochemistry

The only geochemical surveys performed in the area were the rock geochemistry programmes described above.

9.5 Exploration Potential

Exploration to date has focused principally on the Vinberget and Rönnbäcksnäset deposits and more recently on Sundsberget.

The Rönnbäcksnäset deposit is open down dip of the existing drill data and the pit optimisation studies undertaken by SRK and described later in this report show that if the minerallisation does continue at the same grades and thickness then it does have potential to be exploited economically and therefore to add to the overall Mineral Resource. SRK understands that the Company has recently drill tested the depth extension of mineralisation at Rönnbäcksnäset, that these holes have intersected serpentinite at expected depths and that initial indications are sulphide nickel grades are broadly consistent with the rest of Rönnbäcksnäset. Final assay results and a revised geological model from this phase of drilling are pending.

Based on surface sampling and interpretation of ground magnetic data, the Company has drilled at three target areas outside the main deposits; Area 11, -13 and -7. Two of these targets (Area 11 and Area 7) are reported by the Company to have returned encouraging results and may add to the Project mineral resource indue course, should continued exploration activity at these sites prove to be successful (Figure 9-4). A discussion of the preliminary results obtained for each area is provided below.

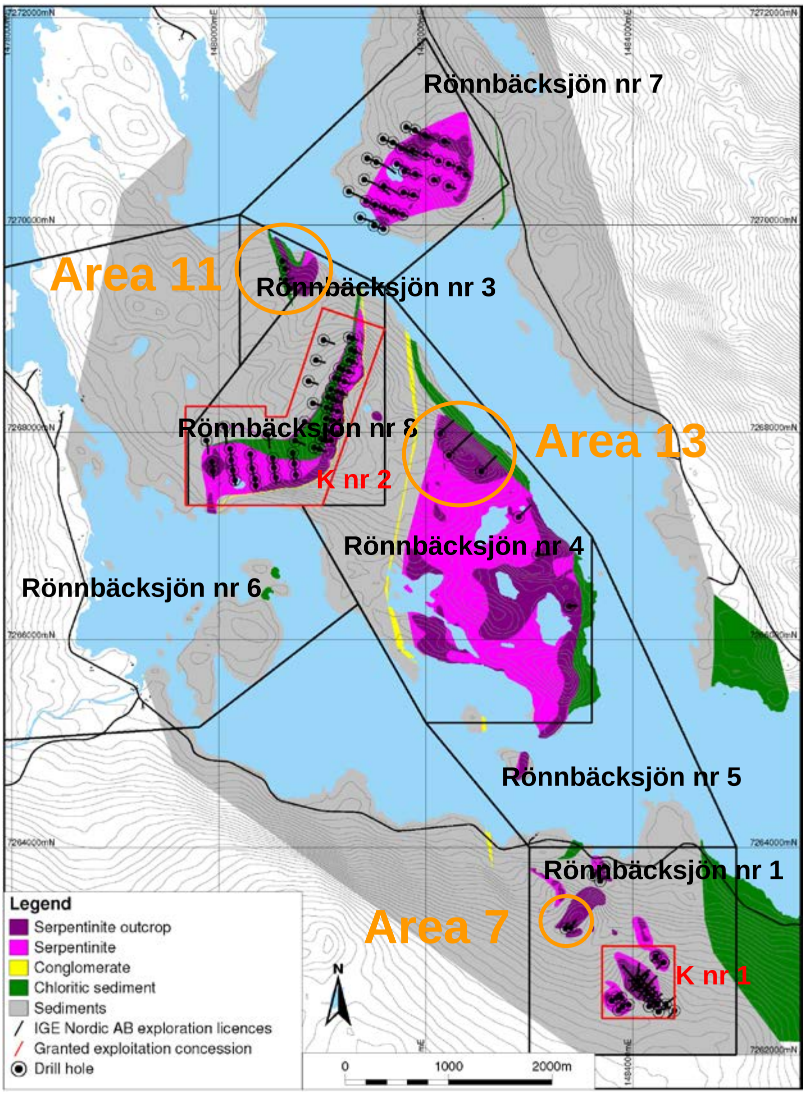

Figure 9-4: Bedrock map illustrating exploration drilling in the Project area outside the main deposits.

Area 7

Two holes (VIN125-126) were drilled on a large ultramafic outcrop of rock northwest of the Vinberget deposit. Only two holes has been drilled to-date on the southern border of this outcrop, but, initial results from these holes are encouraging, and justify further drilling in this area in order to determine its potential.

Area 11

Four holes (RON207-210) were drilled on an outcrop of ultramafic rock on Rönnbäcksnäset Island, located just opposite to the Sundsberget deposit. The holes were drilled on the western side of the outcrop at a spacing of approximately 100 m. The positive assay results from this area justify further drilling in the area in order to determine the resource.

Area 13

The last area to be drilled is situated east of the Rönnbäcksnäset deposit. Six holes (RON211-216) have been drilled to test out the eastern side of the island. Assay results and core logging do not indicate any strong serpentinization in the area.

The following tables give the drill-hole coordinates and assay results for the drill-holes from area 7 and 11.

Table 9-1: Drillhole collar data (Coordinates in RT90 2.5 gon V 0:-15)

Hole	North	East	Azimuth	Dip	Total length
	(m)	(m)	(°)	(°)	(m)
Area 7 (Vinberget)					
VIN125	7263225	1483400	30	-45	137.0
VIN126	7263220	1483348	20	-45	68.6
Area 11 (Rönnbäcksnäset)					
RON207	7269461	1480637	70	-45	127.0
RON208	7269385	1480690	40	-45	9.6
RON209	7269577	1480636	90	-45	59.0
RON210	7269654	1480609	90	-45	60.8
Area 13 (Rönnbäcksnäset)					
RON211	7267625	1482533	45	-45	283.1
RON212	7267779	1482222	45	-45	428.0
RON213	7267988	1482133	45	-45	262.0
RON214	7267182	1482894	90	-45	241.9
RON215	7266824	1483076	90	-45	181.6
RON216	7266328	1483386	90	-45	118.5

Table 9-2: Drillhole assay data

	From		Length	Total Ni	Ni-AC	Co	S
Hole #	(m)	(m)	(m)	(%)	(%)	(%)	(%)
Area 7 (Vinberget)							
VIN125	76	120	44	0,20	0,12	0,010	0,09
VIN126	0,2	60	59,8	0,22	0,09	0,011	0,04
Area 11 (Rönnbäcksnäset)							
RON207	20	108	88	0,22	0,09	0,011	0,04
RON208	54	81,4	27,4	0,21	0,11	0,010	0,09
RON209	0,4	44	43,6	0,21	0,08	0,011	0,03
RON210	0	34	34	0,21	0,09	0,011	0,03
Area 13 (Rönnbäcksnäset)							
RON211	1,4	204	202,6	0,23	0,03	0,011	0,01
RON212	104	398	294	0,23	0,04	0,012	0,02
RON213	3,2	252	248,8	0,23	0,06	0,011	0,02
RON214	0,9	241,9	241,1	0,25	0,01	0,012	0,01
RON215	18,9	181,6	162,7	0,26	0,02	0,012	0,01
RON216	0	118,2	118,2	0,23	0,02	0,011	0,01

Ni-AC (represents nickel in sulphide, which is extractable) analyses were conducted by Labtium Oy, Rovaniemi, Finland. Total Ni, Co and S analyses were conducted by ALS Laboratory Groups, Vancover, Canada, and check assays were performed by ACMELabs, Vancover, Canada.

10 DRILLING

10.1 Introduction

The Company reported that a total of 21 historic holes were drilled by Boliden in the 1970s for 443.5 m.

The Company commenced its Phase 1 drilling campaign, comprising approximately 8,000 m, in April 2008. Phase 2, also approximately 8,000 m, started in October 2008, with drilling completed in January 2009. Subsequent to this, a drill programme was initiated in December 2009 and continued during 2010. Drilling at Rönnbäcksnäset was on-going during SRK's field visit in February 2011. All drilling has been diamond core drilling.

All diamond drilling by the Company to date has been performed by the contactor Styrud Arctic AB (Styrud), previously known as Bergteamet AB and RATE Diamantborrning AB. Initially, two Onram 1000 drill rigs were used. These were later changed to Atlas Copco DIAMEC U6 rigs. Both drill rig types were mounted on Morooka 1500 band dumpers to drill BTW core (42 mm). Drilling in the Project area has consistently been undertaken using environmentally certified hydraulic fluids to minimise environmental impacts in the event of leakage.

Figure 10-1: Drill rig in operation on Vinberget

The Rönnbäcken drillhole database provided by the Company and used by SRK to derive the mineral resource estimates for Rönnbäcksnäset and Vinberget includes information for 110 drillholes, for a total of 17,193.35 m. The database does not include any holes drilled by Boliden and as such no historic drill data was used. The drillhole database provided by MRG for review by SRK included 33 drillholes for a total of 7,111 m.

Due to an absence of supporting information at the time of the Mineral Resource Estimate (MRE) conducted by SRK, certain drillholes were excluded from the database as received from the Company, as presented below in Table 10-2.

A summary of drillholes, total metres drill and associated total number of nickel assays (ammonium citrate method) used to derive the mineral resource estimates presented in this report is summarised in Table 10-1 below.

Table 10-1: Summary of drillholes by deposit used in this Mineral Resource estimate (Ni_AC = Nickel in sulphide by ammonium citrate leach and ICP-AES finish)

Deposit	Number of drill holes	Metres drilled	Metres assay by Ni_AC method
Rönnbäcksnäset	54	7,770	5,124
Vinberget	38	7,602	6,723
Sundsberget*	33	7,111	5,856

*database supplied by MRG

Table 10-2: Company drillholes excluded from the database

Hole ID	Reason for exclusion		
	Absent Geological Data	Absent Assay Data	Absent Survey Data
VIN113	X	X	
VIN114	X	X	
VIN115	X	X	
VIN116	X	X	
VIN117	X	X	
VIN118	X	X	
VIN119	X	X	
VIN120	X	X	
VIN121	X	X	
VIN122	X	X	
VIN123	X	X	
VIN124	X	X	
VIN111			X
VIN22		X	X
VIN50			X
VIN119			X
VIN25		X	
RON51			X
RON76			X
RON200			X

10.2 Vinberget

Steep slopes on either side of the Vinberget ridge dictated the drilling pattern at Vinberget. Drilling was carried out in fans from a several positions at the top of the ridge and designed to achieve a horizontal distance between holes of 50 to 60 m at a downhole depth of 150 m. The drillhole locations are shown in Figure 10-3.

The Company reports that drilling conditions were for the most part favourable, with occasional clay zones causing bogging of the drill rods, particularly when drilling towards the southwest.

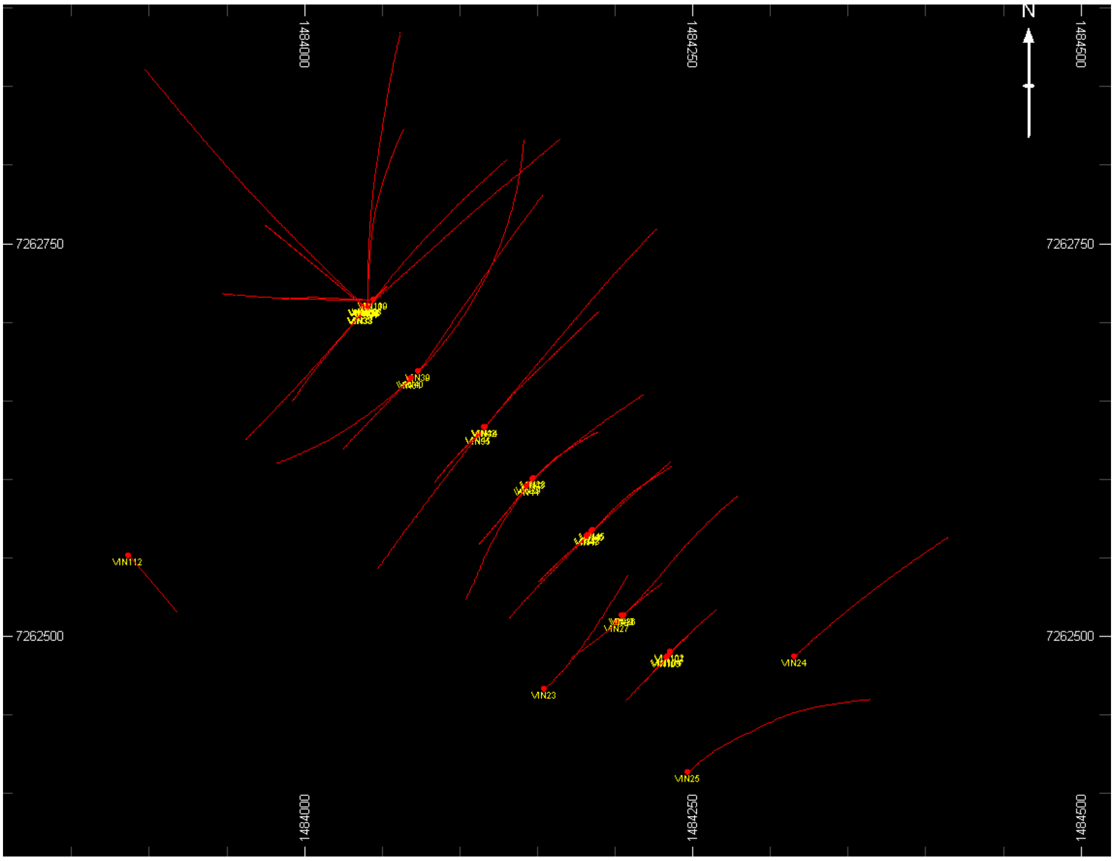

Figure 10-2: Vinberget drillhole collar locations

10.3 Rönnbäcksnäset

Drilling began on Rönnbäcksnäset northeast with three drill holes at 50 m intervals along 100 m sections. Thereafter, a fourth hole was drilled in every second section to check for a possible downward extension of the mineralization and to investigate the associated magnetic anomaly. The drillhole locations are shown in Figure 10-3.

Drilling conditions at Rönnbäcksnäset were found to be more variable than in Vinberget given the lower competency of some units.

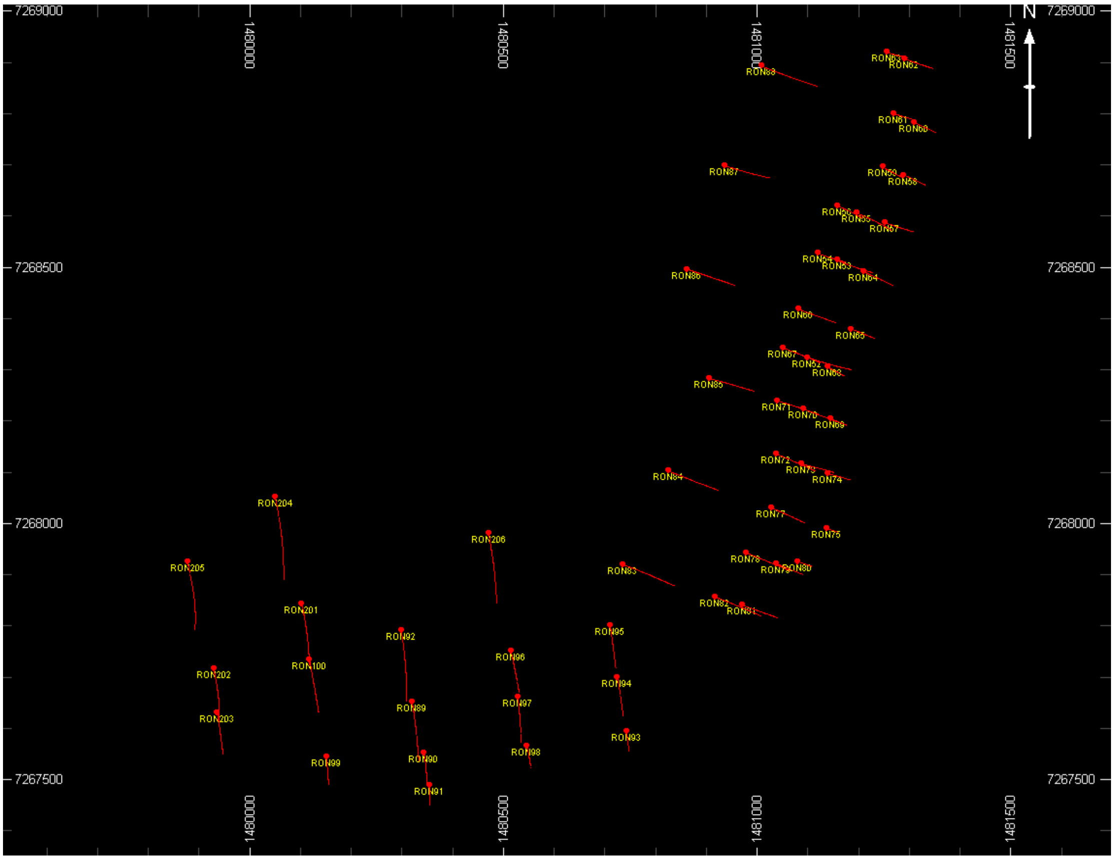

Figure 10-3: Rönnbäcksnäset drillhole collar locations

10.4 Sundsberget

The drillhole database supplied by MRG shows that drilling at Sundsberget was carried out on a 200 m line spacing with an approximate 80 m across strike spacing. The drillhole locations are shown in Figure 10-4.

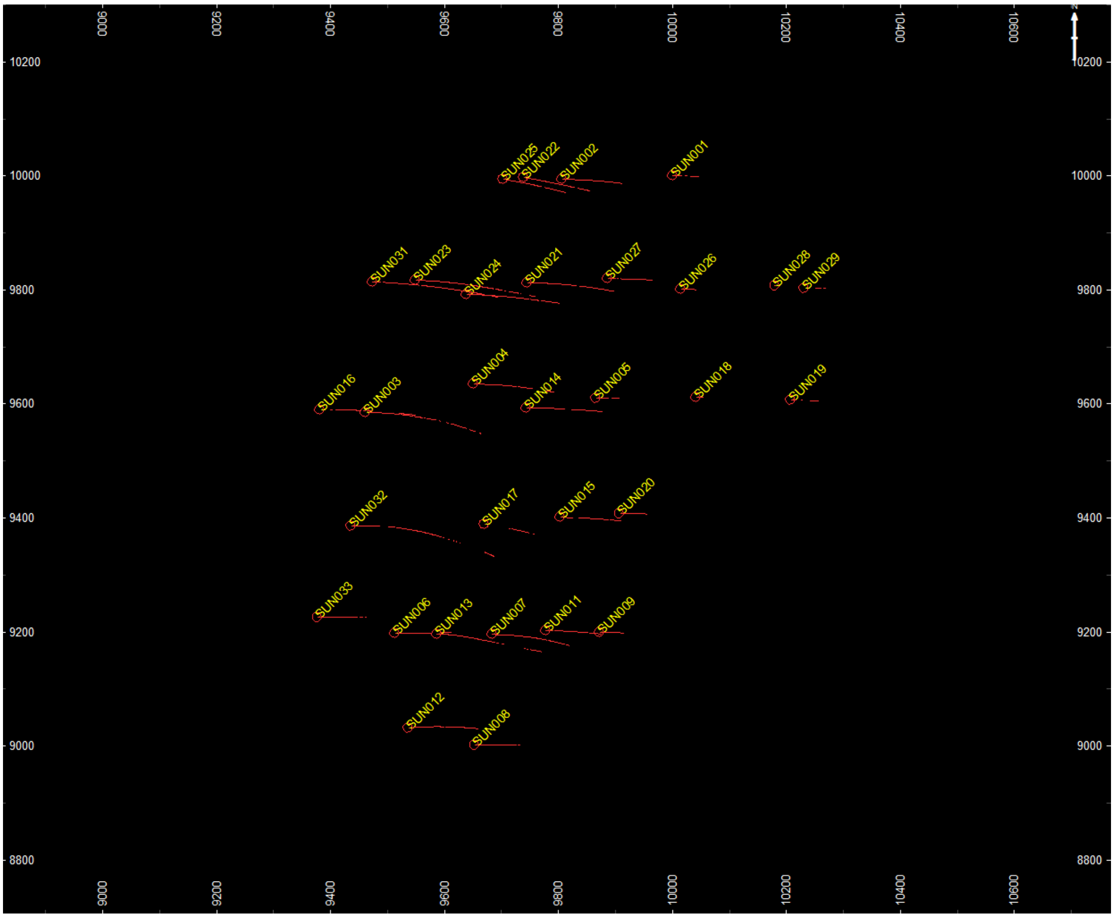

Figure 10-4: Sundsberget drillhole collar locations

10.5 Casing

The Company has indicated that casings above ground level were cut in accordance with Swedish Association of Mines, Mineral and Metal Producers' (SveMin) guidelines to less than 10 cm above ground, and sealed with the cap stamped with the drillhole number. SRK was however unable to confirm this due to the depth of snow cover at the time of the site visit.

10.6 Downhole Surveys

All the deviation surveys were performed using a Reflex Maxibor II instrument which measures the trace of the drillhole with optical technology. Surveys from Vinberget and Rönnbäcksnäset were mainly carried out by Company staff and, to a lesser extent, by contractors Sten Wikström, Skellefteå Bergsupport AB and/or Elin Broström, Styrud.

The database received by SRK from the Company includes 50 holes from Vinberget, for a total of 3,512 records and 54 holes from Rönnbäcksnäset for a total of 2,949 records.

The database received by SRK from MRG includes 33 holes from Sundsberget, for a total of 2,064 records. These surveys were mainly carried out by Company staff and, to a lesser extent, by contractor Sten Wikström.

10.7 Collar Surveys

Drillhole locations were set out using a hand-held GPS. The collars were later surveyed by Tyréns and Mikael Norén using Leica System 1200 GPS technology, using the following projection and with the following measurement accuracy as presented in the final report, 2008-11-03:

- plan projection: RT 90 2,5 gon V 0:-15;
- accuracy in plan projection ±2 to 3 cm;
- vertical projeciton: RH 70; and
- accuracy in vertical projection ±3 to 4 cm.

The Company indicated that drill collar azimuths were calculated from two survey points, one from the top of the casing and another at the top of a 3 m long steel rod that were put down 1 m inside the casing.

Holes drilled after 2008-11-03 were surveyed by hand-held Garmin 60csx GPS only. A list of these holes is presented in Table 10-3 below.

Table 10-3: Drillhole surveyed with handheld GPS

Vinberget	Rönnbäcksnäset	Sundsberget	
VIN109	RON99	SUN01	SUN18
VIN110	RON100	SUN02	SUN19
	RON200	SUN03	SUN20
	RON201	SUN04	SUN21
	RON202	SUN05	SUN22
	RON203	SUN06	SUN23
	RON204	SUN07	SUN24
	RON205	SUN08	SUN25
	RON206	SUN09	SUN26
		SUN10	SUN27
		SUN11	SUN28
		SUN12	SUN29
		SUN13	SUN30
		SUN14	SUN31
		SUN15	SUN32
		SUN16	SUN33
		SUN17	

10.8 Core Logging

A Company geologist or field technician inspected drill core at the site during drilling on a continuous basis and stopped the drilling at a predetermined depth in mineralized material or at a particular lithological intersection. The drill contractor was responsible for transportation of the drill core to the Company's core archive and logging facility in Skellefteå.

The core was photographed and logged at the Company's logging facility. All of the Company's drill cores were logged by staff members or sub-contractors to capture relevant geological and geophysical (susceptibility logs) information. The geologic logging intervals were based on lithological variations in the rock and in addition a qualitative estimate of fibrous asbestiform mineral content was noted.

Rock Quality Designation (RQD) measurements were taken on the basis of the assay intervals (roughly every 2 m).

Initially, magnetic susceptibility was measured at every metre in mafic and ultramafic intersections, using a SM-20 instrument manufactured by GF Instruments. This procedure was abandoned in the 2009-2010 drilling campaign, with susceptibility measurements henceforth being taken only on coarse rejects representing assayed intervals. The database received by SRK from the Company contains a total of 2,703 magnetic susceptibility measurements from Rönnbäcksnäset drill core and 3,414 from Vinberget drill core. Magnetic susceptibility data was not supplied by MRG for the Sundsberget deposit.

Density measurements were carried out by Company staff members or sub-contractors at the core logging facility using the water immersion method on unsealed drill core. Within the serpentinite, density measurements were taken at every assayed interval (every 2 m). Representative density measurements were also taken for the main waste rock lithologies. The Company database contains a total of 2,701 measurements for Rönnbäcksnäset, 3,416 measurements for Vinberget and 2,972 measurements for Sundsberget, as reported by MRG.

The specific gravity of the core was measured to obtain densities for use in the resource estimation procedure but also to get a value of the degree of serpentinization. The transformation of olivine to serpentine lowers the density from greater than 3.0 g/cm^3 to 2.7 g/cm^3. For similar reasons, the magnetic susceptibility has been surveyed on drill core, outcrops, and on sample bags of the coarse rejects from the sample preparation. Magnetite is formed as a secondary product during serpentinization.

All core logging data was recorded onto paper and later entered into Microsoft Excel spreadsheets. A hardcopy check list was prepared and completed as standard for each drillhole by the supervising geologist / technician to maintain data capture protocols.

10.9 Interpretation of Results

On the basis of the Company's drilling, mineralisation wireframes were digitized by SRK for Rönnbäcksnäset Vinberget using Datamine software.

Rönnbäcksnäset is the larger of the two mineralised wireframes and contains the most drillhole intercepts. It measures 2.5 km along strike, 1.6 km on a 16^0 azimuth, in the northwest, and 1.2 km along strike on an 85^0 azimuth in the southeast. The south-eastern portion has a maximum true thickness of roughly 350 m and dips at 25^0 towards the north-northwest, while the northeast portion has a maximum true thickness of roughly 60m and dips at 40^0 towards the west-northwest.

The Rönnbäcksnäset wireframe was modelled to an elevation of -1m (ASL), and contains 337.4 Mm^3 of material.

Vinberget measures 686 m along strike, on an azimuth of 321º, and 300 m across strike at the widest point. It was modeled to a depth of 307m (ASL), with a sub-vertical dip. The Vinberget wireframe contains 22.3 Mm^3 of material.

On the basis of the Company's drilling, a single mineralisation wireframe was digitized by MRG for Sundsberget again using Datamine software. Sundsberget measures 1,200 m along strike on an azimuth of 10º, and 500 to 600 m across strike at the widest point. It was modeled to a maximum down dip depth of approximately 500 m from surface, with a dip of 40° to the west and contains 183.5 Mm^3 of material.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Samples for Assay

All serpentinite core intersections were sampled, along with most of the weakly mineralised mafic unit at Rönnbäcksnäset to ensure that all sulphide nickel mineralisation was entirely captured.

The core was marked for sampling by Company staff geologists or sub-contracting technicians, starting at the contact of the mineralization and then every two metres beginning at the first even numbered metre. Consequently every sample at the footwall and hanging wall of the mineralized material represents a non-regular length.

Two metre sample intervals were initially selected to better understand the distribution of the accessory mineralization and to provide sufficient detail to correlate possible layered ultramafics. Sample intervals and numbers were either recorded onto paper and then entered into a Microsoft Excel spreadsheet, or entered directly into Microsoft Excel. SRK considers that the Company has sampled the host serpentinite in an unbiased fashion using a consistent technique for all intersections.

Once assay results were issued by the laboratory (Labtium) in Excel format, they were merged with the sample interval data in Excel by either the Company's exploration manager or the project geologist. The merged files were imported into Micromine and validated.

SRK notes that no systematic logging of core recovery has been carried out by the Company. However, serpentinite intersections in drill core observed during the field visit to Rönnbäcken and the Company's core logging and storage facilities in Skellefteå in 2010, showed very good recovery and generally good quality core. SRK does not consider core loss to be a material issue with regards resource estimation in this case.

Figure 11-1: Typical example of drill core quality in Serpentinite from the Project area (Photo represents core from hole 16 at a depth of 275 m, Sundsberget)

11.2 Thin Section Samples

SRK understands that samples were systematically collected by the Company for thin section work at approximately 40 m intervals.

11.3 Samples for Metallurgical Tests

Two samples of 20 to 30 kg each were taken from the old Boliden test pit and were tested at Minpro AB. The pit is located at the road one kilometre north of the drilled area at Vinberget. The sample was a composite sample comprised of small fragments collected from throughout the pit.

In an early stage of the drill programme, five samples of 30 to 35 kg each were taken for new tests at the Outokumpu Research Centre (ORC), three from Vinberget and two from Rönnbäcksnäset. The three samples collected from Vinberget comprise coarse reject from the sample preparation of two drillholes, VIN30 and VIN29. The sample from VIN30 represented one lower grade zone higher up and one higher grade zone deeper down in the hole. A third similar type of sample was collected in VIN29. At Rönnbäcksnäset, two samples were collected from drillhole RON53 in the same way as in VIN30. RON53 is located in the north-eastern part of the Rönnbäcksnäset deposit. For the second phase of testing at ORC, two composite samples were prepared using coarse rejects from the two drilled areas at the end of the drill programme.

At Vinberget, all sample rejects were composited into a 2.5 t sample from selected holes. The 1,008 sample intervals and 1,216 m of core drilling represented intersected the mineralization at depths of between 630m to 500 MASL. At Rönnbäcksnäset, the samples were selected from coarse rejects from 15 core holes drilled in the south-western part of Rönnbäcksnäset.

The samples were split in two halves, with one half included in the sample. A total of 264 samples were included, weighing 366 kg and representing 528 m core drilling. This in turn represents approximately two years of production from the area down to 400 MASL. The sample was dominated by the upper pyroxene bearing serpentinite and comprises relatively little of the higher grade lower serpentinite zone with similarities to the Vinberget serpentinite. The sample also does not include any of the low grade mafic intrusion material or the low grade zone with almost no sulphides.

SRK has recommended to the Company that it does further work to clarify metallurgical sample provenance (hole number, interval and sample weight) for more samples to support future studies and specifically to verify metallurgical sample representivity, to understand test work results in the context of deposit geology and to provide support to core sample assays via reconciliation of concentrate grade with original sample grade.

11.4 Sample preparation

The sample preparation was conducted by a company independent of the Company, namely ALS Chemex in Piteå, Sweden.

The aim of the sampling has been to delineate mineralization that could be recovered by established metallurgical methods, i.e., flotation of sulphide minerals. The adapted assay technique was therefore a partial-leach that selectively dissolves nickel in sulphides and leaves the nickel bearing silicates and oxides unaffected. As the sulphur content is low, analyses of sulphur must be performed by methods with low detection limits, better than or equal to 0.01% S.

As the selective nickel leaching technique is not an accredited method for assaying nickel in sulphides, other accepted methods were included in the assay package such as Aqua Regia leach and Near Total Four Acid Leach. To support the values of the grades of nickel in sulphides, mineralogical studies and metallurgical tests were also carried out by the Company and are discussed elsewhere in this report.

Sulphur assays from Four Acid and Aqua Regia digestion give higher sulphur values, when compared with associated sulphur-AC results. Sulphur assays using the ammonium citrate technique are thought to dissolve the free milled and the exposed sulphides at oxide and silicate mineral surfaces and thereby present a better indication of the nickel sulphides amenable to recovery by conventional milling and flotation techniques.

11.5 Chain of Custody and Sample Preparation

The drill contractor was responsible for transportation of the drill core from site to the Company's core archive and logging facility in Skellefteå.

During the logging stage, the core was measured and sample intervals selected by staff geologists or sub-contracting technicians for sample analyses. These intervals were marked on the core and on the core boxes.

ALS Sweden AB, a subsidiary of ALS Chemex (ALS), was contracted to split the core and carry out the sample preparation. A separate room for sample preparation was set up for the Project as a precaution against the health risks associated with asbestos.

The samples were logged in the tracking system, weighed, and split with a diamond saw (Almonte Core Saw). One half of the sawed core was treated according to ALS code PREP-31, which included drying and crushing to 70% -2 mm (Tyler 9 mesh, US Std Nr 10). A split of up to 300 g was taken and pulverized to 85% -75 µm (Tyler 200 mesh, US Std Nr 200). The 300 g sample pulp was then split in two or three subsamples and sent to two different primary assay laboratories (Labtium and ALS Chemex). A third laboratory (ACME) was used for the control assays.

The remainder of the coarse reject was labelled with the analytical number and stored at the assay laboratories. After a holding period at the laboratories, all of the rejects and pulps were returned to the the Company storage facility in Skellefteå. The pulps at Labtium Oy in Rovaniemi, Finland (Labtium), duplicates of the pulps stored in Skellefteå, have been discarded.

A more detailed description of the sample preparation process is illustrated in the flowchart in Figure 11-2. Note that the sample split was modified to up to 300 g instead of 250 g.

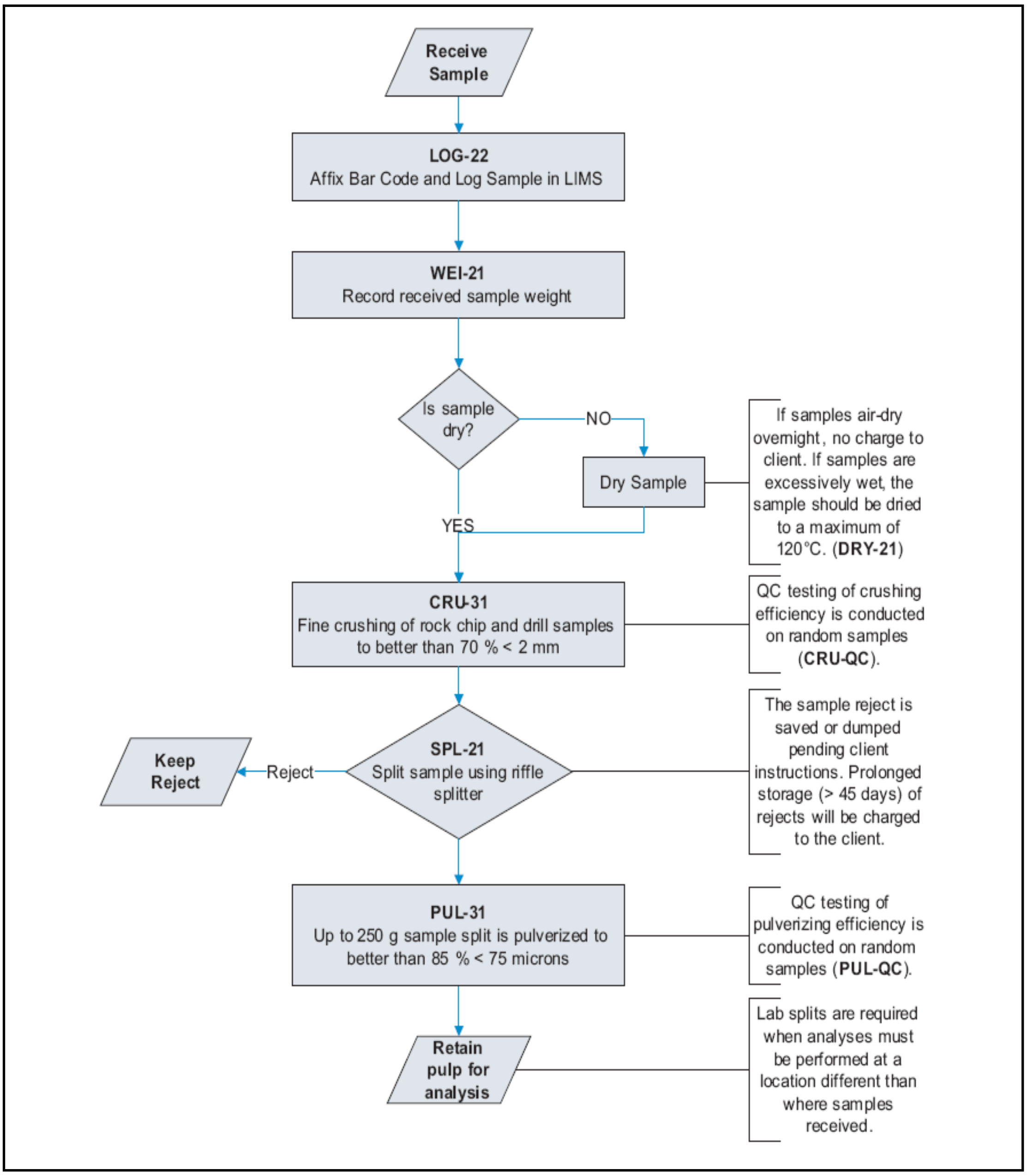

Figure 11-2: Sample preparation flow sheet (modified from ALS Chemex 2009)

11.6 Sample Analysis

Two assay laboratories were contracted for the analyses: Labtium in Rovaniemi, Finland, and ALS in Vancouver, Canada. Check analyses were mainly performed by Acme Analytical Laboratories Ltd (Acme) in Vancouver, Canada. The analyses carried out by the three laboratories are summarised Table 11-1 and Table 11-2 below.

Table 11-1: Analytical methods 2008-2009, Vinberget & Rönnbäcksnäset

Lab	Lab code	Sample digestion	Type	Sample size (g)	Analytes	Main interest	Use
ALS Chemex	ME-MS81	Four-acid	Near total	0.25	38	Ni, Co, S	Original
	ME-4ACD81	Four-acid	Near total	0.25	9	Ni, Co	Original
	ME-ICP06	Four-acid	Near total	2	14	Major Element	Original
	ME-ICP61	Four-acid	Near total	0.25	33	Ni, Co, S	Original
	PGM-ICP23	Fire Assay	Total	30.00	3	Au, Pt, Pd	Original
Labtium	240P	H2O2+NH4 citrate	Sulphides	0.15	3	Ni, Co	Original
	510P	Aqua regia	Partial	0.15	14	Ni, S	Original
Acme	G7TD	Hot four-acid	Near total	0.50	23	Ni, S	QC
	8NiS	H2O2+NH4 citrate	Sulphides	1.00	1	Ni	QC
Labitum	307P	HF +HClO4	Near total	0.20	13	Ni	QC
	720P	Na2O2 Fusion	Total	0.20	12	NI, S	QC

Table 11-2: Analytical methods 2009-2010, Sundsberget

Lab	Lab code	Sample Digest	Digest Type	Analy.	Samp. size (g)	Analytes	Main interest	Use
ALS Chemex	ME-4ACD81	Four acid	Near total	ICP-AES	0.25	9	Ni, Cu, Co	Original
	ME-MS81	Lithium borate fusion	Total	ICP-MS	0.2	38	Ni, Cu, Co	Original
	ME-ICP06	Lithium borate fusion	Total	ICP-AES	0.2	13	Whole rock	Original
	ME-MS42	Aqua regia	Near total	ICP-MS	0.5	6	As, Bi, Hg, Sb, Se, Te	Original
	OA-GRA05	Fusion	Total	Gravimetric	1	1		Original
	TOT-ICP06	Calculation based on LOI and ME-ICP06				1		Normal
	PGM-ICP23	Fusion	Total	Fire Assay (ICP-AES)	30	3	Au, Pd, Pt	Original
	C-IR07	High temp evolution	Total	Leco furnace		1	C	Original
	S-IR08	High temp evolution	Total	Leco furnace		1	S	Original
Labtium	240P	H2O2 + NH4 citrate	Sulphides	ICP-AES	0.15	4	Ni-AC, S-AC	Original
Acme	7TD	Hot four acid	Near total	ICP-AES	0.5	22	Ni, Cu, Co	QC
	8NiS	H2O2 + NH4 citrate	Sulphides	ICP-AES	1	1	Ni-AC, S-AC	QC

For exploration programmes during the past 12 months, the ME-ICP61 method was replaced by a “Complete Characterisation Package” which includes the methods ME-ICP06, ME-4ACD81, ME-MS81. The new package is intended to provide additional information on rock type to aid in the geological interpretation.

The database received by SRK from the Company, contained a total of 6,747 analyses, of which 6,125 related to primary core samples while 622, or 10%, comprised a variety of QA/QC analyses. This is considered by SRK to be a reasonable number of check assays. A summary of the analyses is presented in Table 11-3 below. MRG report a total of 293 QA/QC analyses, being 10% of the assays, for Sundsberegt.

Table 11-3: Analysis Summary

Deposit	Core	Duplicates	UM-4 (reference material)	Blank	Acme check	Coarse reject	Sub-total QC	Total assay	Labtium internal duplicates
VIN	3419	107	68	76	68	15	334	3753	130
RON	2706	94	58	66	56	14	288	2994	105
Total	6125	201	126	142	124	29	622	6747	235
SUN*	2934	116	72	72	33	-	293	2934	-

*MRG data

SRK notes that for low grade sulphide deposits such as Rönnbäcken, the silicate nickel contribution to the nickel assay can be significant. For this reason the Company has elected to characterise both the total nickel content and the nickel content in sulphide, the latter by partial leach methods, namely Labtium’s 240P method (ammonium citrate and hydrogen peroxide).

11.6.1 Labtium

Labtium has FINAS T025 accreditation ISO/IEC 17025:2005. According to FINAS, “a laboratory's fulfilment of the requirements of ISO/IEC 17025:2005 means the laboratory meets both the technical competence requirements and management system requirements that are necessary for it to consistently deliver technically valid test results and calibrations. The management system requirements in ISO/IEC 17025:2005 are written in language relevant to laboratory operations and meet the principles of ISO 9001:2008 Quality Management Systems Requirements and are aligned with its pertinent requirements”. This accreditation represents a higher standard than ISO 9001:2000. According to the website of Labtium, “Labtium’s quality system fulfills the requirements of the Standards Council of Canada (CAN-P-1579), Guidelines for Accreditation of Mineral Analysis Testing Laboratories”. However, the ammonium citrate leach procedure is not covered by the accreditation as the method is relatively new to Labtium.

Ammonium citrate hydrogen peroxide leach (AC), Labtium code 240P, is described as follows. A 0.15 g subsample is leached in a mixture of ammonium citrate and hydrogen peroxide (1:2; total volume 15 mL). The leach is done on a shaking table for two hours at room temperature. The solution is decanted from the sample powder directly after the leach. The solutions are diluted (5:1) and measured with ICP atomic emission spectroscopy (ICP-AES). It is a partial leach and is selective at dissolving nickel, cobalt, and copper from sulphide mineral species while leaving those elements in silicates unaffected. The detection limits are 10 ppm.

This method was used to determine the recoverable nickel content for this Project, that is, specifically to obtain accurate estimates of the metals that can be recovered by established metallurgical methods, such as flotation.

Aqua regia digestion, laboratory code 510P at Labtium, is described as follows. A 0.15 g subsample is digested with aqua regia (3:1 mixture of concentrated hydrochloric acid and concentrated nitric acid) by heating at 90°C in an aluminium-heating block for 1.5 hours and diluted to 15 mL with water. An aliquot is centrifuged before instrumental analysis. Aqua regia is a partial leach for silicates, but is an almost complete leach for sulphides and oxides. It is a much better leach for this Project than the “near total” leach, however, as silicates are partially dissolved, even this method will overestimate the metal content. It is mainly included as a comparison to the sulphide nickel method for the sulphur content and other elements, such as arsenic, that can exist in sulphide phases.

The results from Labtium are reported with three significant digits (zero uncounted) or <X where X is the detection limit. The latter is preferable to the ALS reporting method, even if the last digits are not significant.

The Company report that for the 2009-2010 exploration programme, aqua regia digestion has been abandoned in preference for the 240P method.

11.6.2 ALS

ALS is accredited by ISO 9001:2000 overall and conforms to the requirements of CAN-P-1579 and CAN-P-4E (ISO/IEC 17025:2005) by the Standards Council of Canada (SCC) for a number of specific test procedures, including the two methods employed by the Company.

ALS code ME-ICP81 requires the pulp to be digested with perchloric, nitric, hydrofluoric, and hydrochloric acids (HNO3-HClO4-HF-HCl). The residue is topped up with dilute hydrochloric acid and the resulting solution is analyzed by ICP-AES. Results are corrected for spectral inter-element interferences. Four acid digestions are able to dissolve most minerals. However, although the term “near-total” is used, depending on the sample matrix, not all elements are quantitatively extracted. Therefore, the leach is less useful to the Project as an estimate of recoverable metals. It is mainly included to demonstrate the need of the partial leach method and to provide an extra check of sulphur content. The elements analyzed and ranges of the procedure are shown in Table 11-4,

Table 11-4: Analytes and Ranges of ME-ICP81

Analytes & Ranges (ppm)							
Ag	1-1,000	Ga	0.1-1,000	Pb	5-10,000	Tm	0.01-1,000
Ba	0.5-10,000	Gd	0.05-1,000	Pr	0.03-1,000	U	0.05-1,000
Ce	0.5-10,000	Hf	0.2-10,000	Rb	0.2-10,000	V	5-10,000
Co	0.5-10,000	Ho	0.01-1,000	Sm	0.03-1,000	W	1-10,000
Cr	10-10,000	La	0.5-10,000	Sn	1-10,000	Y	0.5-10,000
Cs	0.01-10,000	Lu	0.01-1,000	Sr	0.1-10,000	Yb	0.03-1,000
Cu	5-10,000	Mo	2-10,000	Ta	0.1-10,000	Zn	5-10,000
Dy	0.05-1,000	Nb	0.2-10,000	Tb	0.01-1,000	Zr	2-10,000
Er	0.03-1,000	Nd	0.1-10,000	Th	0.05-1,000		
Eu	0.03-1,000	Ni	5-10,000	Tl	0.5-1,000		

More detailed descriptions of ALS codes ME-4ACD81 and ME-MS81 follow. For ME-4ACD81, a prepared sample (0.25 g) is digested with perchloric, nitric, hydrofluoric and hydrochloric acids. The residue is topped up with dilute hydrochloric acid and the resulting solution is analyzed by inductively coupled plasma-atomic emission spectrometry. The results are corrected for spectral inter element interferences. For ME-MS81, a prepared sample (0.200 g) is added to lithium metaborate flux (0.90 g), mixed well and fused in a furnace at 1000°C. The resulting melt is then cooled and dissolved in 100 mL of 4% nitric acid. This solution is then analyzed by inductively coupled plasma-mass spectrometry.

Table 11-5: Elements analysed and their ranges for ME-4ACD81 and ME-MS81

Analytes and Ranges (ppm)							
ME-4ACD81							
Ag	0.5 – 1,000	Co	1 – 10,000	Ni	1 – 10,000		
As	5 – 10,000	Cu	1 – 10,000	Pb	2 – 10,000		
Cd	0.5 – 500	Mo	1 – 10,000	Zn	2 – 10,000		
ME-MS81							
Ag	1 – 1,000	Ga	0.1 – 1,000	Pb	5 – 10,000	Tm	0.01 – 1,000
Ba	0.5 – 10,000	Gd	0.05 – 1,000	Pr	0.03 – 1,000	U	0.05 – 1,000
Ce	0.5 – 10,000	Hf	0.2 – 10,000	Rb	0.2 – 10,000	V	5 – 10,000
Co	0.5 – 10,000	Ho	0.01 – 1,000	Sm	0.03 – 1,000	W	1 – 10,000
Cr	10 – 10,000	La	0.5 – 10,000	Sn	1 – 10,000	Y	0.5 – 10,000
Cs	0.01 – 10,000	Lu	0.01 – 1,000	Sr	0.1 – 10,000	Yb	0.03 – 1,000
Cu	5 – 10,000	Mo	2 – 10,000	Ta	0.1 – 10,000	Zn	5 – 10,000
Dy	0.05 – 1,000	Nb	0.2 – 10,000	Tb	0.01 – 1,000	Zr	2 – 10,000
Er	0.03 – 1,000	Nd	0.1 – 10,000	Th	0.05 – 1,000		
Eu	0.03 – 1,000	Ni	5 – 10,000	Tl	0.5 – 1,000		

The detection limits of PGM-ICP23 are 1 ppb for Au and Pt and 5 ppb for Pd. The upper limit is 10 ppm and has never been reached.

The results from ALS are reported by increments of the detection limits. For example, if the detection limit is 1, the result given is <1, 1, 2, 3, etc, with some exceptions such as Pb (<2, 2, 3, 4, etc.).

11.6.3 ACME

Acme is accredited as complying with ISO 9001:2000. Check assays were mostly done at Acme using the four acid digestion and ammonium citrate methods.

11.7 Quality Assurance and Quality Control (QAQC)

The Company Quality Control/Quality Assurance (QA/QC) programme comprised submitting sample blanks, standard reference samples, sample duplicates, and inter-laboratory check samples. The approximate rate of sample submissions is summarized in Table 11-6 below.

Table 11-6: QC Sample Frequency

Sample Type	Frequency
Blank	1/50
UM-4 (Reference material)	1/50
Duplicate	1/25
Interlab Check Assays	1/50

Additional checks were done on near total and total nickel on coarse rejects. In addition, the laboratories performed analyses of duplicates, in-house standards, etc, which were also forwarded to the Company. The QA/QC results from the laboratory were checked as they were returned.

11.7.1 Sample Blanks

For the Rönnbäcksnäset and Vinberget deposits, the Company submitted 142 sample blanks into the sample stream to check for contamination and drift. The blanks were prepared from pale coloured granite and were inserted by the sample preparation laboratory (ALS Chemex, Piteå). For Sundsberget, MRG report that 72 sample blanks were submitted into the sample stream to check for contamination and drift. The blanks were prepared from pale coloured granite and were inserted by the sample preparation laboratory (ALS Chemex, Piteå). Of the 72, 56 were also analysed through Labtium.

The relevant checks are for Ni, Ni-AC, and Co-AC and their detection limits are 1 ppm, 10 ppm, and 1 ppm, respectively.

11.7.2 Reference Material

Reference samples were inserted in the sample stream to check the accuracy of the assay laboratory. Reference UM-4 sample was purchased from CANMET Mining and Mineral Sciences Laboratories (CANMET) and originated from the Werner Lake - Gordon Lake district of north-western Ontario, Canada. The reference sample is intended as a reference material for the determination of ascorbic acid/hydrogen peroxide-soluble copper, nickel, and cobalt in ultramafic rocks. There are no certified standards for the sulphide selective leach method used, mostly due to the lack of laboratories offering such analytical services. Therefore, no Round Robin Test was done and no performance gates were recommended which are normally based on the Round Robin statistics. The reference grades recommended by

CANMET are 0.19% Ni and 0.007% Co.

For the Rönnbäcksnäset and Vinberget deposits, the Company submitted 126 UM-4 samples for analysis by the ammonium citrate method (Ni- AC) described in Section 11.6.1 above. For Sundsberget, MRG report that Nickel Mountain submitted 72 UM-4 samples for analysis of which 60 were also analysed through Labtium by the ammonium citrate method (Ni-AC).

11.7.3 Duplicate Pulp Samples

For the Rönnbäcksnäset and Vinberget deposits the Company renumbered and submitted 201 sample pulps to Labtium for assay as duplicates. For Sundsberget, MRG report that the Company renumbered and submitted 126 sample pulps to Labtium for assay as duplicates.

11.7.4 Duplicate Coarse Reject Samples

In the case of the Rönnbäcksnäset and Vinberget deposits, 28 samples of coarse rejects were renumbered and resubmitted for assay to test if the 70% -2 mm crush size would achieve repeatable results. For Sundsberget, MRG report that 116 samples were assayed as laboratory repeats.

11.7.5 Interlaboratory Check Assays

For the Rönnbäcksnäset and Vinberget deposits a total of 123 samples originally assayed at Labtium were submitted for assay at Acme principally as a check on the accuracy of the Ni-AC results. For Sundsberget, MRG report that 33 samples were assayed at Acme.

11.7.6 SRK Consulting Duplicate Samples

During a visit to the Company's exploration office and core archive facilities in Skellefteå, SRK collected 16 sample pulps at random from the sample pulp archive originating from the Project area. These sample pulps were re-bagged, assigned a new sample numbers and sent to Labtium for assay by method code 240P.

11.7.7 Density Measurements

Specific gravity dterminations were carried out at ALS Chemex (Piteå) on a total of 79 samples using the water immersion method. Of these samples, 44 were from Vinberget and 35 from Rönnbäcksnäset. MRG report that 2,972 dterminations were also undertaken by the Company at its base in Skellefteå using the water immersion method.

11.8 Security

11.8.1 Storage Of Drill Cores

Drill core, coarse rejects, and pulps are stored in a locked unheated storage building inside a fenced area at the Company's core depot in Skellefteå and a second secure facility at Bastuträsk.

11.8.2 Database

All project data are stored on the Company's exploration office server, with data backup. In addition, a full version of the database is managed through MRG in Perth, Western Australia, using industry standard DataShed™ software.

11.9 Summary comments

In SRK's opinion, the Company has developed appropriate logging and sample preparation procedures that enable the logical flow of the core from the drill rig through to sample dispatch; the core shed, logging, sampling and preparation facilities are clean, organised and appear well managed; appropriate security procedures are in place and the assaying has been carried out using appropriate techniques and by qualified laboratories.

12 DATA VERIFICATION

12.1 Introduction

The following sections present SRK's analysis of the QAQC data provided by the Company for Rönnbäcksnäset and Vinberget deposits. This includes blanks, reference material and duplicates as described above. The results of the QAQC have not been split into the individual deposits, but include all data supplied to SRK for both deposits combined.

12.2 Reference Material (UM-4)

Figure 12-1 shows the performance of the Labtium laboratory analysis of Ni-AC in reference material UM-4. The majority of results lie within 5% of the reference grade recommended by CANMET (0.19% Ni), there does not appear to be a bias over time and the results appear to be evenly distributed about the recommended grade.

Figure 12-2 shows the performance of the Labtium laboratory analysis of Co-AC in reference material UM-4. The majority of results lie within plus 10% to minus 5% of the reference grade recommended by CANMET (0.007% Co) but the results exhibit a slight positive skew and appear to be distributed around plus 5% of the recommended grade. There does not, however, appear to be a bias over time.

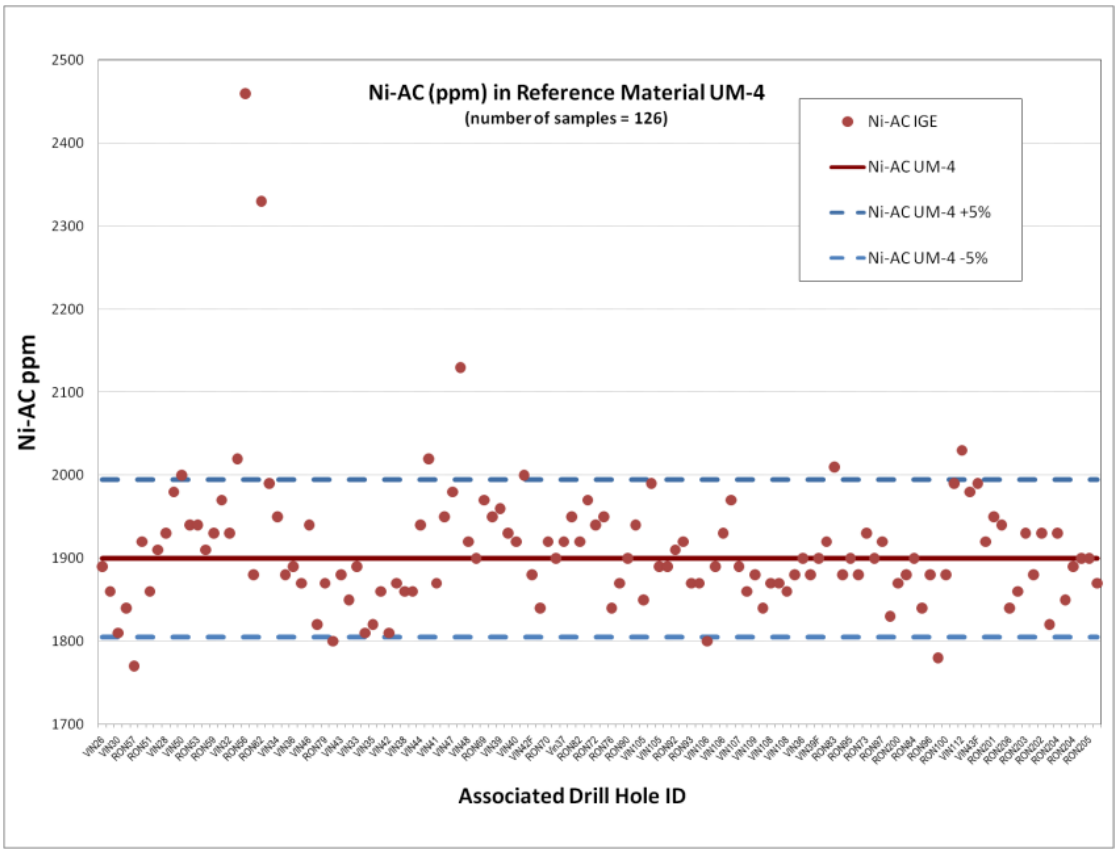

Figure 12-1: Labtium Ni-AC in Reference Material UM-4

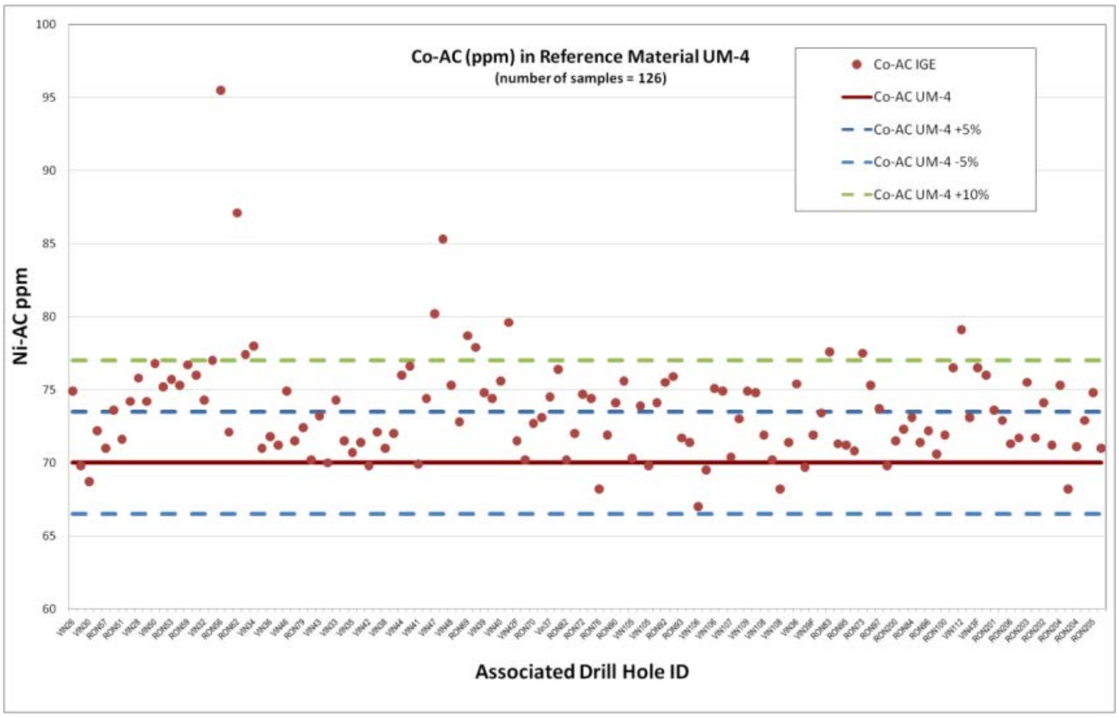

Figure 12-2: Labtium Co-AC in Reference Material UM-4

Summary - Standards

The results of the QAQC standards show that the majority of the samples fall within an acceptable range relative to the nickel and cobalt grades recommended by CANMET. Given that CANMET's recommended grades of the UM-4 reference material were obtained through a different dissolution procedure compared the methodology used by Labtium (ascorbic acid hydrogen peroxide leach as opposed to ammonium citrate hydrogen peroxide leach), SRK considers that these results indicate acceptable accuracy of assays for nickel and cobalt in sulphides.

However, SRK notes that in addition to being referenced against a different assay method, the recommended nickel grade of the UM-4 reference material lies well above typical sulphide nickel grades found in the Project serpentinites. SRK has recommended therefore that the Company considers creating reference material from a composite of Rönnbäcken serpentinite as a more suitable means of gauging future exploration Ni-AC assay precision.

12.3 Blanks

Figure 12-3 shows the performance of the Labtium laboratory analysis of Ni-AC in sample blanks. The Company replaced all results reporting at less than the detection limit to 0.5 times the detection limit, or 5 ppm Ni-AC. A total of 11 samples (8 %) had laboratory results which were at or above the detection limit of 10 ppm Ni-AC. Four samples (3%) assayed greater than twice the detection limit; that is, >20 ppm Ni-AC.

The results indicate a potential for minor contamination during sample preparation at ALS Chemex or instrument drift during assaying at Labtium. The grouped nature of slightly high results may indicate periods in which the routine cleaning of equipment between samples was not undertaken thoroughly. An alternative explanation, though perhaps more unlikely, could be that minor silicate Ni from mafic minerals in the granite was leached in these instances.

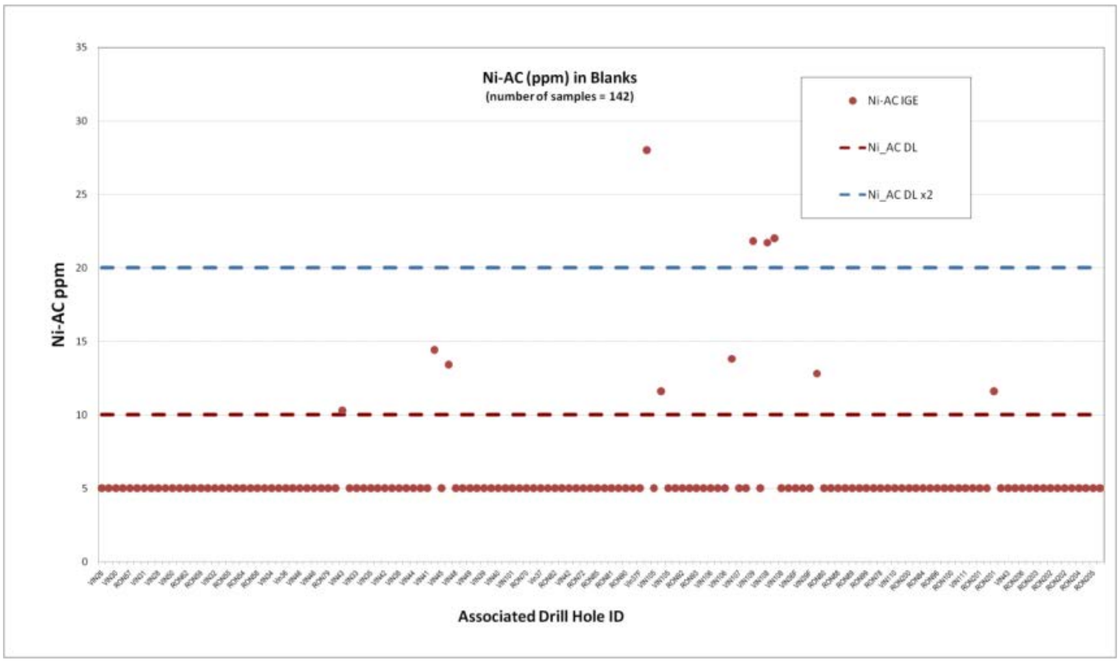

Figure 12-3: Sample Blanks Ni-AC

Figure 12-4 shows the performance of the Labtium laboratory analysis of Co-AC in sample blanks. The Company replaced all results reporting at less than the detection limit to 0.5 times the detection limit, or 5 ppm Co-AC. All samples returned values below detection limit.

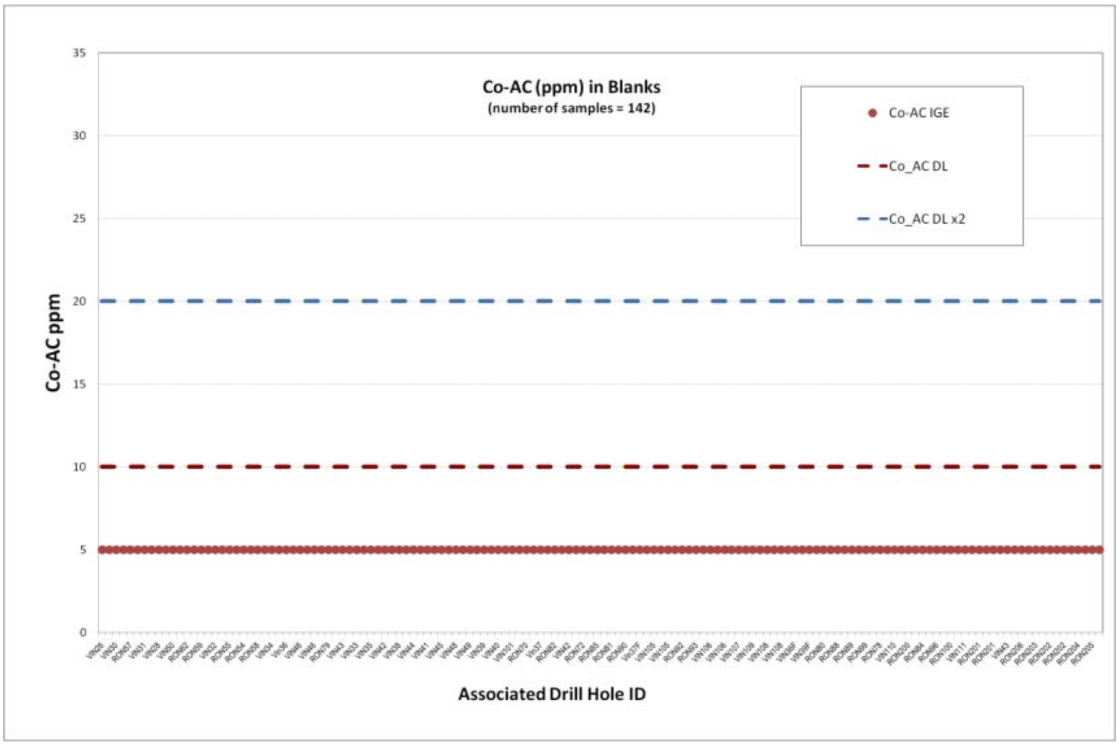

Figure 12-4: Sample Blanks Co-AC

Summary - Blanks

In SRK's opinion, the results of the sample blank assays indicate an acceptable level of contamination and drift at the sample laboratories. SRK has, however, recommended to the

Company that it considers using barren quartz material for sample blanks to eliminate any potential for contamination of Ni from mafic minerals in the granite currently used as sample blank.

12.4 Duplicates

Figure 12-5 and Figure 12-6 show the results of the laboratory duplicates for Ni-AC and Co-AC. The duplicate samples show a strong correlation to the original sample and in SRK's opinion, sample preparation and analysis shows an acceptable level of repeatability.

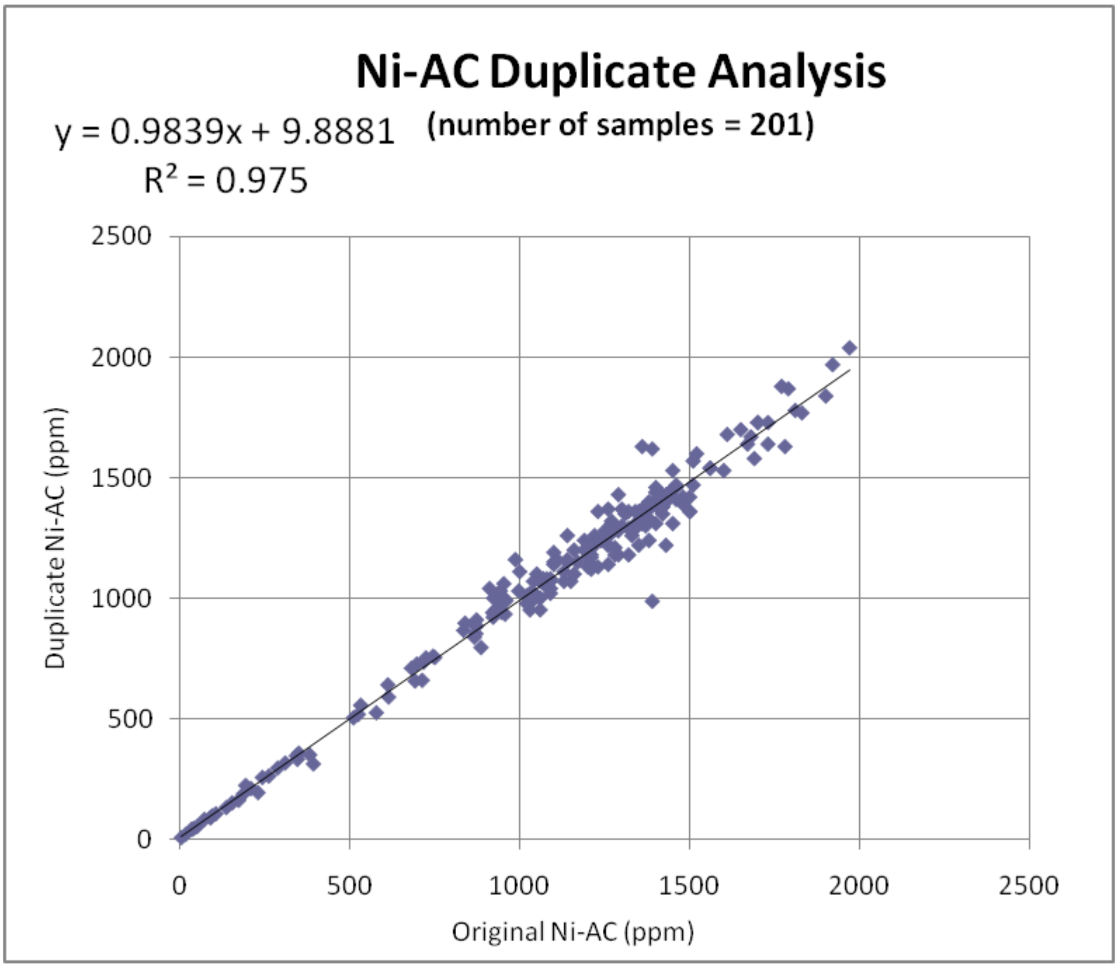

Figure 12-5: Ni-AC Duplicate Analysis

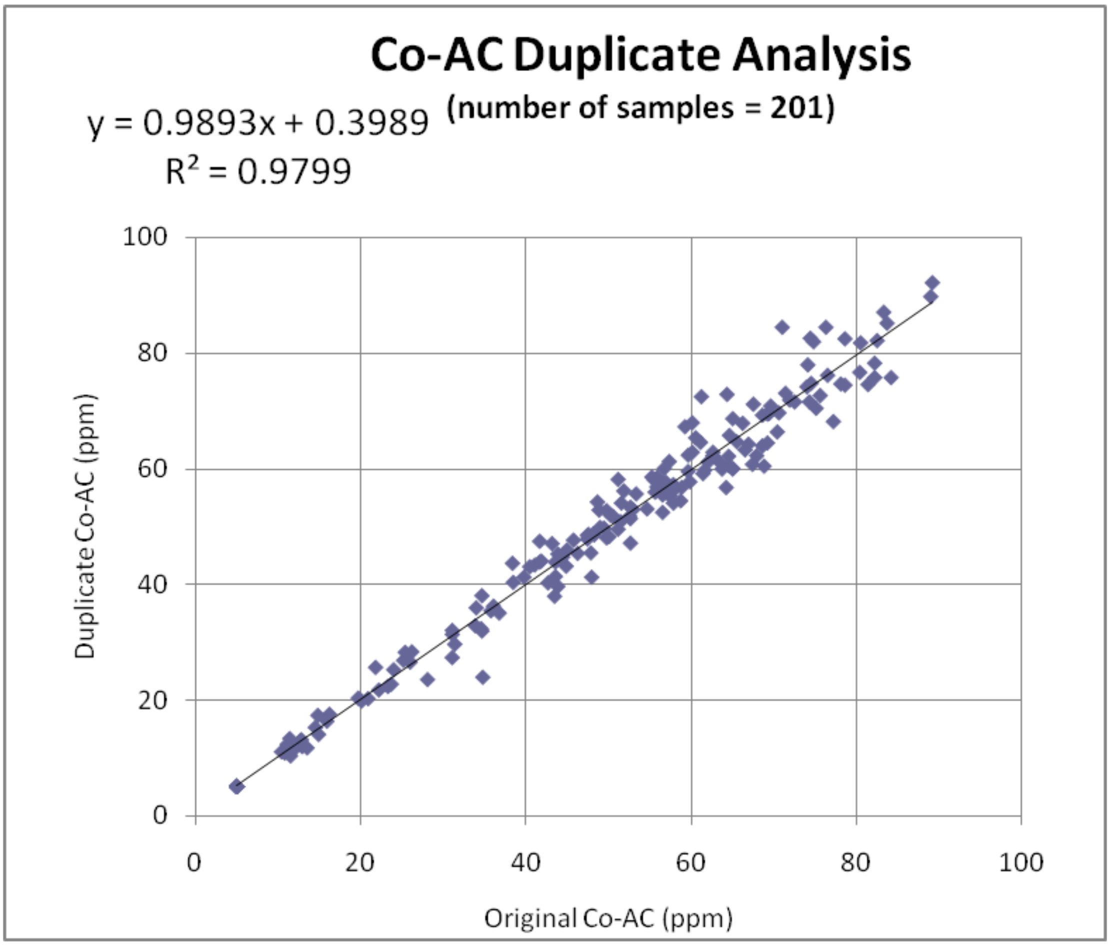

Figure 12-6: Co-AC Duplicate Analysis

12.5 Duplicate Coarse Reject Samples

Figure 12-7 and Figure 12-8 show the results of the coarse reject duplicates for Ni-AC and Co-AC. The coarse reject duplicate samples show a strong correlation to the original sample. In SRK's opinion, Ni-AC and Co-AC grades in coarse rejects exhibit an acceptable level of repeatability.

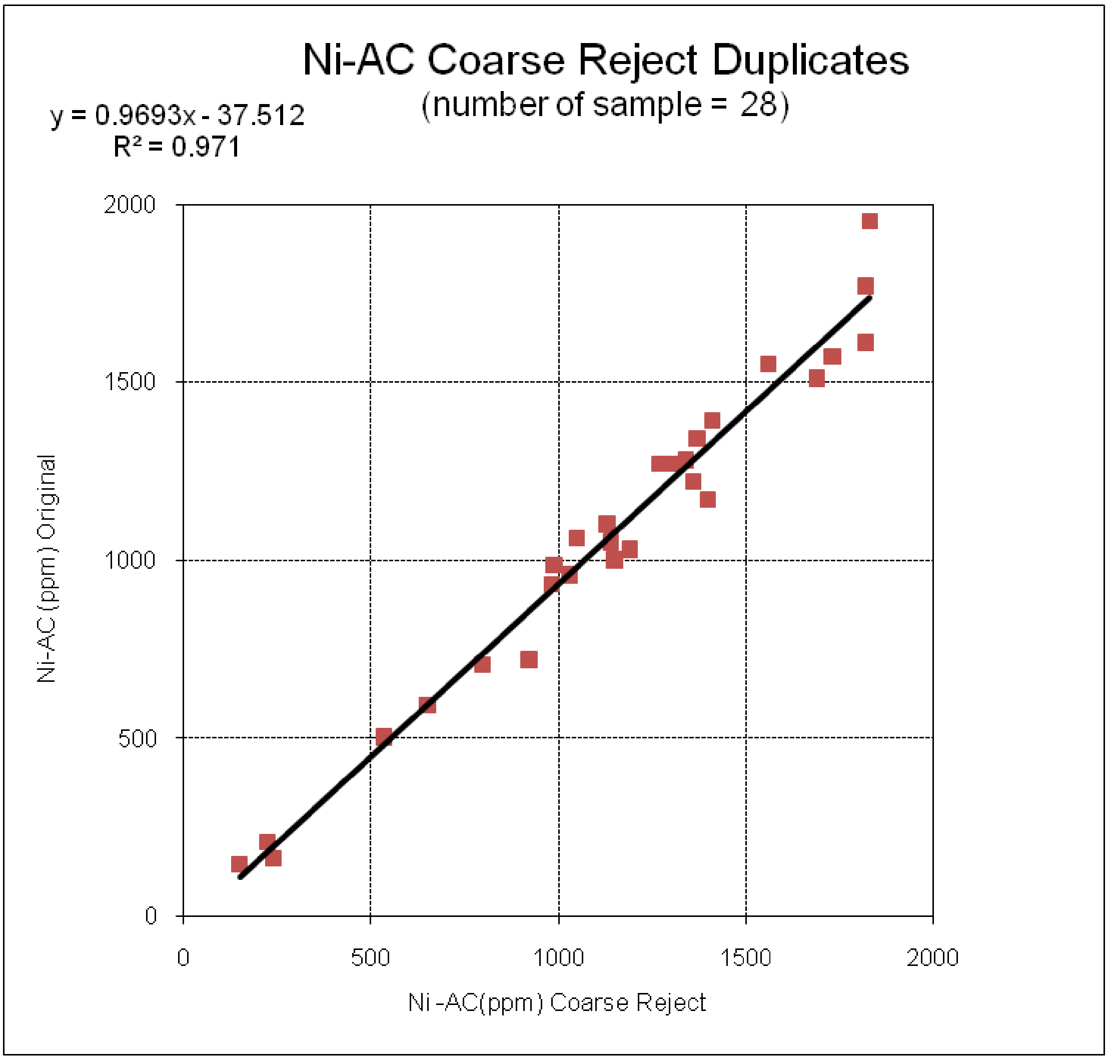

Figure 12-7: Ni-AC Coarse Reject Duplicate Analysis

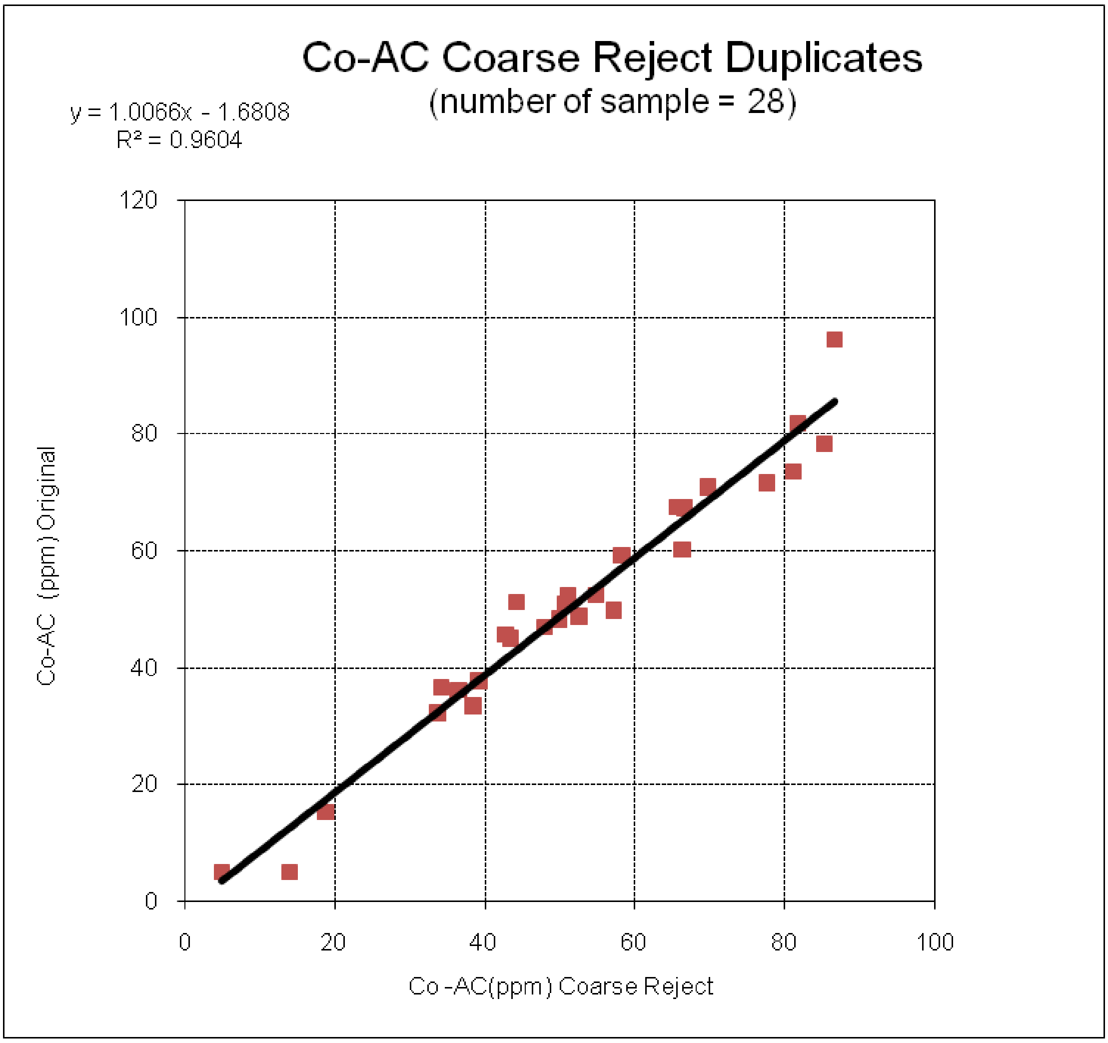

Figure 12-8: Co-AC Coarse Reject Duplicate Analysis

12.6 Interlaboratory Check Assays

Figure 12-9 shows the results of the control analysis for Ni-AC carried out at Acme, against the original Ni-AC analysis carried out at Labtium. The control assays display a strong correlation to the original assays, with similar mean grades of 1126 ppm Ni-AC and 1133 ppm Ni-AC for Acme and Labtium respectively. In SRK's opinion, the inter-laboratory check assays performed at Acme provide good support for Ni-AC assays carried out by Labtium.

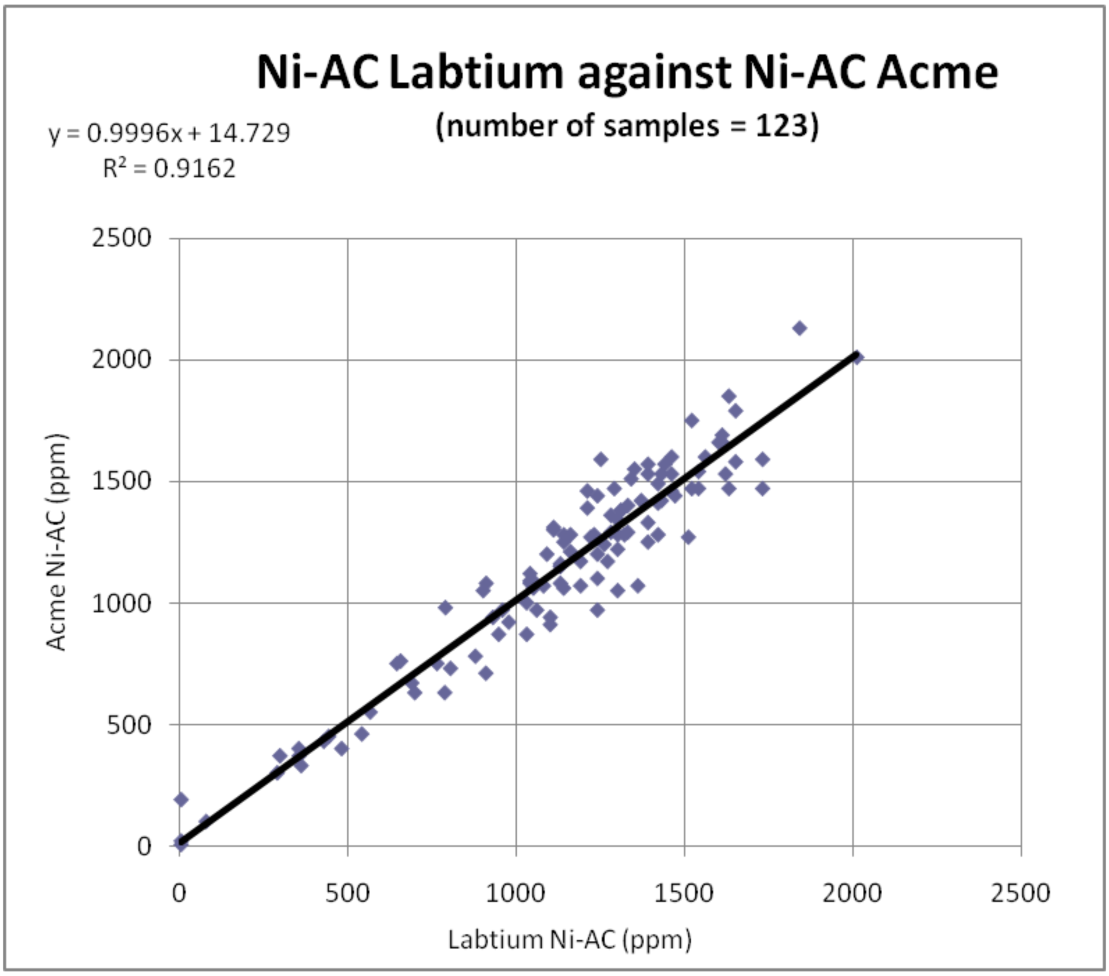

Figure 12-9: Control samples Ni-AC Labtium against Ni-AC Acme

12.7 SRK Consulting Duplicate Samples

Table 12-1 details assay results received by SRK for 16 pulp samples sent to Labtium for analysis by method code 240P. Corresponding original assay results are presented for comparison. Results below detection have been amended to half the detection limit for consistency with the Company's data.

Figure 12-10 and Figure 12-11 illustreate the results of SRK's duplicate sample analysis for Ni-AC and Co-AC carried out at Labtium by method 240P, against the Company's original Ni-AC and Co-AC analysis carried out at Labtium using the same method. SRK duplicates display a strong correlation to the original assays, with similar mean grades of 1116 ppm Ni-AC and 1064 ppm Ni-AC, and 42 ppm Co-AC and 43 ppm Co-AC for SRK duplicates and originals respectively.

The number of duplicate samples selected by SRK represents a very small proportion of the overall number of Ni-AC and Co-AC analysis carried out on the Project to date. Notwithstanding this, the results add further support to the repeatability of Labtium's 240P method for nickel and cobalt.

Table 12-1: Details of SRK Duplicate assays for Ni-AC and Co-AC with respect to original assay results

SRK Sample ID	Co-AC (SRK)	Ni-AC (SRK)	Original Analysis ID	Co-AC (Original)	Ni-AC (Original)
314201	68	1490	VIN1071116	70	1440
314202	44	1390	VIN291047	38	1270
314203	73	1650	VIN1071040	77	1630
314206	70	1440	VIN411012	79	1480
314207	5	133	RON2021091	5	137
314209	66	1430	VIN0392058	66	1360
314210	69	1370	VIN301086	72	1310
314211	10	182	RON0701035	5	164
314212	40	923	RON0881023	33	788
314213	45	1110	RON2041121	45	1110
314215	50	842	VIN281006	50	792
314216	23	1640	RON0981004	22	1490
314217	5	933	RON0921015	5	773
314218	49	999	VIN381029	51	949
314219	59	1240	VIN0392034	66	1310
314220	5	1090	RON0891016	5	1020

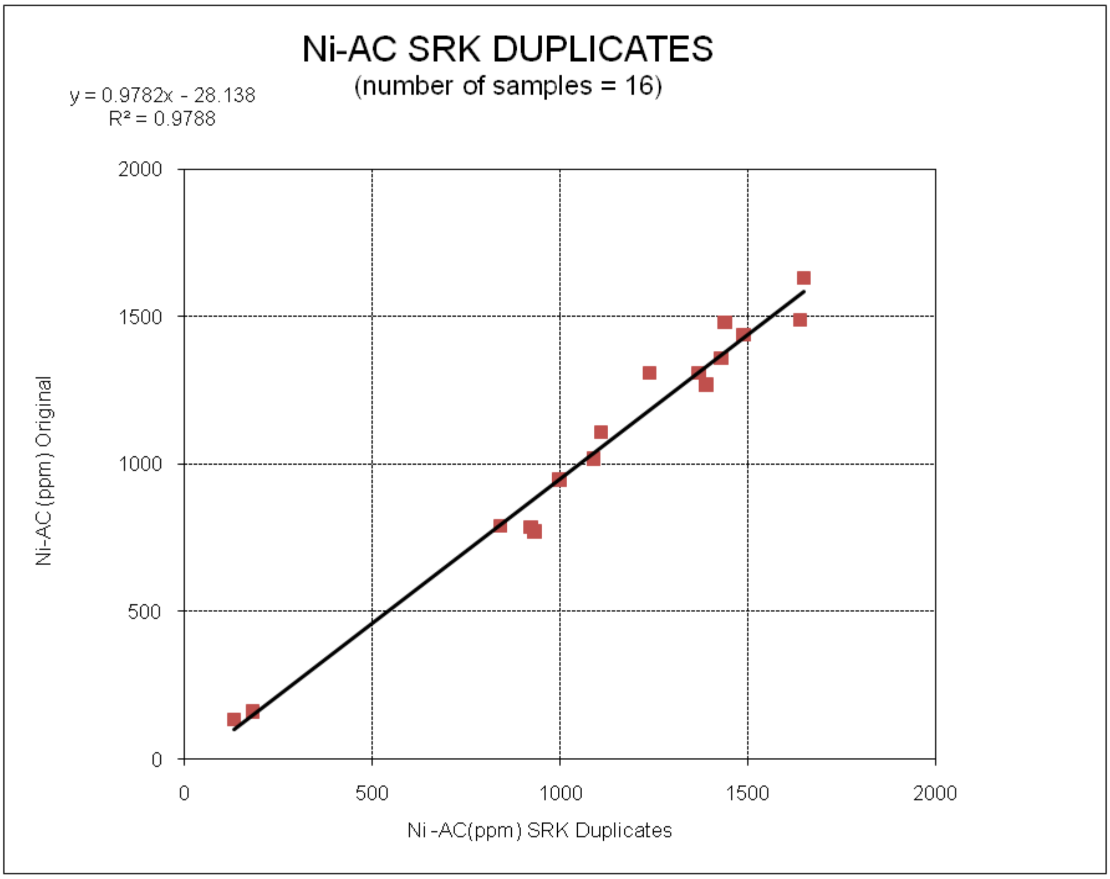

Figure 12-10: Ni-AC SRK Duplicate Samples against Originals

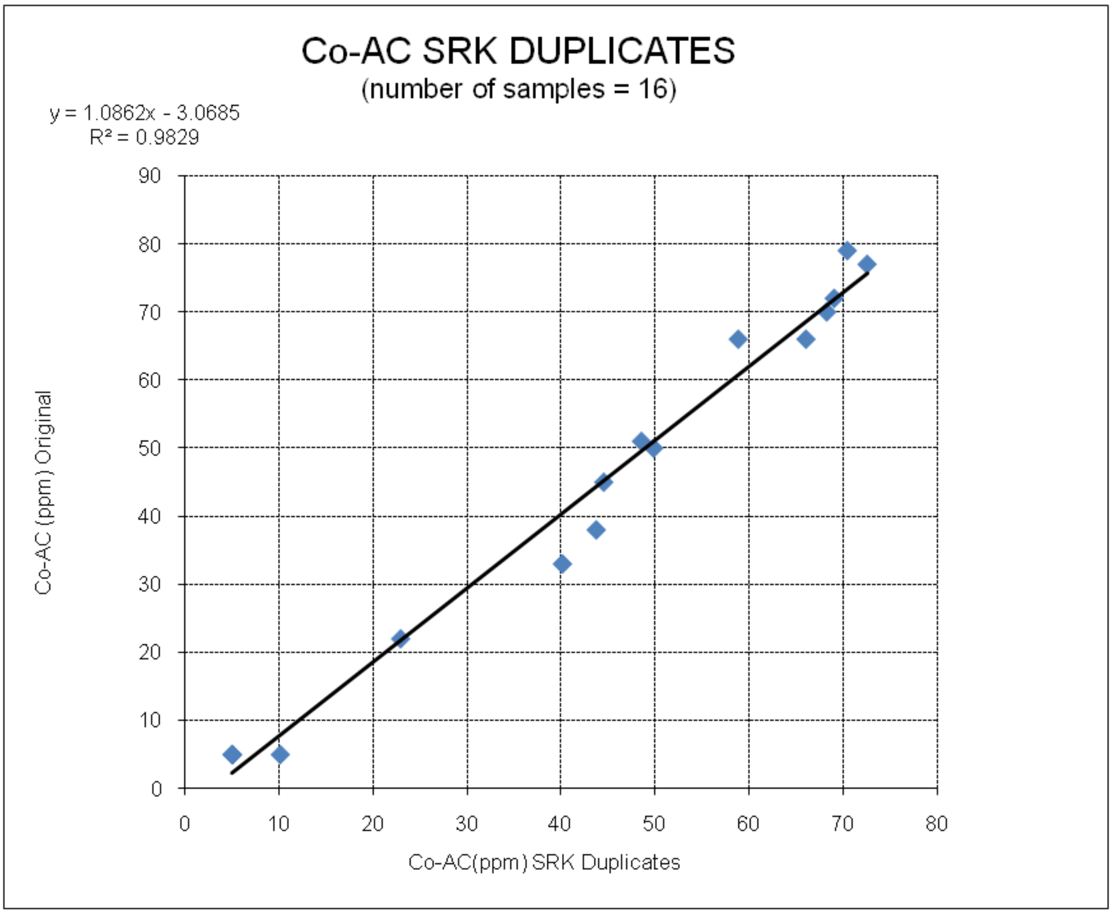

Figure 12-11: Co-AC SRK Duplicate Samples against Originals

12.8 Density Measurements

Figure 12-12 illustrates the specific gravity of 79 samples measured at ALS Chemex and using the water immersion method, as compared to the Company's density measurements using the same method. With the exception of a single outlier, the results from ALS Chemex provide good support for density measurements taken by the Company.

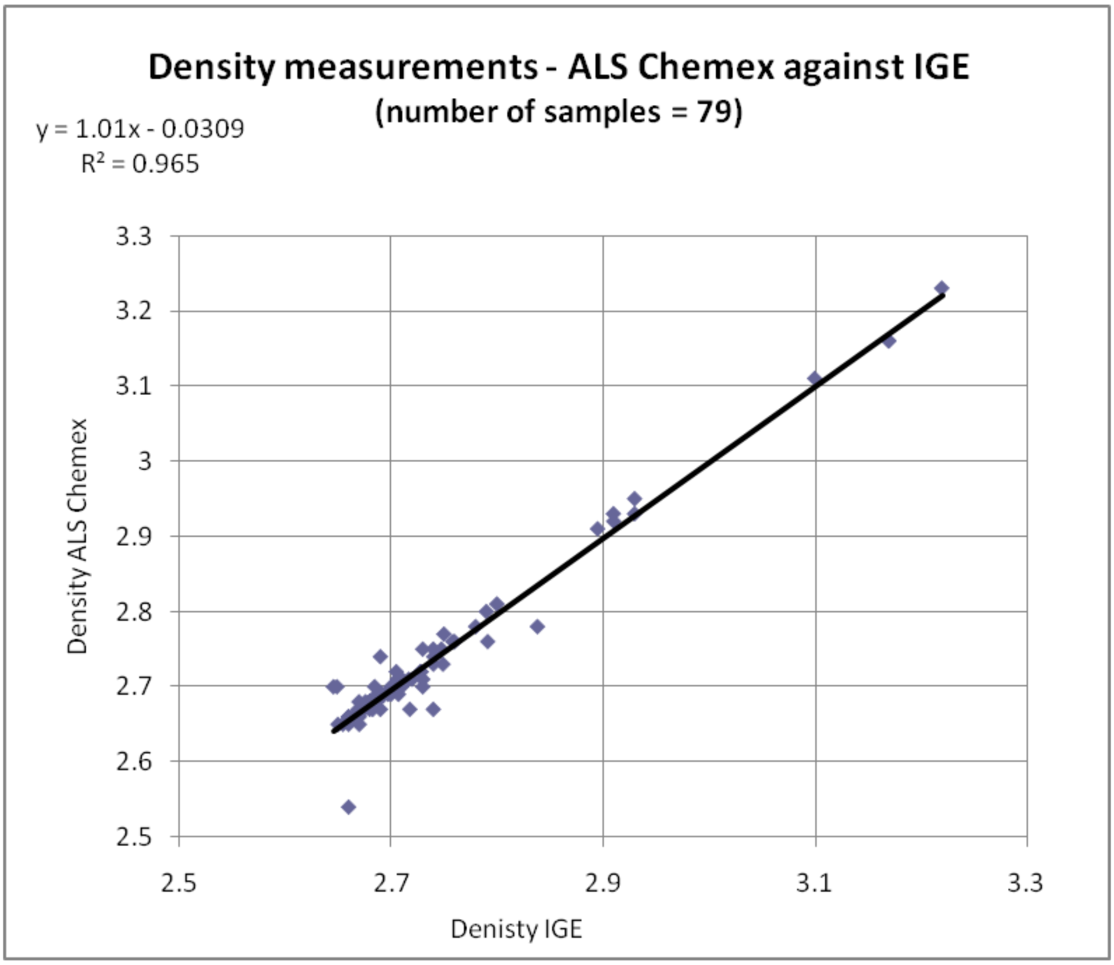

Figure 12-12: Density measurement comparison

12.9 MRG QAQC Analysis – Sundsberget

For the Sundsberget deposit, MRG undertook an analysis of the QAQC data provided by the Company. This includes blanks, reference material and duplicates as described above.

For a detailed description of the results obtained the reader is directed to the MRE report generated by MRG. MRG concluded that the results of the Sundsberget QA/QC were acceptable and within the accepted accuracy and precision limits and that the database supplied by the Company was suitable for use in the Mineral Resource estimate for Sundsberget.

12.10 Summary Comments

SRK is of the opinion that the assay and density information available of sufficient quality to support the estimates of mineral resources presented later in this report.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

A significant amount of metallurgical testwork has been undertaken and is commented upon in Section-17 “Recovery Methods”.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

This section of the report describes the work undertaken by SRK in developing Mineral Resource Statements for the Rönnbäcksnäset, Vinberget and Sundsberget deposits. SRK's initial Mineral Resource estimates for Vinberget and Rönnbäcksnäset were produced in April 2010, at which time this was restricted to mineralisation which SRK considered had reasonable prospects for eventual economic extraction by only reporting that material which fell within an optimised pit outline generated using parameters appropriate at that time. The most up to date mineral resource estimate available at that time for the Sundsberget deposit was the MRG's estimate produced in December 2010.

The pit optimisation parameters used by SRK in April 2010 have been updated for use in this current study and while no additional assay data has been supplied to SRK by the Company since the generation of the April 2010 models, the Mineral Resource estimates presented here for Vinberget and Rönnbäcksnäset have been re-reported using an updated pit outline produced using these. In addition, given the now identified potential for recovery of a magnetite product, Fe has for the first time also been interpolated into the models. In addition, SRK has also produced an updated Mineral Resource estimate for the Sundsberget deposit using the same methodology as used at Vinberget and Rönnbäcksnäset in order to ensure consistency between the estimates.

14.2 Statistical Analysis and Geological Domaining

14.2.1 Introduction

A statistical study of the data made available for the Rönnbäcksnäset, Vinberget and Sundsberget deposits was undertaken to determine suitable geological domains to be used. It was clear that the dominant Ni mineralisation is limited to the serpentinite bodies in each case and that there is a hard contact between these and the host rock metasediments. Internal mafic units also contain low levels of Ni mineralisation that are present at Rönnbäcksnäset and Sundsberget in addition to internal zones of non mineralised serpentinite that are present at Rönnbäcksnäset and Sundsberget. The Vinberget deposit is a single body of serpentinite that does not contain any inclusions of mafics or internal non mineralised zones within the serpentinite.

14.2.2 Rönnbäcksnäset

The Rönnbäcksnäset deposit consists of a single serpentinite body that strikes in an east-west orientation in the south-western portion of the deposit and a north-south orientation in the north-eastern portion of the deposit. Figure 14-1 shows the drillhole distribution and solid wireframe created for the serpentinite body and Figure 14-2 shows the histogram of Ni-AC assays associated with the mineralised serpentinite body. As shown in Figure 14-2, two clear populations of data exist in the Rönnbäcksnäset deposit. Figure 14-3 shows the probability plot for Ni-AC for the same data with grade breaks being evident at 0.04% Ni-AC, 0.08% Ni-AC and 0.15% Ni-AC. When applying the identified grade breaks to the drillhole file, it is clear that a low grade domain exists in the north-eastern portion of the Rönnbäcksnäset deposit on the hanging wall side of the serpentinite body. Figure 14-4 shows a cross section containing drillholes RON54, RON53 and RON64 with the low grade Ni-AC domain being highlighted.

The section shown also highlights the mafic units within the serpentinite.

The Ni-AC distribution of the identified grade domains for the Rönnbäcksnäset deposit are shown in Figure 14-5 to Figure 14-8. The histograms show a near normal distribution within the mineralised serpentinite with the exception of a small low grade tail within the high grade population. This is related to individual low grade samples that cannot be domained out. Conversely, the mafic domain and internal waste domain show a high grade tail where individual samples occur within the larger modelled domain.

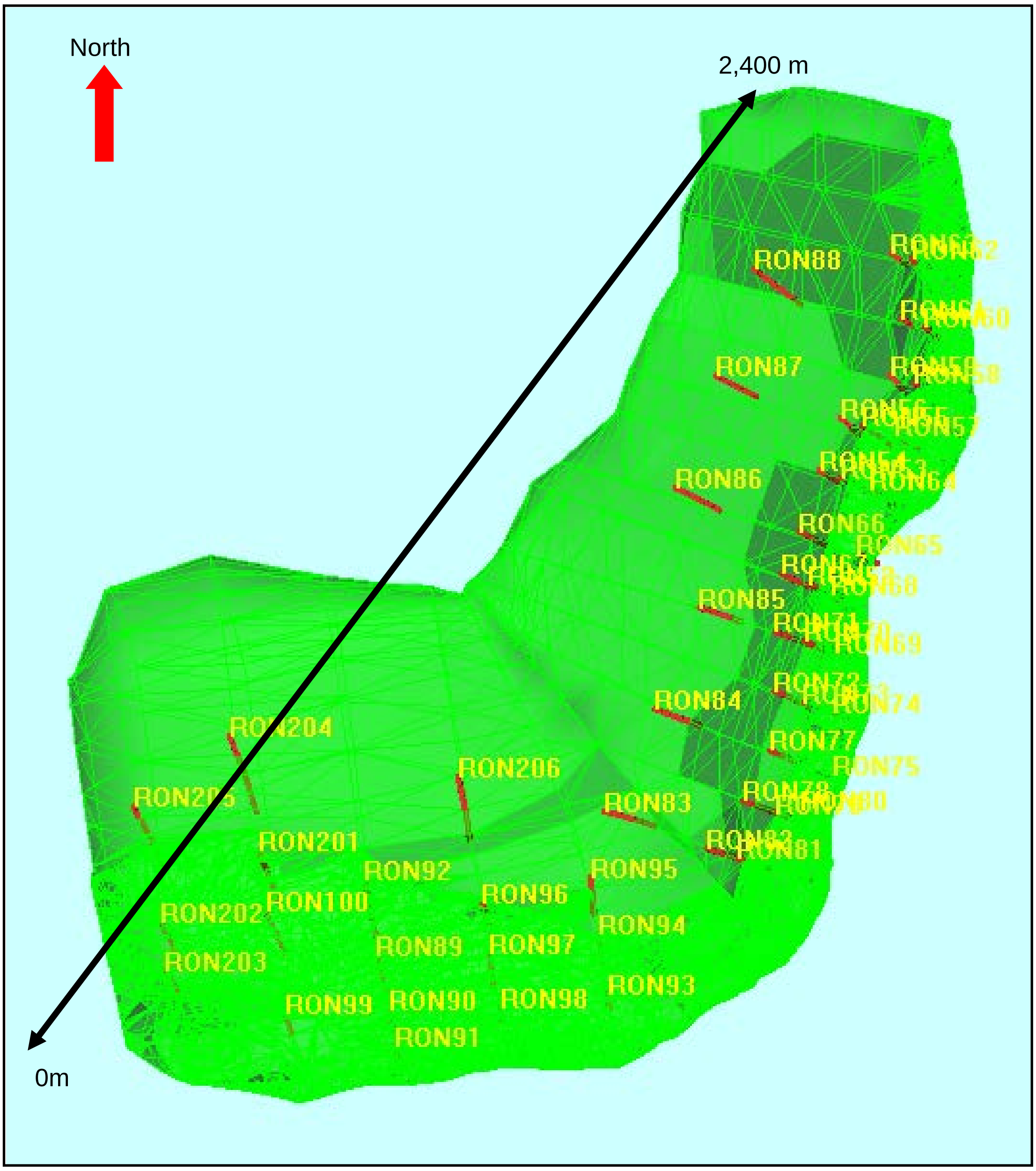

Figure 14-1: Rönnbäcksnäset serpentinite body and drillhole locations

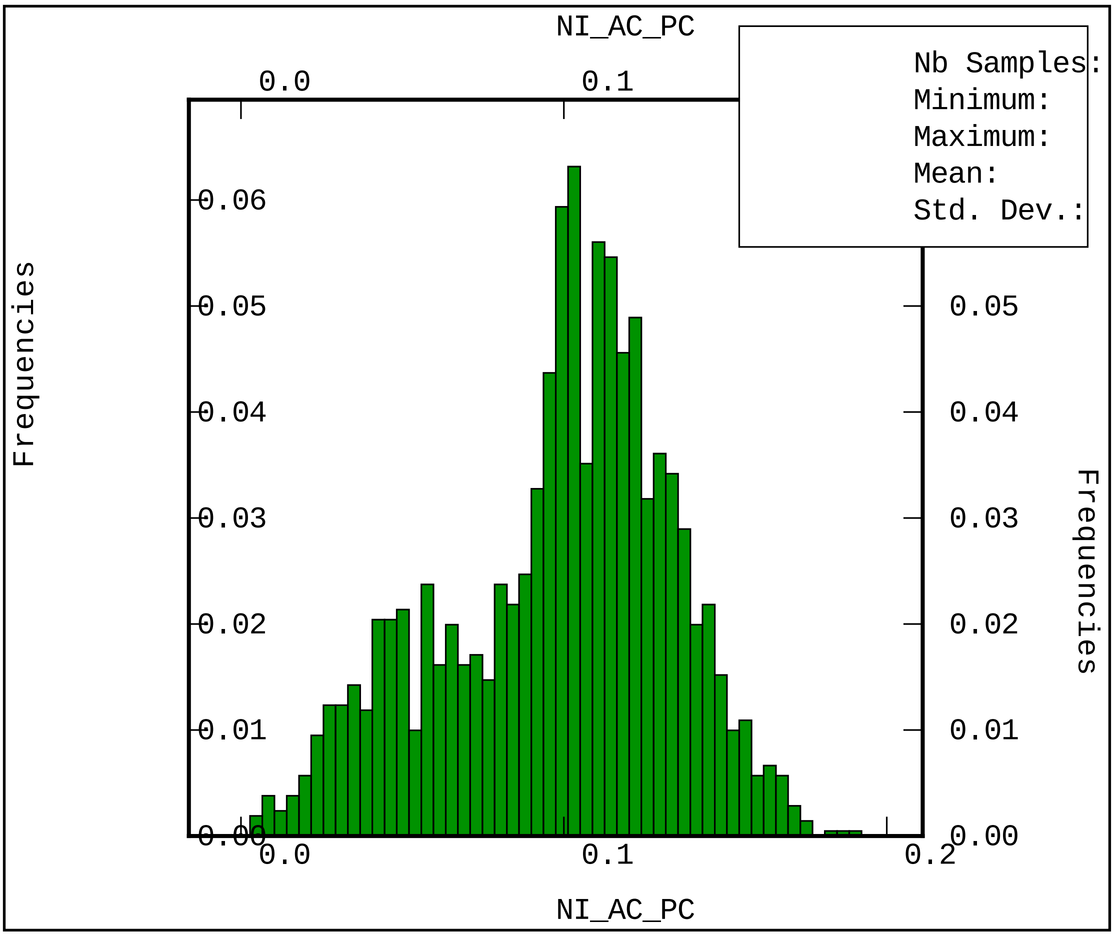

Figure 14-2: Ni-AC histogram for the Rönnbäcksnäset serpentinite

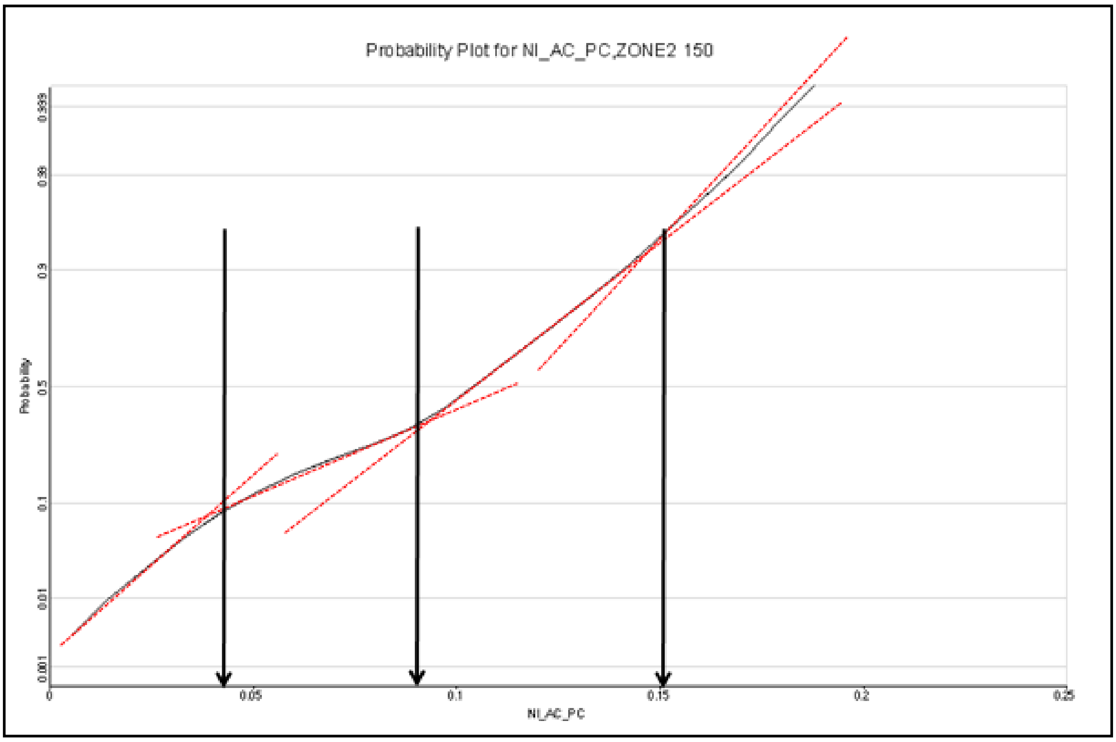

Figure 14-3: Probability plot for the Rönnbäcksnäset serpentinite

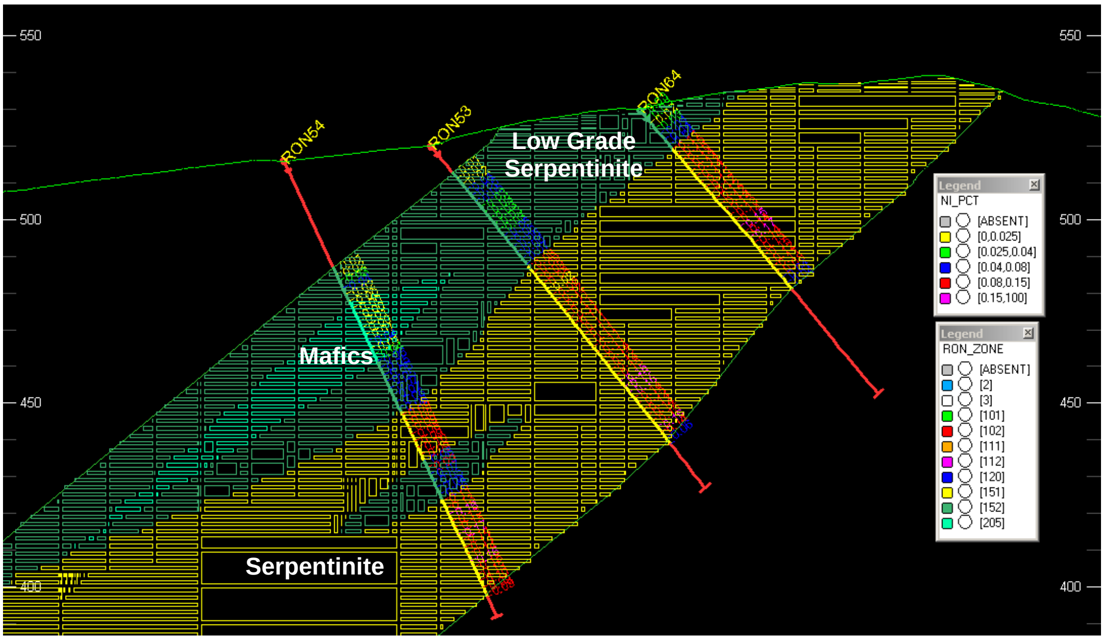

Figure 14-4: Cross section showing low grade Ni-AC domain on the hangingwall side of the serpentinite body (view looking north)

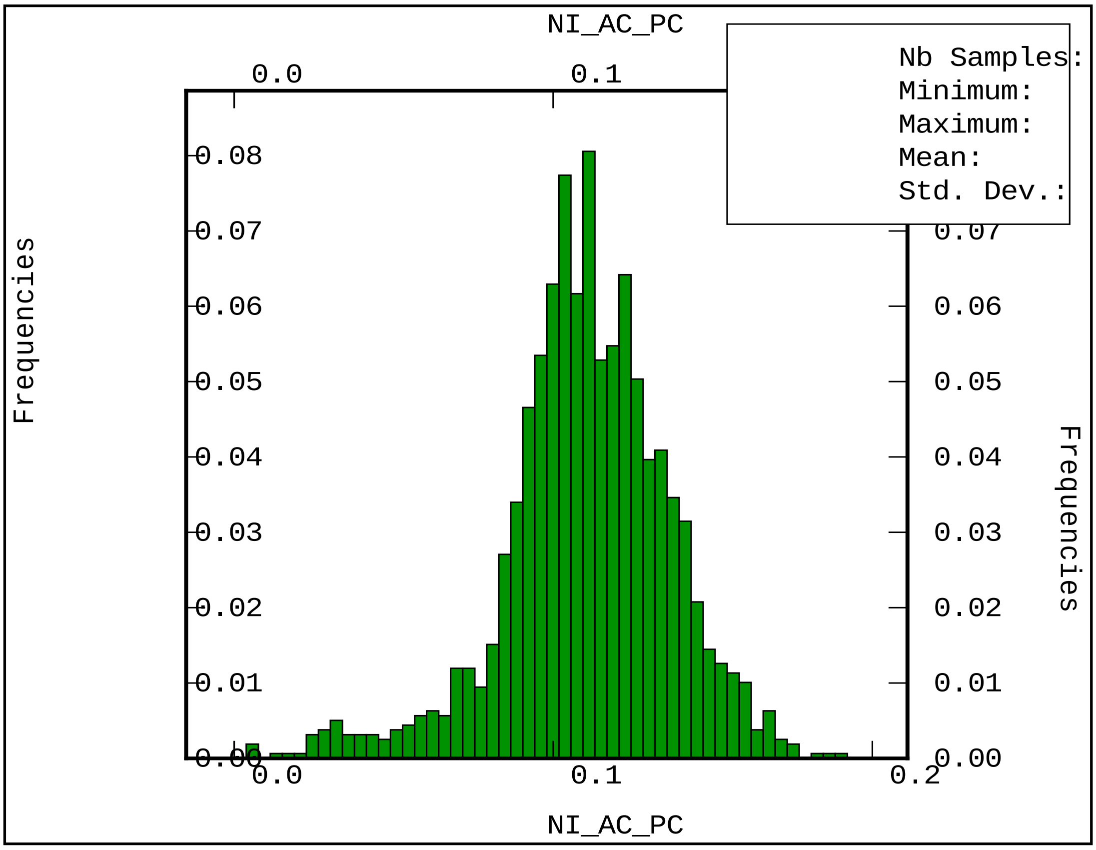

Figure 14-5: Ni-AC distribution of the high grade serpentinite

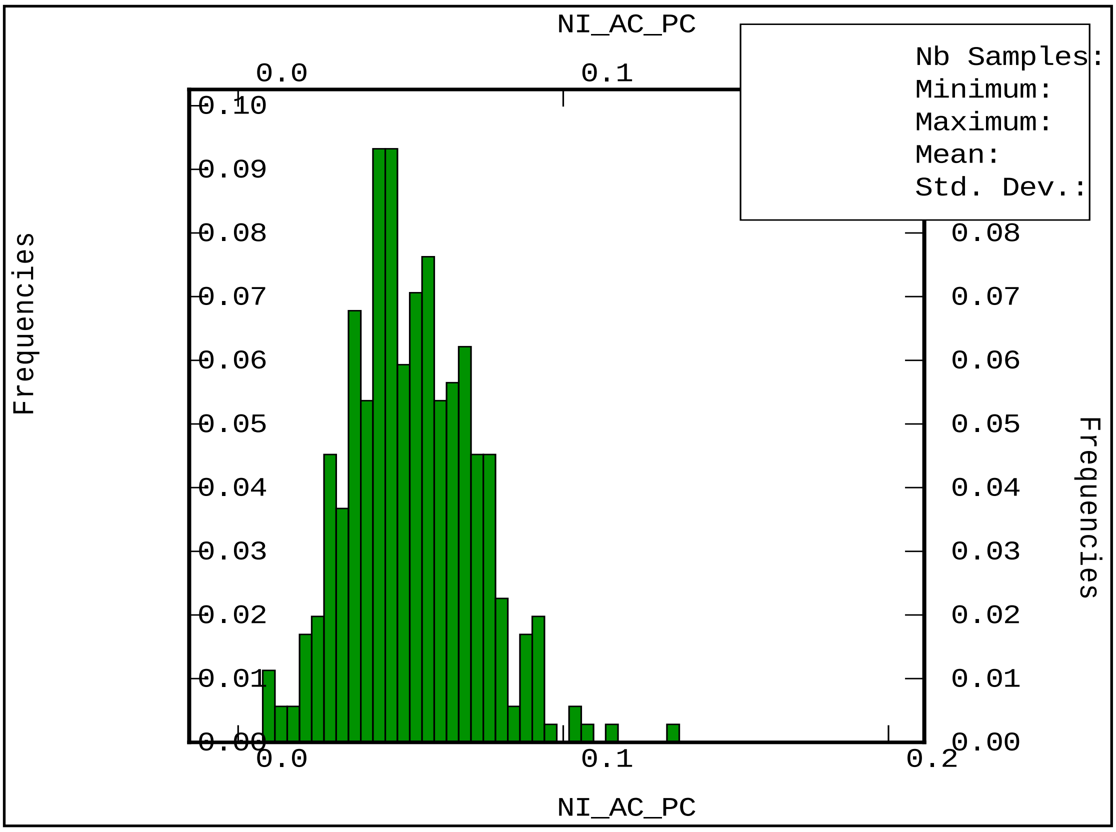

Figure 14-6: Ni-AC distribution of the low grade serpentinite

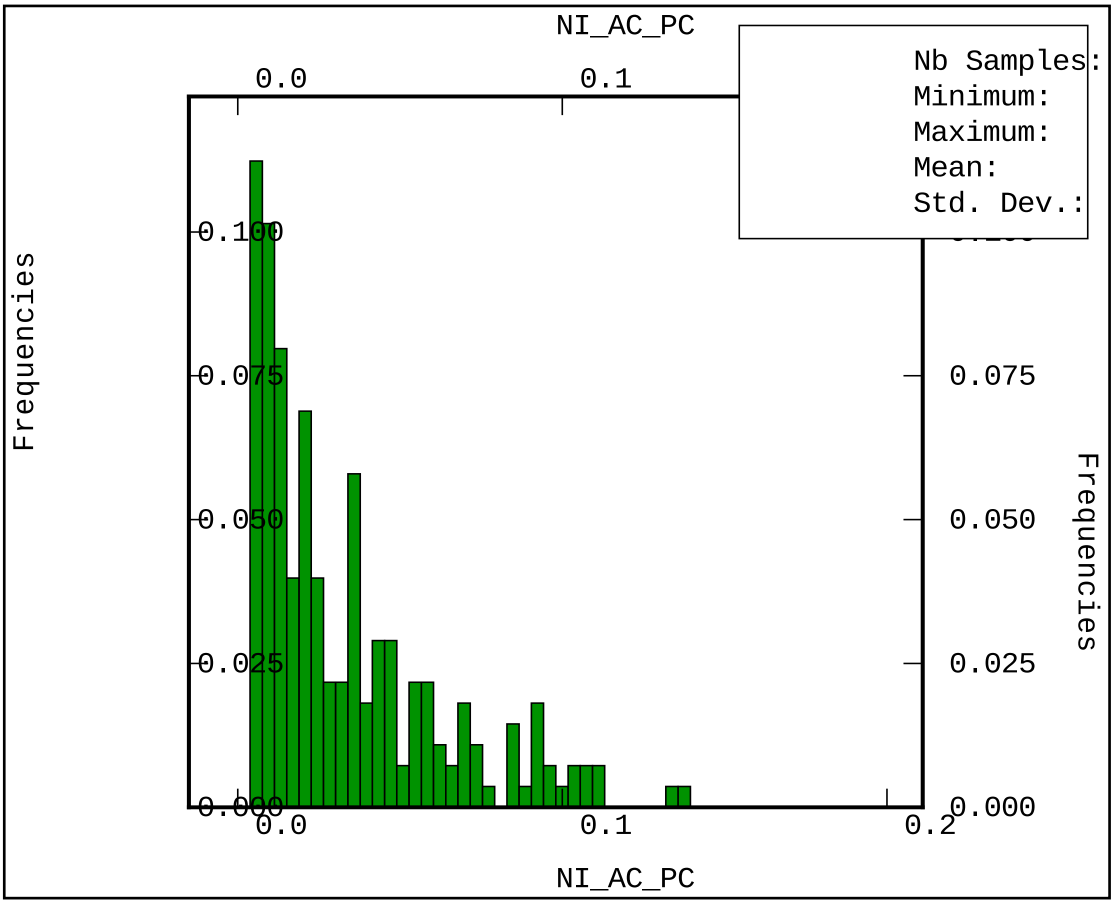

Figure 14-7: Ni-AC distribution of the mafic unit

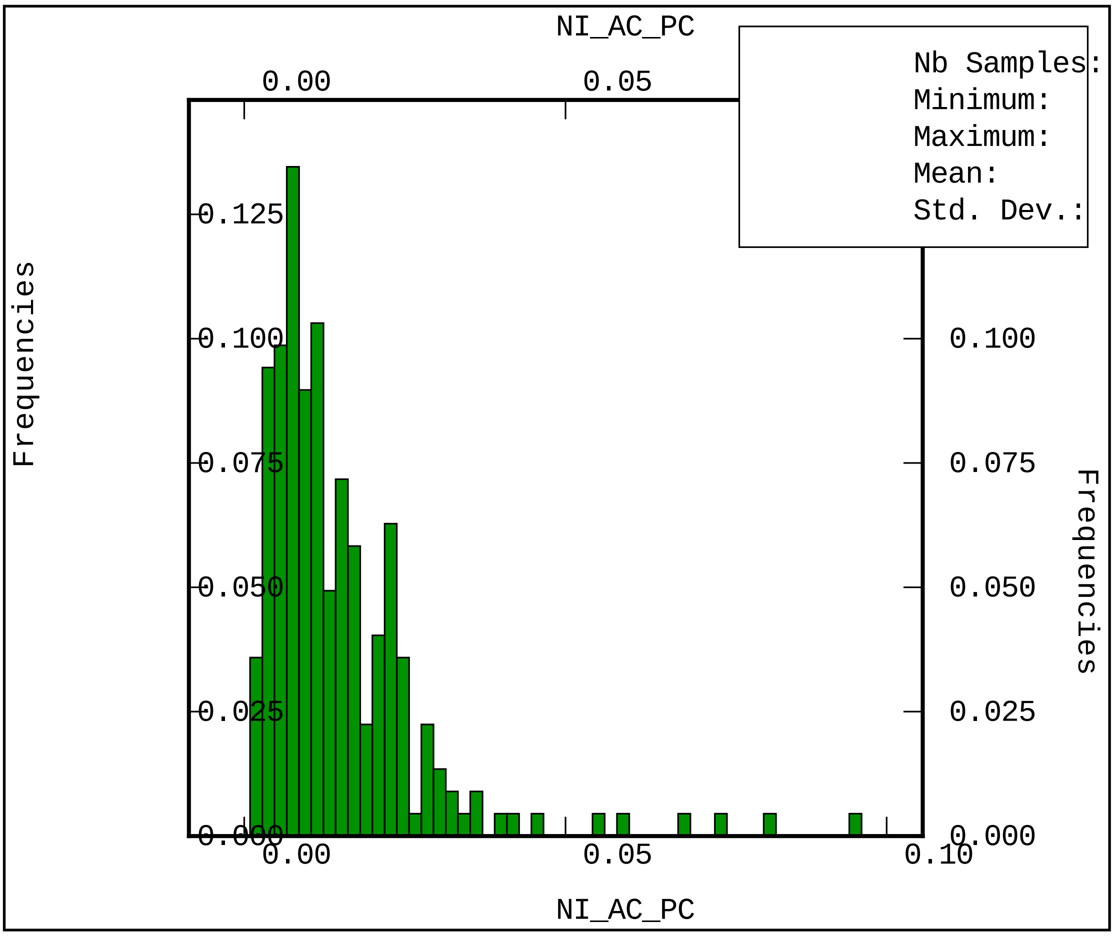

Figure 14-8: Ni-AC distribution of the internal waste domain

14.2.3 Vinberget

The Vinberget deposit consists of a single serpentinite body that strikes in a northwest-southeast orientation. Figure 14-9 shows the drillhole distribution and solid wireframe created for the serpentinite body and Figure 14-10 shows the histogram of Ni-AC assays associated with the mineralised serpentinite body. As shown in Figure 14-10, a near normal population of data exists in the Vinberget deposit. Figure 14-11 shows the probability plot for Ni-AC for the same data with subtle grade breaks being evident at 0.075% Ni-AC and 0.12% Ni-AC. When applying the identified grade breaks to the drillhole file, no clear trends in the mineralisation are observed. This is shown in Figure 14-12. The serpentinite body has therefore not been domained in any greater detail.

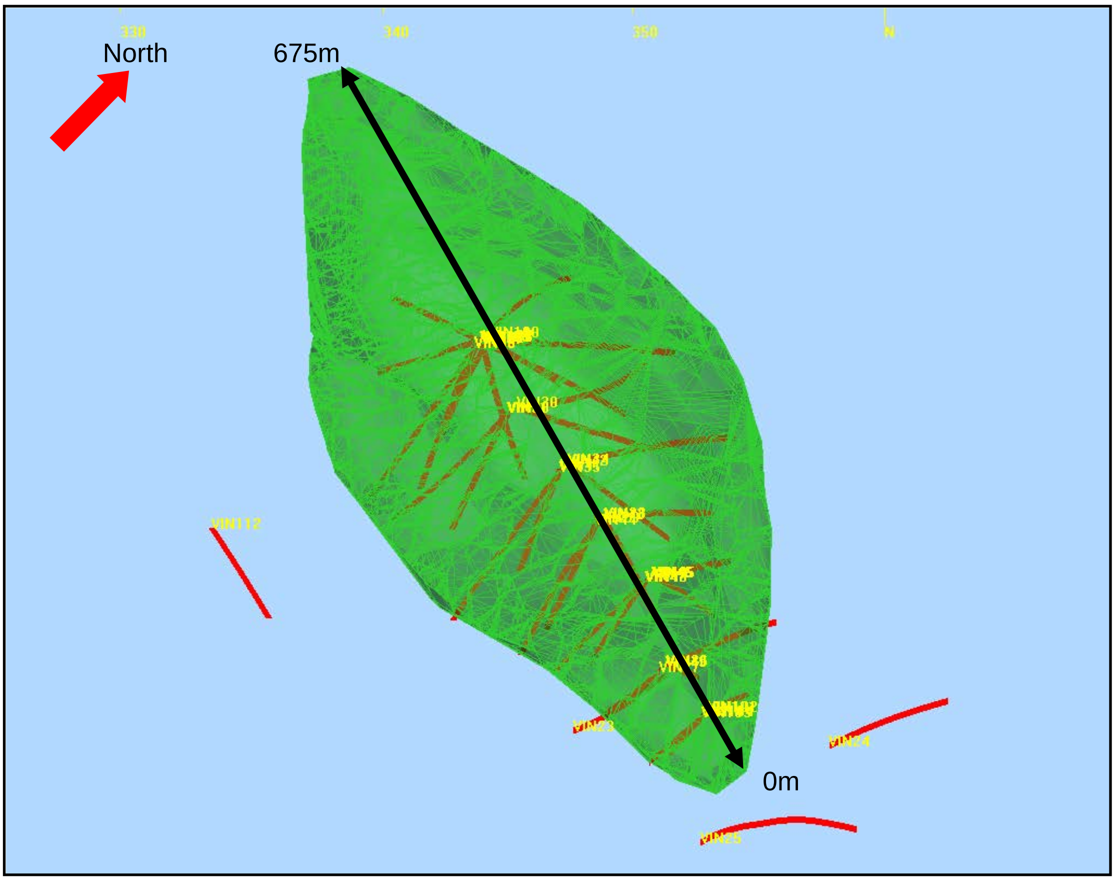

Figure 14-9: Vinberget serpentinite body and drillhole locations

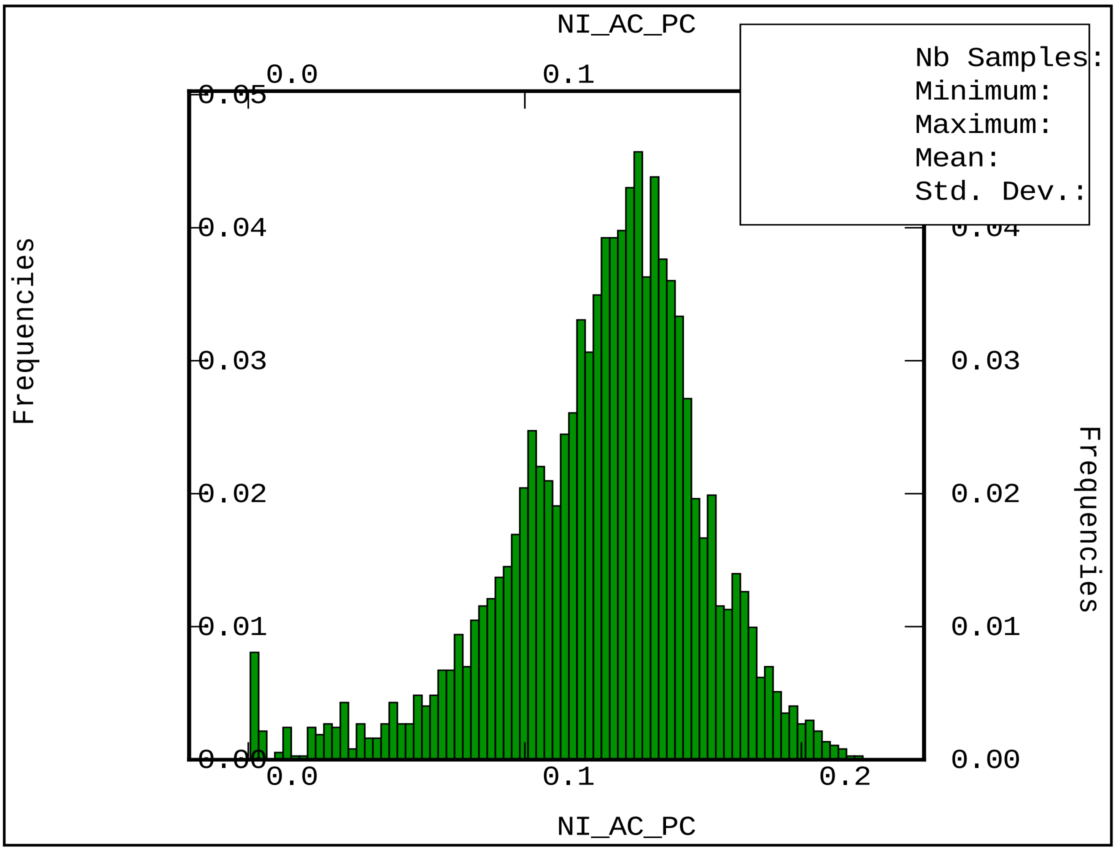

Figure 14-10: Ni-AC histogram for the Vinberget serpentinite

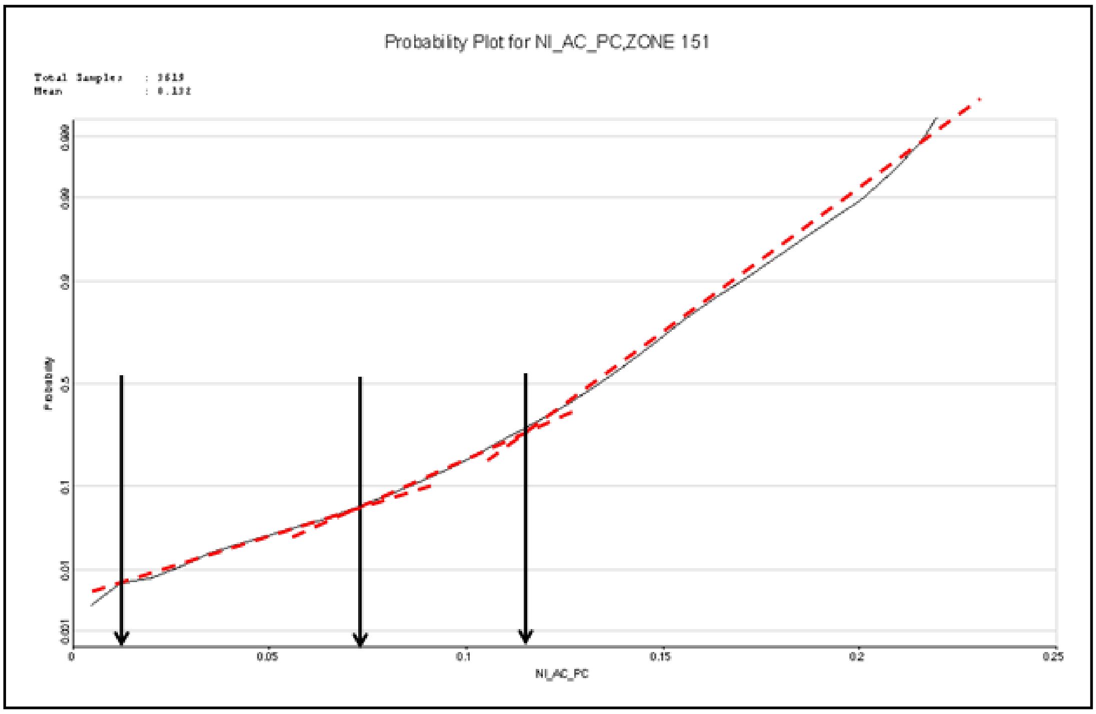

Figure 14-11: Probability plot for the Vinberget serpentinite

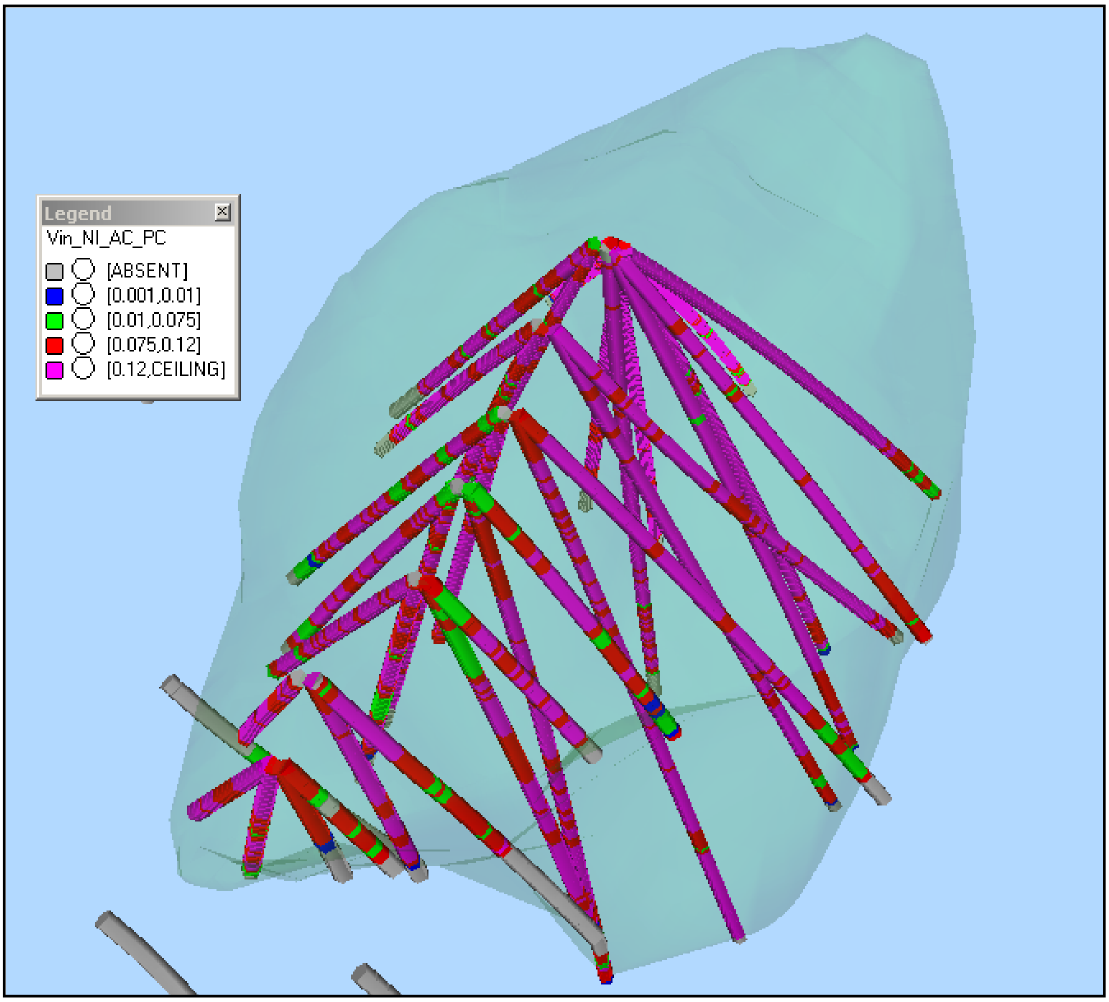

Figure 14-12: Vinberget drillhole file coloured by grade domains identified (looking west-northwest)

14.2.4 Sundsberget

The Sundsberget deposit consists of one main mineralised serpentinite body and a smaller unit towards the east. The main serpentenite unit contains a non-mineralised mafic unit as well as two lower grade areas. The main unit strikes in a north-northeast to south-southwest orientation and dips at roughly 30° to the west northwest. The deposit extends for roughly 1.2 km along strike and is between 500 m and 600 m in width.

Figure 14-13 shows the drillhole distribution along with the mineralised serpentenite solid wireframes and mafic unit.

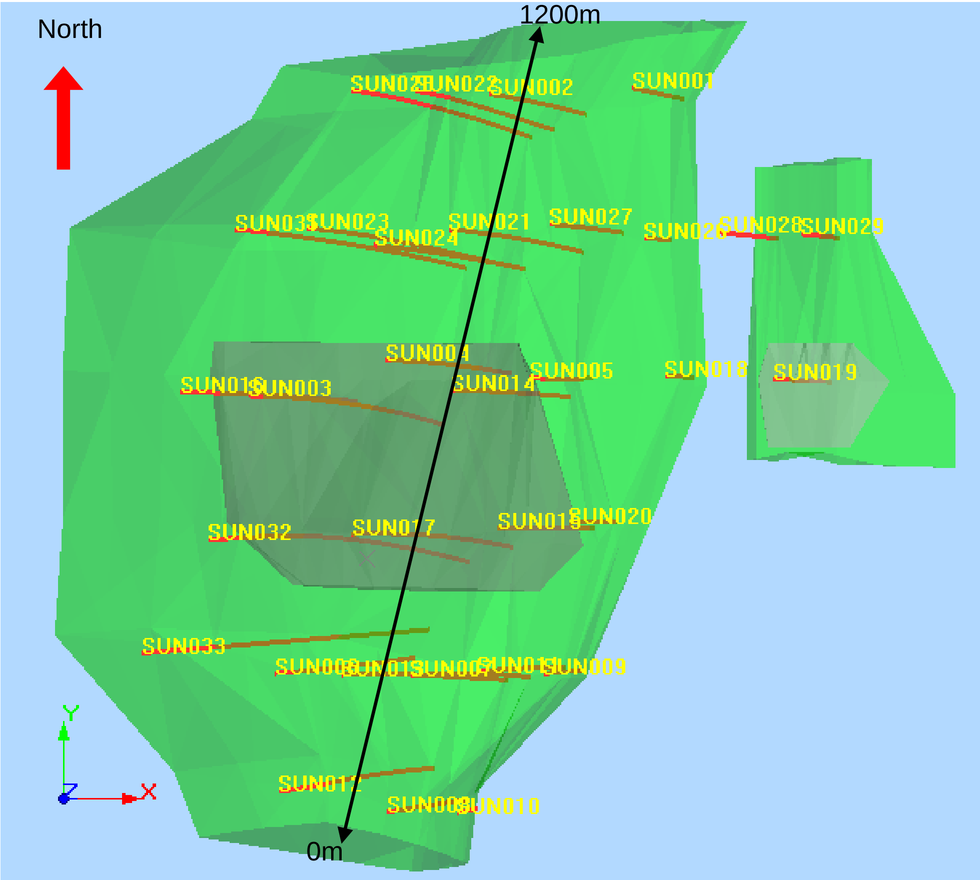

Figure 14-13: Sundsberget mineralised body and internal mafic unit with drillhole locations. Plan view.

Figure 14-14 shows the probability plot for Ni-AC at the Sundsberget deposit. It shows a smooth population, with very minor population breaks, which was difficult to use for domaining. Figure 14-15 shows the probability plot for Ni PCT (Ni tot) for the Sundsberget deposit, which defines clear population breaks at 0.07% Ni-PCT and 0.16% Ni-PCT. These separate populations can be identified in Figure 14-16 which shows a scatterplot of Ni-AC against Ni PCT (Ni total). These appear to be different Ni sulphide mineral species, and so Ni PCT was used to create unique domains to control the estimation.

When the domaining criteria was applied, the two populations identified above were captured (high grade Ni PCT Zone 151, low grade Ni PCT Zone 153), along with the low grade mafic unit within the high grade serpentenite zone (Zone 205). When the statistics were re-run using these domains, it was evident that a low-grade Ni AC zone was also present along the footwall of the deposit. This was domained separately using a 0.05% Ni AC cut-off grade (Zone 152).

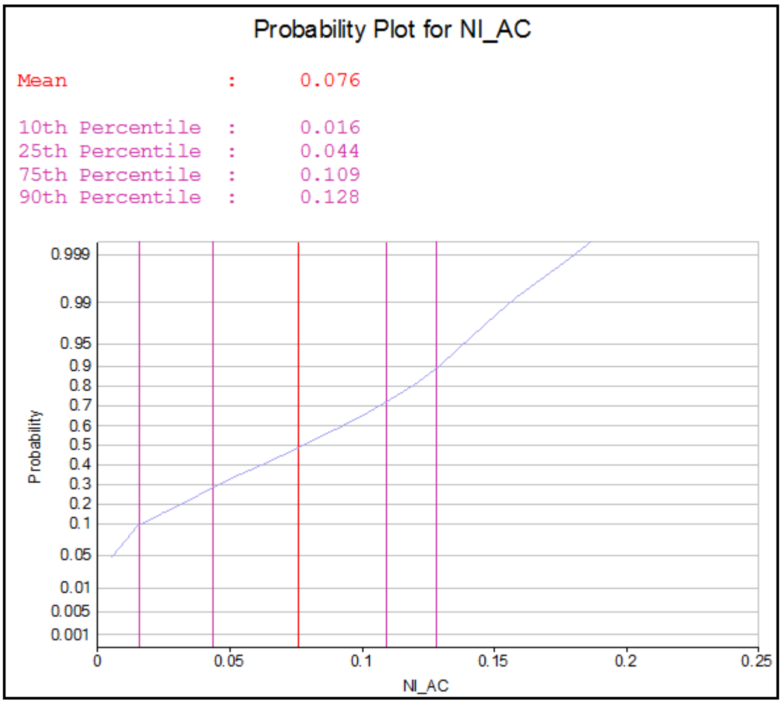

Figure 14-14: Probability plot for Ni-AC for the Sundsberget deposit.

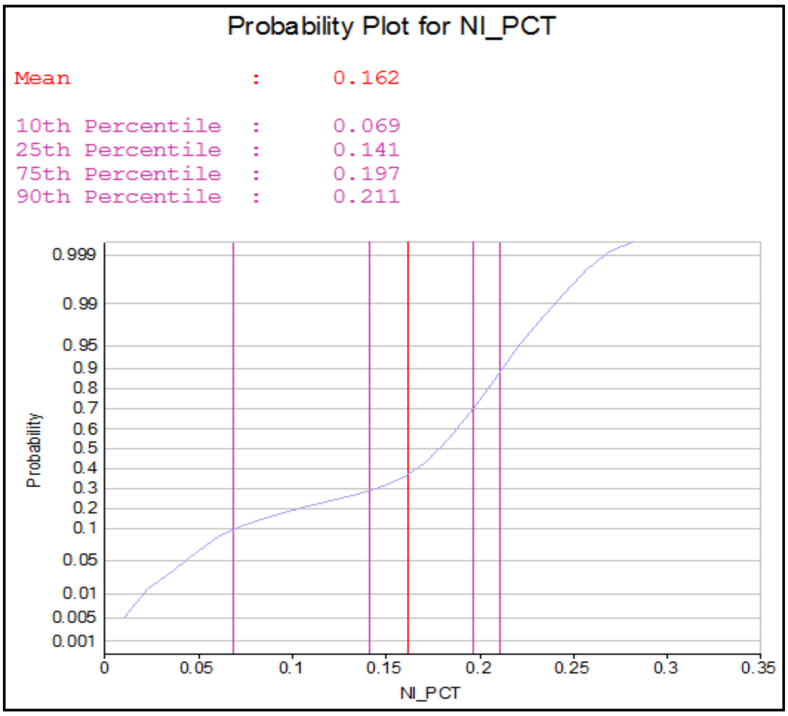

Figure 14-15: Probability plot for Ni% for the Sundsberget deposit.

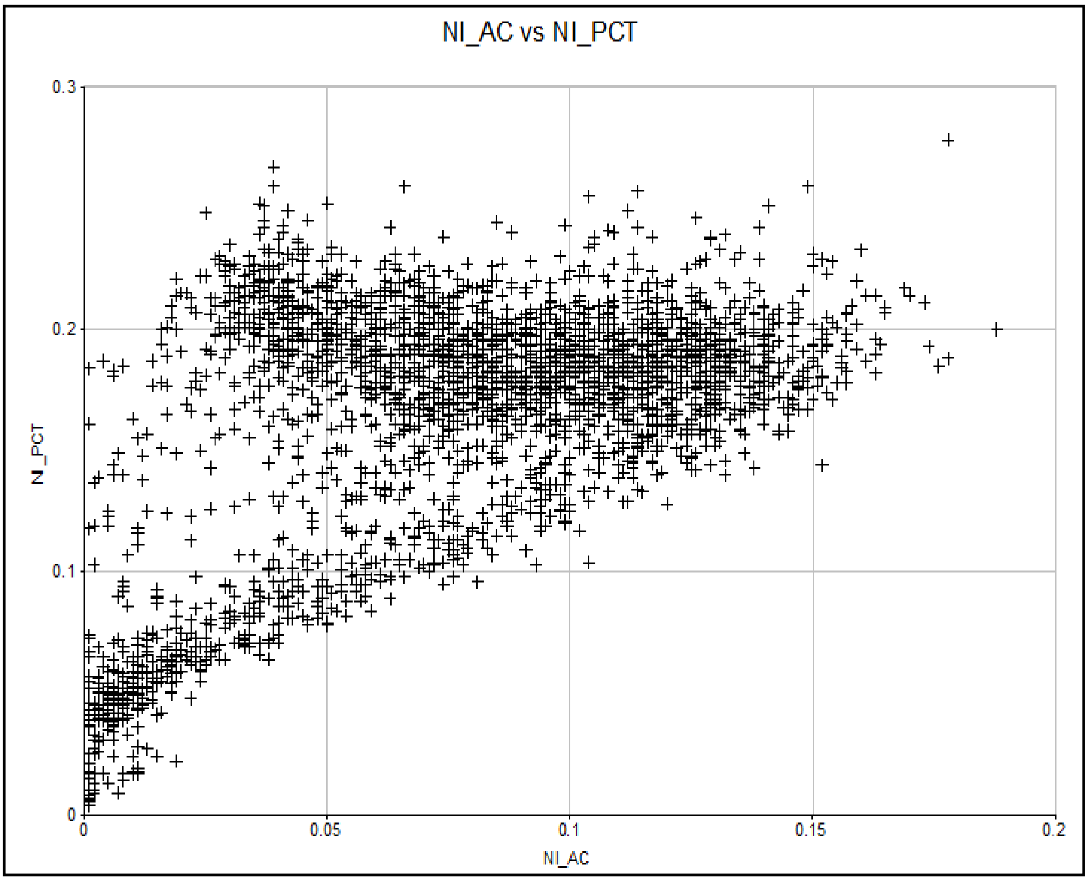

Figure 14-16: Scatterplot of Ni-AC vs Ni Total (Ni PCT)

The Ni-AC distribution of the identified grade domains for the Sundsberget deposit are shown in Figure 14-17 to Figure 14-20. The histograms show near normal distributions of Ni AC within the domains. The scatterplots of Ni AC against Ni PCT shown in Figure 14-21 and Figure 14-22, confirms the domaining strategy adopted.

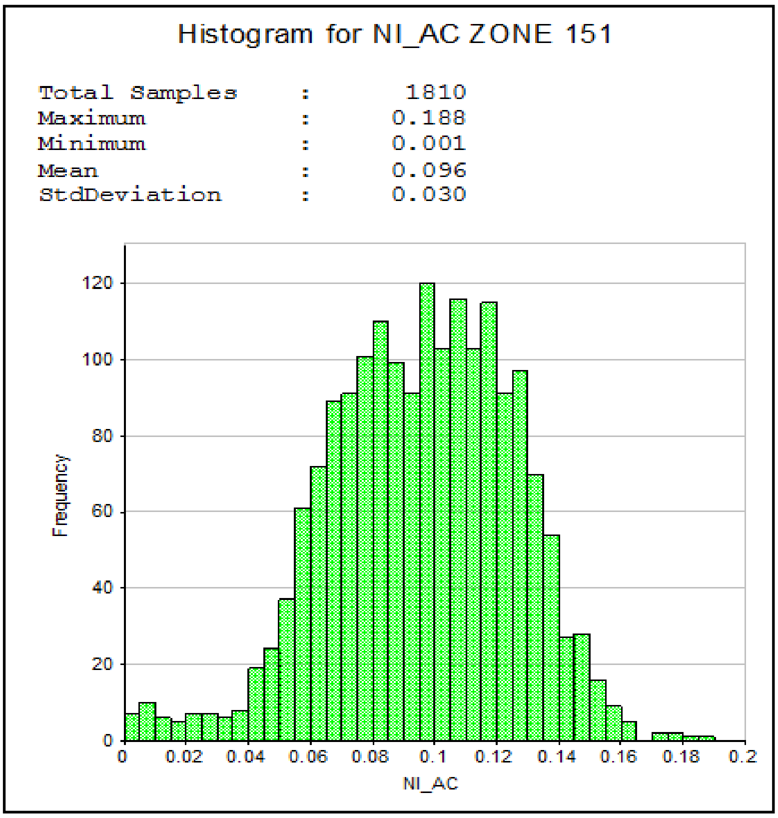

Figure 14-17: Ni-AC histogram for zone 151, high Ni-AC grade.

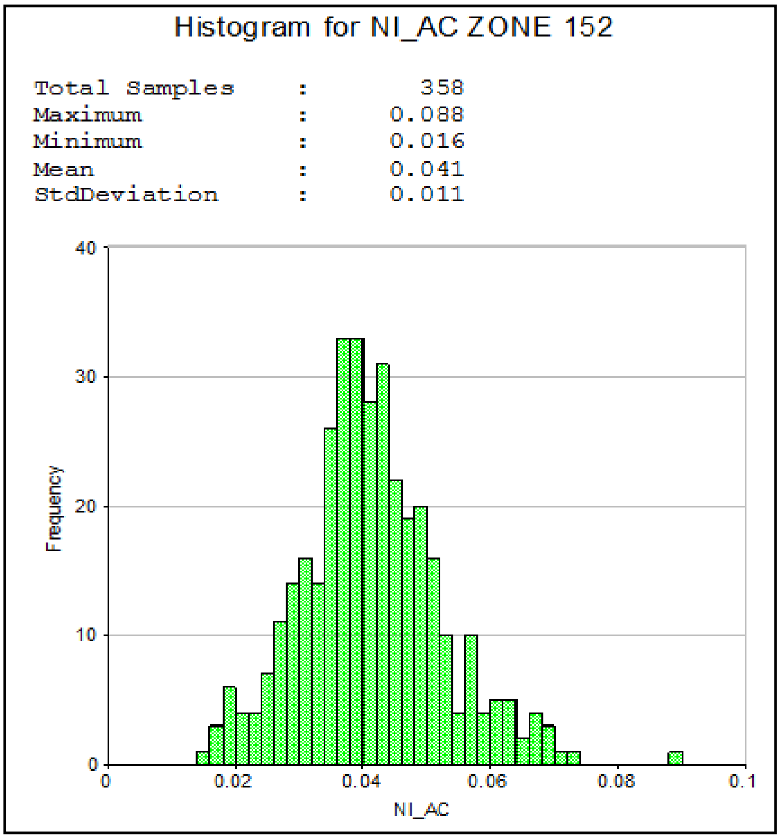

Figure 14-18: Ni-AC histogram for zone 152, low Ni-AC grade.

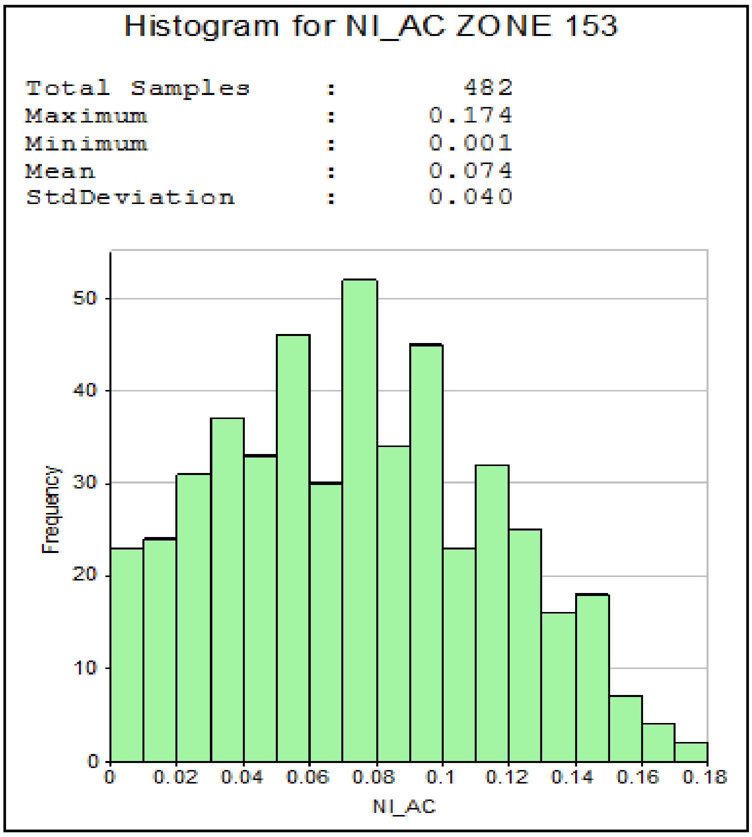

Figure 14-19: Ni-AC histogram for zone 153, high total Ni, low Ni-AC.

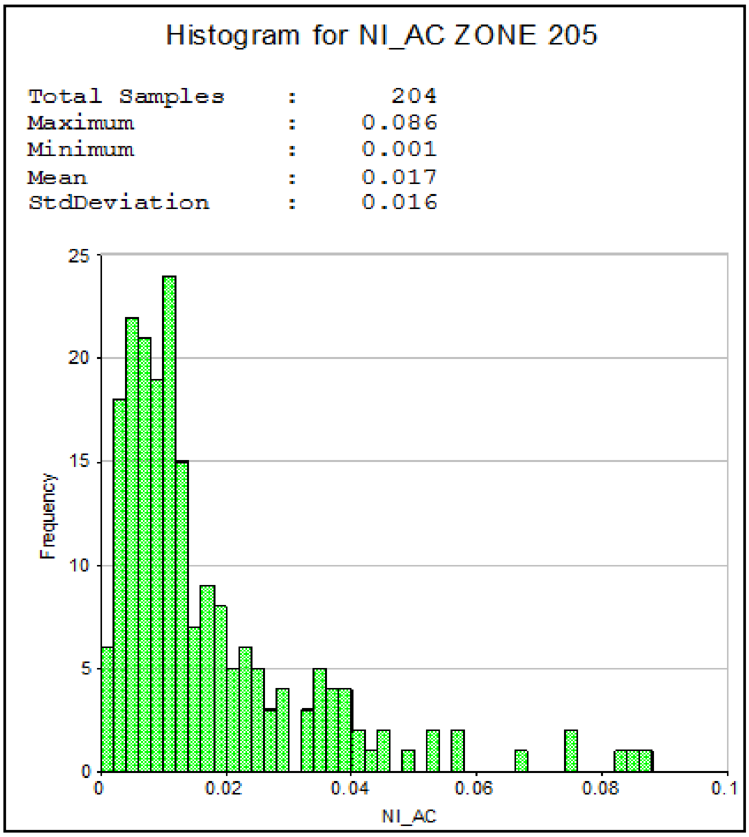

Figure 14-20: Ni-AC histogram for zone 205; internal waste.

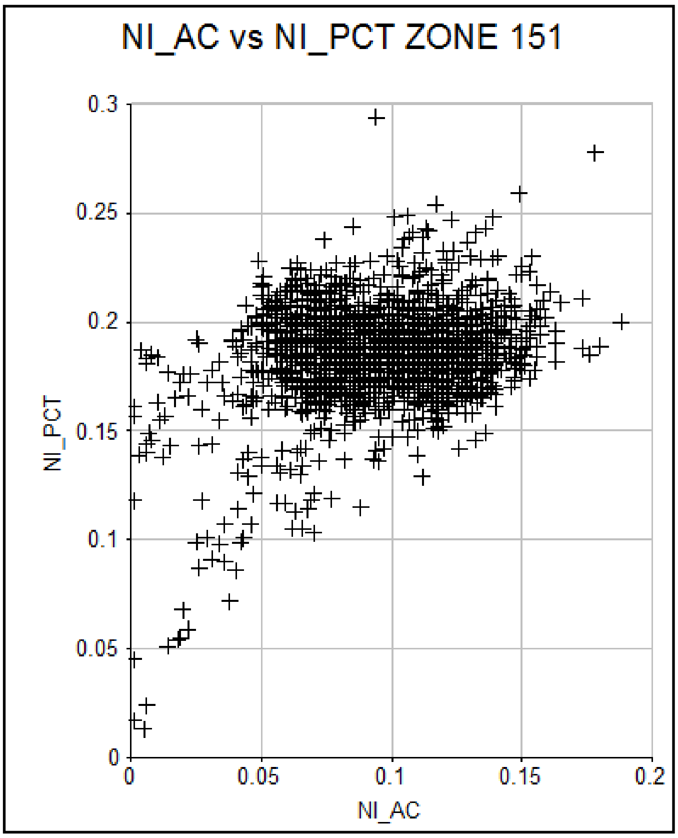

Figure 14-21: Scatterplot of Ni AC vs Ni PCT for Zone 151

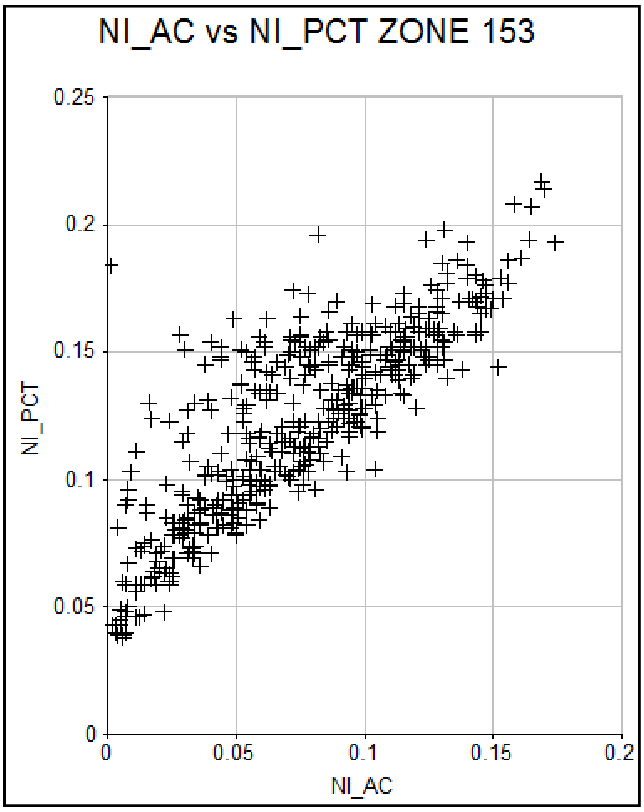

Figure 14-22: Scatterplot of Ni AC vs Ni PCT for Zone 152

Figure 14-23 shows a cross section with high and low grade Ni-AC domains, internal waste as well as a domain with high total Ni, but low grade Ni-AC.

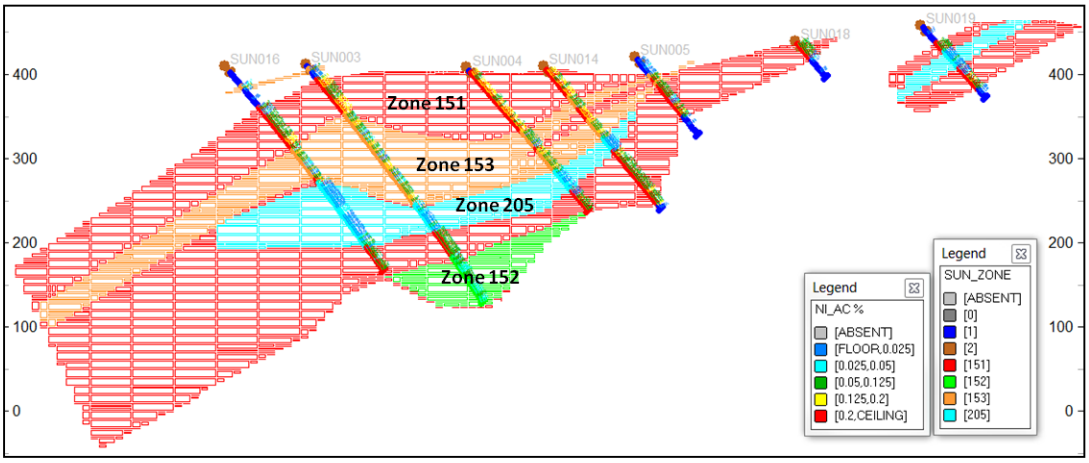

Figure 14-23: Cross section showing zone 151, 152, 153, and 205. (view looking north)

14.3 Geological Modelling and Block Model Creation

In all cases, the geological modelling was conducted in Datamine Studio 3 software and comprised the following:

- importing the collar, survey, assay and geology data into Datamine to create a desurveyed drillhole file;
- importing the topography data file;
- the creation of mineralisation wireframes based on the logged serpentinite body and the grade domains outlined above; and
- the creation of an empty block model coded by zone to distinguish the different geological domains identified (Figure 14-24 to Figure 14-26 and Table 14-1)
- The empty block model created used a parent cell size of 50mN by 50mE by 10mZ for the Rönnbäcksnäset deposit and a 25mN by 25mE by 10mZ for the Vinberget deposit and a 50mN by 50mE by 10mZ for the Sundsberget deposit, representing a division of the current drillhole spacing observed at each deposit (Table 14-2).

Table 14-1 shows the coding applied to the various geological domains in the Rönnbäcksnäset, Vinberget and Sundsberget geological models.

Table 14-2 shows the block model parameters used to build the empty block models for the Rönnbäcksnäset, Vinberget and Sundsberget deposits.

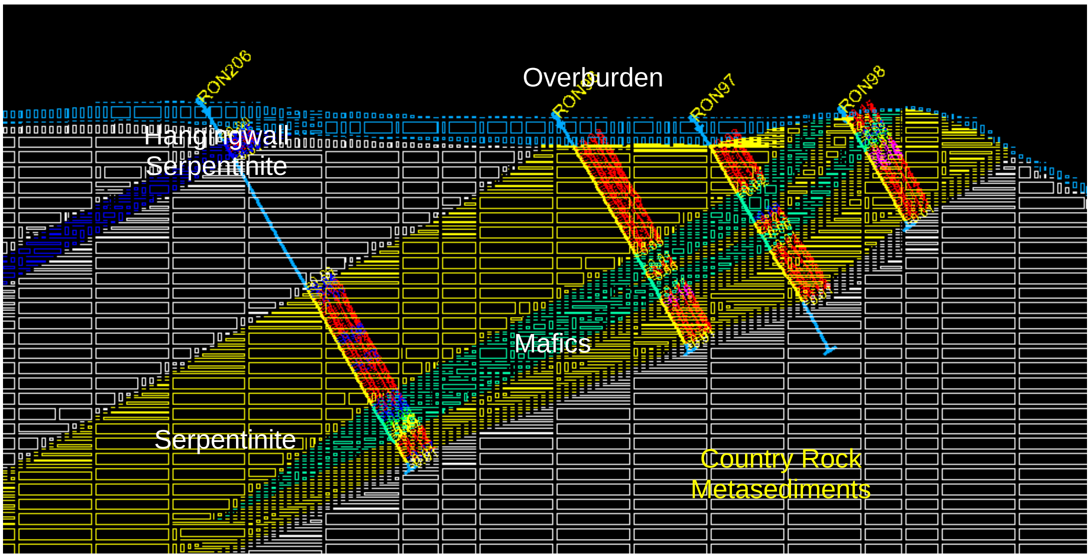

Figure 14-24: Rönnbäcksnäset empty block model (looking east, block height = 10 m)

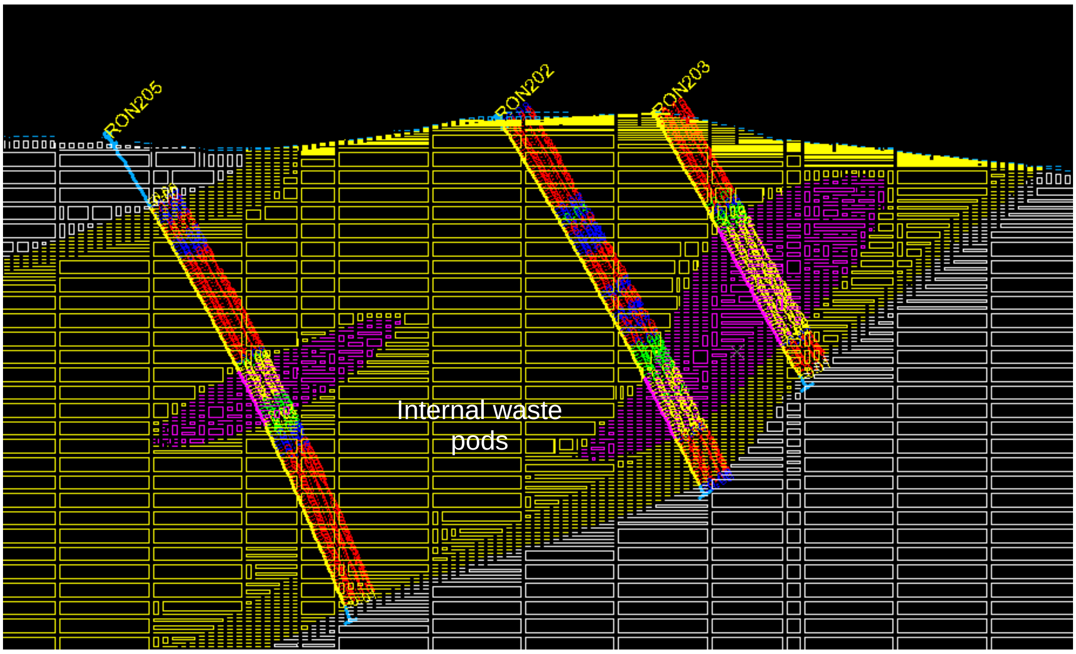

Figure 14-25: Rönnbäcksnäset empty block model showing internal waste pods (looking east, block height = 10 m)

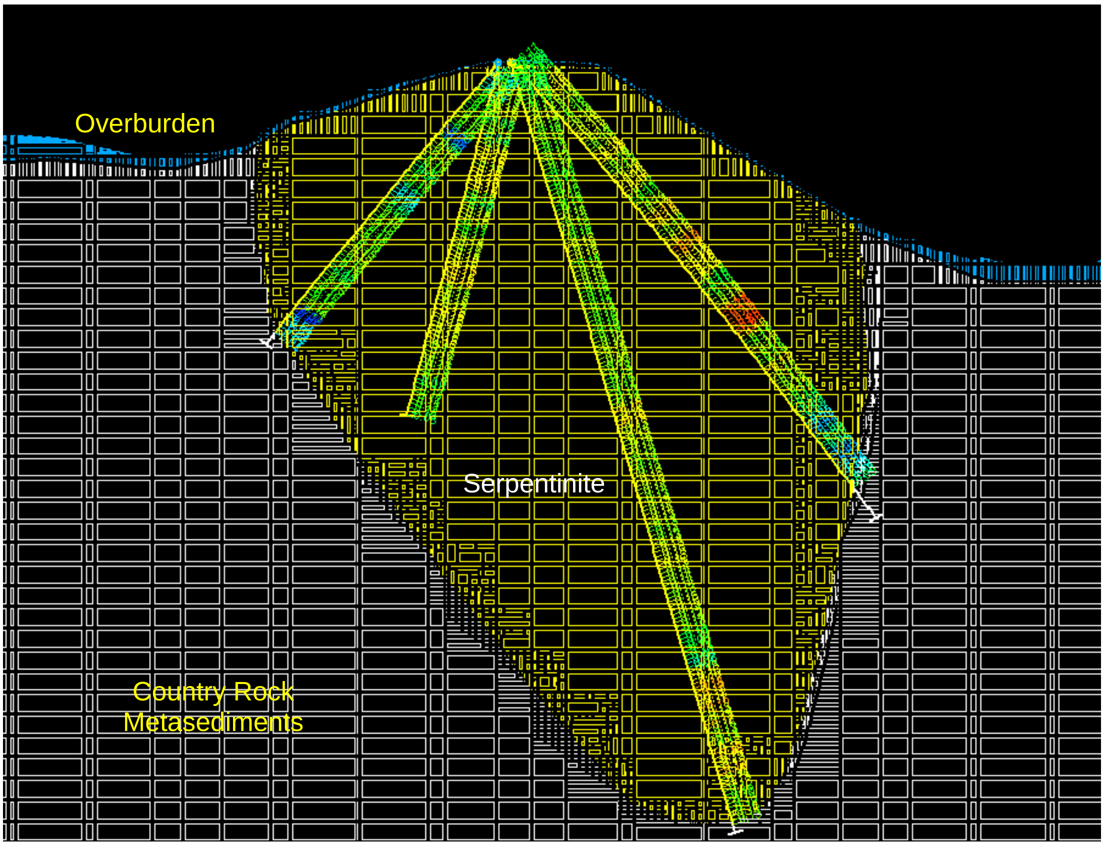

Figure 14-26: Vinberget empty block model (looking northwest, block height = 10 m)

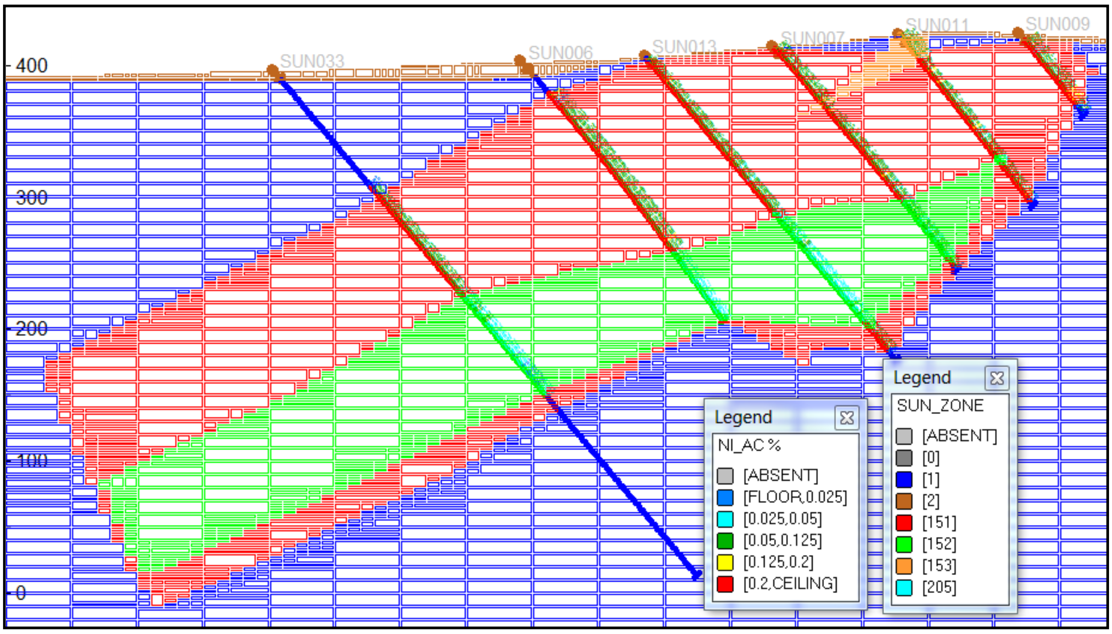

Figure 14-27: Sundsberget empty block model (looking north, block height= 10m)

Table 14-1: Zone codes created for Rönnbäcken Nickel Sulphide Project

Deposit	Geology	Code
Rönnbäcksnäset	Metasediments	0
	Overburden	2
	Internal waste (NE)	102
	Internal waste (SW)	112
	High Grade Serpentinite	151
	Low Grade Serpentinite	152
	Mafics	205
Vinberget	Metasediments	0
	Overburden	2
	Serpentinite	151
Sundsberget	Metasediments	1
	Overburden	2
	High Grade Serpentinite	151
	Low Grade Serpentinite (low Ni-AC%)	152
	Low Grade Serpentinite (low Ni-PCT %)	153
	Mafics (internal waste)	205

Table 14-2: Block model parameters

DEPOSIT	DIRECTION	ORIGIN	BLOCK SIZE	NR. OF BLOCKS
Rönnbäcksnäset	X	1479250	50	55
	Y	7267000	50	60
	Z	0	10	75
Vinberget	X	1483600	25	50
	Y	7262200	25	50
	Z	250	10	100
Sundsberget	X	8550	50	45
	Y	8180	50	55
	Z	-300	10	110

14.4 Available Data

The Rönnbäcksnäset deposit consists of 54 diamond drillholes for a total of 7,770 drilled meters. Of this, 5,124 drilled meters have been assayed for Ni-AC. The Vinberget deposit consists of 38 diamond drillholes for a total of 7,602 drilled meters. Of this, 6,723 drilled meters have been assayed for Ni-AC. The Sundsberget deposit consists of 32 diamond drillholes for a total of 6,888 drilled meters. Of this, 5,856 drilled meters have been assayed for Ni-AC. This is summarised in Table 14-3 below.

Table 14-3: Available data

DEPOSIT	Nr. of drillholes	Total Meters Drilled	Ni-AC Assayed Meters
Rönnbäcksnäset	54	7,770	5,124
Vinberget	38	7,602	6,723
Sundsberget	32	6,888	5,856

14.5 Data Validation

All available data was validated through the production of histograms and scatterplots and the use of the Datamine drillhole validation tools upon creation of a desurveyed drillhole file. Twenty drillholes were removed from the database (17 drillholes from Vinberget and three from Rönnbäcksnäset) due to a lack of associated survey, assay and/or geological data, but no additional errors were found in the data files provided. SRK is satisfied that the resulting data is suitable to be used in the Mineral Resource Estimate.

14.6 Raw Statistics

Table 14-4 shows the raw statistics for the domains modelled at the Rönnbäcksnäset, Vinberget and Sundsberget deposits. The main serpentinite zones are highlighted in red. As shown, the mean Ni-AC grade of the Rönnbäcksnäset high grade serpentinite is 0.111% and the mean grade of the low grade serpentinite is 0.054%. The mean grade of the Vinberget serpentinite is 0.131% Ni-AC. The mean Ni-AC grade of the Sundsberget high Ni PCT grade serpentinite (Zone 151) is 0.096% and the mean Ni-AC grade of the lower grade Ni PCT serpentinite (Zone 153) is 0.074%.

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 a significant impact on the final estimation. Within the main serpentinite domains, Table 14-4 shows that CoV values are very low, being 0.23, 0.25 and 0.32 respectively and indicating the low variability of the data.

Table 14-4: Length weighted Ni-AC statistics for the Rönnbäcksnäset Vinberget and Sundsberget deposits

DEPOSIT	ZONE	NSAMPLES	MIN	MAX	RANGE	MEAN %	VAR	SDEV	CoV
Ron	102	131	0.001	0.081	0.08	0.015	0	0.012	0.8
	112	104	0.002	0.096	0.094	0.016	0	0.014	0.88
	120	67	0.002	0.116	0.114	0.037	0.001	0.033	0.89
	151	**1,684**	**0.003**	**0.192**	**0.189**	**0.111**	**0.001**	**0.026**	**0.23**
	152	**410**	**0.004**	**0.154**	**0.15**	**0.054**	**0**	**0.02**	**0.37**
	205	315	0.001	0.139	0.138	0.03	0.001	0.029	0.97
Vin	**151**	**3,632**	**0.001**	**0.222**	**0.221**	**0.131**	**0.001**	**0.033**	**0.25**
Sun	**151**	**1 941**	**0.0005**	**0.206**	**0.206**	**0.096**	**0.001**	**0.031**	**0.32**
	152	374	0.0159	0.088	0.072	0.041	0.000	0.011	**0.26**
	153	**596**	**0.0005**	**0.174**	**0.174**	**0.074**	**0.002**	**0.040**	**0.55**
	205	221	0.0005	0.126	0.126	0.017	0.000	0.016	0.97

14.7 Compositing

Data compositing is commonly 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 majority of samples in all the Rönnbäcksnäset, Vinberget and Sundsberget drillhole files are 2 m in length with smaller samples being present at the geological contacts. Due to the very low CoV observed in the database and the near normal populations shown in the histograms of the raw data, all samples have been composited to 2 m as increasing the sample to a larger composite length has little impact on the variability of the database.

Table 14-5: 2 m composite Ni-AC statistics for the Rönnbäcksnäset, Vinberget and Sundsberget deposits

DEPOSIT	ZONE	NSAMPLES	MIN	MAX	RANGE	MEAN %	VAR	SDEV	CoV
Ron	102	109	0.001	0.077	0.076	0.015	0.000	0.011	0.73
	112	103	0.004	0.075	0.071	0.016	0.000	0.013	0.81
	120	67	0.002	0.100	0.098	0.037	0.001	0.033	0.89
	151	**1 589**	**0.004**	**0.192**	**0.188**	**0.111**	**0.001**	**0.026**	**0.23**
	152	**354**	**0.008**	**0.135**	**0.127**	**0.054**	**0.000**	**0.019**	**0.35**
	205	250	0.001	0.139	0.138	0.029	0.001	0.029	1.00
Vin	**151**	**3 275**	**0.001**	**0.215**	**0.214**	**0.131**	**0.001**	**0.032**	**0.24**
Sun	**151**	**1 810**	**0.001**	**0.188**	**0.188**	**0.096**	**0.001**	**0.030**	**0.31**
	152	358	0.016	0.088	0.072	0.041	0.000	0.011	0.26
	153	**482**	**0.001**	**0.174**	**0.174**	**0.076**	**0.002**	**0.040**	**0.53**
	205	204	0.001	0.086	0.086	0.017	0.000	0.016	0.96

14.8 Density Analysis

A comprehensive density dataset has been generated by the Company using the methodology described in Section 10.8. In total, 2,701 density measurements are present for the Rönnbäcksnäset domains and 3,416 density measurements are present for the Vinberget domains and 3,291 density measurements are present for the Sundsberget domains.

Table 14-6 shows the breakdown of samples per domain for Rönnbäcksnäset and Vinberget deposits. Density measurements have also been acquired for the waste domains allowing accurate tonnages to be determined for all material types.

Table 14-6: Density measurements

DEPOSIT	DOMAIN	NSAMPLES	MIN	MAX	RANGE	MEAN (g/cm^3)	VAR	SDEV
Ron	0	114	2.57	3.14	0.57	2.74	0.01	0.102
	102	118	2.64	3.25	0.61	2.79	0.033	0.181
	112	104	2.6	3.03	0.43	2.7	0.004	0.063
	120	65	2.48	3.04	0.56	2.69	0.006	0.077
	151	**1681**	**2.48**	**3.38**	**0.9**	**2.73**	**0.008**	**0.087**
	152	**435**	**2.54**	**3.24**	**0.7**	**2.73**	**0.012**	**0.107**
	205	307	2.65	3.26	0.61	2.96	0.035	0.187
Vin	0	94	2.63	3.21	0.58	2.8	0.01	0.1
	2	1	2.67	2.67	0	2.67	-	-
	151	**3619**	**2.46**	**3.06**	**0.6**	**2.71**	**0**	**0.05**
Sun	1	190	2.66	3.27	0.61	2.86	0.023	0.152
	151	**1915**	**2.52**	**3.27**	**0.75**	**2.81**	**0.010**	**0.102**
	152	372	2.70	3.20	0.50	2.92	0.009	0.094
	153	593	2.62	3.24	0.62	2.93	0.018	0.134
	205	221	2.69	3.25	0.56	3.07	0.013	0.113

14.9 Geostatistical Analyses

14.9.1 Variography

The 2 m composited drillhole database, coded by the modelled domains, was imported into ISATIS software for the geostatistical analysis. Variography was attempted on the main serpentinite ore domain (151), but due to the low number of samples and poor variograms produced in the low grade serpentinite domain (152) and the waste domains, variography was not possible and those produced for domain 151 were utilised in the interpolation of all other domains.

Directional experimental semi-variograms were produced for Ni-AC and Co-AC for Rönnbäcksnäset and Ni-AC, Co-AC, Fe-PCT and SG for Sundsberget, with omni directional experimental semi-variograms produced for Vinberget. Directional semi-variograms were produced for Rönnbäcksnäset and Sundsberget due to the interpreted dip and strike direction observed for the serpentinite body. Omni directional semi-variograms were produced for Vinberget due to the apparent lack of mineralisation trends in the sub-vertical serpentinite body. The directional semi variograms that were attempted for Vinberget, using a near vertical dip to the search ellipse, produced near identical ranges and structures in the down-dip, along strike and down-hole directions indicating the appropriateness of an omni directional variogram.

The semi-variograms were produced using a 2 m (composite length) lag in the downhole / omni directional direction allowing the short-scale structures and nugget variance to be determined. Along strike and down-dip variograms for the Rönnbäcksnäset and Sundsberget deposits were then produced with the nugget fixed from the downhole variogram, and using a lag spacing of 100 m with a 50% tolerance being applied to the lag spacing.

For Rönnbäcksnäset and Vinberget, variograms were not produced for Fe with the Fe estimation using the variogram parameters determined for the Ni-AC data.

Two sets of variograms were produced for the Rönnbäcksnäset deposit to better model the change in strike direction of the serpentinite body.

Figure 14-28 shows the plane used to define the directional variography for the northeast portion of the Rönnbäcksnäset deposit, using a 016º azimuth, 40º dip to the west and a 45º plunge.

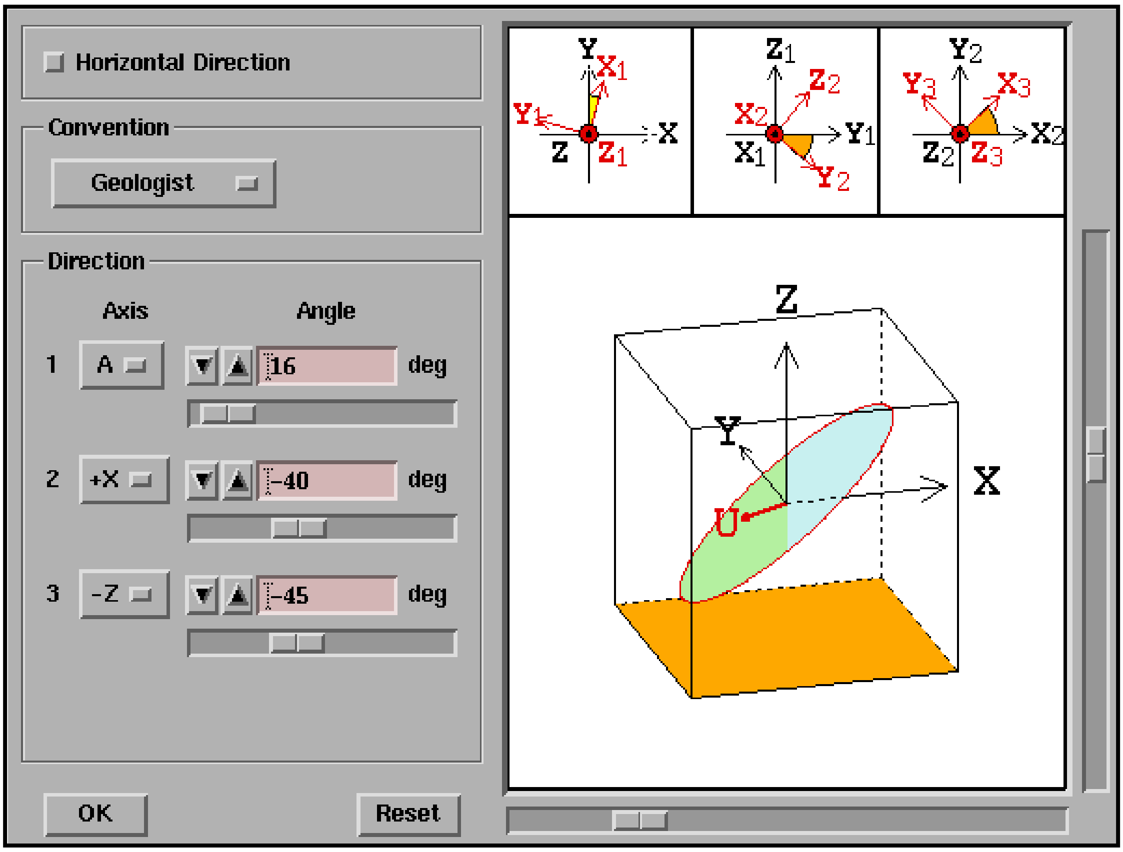

Figure 14-28: Directional variography plane for Rönnbäcksnäset northeast

Figure 14-29 shows the plane used to define the directional variography for the southwest portion of the Rönnbäcksnäset deposit, using a 085º azimuth, 25º dip to the west and no plunge.

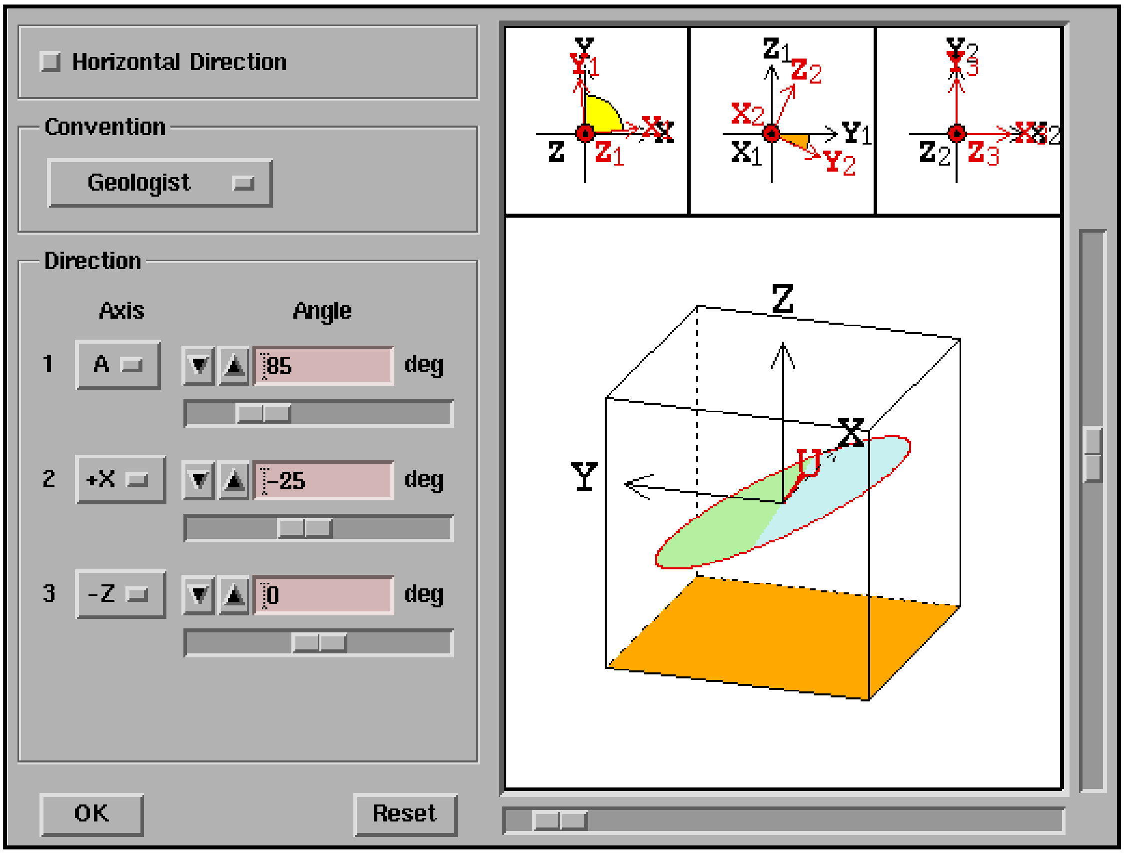

Figure 14-29: Directional variography plane for Rönnbäcksnäset southwest

Figure 14-30 to Figure 14-36 show the Ni-AC semi-variograms for Rönnbäcksnäset and Vinberget. Sample pairs are not displayed on the variograms for easier visualisation purposes; however, they were checked in the variography process with sufficient numbers being used.

Variograms produced for Co-AC showed similar structures and ranges.

Density was modelled using omni directional semi variograms. These are shown in Figure 14-37 to Figure 14-39.

The results of the variography are shown in Table 14-7 to Table 14-9.

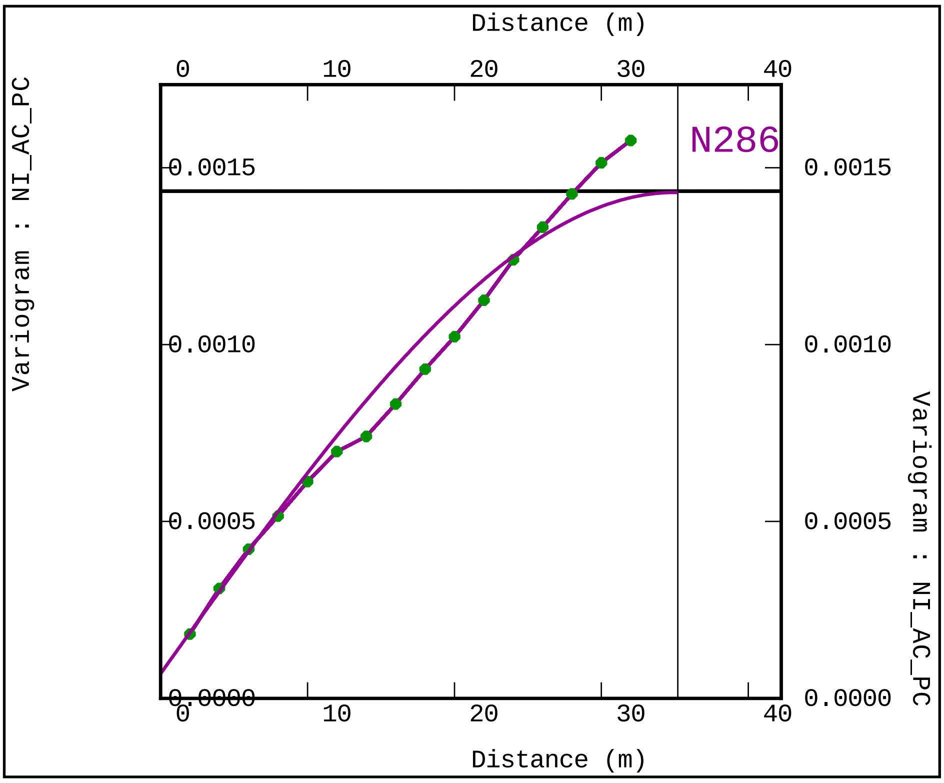

Figure 14-30: Ni-AC downhole semi-variogram for Rönnbäcksnäset northeast

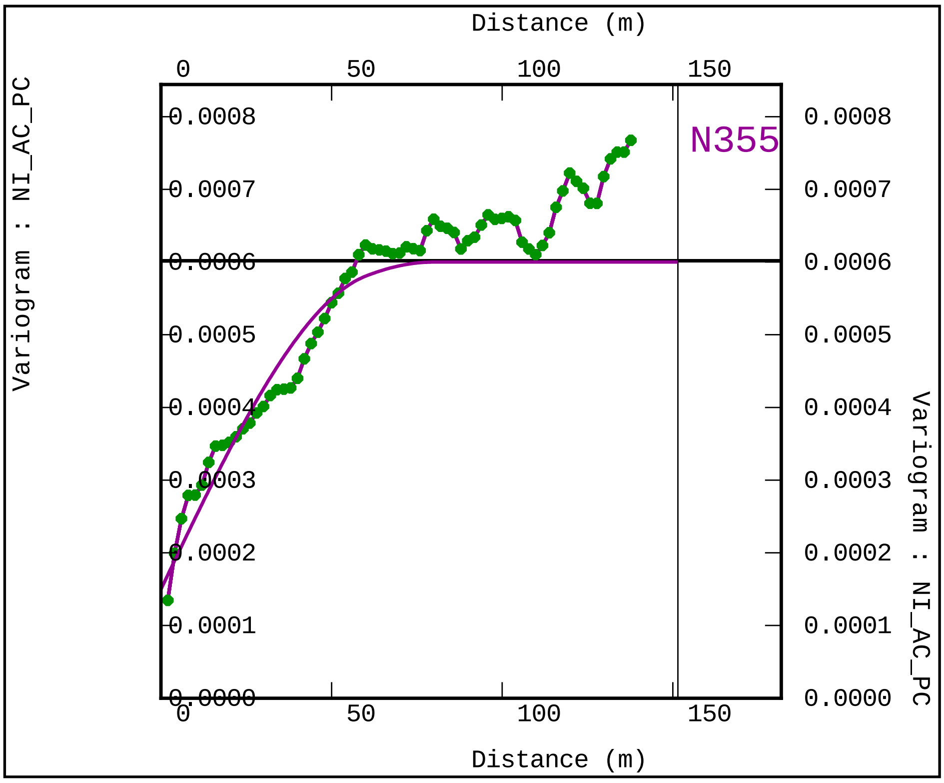

Figure 14-31: Ni-AC downhole semi-variogram for Rönnbäcksnäset southwest

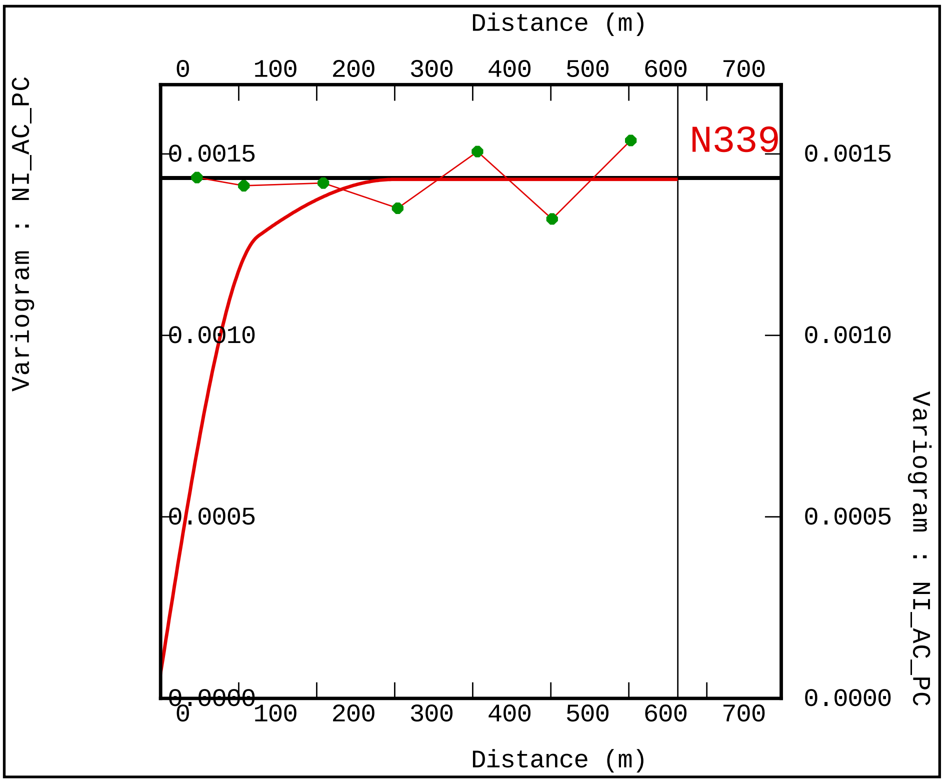

Figure 14-32: Ni-AC along strike semi-variogram for Rönnbäcksnäset northeast (016º)

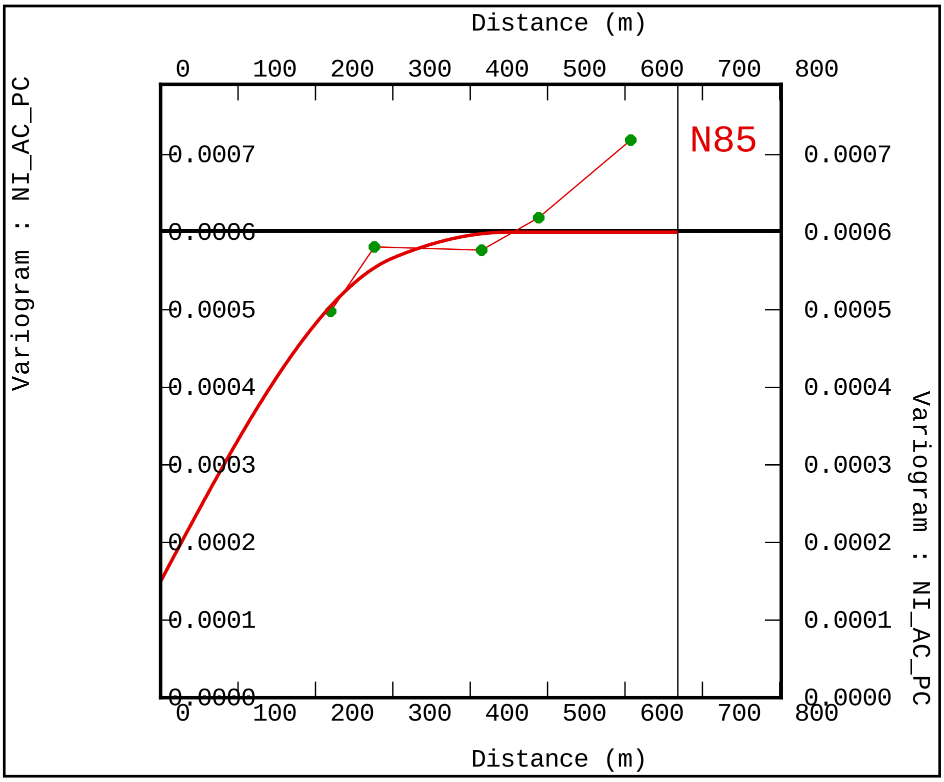

Figure 14-33: Ni-AC along strike semi-variogram for Rönnbäcksnäset southwest (085º)

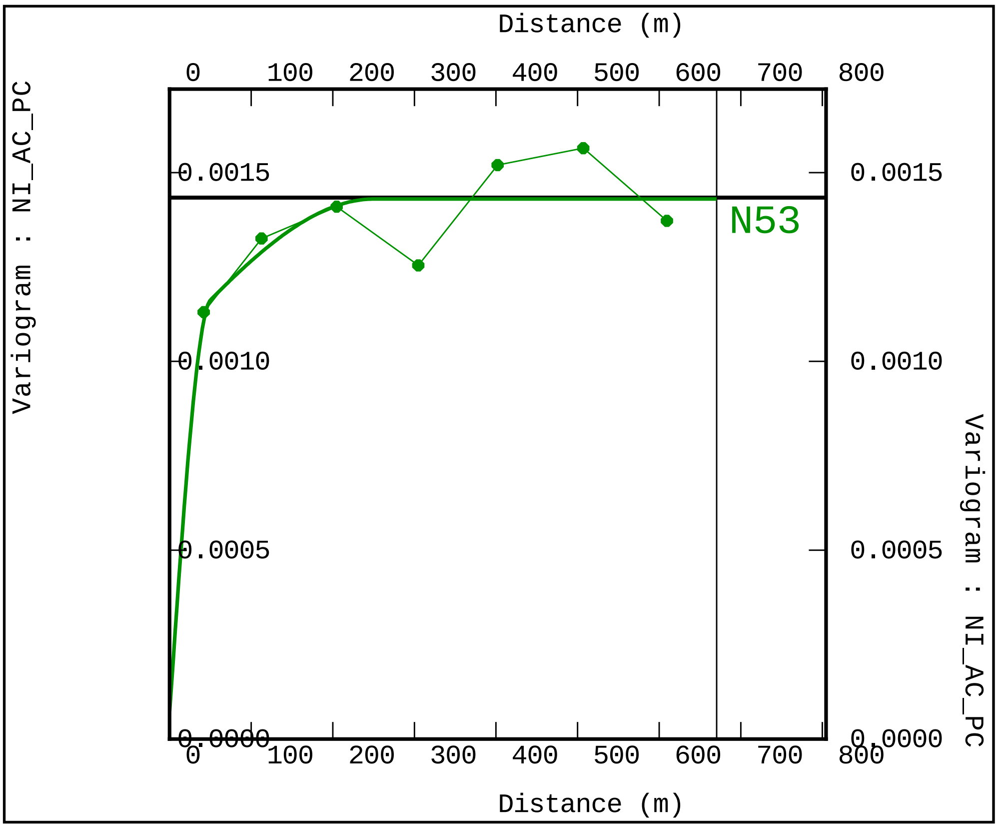

Figure 14-34: Ni-AC down-dip semi-variogram for Rönnbäcksnäset northeast (040º west)

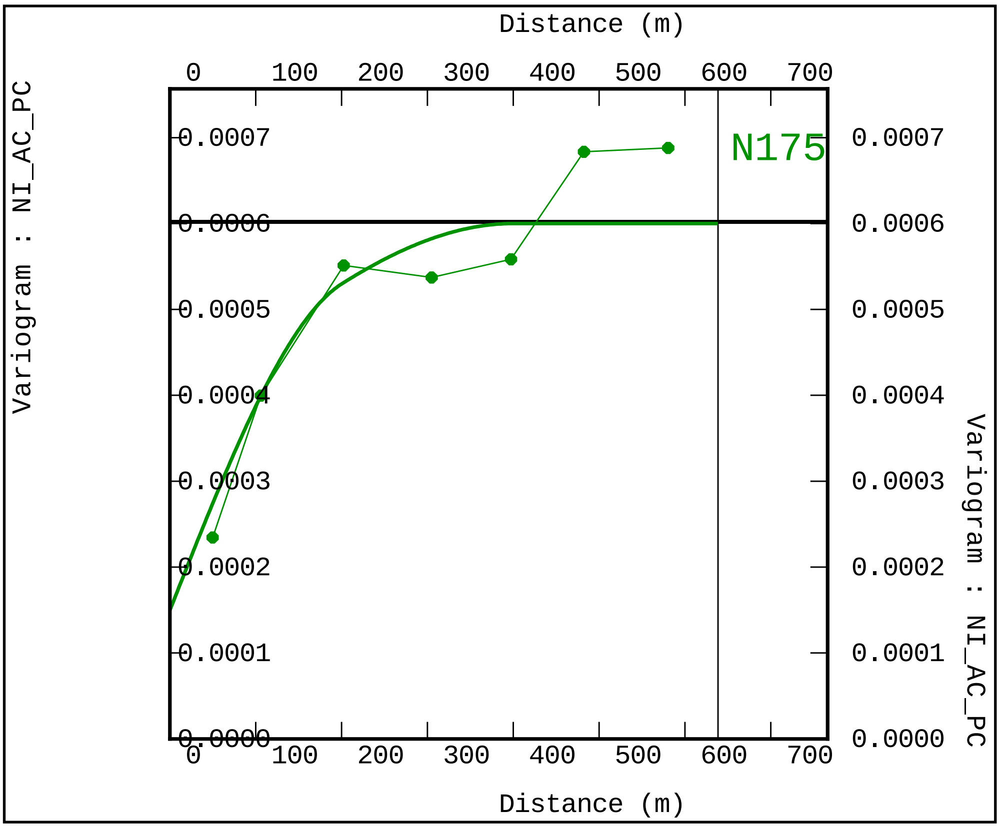

Figure 14-35: Ni-AC down-dip semi-variogram for Rönnbäcksnäset southwest (025º north)

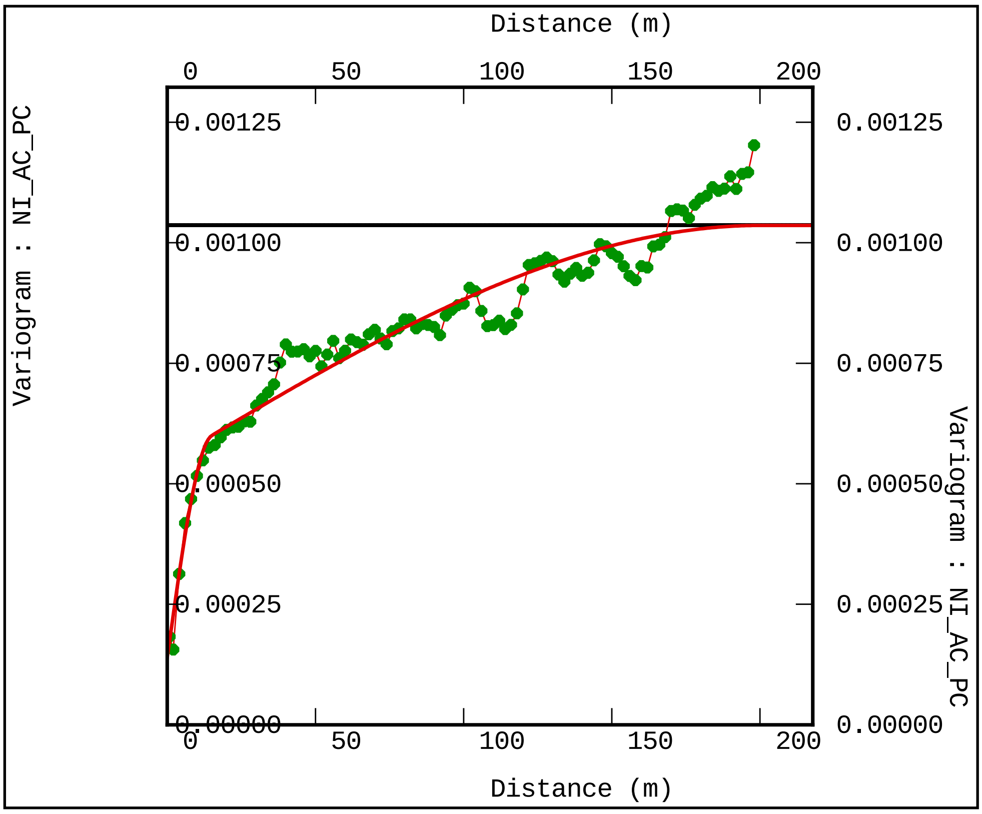

Figure 14-36: Ni-AC omni directional semi-variogram for Vinberget

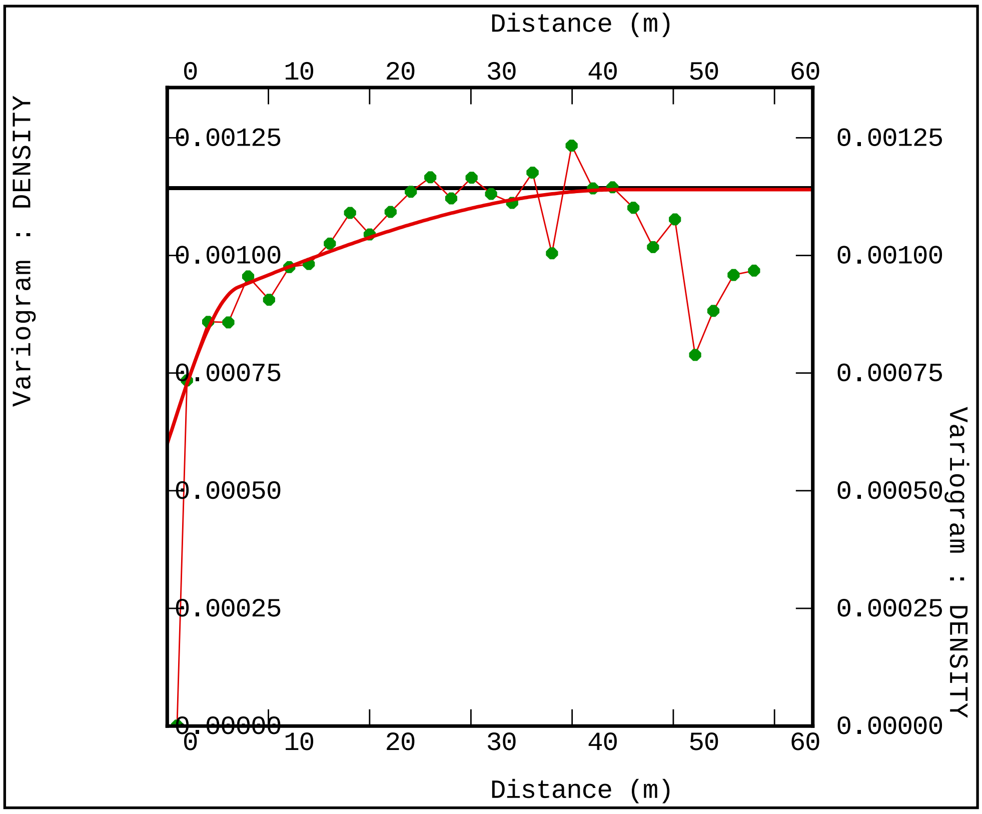

Figure 14-37: Density omni directional semi-variogram for Rönnbäcksnäset northeast

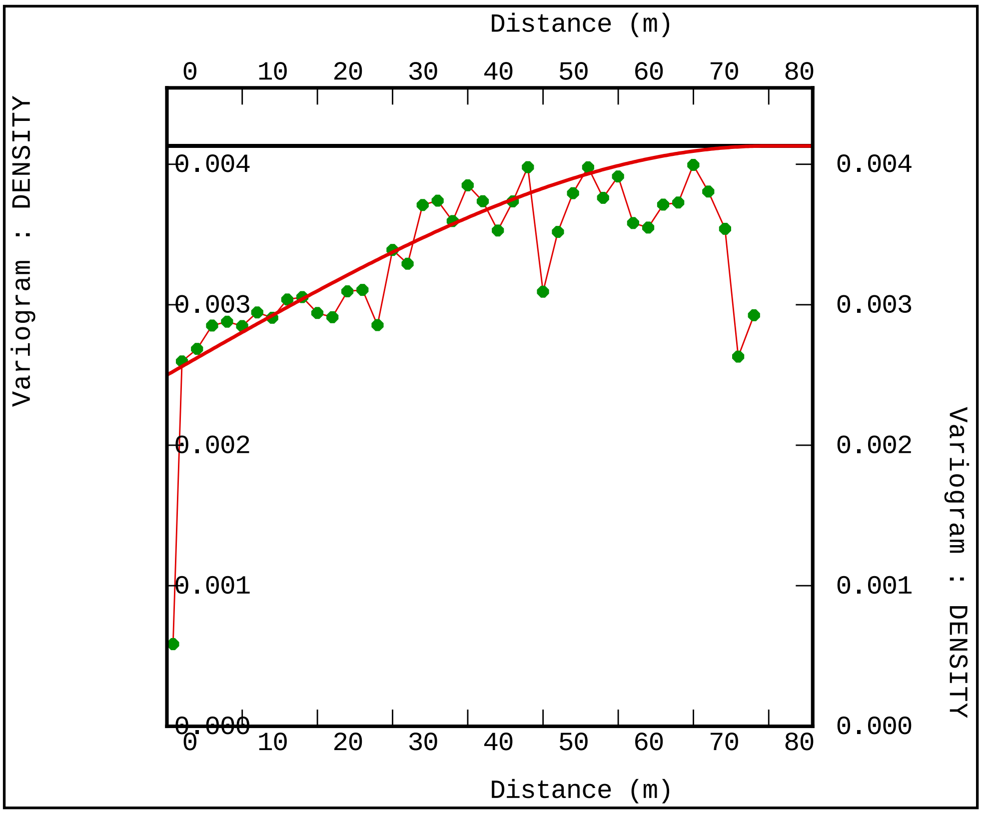

Figure 14-38: Density omni directional semi-variogram for Rönnbäcksnäset southwest

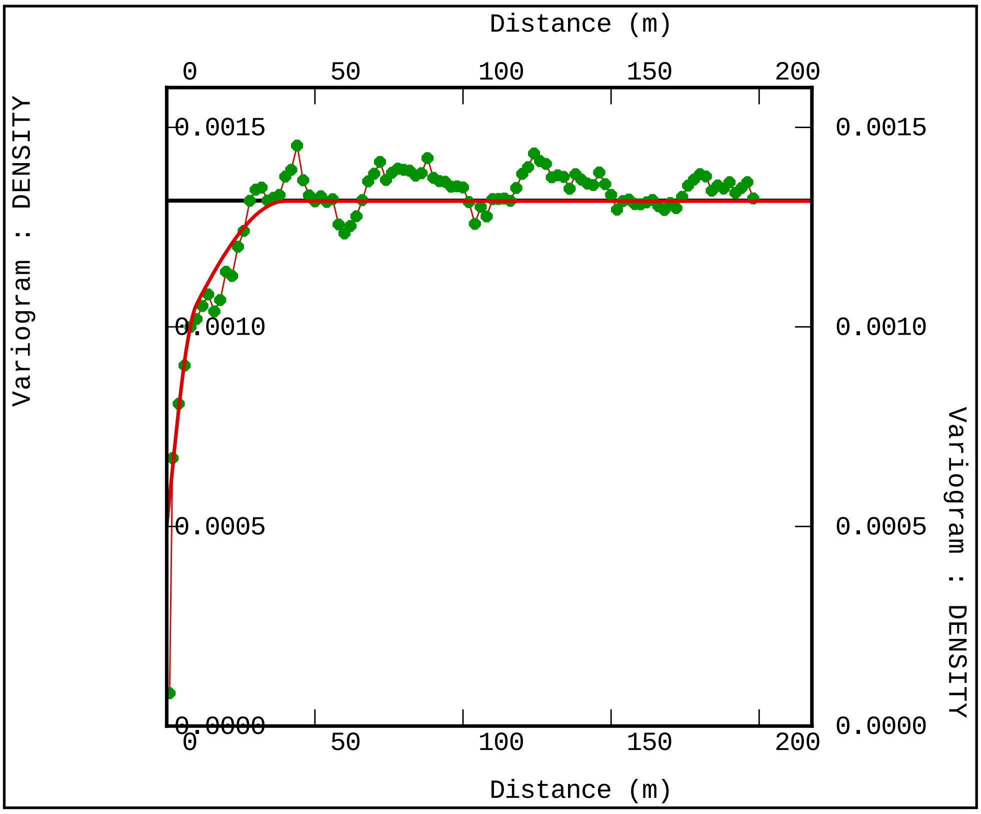

Figure 14-39: Density omni directional semi-variogram for Vinberget

The reference plane for the Sundsberget variograms is shown in Figure 14-40. Figure 14-41 to Figure 14-43 show the Sundsberget directional variograms for Ni-AC in Zone 151.

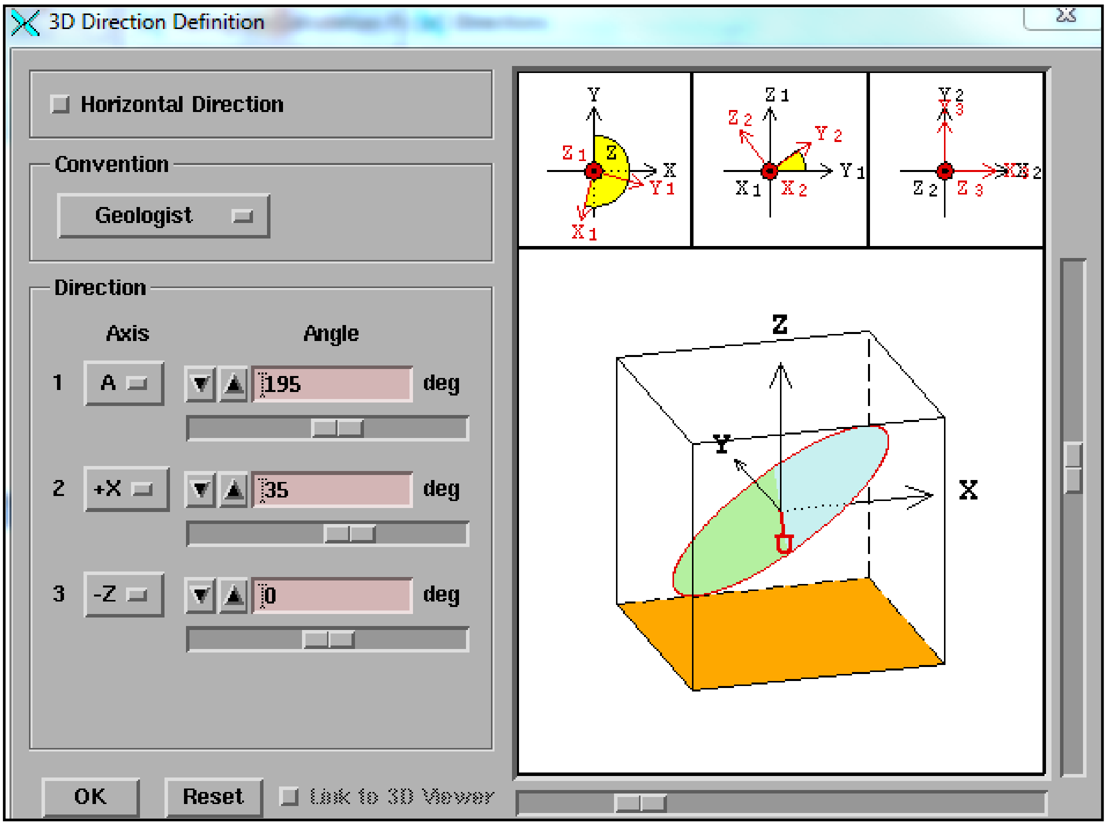

Figure 14-40: Directional variography plane for Sundsberget

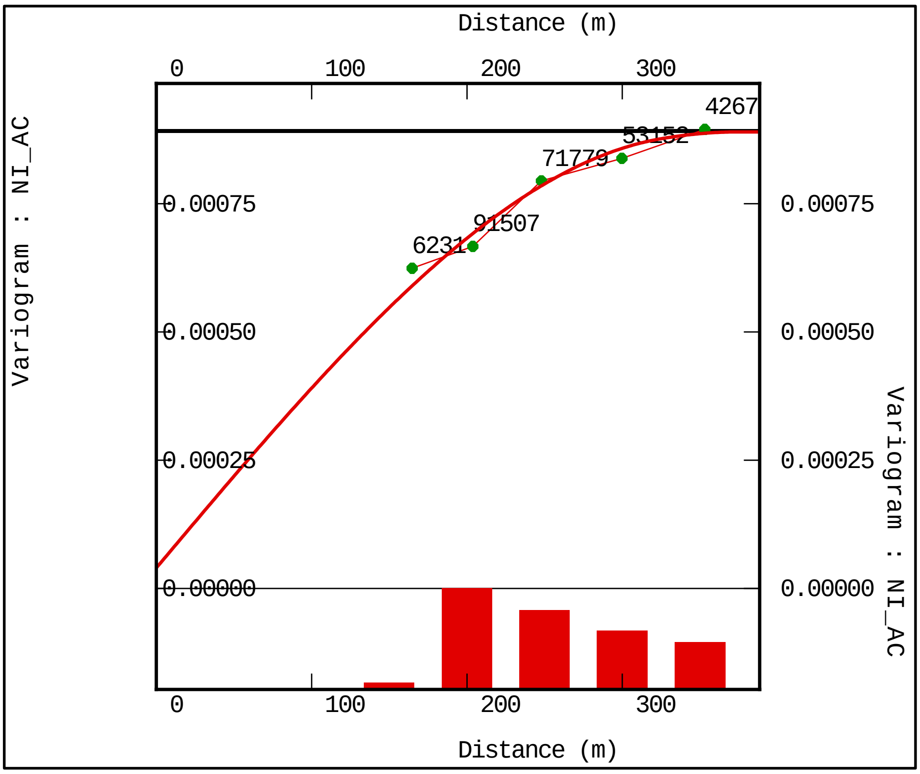

Figure 14-41: Zone 151, along strike (195°), directional semi-variogram for Sundsberget

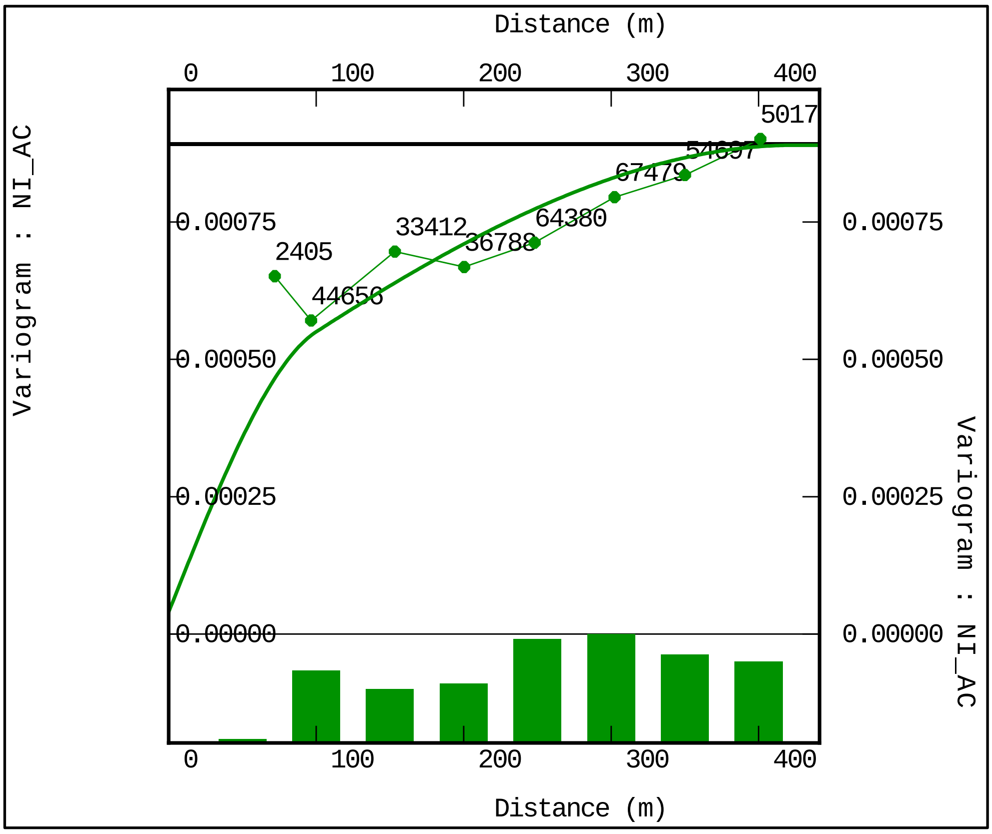

Figure 14-42: Zone 151, down dip (35°), directional semi-variogram for Sundsberget

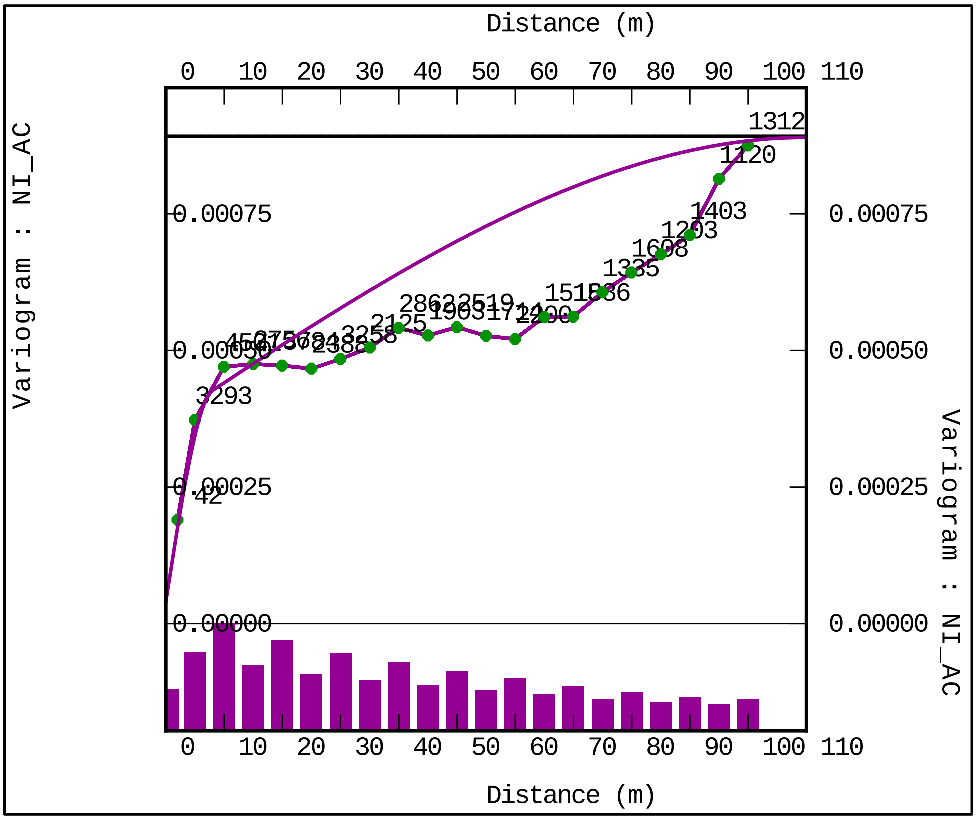

Figure 14-43: Zone 151, down hole, directional semi-variogram for Sundsberget

Table 14-7: Variography Results for Rönnbäcksnäset

Deposit	Assay	Nugget	Structure 1 - Range			Variance	Structure 2 - Range			Variance	Sill	Relative Nugget (%)
			Down-Dip (40º)	Along-Strike (016º)	Downhole		Down-Dip (40º)	Along-Strike (016º)	Downhole			
Ronn NE (151)	Ni-AC	0.00007	50	125	35	0.00098	250	300	35	0.00038	0.00143	5
	Co-PPM*	16.84	100	125	30	235.7	200	300	35	91.4	343.94	5
	Density	0.0006	7	7	7	0.00027	45	45	45	0.00027	0.00114	53

Deposit	Assay	Nugget	Structure 1 - Range			Variance	Structure 2 - Range			Variance	Sill	Relative Nugget (%)
			Down-Dip (25º)	Along-Strike (085º)	Downhole		Down-Dip (25º)	Along-Strike (085º)	Downhole			
Ronn SW (151)	Ni-AC	0.00015	200	300	60	0.000225	400	450	80	0.00038	0.0006	25
	Co-PPM*	31	125	200	60	264.5	450	300	90	264.5	560	6
	Density	0.0025	80	80	80	0.00163	-	-	-	-	0.00413	61

*variography based on Co-PPM due to very low variance observed

Table 14-8: Variography Results for Vinberget

Deposit	Assay	Nugget	Structure 1 - Range			Variance	Structure 2 - Range			Variance
			Down-Dip (Omni)	Along-Strike (Omni)	Downhole (Omni)		Down-Dip (Omni)	Along-Strike (Omni)	Downhole (Omni)	
Ronn NE (151)	Ni-AC	0.00014	15	15	15	0.000405	200	200	200	0.000491
	Co-PPM*	25	25	25	25	60.36	60	60	60	55.36
	Density	0.0005	10	10	10	0.000405	40	40	40	0.00041

Structure 3 – Range			Variance	Sill	Relative Nugget (%)
Down-Dip (Omni)	Along-Strike (Omni)	Downhole (Omni)			
-	-	0.00143	5	0.00104	14
200	200	343.94	5	206.08	12
-	-	0.00114	53	0.001315	38

*variography based on Co-PPM due to very low variance observed

Table 14-9: Variography Results for Sundsberget

Deposit	Assay	Nugget	Structure 1 - Range			Variance	Structure 2 - Range			Variance	Structure 3 – Range			Variance	Sill	Relative Nugget (%)
			Along-Strike (195º)	Down-Dip (35º)	Downhole		Along-Strike (195º)	Down-Dip (35º)	Downhole		Along-Strike (195º)	Down-Dip (35º)	Downhole			
Sundsberget Zone 151	Ni_AC	0.00004	325	100	8	0.0003	375	425	110	0.00	-	-	-	-	0.0009	4.5
	NI_PCT	0.00004	175	140	8	0.0002	250	140	85	0.00	-	-	-	-	0.0005	7.5
	CO_AC	10.00	150	100	6	25.00	225	100	105	155.00	350	650	110	150.00	340.0	2.9
	FE_PCT	0.04	250	100	9	0.14	250	350	110	0.15	-	-	-	-	0.3	12.3
	SG	0.0030	250	85	5	0.0037	275	275	125	0.00	-	-	-	-	0.01	31.3

14.9.2 Summary

The directional experimental semi-variograms produced for Rönnbäcksnäset northeast allowed very robust variogram models to be generated in the downhole and down-dip directions (40° to the west) for Ni-AC and Co-AC. Along strike (016°) variograms were however poor, with little or no structure being observed.

The directional experimental semi-variograms produced for Rönnbäcksnäset southeast allowed very robust variogram models to be generated in the downhole and down-dip directions (25° to the north) for Ni-AC and Co-AC. Along strike (085°) variograms were also modelled for Ni-AC with a simple two structure spherical model. Co-AC variograms were however poor in the along strike direction.

The directional experimental semi-variograms produced for Sundsberget allowed very robust variogram models to be generated in the along strike (195°), downhole and down-dip directions (35° to the west) for Ni-AC, Co-AC, Fe-PCT and SG.

The omni directional experimental semi-variograms produced for Vinberget allowed very robust variogram models to be generated for Ni-AC and Co-AC.

Similarly, the omni directional experimental semi-variograms produced for Rönnbäcksnäset and Vinberget allowed very robust variogram models to be generated for density.

The results of the variography were used in the interpolation to assign the appropriate weighting to the samples 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 2/3rd rule to the total range of the variograms in the search ellipse dimensions forces the interpolation to use samples where covariance between samples exists. As a result of the variography, ordinary kriging (OK) was deemed the most appropriate interpolation technique to be applied to Ni-AC, Co-AC and density.

Table 14-10: Ranges and 2/3rd ranges for Rönnbäcksnäset

Deposit	Parameter	Along Strike	Down Dip	Across Strike
Rönnbäcksnäset (Directional)	Average Total Range	338m	325m	60m
	2/3 Average Range	225m	217m	40m
	Search Ellipse Chosen	225m	217m	20m
Vinberget (Omni Directional)	Average Total Range	200m	200m	200m
	2/3 Average Range	133m	133m	133m
	Search Ellipse Chosen	100m	100m	100m
Sundsberget	Average Total Range	375m	425m	110m
	2/3 Average Range	250m	283m	73m
	Search Ellipse Chosen	250m	280m	70m

14.10 Quantitative Kriging Neighbourhood Analysis (QKNA)

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

14.10.1 QKNA Process

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 al 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 in order 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 et 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 in order 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.

14.10.2 Interpolation Process

Neighbourhood tests were run on ore domains 151 and 152 for Rönnbäcksnäset, domain 151 for Vinberget, and domains 151, 152 and 153 for Sundsberget with the search ellipse dimensions being fixed against the optimum ranges identified in the variography and as highlighted in Table 14-7 to Table 14-9. Search ellipse range parameters were not tested in the QKNA process due to the robust variograms produced highlighting the optimum search ranges that should be used.

Should an insufficient number of blocks be estimated using the optimum ellipse ranges, additional QKNA scenarios should be run to test the optimum ellipse dimensions at an increasing ellipse size. This was however, not necessary and all QKNA models were based on a first pass interpolation.

Table 14-11 outlines the chosen parameters used in the QKNA tests.

Table 14-11: QKNA model parameters

QKNA Model	Search Ellipse Dimension (m)			Min Samples	Max Samples	Max Samples per Drillhole
	Along Strike	Down Dip	Across Strike			
Ron NE	225	217	20	6	12	6
Ron SW	225	217	20	6	12	6
Vin	100	100	100	12	24	3
Sun (Zone 151)	250	280	70	6	36	3
Sun (Zone 152, 153, 205)	250	280	35	6	36	3

The neighbourhood run was checked to ensure that an adequate number of blocks were filled ensuring that meaningful results were generated. The slope of regression was estimated into the individual models and each QKNA run compared. The results of the QKNA checks are shown in Table 14-12. As shown, a high number of blocks have been estimated using the optimum search parameters defined with a relatively high mean slope of regression in Rönnbäcksnäset (0.6 and 0.4), a high mean slope of regression in Vinberget (0.85), and a high mean slope of regression for all Sundsberget Zones (0.76 and 0.77)

Table 14-12: QKNA results: slope of regression and percent block filled

Deposit	Min	Max	Mean	% Filled in Run 1
Ron (151)	0.013	0.981	0.60	62
Ron (152)	0.011	0.803	0.40	94
Vin (151)	0.409	0.969	0.85	75
Sun (151)	0.205	0.999	0.77	70
Sun (152)	0.197	0.999	0.76	64
Sun (153)	0.192	0.999	0.77	75

The distribution of Ni-AC slope of regression values is shown in Figure 14-44 to Figure 14-46 for Rönnbäcksnäset. A high slope of regression (>0.8) can be seen around well-informed blocks with the slope of regression value decreasing towards the base of the model where the blocks are less well-informed with sample data. The slope of regression data shows that the southwest portion of the deposit is better informed with data than the northeast portion of the deposit, despite the closer drill spacing in the northeast portion of the deposit.

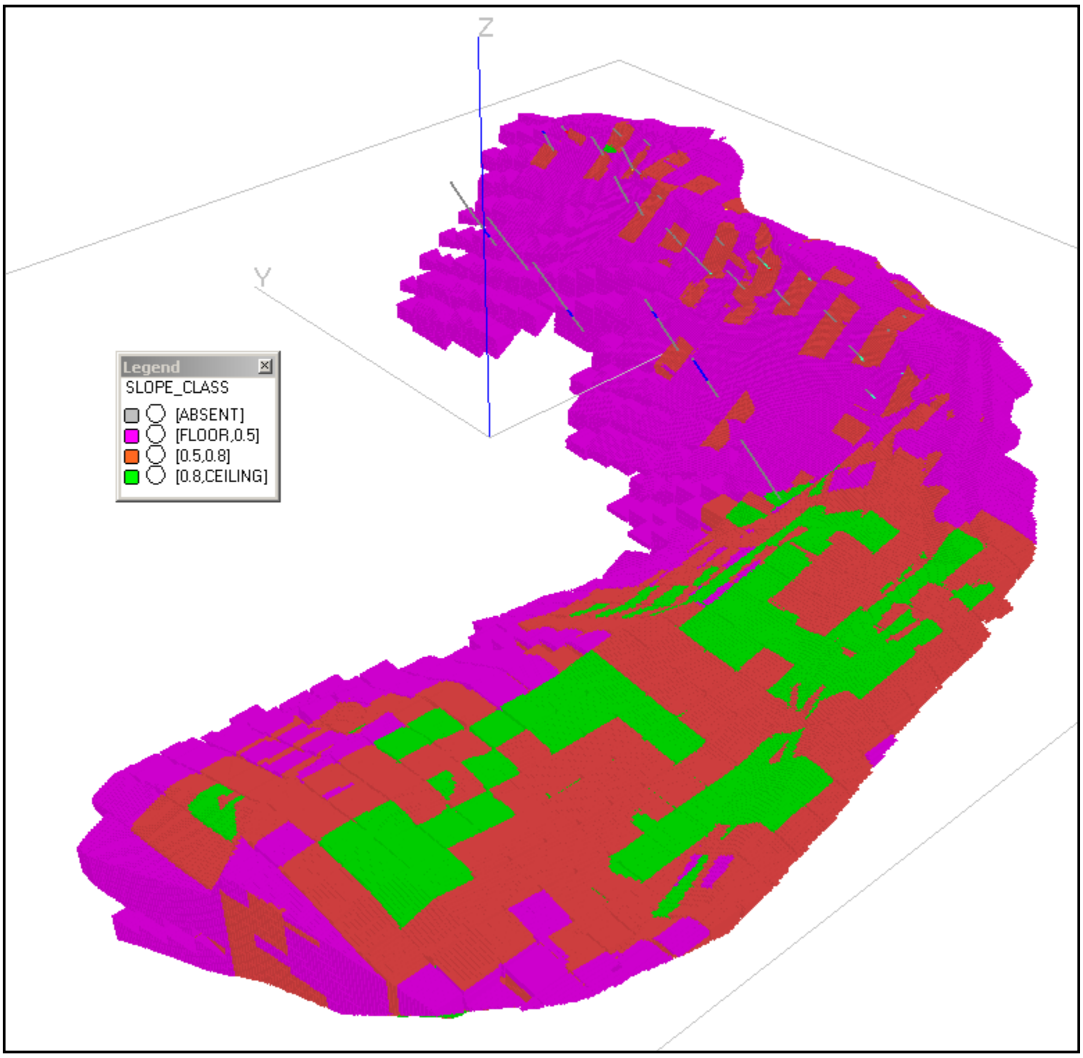

Figure 14-44: Rönnbäcksnäset block model coloured by slope of regression (looking northeast)

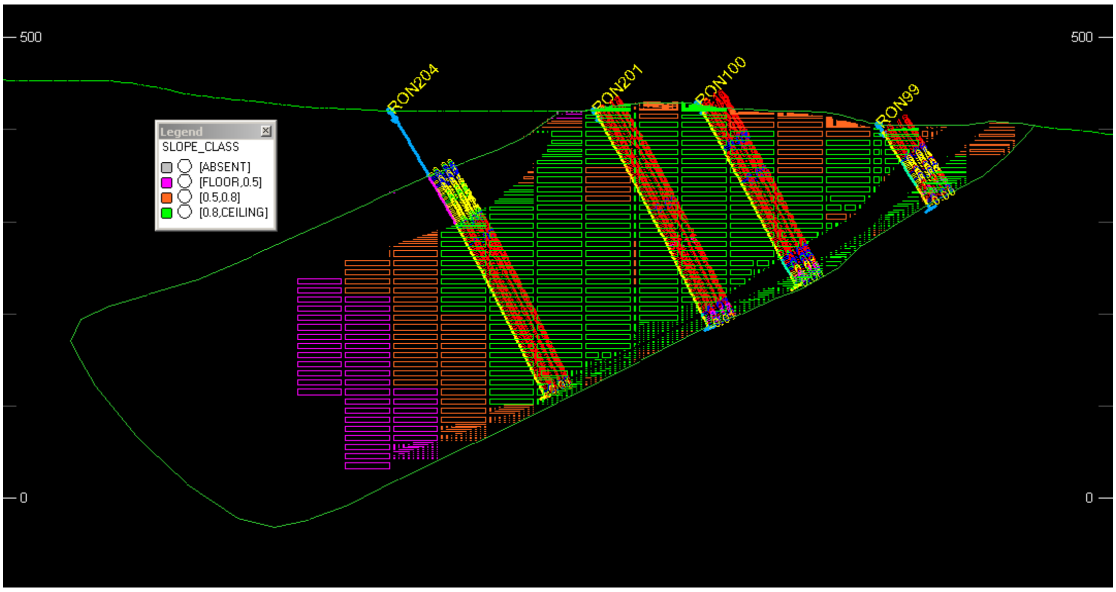

Figure 14-45: Rönnbäcksnäset southwest block model coloured by slope of regression (looking east)

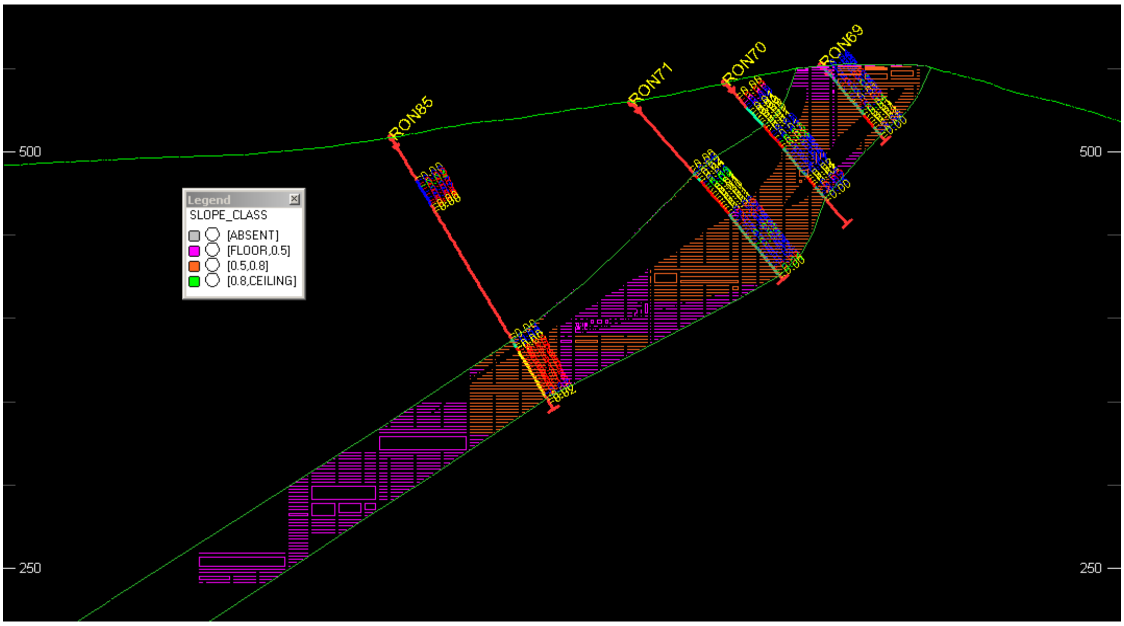

Figure 14-46: Rönnbäcksnäset northeast block model coloured by slope of regression (looking north)

The distribution of Ni-AC slope of regression values is shown in Figure 14-47 and Figure 14-48 for Vinberget. A high slope of regression (>0.8) can be seen around well-informed blocks with the slope of regression value decreasing towards the base and edges of the model where the blocks are less well-informed with sample data.

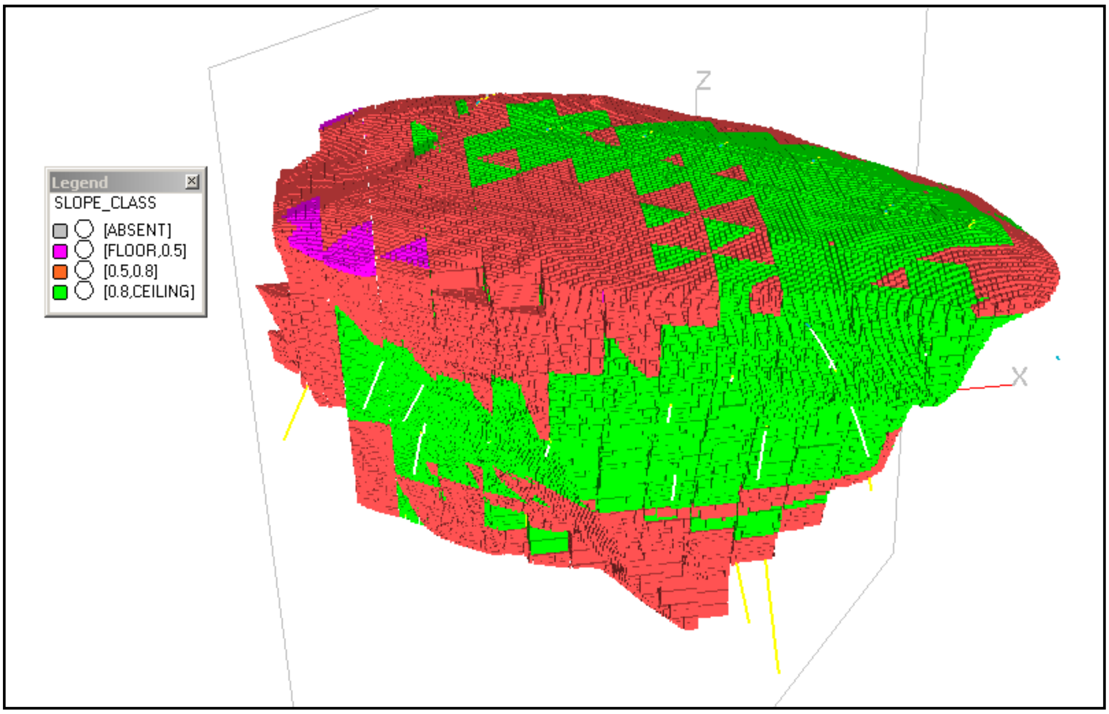

Figure 14-47: Vinberget block model coloured by slope of regression (looking north)

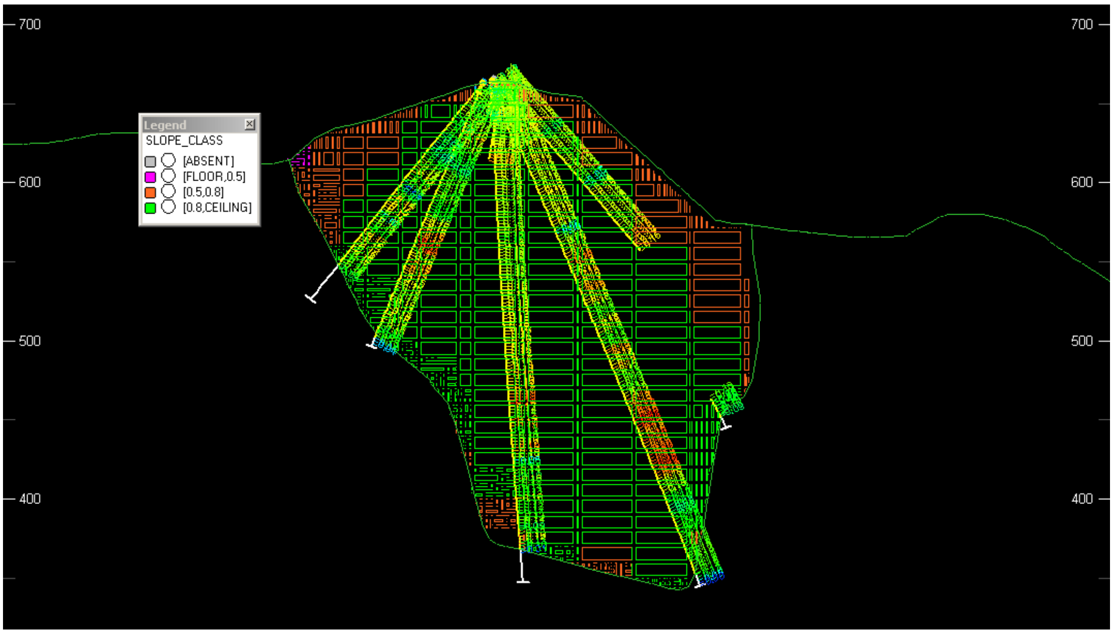

Figure 14-48: Vinberget block model coloured by slope of regression (looking northwest)

The distribution of Ni-AC slope of regression values is shown in Figure 14-49 and Figure 14-50 for Sundsberget. A high slope of regression (>0.8) can be seen around well-informed blocks with the slope of regression value decreasing towards the base and edges of the model where the blocks are less well-informed with sample data.

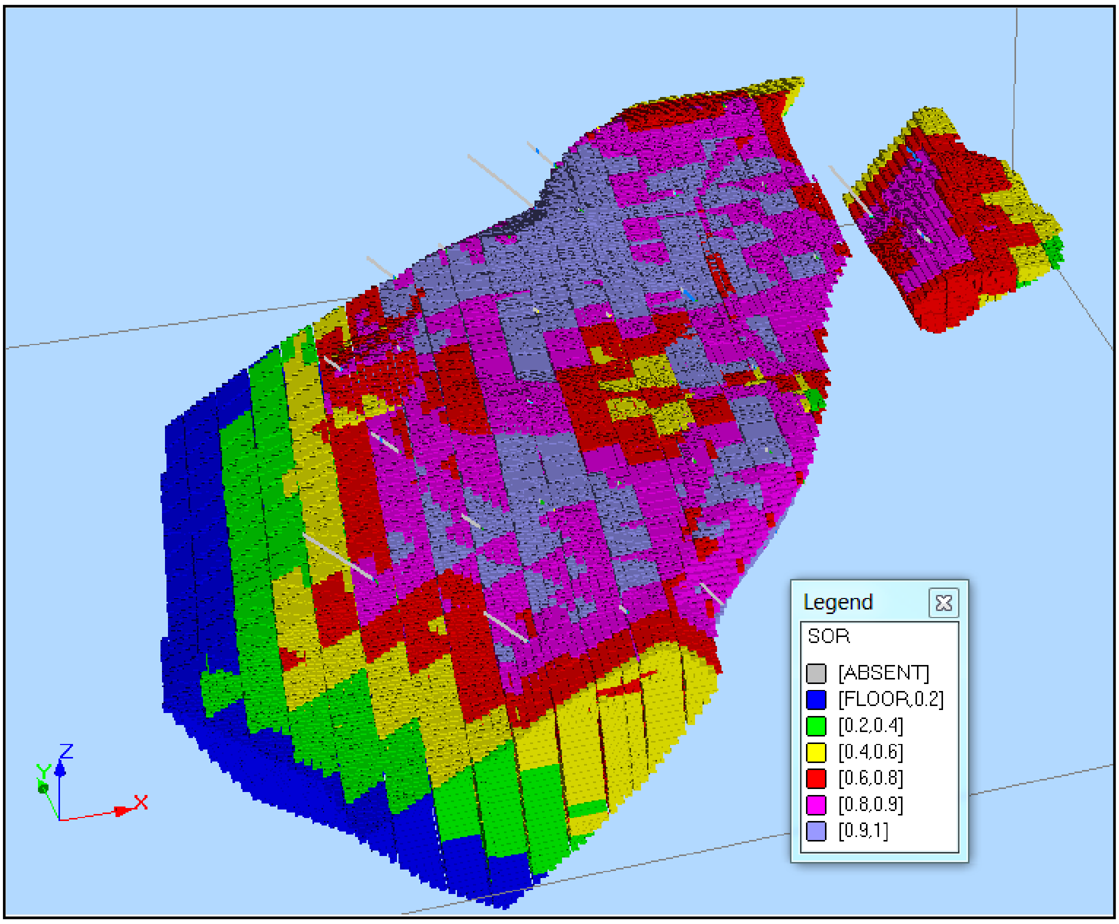

Figure 14-49: Sundsberget block model coloured by slope of regression (looking north north east)

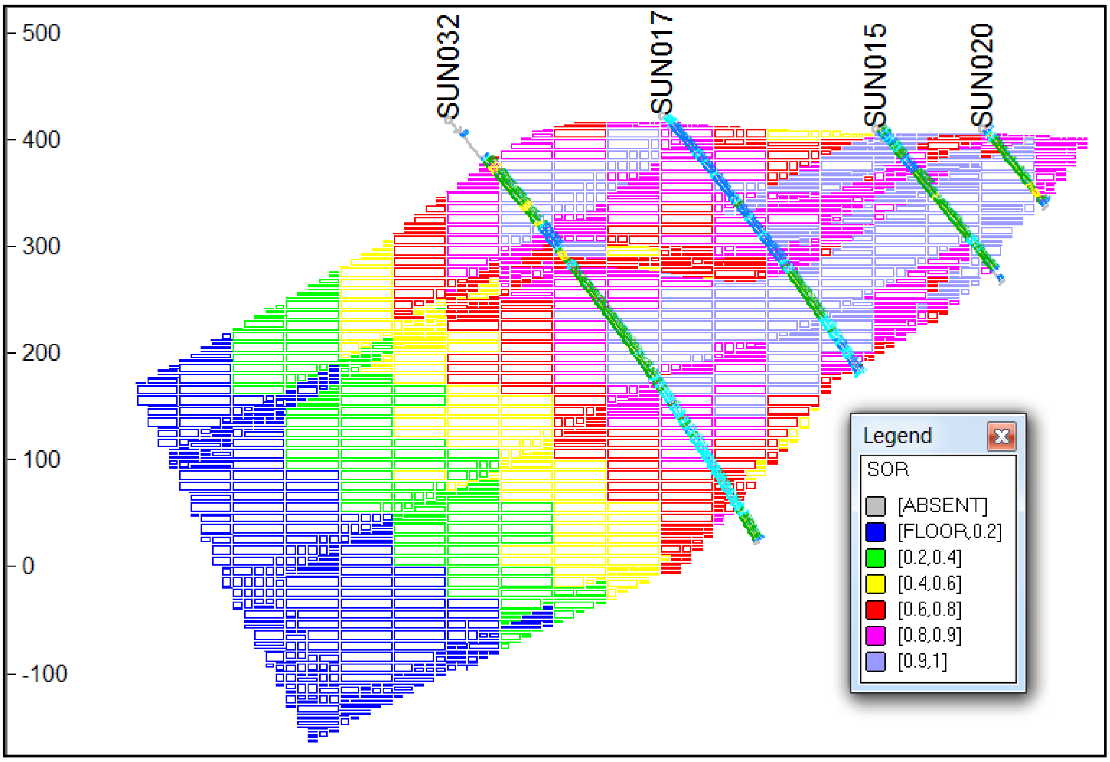

Figure 14-50: Sundsberget block model coloured by slope of regression (looking north northwest)

14.11 Block Modelling

14.11.1 Interpolation

An empty block model was generated using the lithology wireframes with block dimensions as shown in Table 14-13. These block dimensions approximate half the drillhole spacing at Rönnbäcksnäset northeast and at Vinberget. Due to the low nugget effect observed at the Rönnbäcksnäset and Sundsberget deposits, it is deemed appropriate to use blocks smaller than half the drillhole spacing as it is assumed that blocks that are not supported by drillhole intersections are supported by data within the short scale range observed in the variograms. The results of the QKNA study also highlight that the blocks in the Rönnbäcksnäset southwest deposit are well supported by data. A block height of 10 m was chosen, being the assumed working bench height of the operating pit. Table 14-13 summarizes the block model parameters.

Table 14-13: Block Model Framework

DEPOSIT	DIRECTION	ORIGIN	BLOCK SIZE	NR. OF BLOCKS
Rönnbäcksnäset	X	1479250	50	55
	Y	7267000	50	60
	Z	0	10	75
Vinberget	X	1483600	25	50
	Y	7262200	25	50
	Z	250	10	100
Sundsberget	X	8550	50	45
	Y	8180	50	55
	Z	-300	10	110

Grades of Ni-AC, Co-AC and Ni-Total were interpolated into the model using OK and the kriging parameters given in Table 14-7 and Table 14-8. Ni-Total was interpolated using the Ni-AC kriging parameters and represents the nickel present in both silicate and sulphide phases.

All domains were interpolated using OK, with the mafic units and internal waste domains at Rönnbäcksnäset utilising the variography data determined for the main serpentinite domain (151).

14.11.2 Search Ellipse Parameters

The dip and strike of the Rönnbäcksnäset deposit varies, with the strike changing from a near east-west orientation in the southwest to a near north-south orientation in the northeast. Due to the varying nature of the strike, it was necessary to either divide the deposit into two separate domains based on strike, or use the dynamic anisotropy function in Datamine Studio 3 to move the search ellipse with the changing strike direction. Dividing the deposit into unique domains based on strike, would have resulted in an artificial "hard" boundary between the northeast and southwest portions of the deposit beingan inaccurate representation of Rönnbäcksnäset geology, given that the nature of Ni-AC mineralisation appears to be consistent across these strike change boundaries. As such, dynamic anisotropy was selected to provide a continuous estimation and honour the observed geological structure in the estimation process.

Dynamic anisotropy uses angle data generated from the orebody wireframe to assign dip and dip direction to every block in the model. The search ellipse is rotated upon estimation of the block by honouring the associated dip and dip direction of that block. Figure 14-51 shows the search ellipse generated at various points of the Rönnbäcksnäset deposit, with the dip and strike of the ellipse corresponding with the dip and strike of the orebody wireframe.

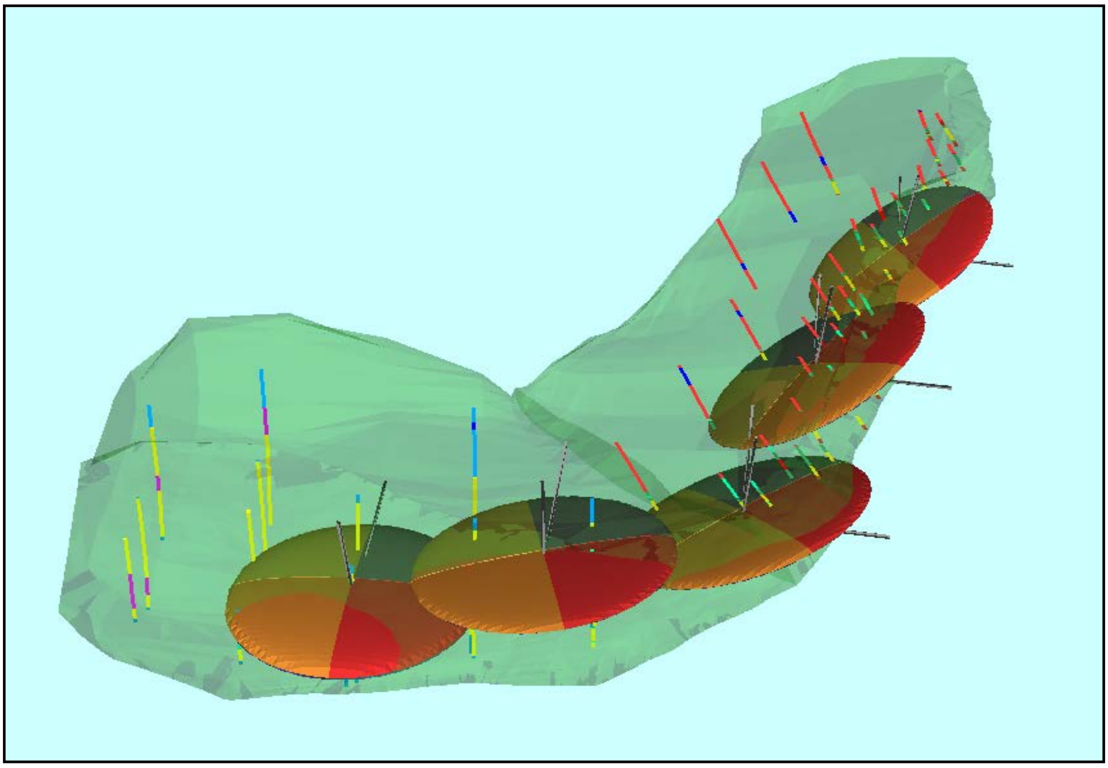

Figure 14-51: First pass search ellipses used in the interpolation of Rönnbäcksnäset (looking north, search ellipse radii measures 225 m along strike)

The Vinberget deposit used an isotropic search ellipse of 100 by 100 by 100 m. This is shown in Figure 14-52.

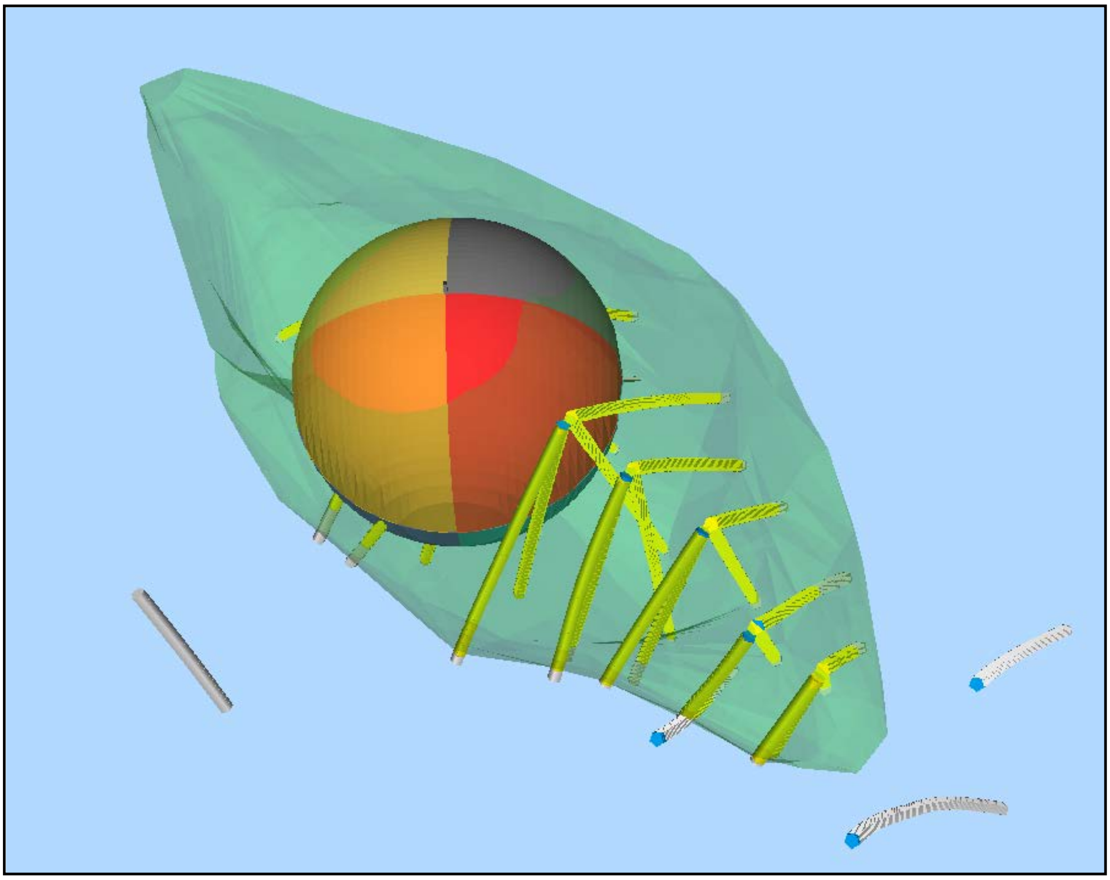

Figure 14-52: First pass search ellipses used in the interpolation of Vinberget (looking north, search ellipse radii measures 100 m)

The Sundsberget deposit also used dynamic anisotropy during the interpolation. An anisotropic search ellipse of 250 by 280 by 70 m for the high grade zone 151 (Figure 14-53), and an anisotropic search ellipse of 250 by 280 by 35 m for the lower grade zones 152, 153 and 205 (Figure 14-54) were utilised.

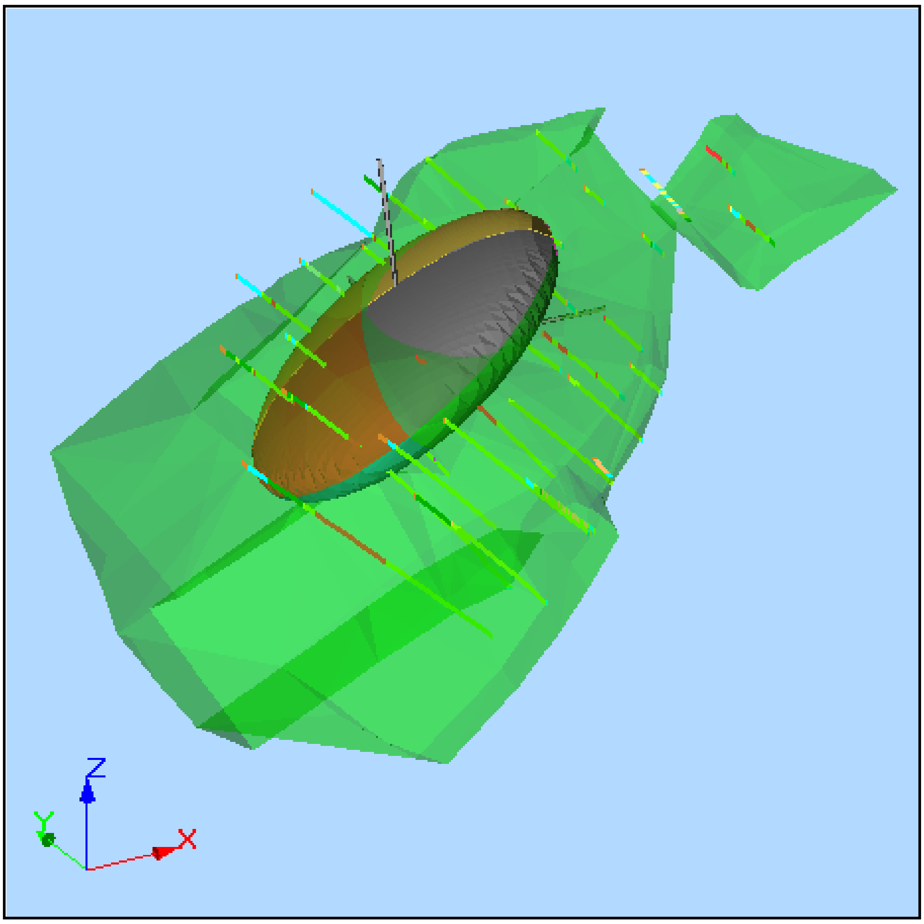

Figure 14-53: High grade mineralisation at Sundsberget, with high grade search ellipse. Drillholes coloured by lithological codes. Looking NNE.

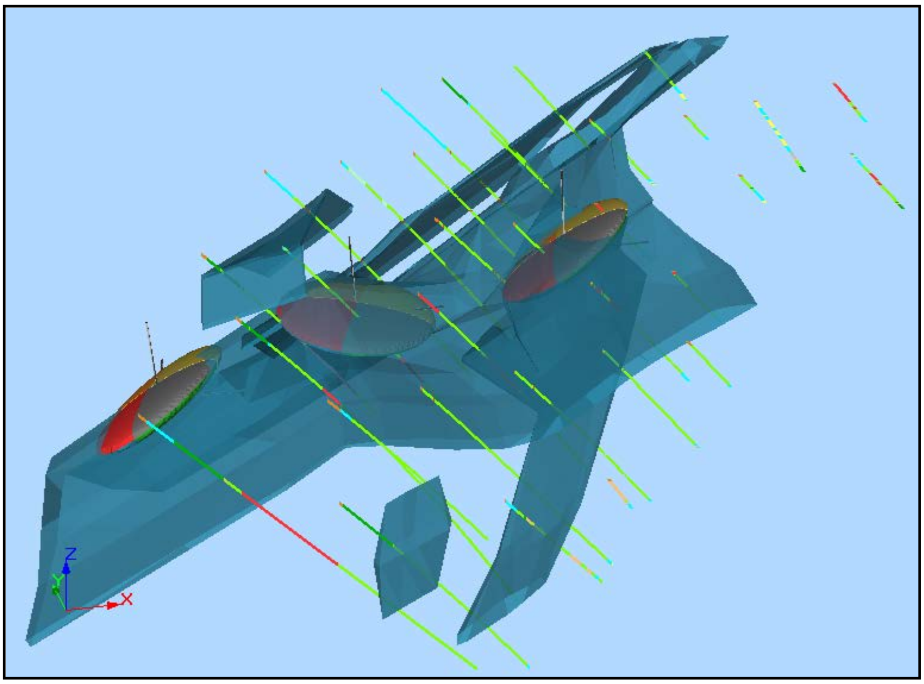

Figure 14-54: Low grade mineralisation at Sundsberget, with low grade search ellipses.

Three different grade interpolations with specific sample criteria were run. The first run used the "optimum" parameters determined by the QKNA testwork. The second run doubled the dimensions of the search ellipse and the third run multiplied the original search ellipse by a factor of ten and required a minimum number of samples too be used. The third run was designed to estimate any blocks not estimated in runs one and two and the confidence in the resulting grades is less as the search ellipse will have encapsulated samples that are outside the geostatistical range of the samples as shown in the geostatistical analysis.

Table 14-14 shows the search ellipse parameters used for the three estimation runs.

Table 14-14: Search ellipse parameters

ZONE	STRIKE (°)	DIP (°)	RUN	ALONG STRIKE RADII	DOWN DIP RADII	ACROSS STRIKE RADII	MIN SAMPS	MAX SAMPS
Rönnbäcks-näset	Defined by dynamic anisotropy		1	225	217	20	6	12
			2	450	450	40	6	12
			3	2250	2250	200	3	12
Vinberget	Isotropic		1	100	100	100	12	24
			2	200	200	200	12	24
			3	1000	1000	1000	12	24
Sundsberget (Zone 151)	Defined by dynamic anisotropy		1	250	280	70	6	36
			2	500	560	140	6	36
			3	2500	2800	700	3	36
Sundsberget (Zone 152, 153, 205)	Defined by dynamic anisotropy		1	250	280	35	6	36
			2	500	560	70	6	36
			3	2500	2800	350	3	36

14.11.3 Block Model Validation

The block model has been validated using the following techniques:

- visual inspection of block grades in plan and section and comparison with drillhole grades;
- comparison of global mean block grades and sample grades.

Visual Validation

Figure 14-55 to Figure 14-57 show examples of the visual validation checks between block Ni-AC grades and the input composite Ni-AC grade for the Rönnbäcksnäset deposit. The grades follow the strike and dip of the orebody showing that the search ellipse orientation has been used appropriately with Figure 14-57 showing that the grade has been interpolated through the change of strike of the serpentinite body.

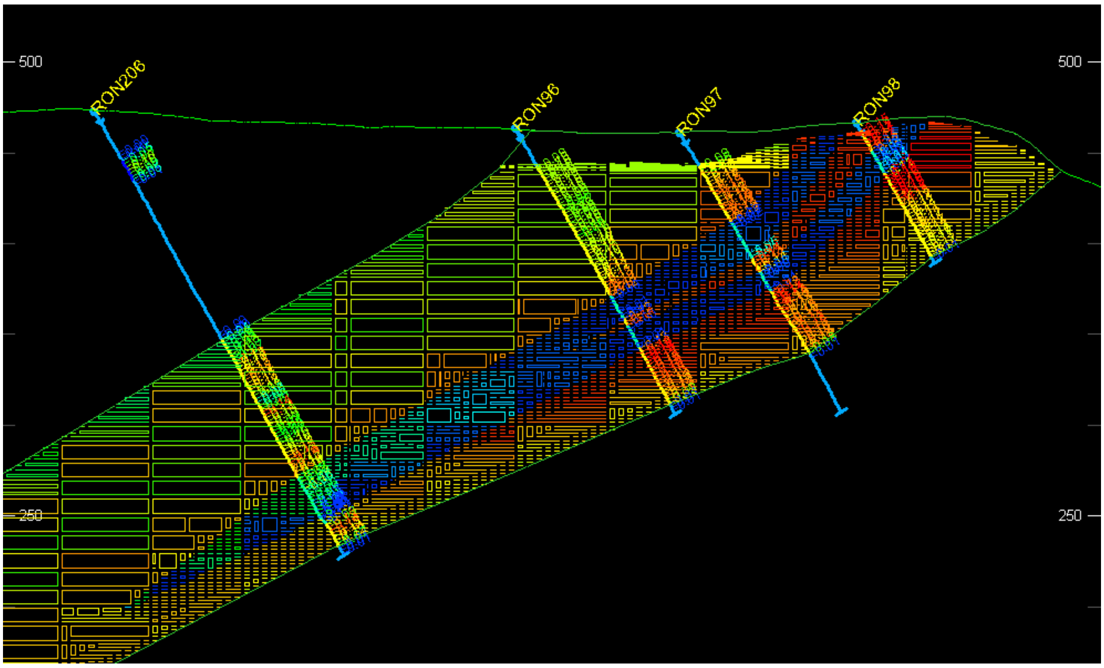

Figure 14-55: Visual validation of block grades against 2 m composite sample grades for Rönnbäcksnäset northeast (looking north)

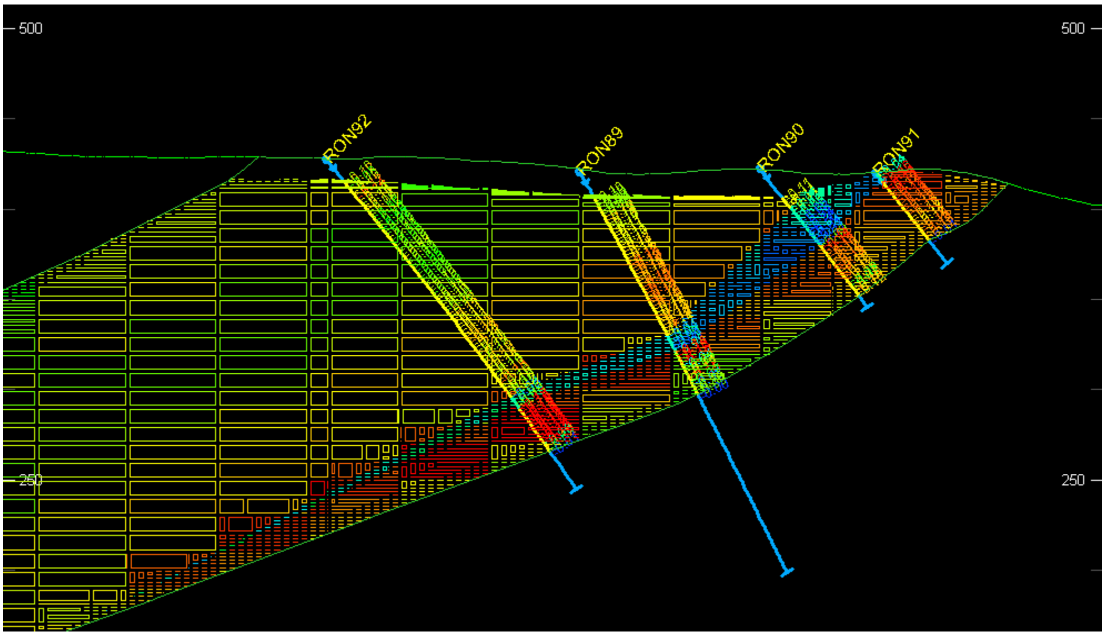

Figure 14-56: Visual validation of block grades against 2 m composite sample grades for Rönnbäcksnäset southwest (looking east)

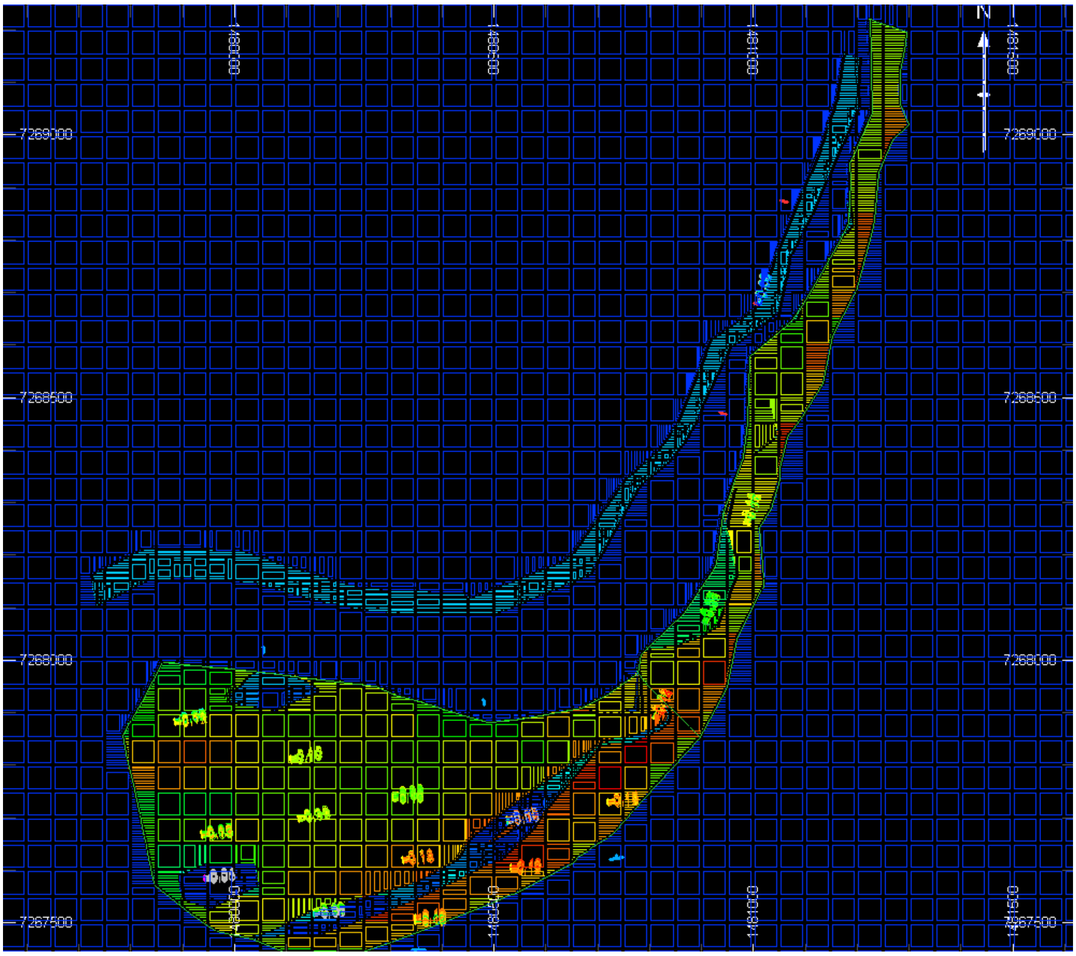

Figure 14-57: Visual validation of block grades against 2 m composite sample grades for Rönnbäcksnäset (plan view, RL set at 360 m)

Figure 14-58 to Figure 14-60 show examples of the visual validation checks between block Ni-AC grades and the input composite Ni-AC grade for the Vinberget deposit. Due to the homogenous nature of the Ni-AC distribution, grade trends are difficult to pick. However, from the cross section, long section and plan view, it is clear that the block model grades correspond well with the composite sample grades.

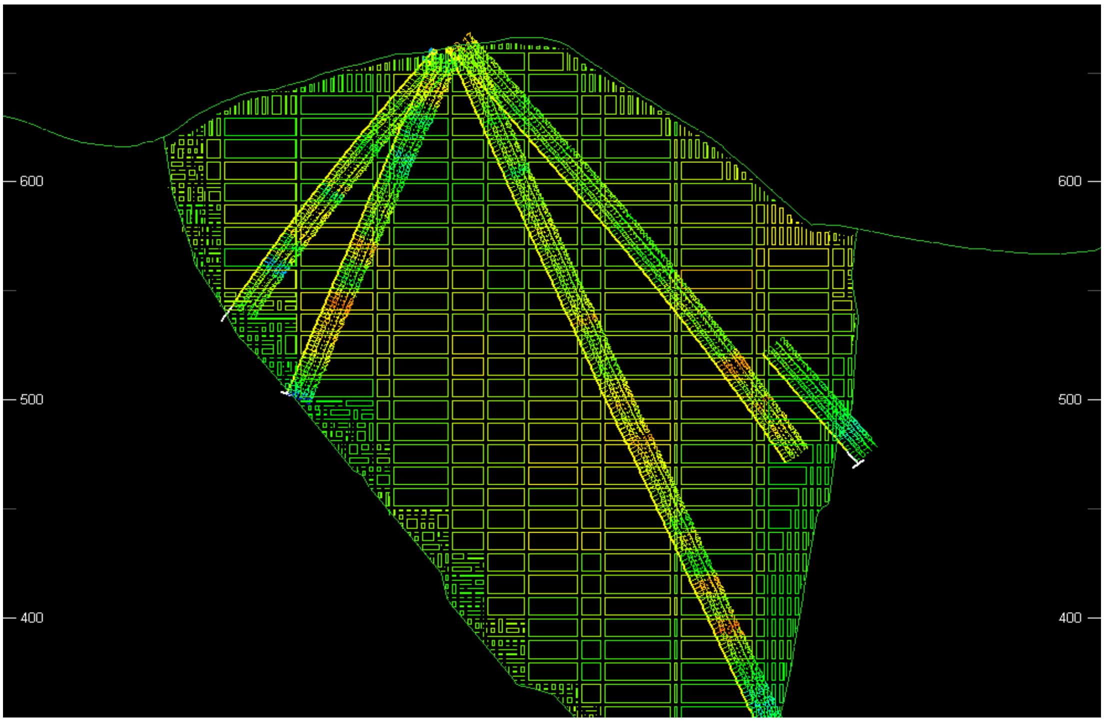

Figure 14-58: Visual validation of block grades against 2 m composite sample grades for Vinberget (looking northwest)

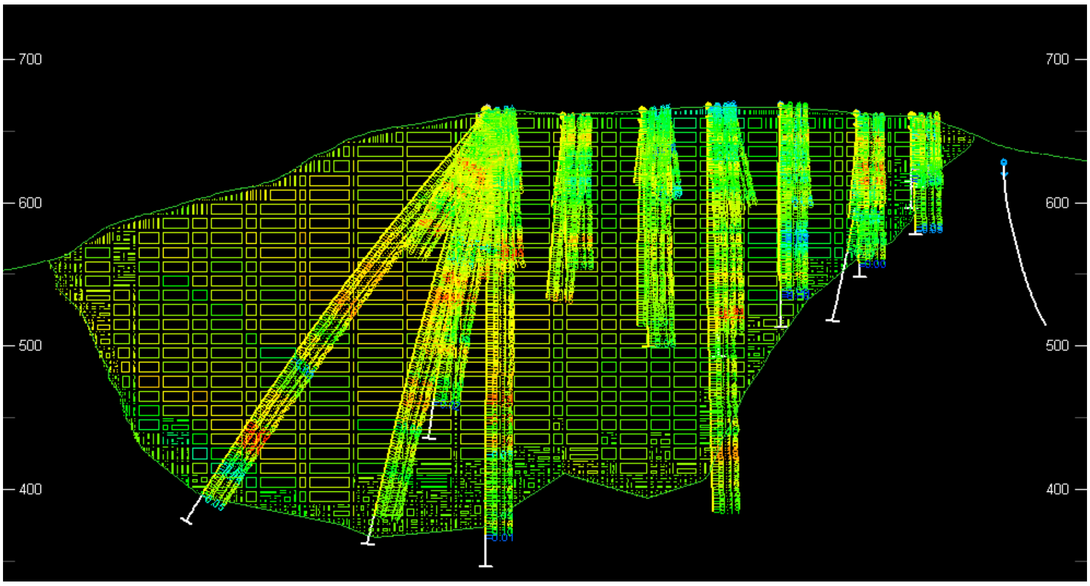

Figure 14-59: Visual validation of block grades against 2 m composite sample grades for Vinberget (looking northeast)

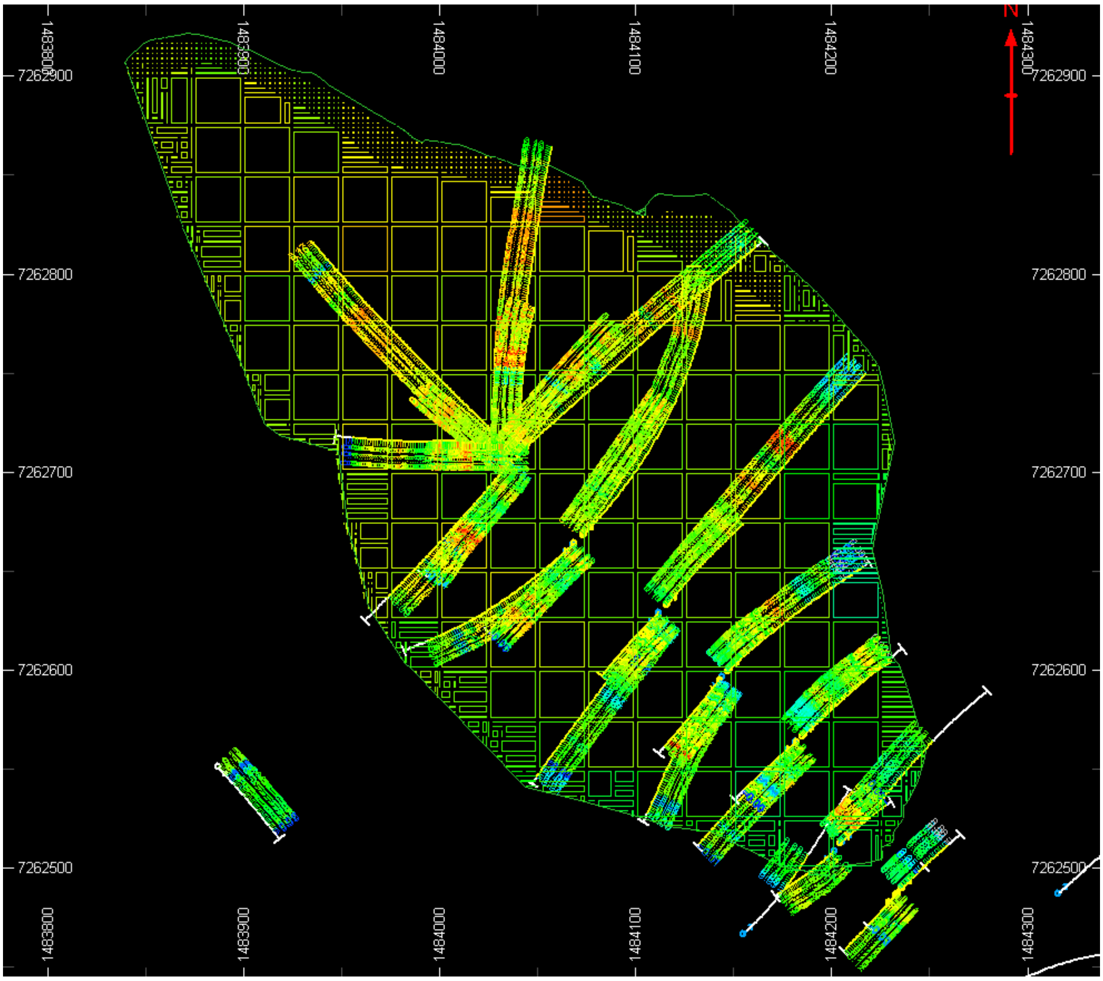

Figure 14-60: Visual validation of block grades against 2 m composite sample grades for Vinberget (plan view, RL set at 570m)

Figure 14-61 and Figure 14-62 show examples of the visual validation checks between block Ni-AC grades and the input composite Ni-AC grade for the Sundsberget deposit. It is clear that the block model grades correspond well with the composite sample grades.

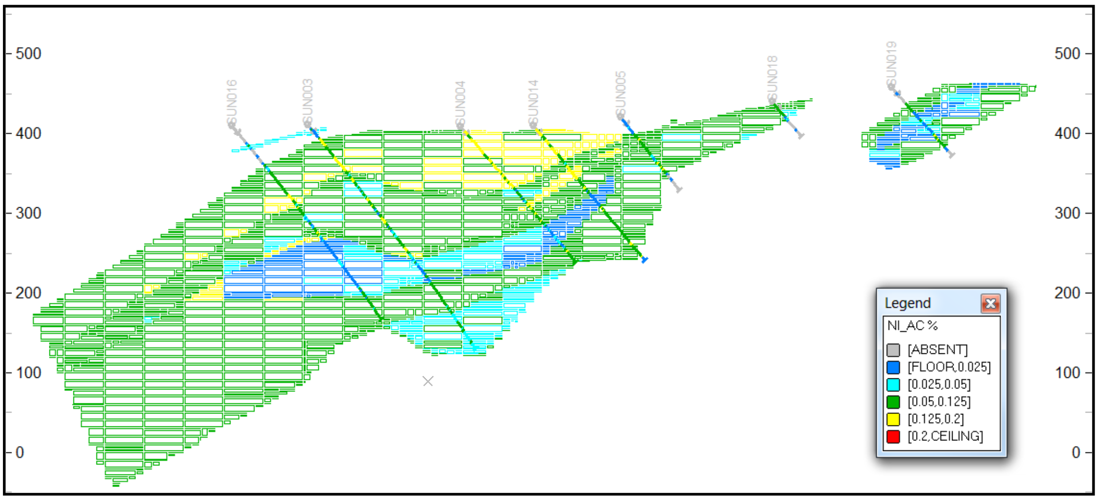

Figure 14-61: Visual validation of block grades against 2 m composite sample grades for centralSundsberget (looking north-northeast)

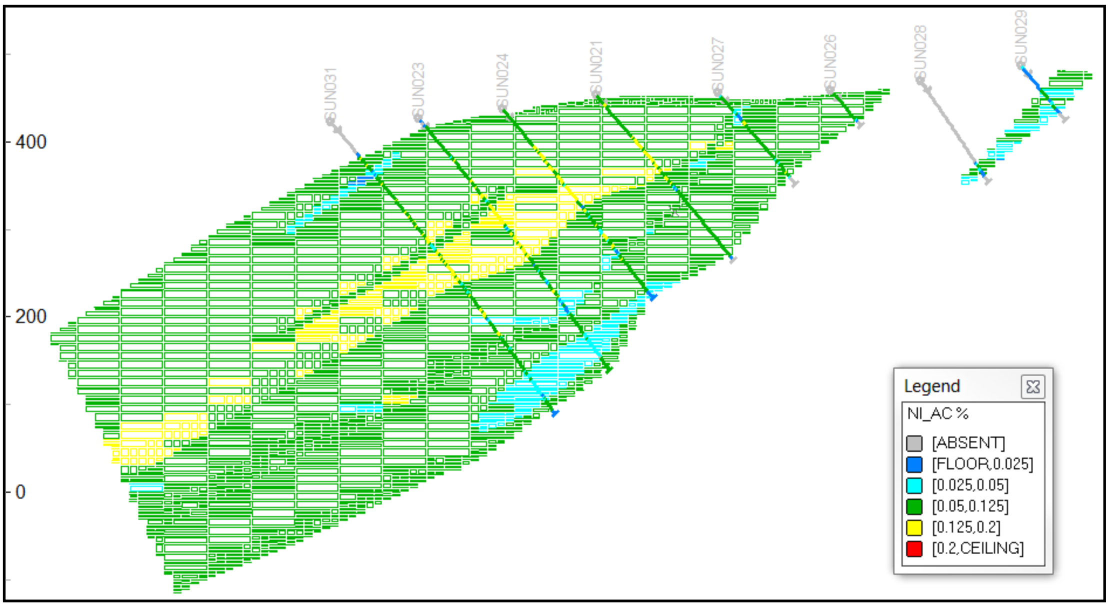

Figure 14-62: Visual validation of block grades against 2 m composite sample grades for southern Sundsberget (looking north-northeast)

Table 14-15 shows a comparison of the global block mean grades with the global sample means grades for Ni-AC, Co-AC and density.

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 14-15: Comparison of block and sample mean grades

DEPOSIT	DOMAIN	FIELD	BLOCK MEAN	COMP MEAN	DIFFERANCE
Rönnbäcksnäset	102	NI_AC	0.016	0.015	0.001
	112	NI_AC	0.018	0.016	0.002
	151	NI_AC	0.106	0.111	-0.005
	152	NI_AC	0.055	0.054	0.001
	205	NI_AC	0.035	0.029	0.006
	102	NI_TOT	0.145	0.148	-0.003
	112	NI_TOT	0.167	0.167	0.000
	151	NI_TOT	0.180	0.183	-0.003
	152	NI_TOT	0.165	0.167	-0.002
	205	NI_TOT	0.072	0.067	0.005
	102	CO_AC	0.001	0.001	0.000
	112	CO_AC	0.001	0.001	0.000
	151	CO_AC	0.003	0.003	0.000
	152	CO_AC	0.003	0.003	0.000
	205	CO_AC	0.002	0.002	0.000
	102	FE	5.42	5.47	-0.05
	112	FE	5.47	5.42	0.05
	151	FE	5.47	5.42	0.05
	152	FE	5.29	5.28	0.01
	205	FE	5.90	5.55	0.35
	102	DENSITY	2.79	2.79	0.00
	112	DENSITY	2.70	2.70	0.00
	151	DENSITY	2.74	2.73	0.01
	152	DENSITY	2.72	2.73	0.00
	205	DENSITY	2.95	2.96	0.00
Vinberget	151	NI_AC	0.132	0.131	0.001
	151	NI_TOT	0.186	0.187	-0.001
	151	CO_AC	0.006	0.006	0.000
	151	FE	5.17	5.19	-0.02
	151	DENSITY	2.71	2.71	0.00
Sundsberget	151	NI_AC	0.09	0.10	0.003
	152	NI_AC	0.04	0.04	0.000
	153	NI_AC	0.07	0.08	0.004
	205	NI_AC	0.02	0.02	-0.001
	151	NI_TOT	0.21	0.21	0.002
	152	NI_TOT	0.12	0.12	0.003
	153	NI_TOT	0.12	0.12	0.003
	205	NI_TOT	0.06	0.06	-0.002
	151	CO_AC	0.002	0.002	-0.00021
	152	CO_AC	0.001	0.001	-0.00003
	153	CO_AC	0.004	0.004	0.00032
	205	CO_AC	0.001	0.001	-0.00003
	151	FE	5.82	5.68	-0.138
	152	FE	5.76	5.77	0.003

DEPOSIT	DOMAIN	FIELD	BLOCK MEAN	COMP MEAN	DIFFERANCE
	153	FE	6.52	6.59	0.060
	205	FE	5.51	5.43	-0.073
	151	DENSITY	2.82	2.81	-0.010
	152	DENSITY	2.91	2.92	0.015
	153	DENSITY	2.94	2.93	-0.016
	205	DENSITY	3.07	3.07	0.002

14.12 Mineral Resource Classification

The definitions given in the following section are taken from the 2000 Canadian Institute of Mining Standing Committee on Reserve Definitions' guidelines on Mineral Resources and Reserves, and comply with the requirements of National Instrument 43-101.

14.12.1 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 Resourc**e** 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.

14.12.2 Classification

Introduction

To classify the Rönnbäcksnäset, Vinberget and Sundsberget deposits, the following key indicators were used:

- geological complexity;
- quality of data used in the estimation:
- QAQC, density analysis

- results of the geostatistical analysis
- variography, and
- QKNA results; and
- quality of the estimated block model.

Geological Complexity

Due to the large amount of drill data, it is possible to see clear geological continuity between sections and deduce a clear geological model for the Rönnbäcksnäset, Vinberget and Sundsberget deposits with all of the mineralisation occurring within the serpentinite body. The drill spacing has allowed for the interpretation of a continuous zone of mafic material with a low associated Ni-AC grade. Internal waste pods have been interpreted that are harder to join from adjacent sections, but these are limited in number and form a small part of the overall serpentinite body.

A statistical study of the Rönnbäcksnäset, Vinberget and Sundsberget data shows a very low variability in the grade distribution with near normal populations of data being present. A continuous low grade serpentinite unit has been identified from the statistical study that was subsequently domained as a separate unit.

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

Quality of the Data used in the Estimation

Quality assurance and quality control (QAQC) checks were implemented throughout the assaying period that included the insertion of standards, blanks, laboratory duplicates and the use of an umpire laboratory. 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 the Company throughout the sampling period that has enabled SRK to estimate density into the model using OK. SRK is therefore confident that the associated tonnages estimated for the Rönnbäcksnäset, Vinberget and Sundsberget deposits should be reasonable.

Results of the Geostatistical Analysis

The data used in the geostatistical analysis resulted in robust variogram models being produced for all three deposits. This enabled the nugget and short-scale variation in grade to be determined with a high level of confidence. 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 Block Model

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

Classification Approach

All three deposits have been classified as containing Indicated and Inferred Resources. Only Rönnbäcksnäset has been classified as containing Measured Resources due to the closer spaced drilling of 100 m section lines.

Measured Resources at Rönnbäcksnäset have been assigned where the folliwing criteria have been met:-

- very low geological complexity;
- drillhole spacing of much less than the 2/3rd geostatistical range;
- all blocks were estimated in search volume one, using the optimum search parameters determined; and
- slope of regression values dominantly greater than 0.8.

At Sundsberget and Vinberget the same criteria as explained above for Measured Resources have been used to assign Indicated Resources with the exception that the minimum slope of regression was set at 0.5. Indicated Resources at Rönnbäcksnäset have been extended approximately 100 m down dip of the last drillhole intersection on the section line.

Inferred Resources at Rönnbäcksnäset and Viberget have been calculated by extending the Indicated boundary 50 m down-dip and by including areas where internal waste pods are defined and unsupported by more than two drillholes on a section line. Due to the regular drilling pattern and the simple geometry at Sundsberget, the Indicated blocks account for all the well-informed blocks, therefore no Inferred Resources were assigned.

In all cases, the above have been used to model zones for the each of the classification categories for each deposit rather than to assign this on a block by block basis.

Figure 14-63, Figure 14-64 and Figure 14-65 show the block models coloured by classification for Rönnbäcksnäset, Vinberget and Sundsberget, respectively.

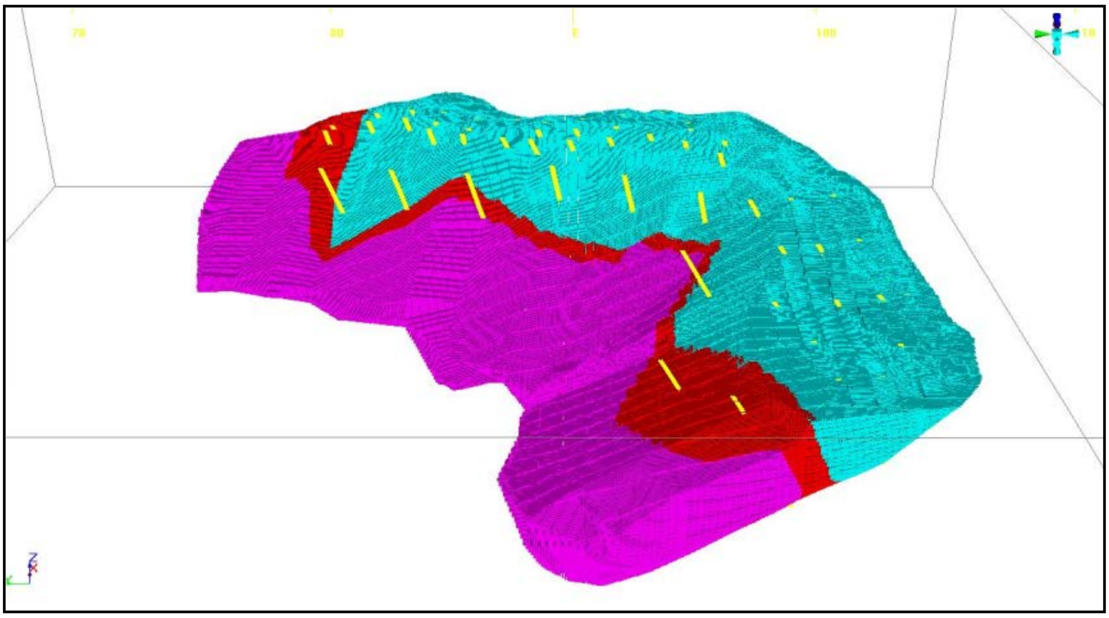

Figure 14-63: Rönnbäcksnäset classification. Blue = Indicated; Red = Inferred; Purple = Unclassified (looking east)

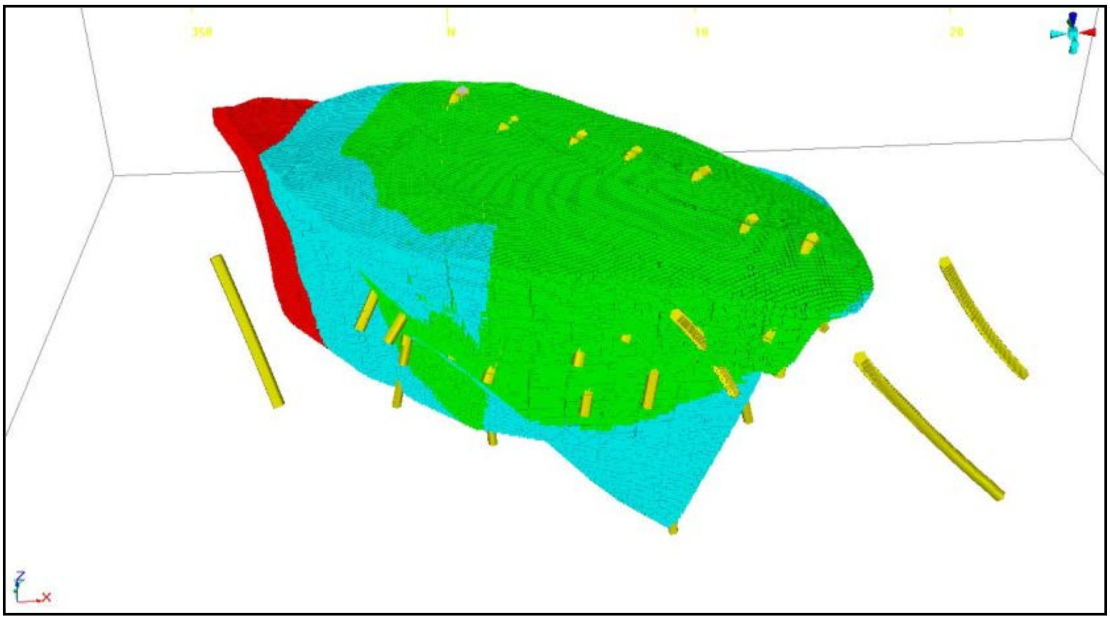

Figure 14-64: Vinberget classification. Green = Measured; Blue = Indicated; Red = Inferred (looking north)

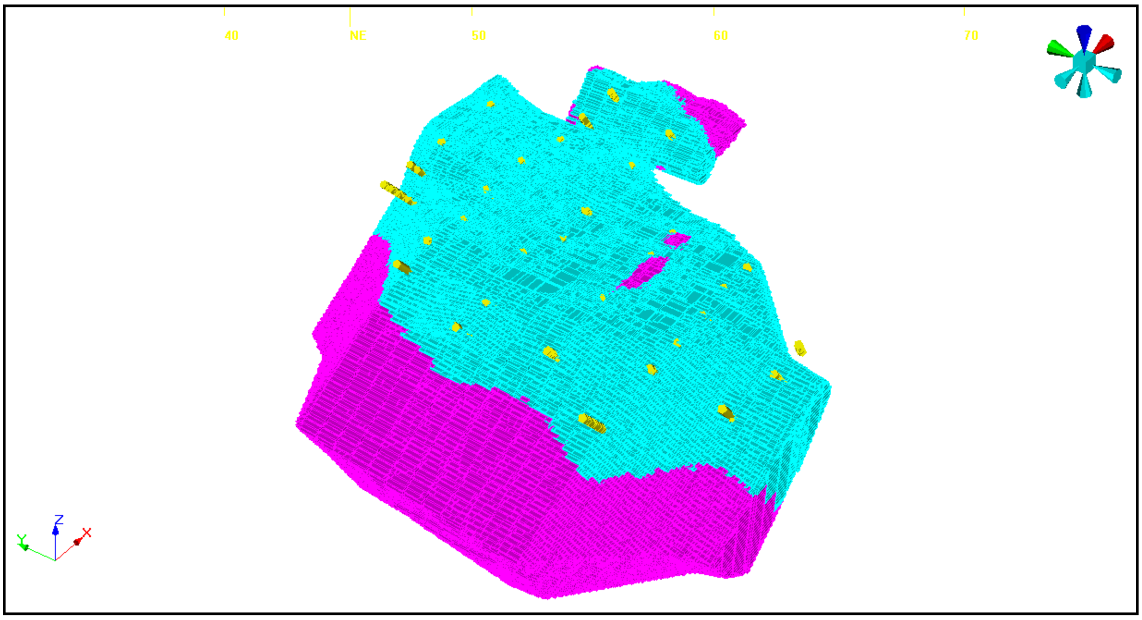

Figure 14-65: Sundsberget classification. Blue = Indicated; Purple = Unclassified (looking north)

14.13 Pit Optimisation

In order to derive the final Mineral Resource Statement, and so as to comply with the requirement that the resulting Mineral Resource must have reasonable prospects of economic extraction, the resulting blocks have been subjected to a Whittle pit optimisation exercise.

The Whittle optimisation requires the input of reasonable processing and mining cost parameters in addition to appropriate pit slope angles and processing recoveries.

Table 14-16 shows the assumptions applied in the Whittle optimisation.

The Whittle optimisation has assumed that the serpentinite domains (151, 152 and 153) are to be treated as the key potential ore material types.

Table 14-16: Whittle parameters

Revenue	
Ni Price	USD 11/lb
Govt Royalty	0.05%
Private Royalty	0.15%
Discount Rate	0%
Process and Mining Statistics	
Overall Slope Angle	48° (Ronn), 50° (Vin), 49° (Sun)
Mining Recovery	95%
Mining Dilution	2.5%
Process Recovery	80%
OP Base Mining Cost	1.35 USD/tonne
Incremental Mining Cost above surface	0.05 USD/tonne/10m
Incremental Mining Cost below surface	0.07 USD/tonne/10m
Processing Cost	4.96 USD / ore tonne
Effective charges per lb Ni in smelter feed	1.14 USD / lb
General & Administration	0.4 USD / ore tonne processed
Rail / Road Transport Cost	0.1 USD / ore tonne processed
Concentrate Grade	25.0%

14.14 Mineral Resource Statement

The Mineral Resource Statement generated by SRK has been restricted to all classified material falling within the Whittle shell representing a metal price of USD11/lb and through the application of the parameters outlined in Section 14.13. SRK assumed a nickel price of USD11.00/lb in a Whittle open pit optimisation exercise to limit the material reported to that which SRK considers has reasonable prospects for eventual economic extraction and applied a cut off grade of 0.031% Ni-AC representing the calculated marginal cut off grade for the deposits. The USD11.00/lb nickel price includes a 30% premium above the consensus long-term nickel price, determined from over 30 market forecasts.

Table 14-17 shows the resulting Mineral Resource Statement for Ni, Co and Fe Total, for Rönnbäcksnäset, Vinberget and Sundsberget. Due to the large relative proportion of non-sulphide nickel in the Ronnbacken ores, resources are reported in terms of sulphide (AC) nickel along with the more conventional total nickel. A high proportion of the sulphide nickel is recovered in the flotation process whereas the non-sulphide nickel reports predominantly to tailings.

The effective date for these statements is 21 October 2011.

Mineral resources that are not mineral reserves do not have demonstrated economic viability. Notwithstanding this, neither SRK nor the Company are aware of any factors (environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors) that could materially affect the potential of these to be exploited.

Table 14-17: Mineral Resource Statement (reported above a Marginal cut off grade of 0.031% Ni-AC)

DEPOSIT	CLASSIFICATION	TONNES (Mt)	Ni-Total %	Sulphide Ni (Ni-AC)	Sulphide Co (Co-AC)	Fe-Total %	Ni-Total ktonnes	Sulphide Ni ktonnes
Rönnbäcksnäset	Measured							
	Indicated	225.4	0.176	0.101	0.003	5.41	397	227
	Measured + Indicated	225.4	0.176	0.101	0.003	5.41	397	227
	Inferred	86.5	0.177	0.100	0.003	5.17	153	86
Vinberget	Measured	28.3	0.188	0.132	0.006	5.19	53	37
	Indicated	23.3	0.183	0.133	0.006	5.14	43	31
	Measured + Indicated	51.5	0.186	0.133	0.006	5.14	96	68
	Inferred	6.8	0.183	0.138	0.007	5.58	12	9
Sundsberget	Measured							
	Indicated	296.9	0.170	0.088	0.003	5.93	505	260
	Measured + Indicated	296.9	0.170	0.088	0.003	5.93	505	260
	Inferred							
TOTAL (Measured & Indicated)	**Measured**	**28.3**	**0.188**	**0.132**	**0.006**	**5.19**	**53**	**37**
	Indicated	**545.6**	**0.173**	**0.095**	**0.003**	**5.68**	**945**	**519**
	Measured + Indicated	**573.9**	**0.174**	**0.097**	**0.003**	**5.66**	**998**	**556**
TOTAL (Inferred)	**Inferred**	**93.2**	**0.177**	**0.103**	**0.003**	**5.55**	**166**	**96**

In total, the three deposits have a combined Measured and Indicated Mineral Resource of 573.9Mt with mean grades of 0.174 Ni-Total, 0.097% Ni-AC and 0.003% Co-AC. Of this, 28.3 Mt with mean grades of 0.188% Ni-Total, 0.132% Ni-AC and 0.006% Co-AC is in the Measured category and 545.6Mt with mean grades of 0.173% Ni-Total, 0. 095% Ni-AC and 0.003% Co-AC is in the Indicated category. In addition to the Measured and Indicated Mineral Resource, SRK has derived an Inferred Mineral Resource of some 93.2 Mt with mean grades of 0.103% Ni-AC and 0.003% Co-AC

SRK's Mineral Resource Statement includes material domained as mafic material (domain 205) at Rönnbäcksnäset and Sundsberget. This material, equating to14 Mt, or 2% of the total resource and with a mean grade of 0.052% Ni-AC in Rönnbäcksnäset and with a grade of 0.043% NI-AC in Sundsberget has been included due to the presence within this of potentially recoverable Ni above the calculated marginal cut off grade. The economic model presented later in this report does assumes that this material is not processed, but SRK considers the inclusion of this material in the Mineral Resource Statement to be justified in that it represents potential upside to the project.

Figure 14-66 shows the Rönnbäcksnäset Whittle pit shell generated using a metal price of USD11/lb and through the application of the parameters outlined in Section 14.13. Figure 14-67 shows the Vinberget Whittle pit shell generated using a metal price of USD11/lb and through the application of the parameters outlined in Section 14.13.

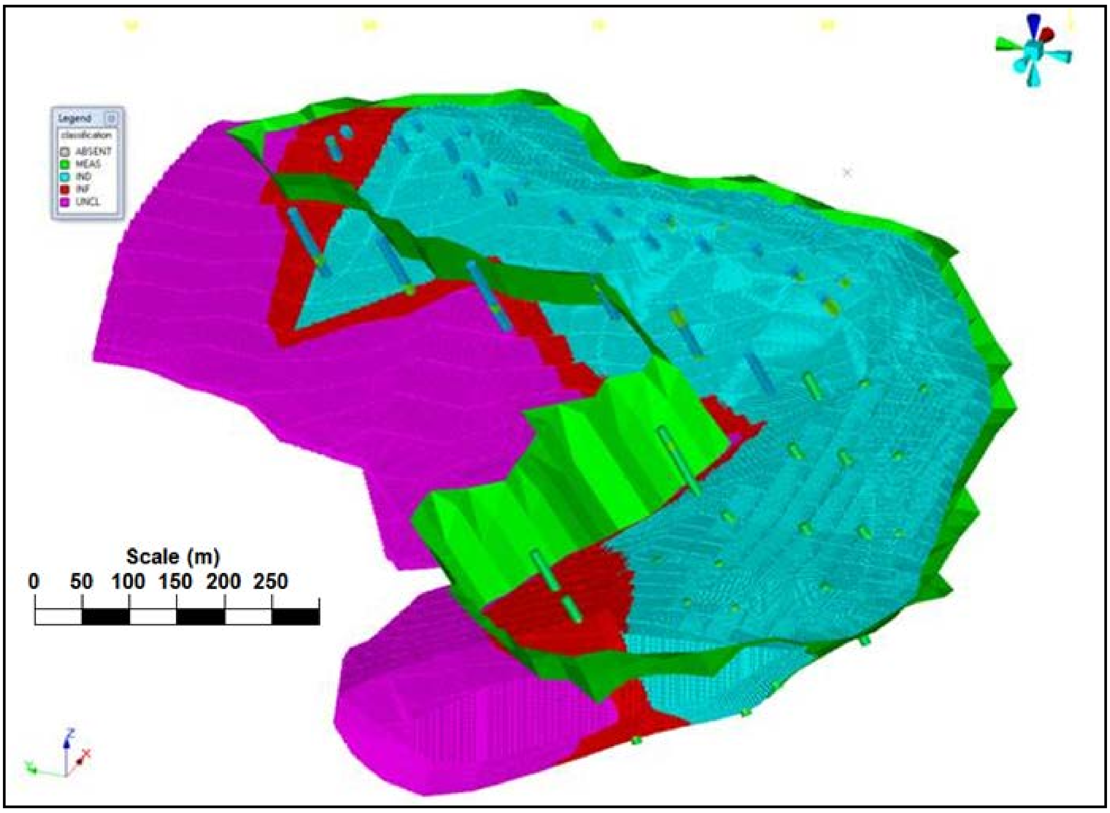

Figure 14-66: Rönnbäcksnäset pit shell with scale, looking east.

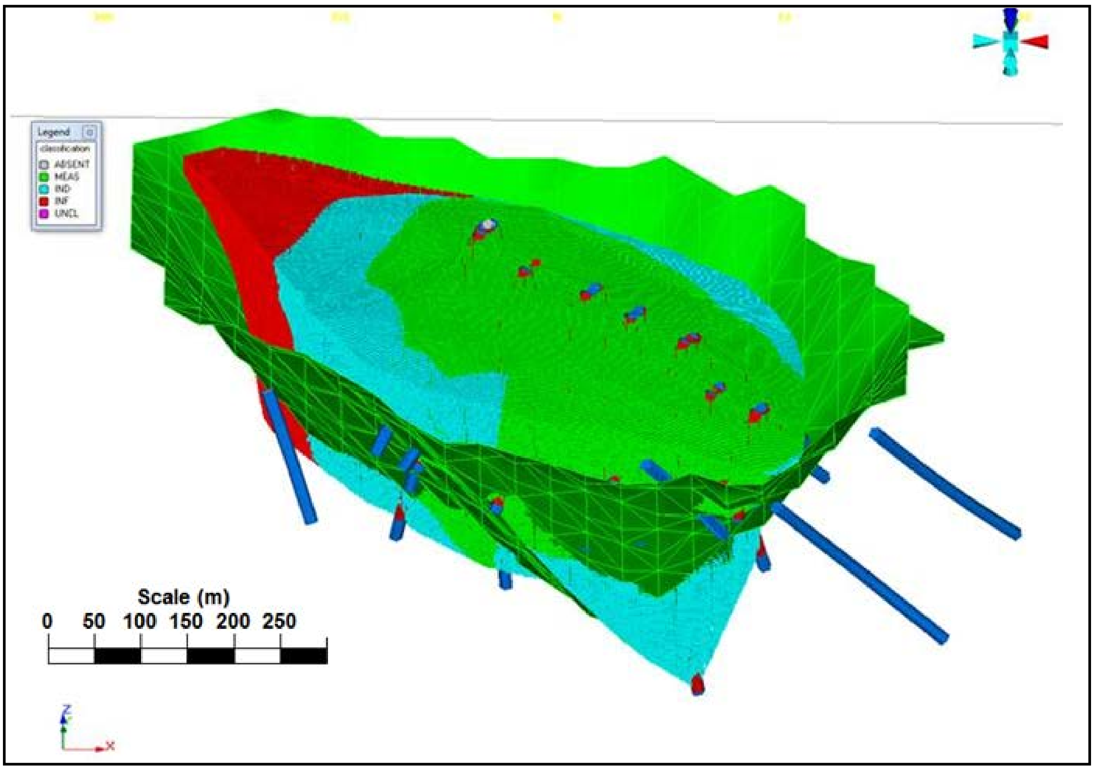

Figure 14-67: Vinberget pit shell with scale, looking north.

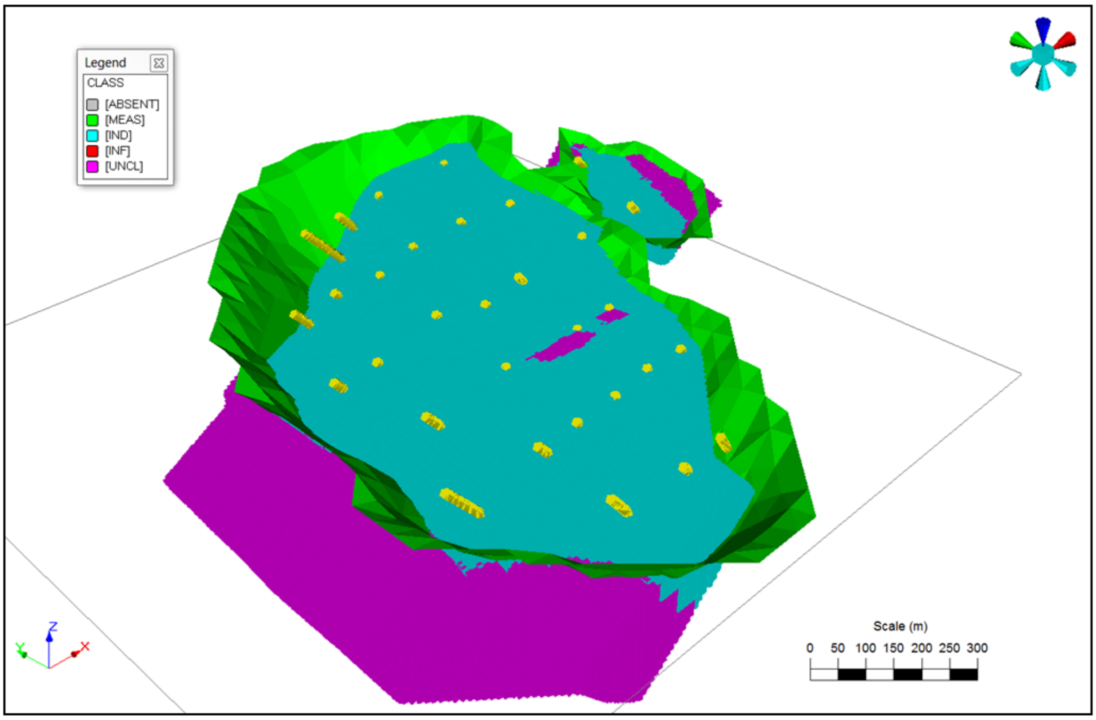

Figure 14-68: Sundsberget pit shell with scale, looking northeast.

14.15 Grade Tonnage Curves

Grade-tonnage curves for Rönnbäcksnäset, Vinberget and Sundsberget are shown in Figure 14-69, Figure 14-70 and Figure 14-71 for Ni-AC%. The curve shows the relationship between the modelled tonnage and grade at increasing Ni-AC% cut-offs.

The Rönnbäcksnäset grade-tonnage curve shows a gentle decreasing tonnage with an associated gentle increasing Ni-AC% grade up to a Ni-AC cut off of approximately 0.09%. This low grade material relates to the low grade serpentinite unit at Rönnbäcksnäset northeast. Above a cut off of 0.09% Ni-AC, the tonnage drops from approximately 225 Mt with an associated sharp increase in Ni-AC%.

The Vinberget grade-tonnage curve shows that all material is above the marginal cut off grade of 0.031% Ni-AC with a steadily dropping tonnage and increasing Ni-AC% from approximately 0.1% Ni-AC cut off.

The Sundsberget grade-tonnage curve shows a gentle decreasing tonnage with an associated gentle increasing Ni-AC% grade up to a Ni-AC cut off of approximately 0.06%. Above 0.06% the tonnage drops steadily associated with increasing Ni-AC% grade.

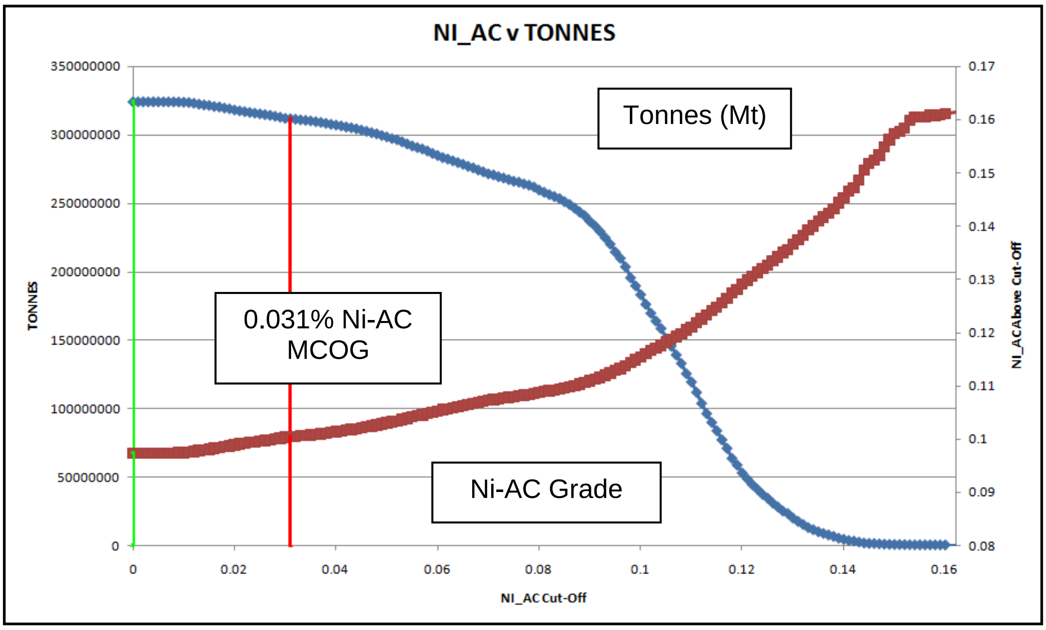

Figure 14-69: Rönnbäcksnäset Grade Tonnage Curve – All classification categories above Whittle pit shell (green line marks 0.031% marginal cut off grade)

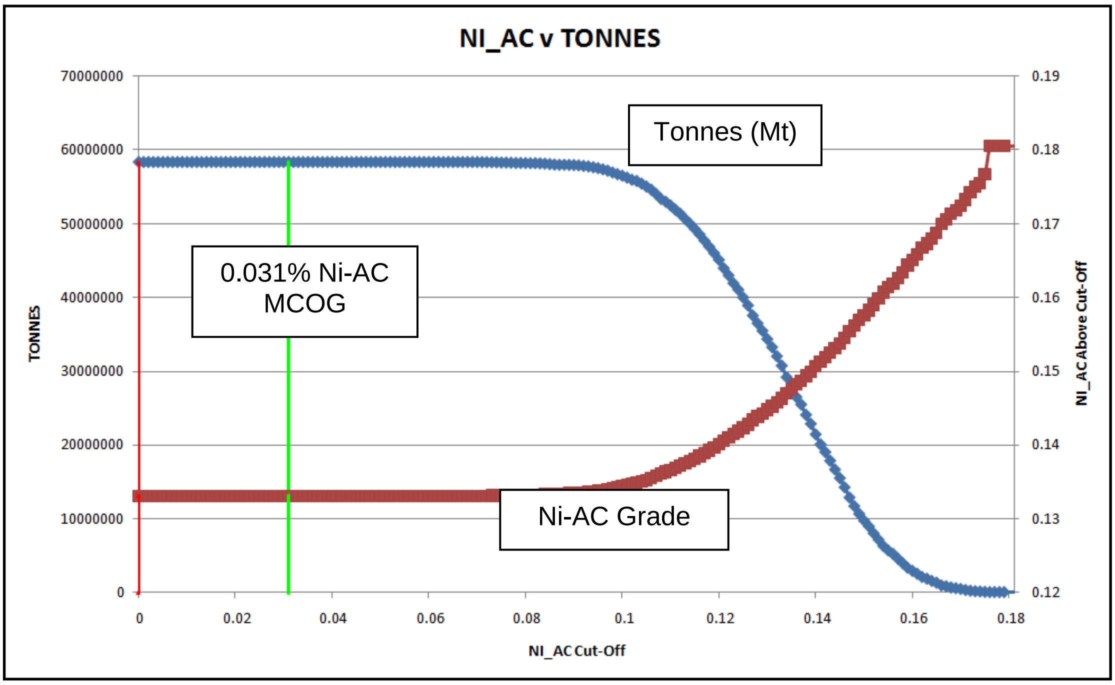

Figure 14-70: Vinberget Grade Tonnage Curve – All classification categories above Whittle pit shell (green line marks 0.031% marginal cut off grade)

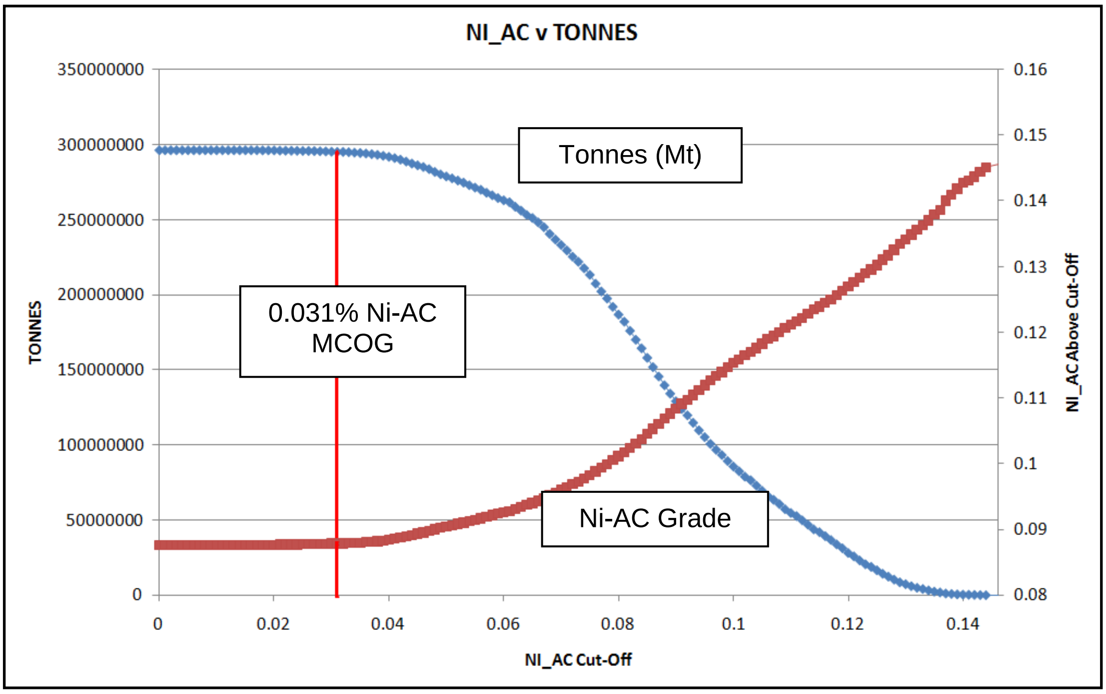

Figure 14-71: Sundsberget Grade Tonnage Curve – All classification categories above Whittle pit shell (red line marks 0.031% marginal cut off grade)

14.16 Exploration Potential

There are several million tonnes of additional mineralised material within the Whittle pit shell at Rönnbäcksnäset that has not been included in SRK's Mineral Resource Statement as it occurs in areas where more drilling is required to support sufficiently reliable estimates of tonnage and grade. This material, however, has a good chance of being potentially economic and adding to the mineral resource presented here following further drilling.

The Company is currently undertaking drilling at Rönnbäcksnäset to target down dip extensions to the mineralisation that was identified as potentially economic through an optimisation exercise which includes the above material. SRK considers this work justified by the potential and that further such optimisation exercises will justify more drilling programmes which may add to the Mineral Resource even further in due course.

14.17 Comparisons between SRK and MRG Sundsberget estimates

Table 14-18 shows the Mineral Resource Statement generated by MRG for the Sundsberget deposit in October 2010. This was restricted to all classified material falling within the Whittle shell representing a nickel price of USD9/lb and applied a cut-off grade of 0.05% Ni-AC representing the calculated marginal cut-off grade for the deposit. The Mineral Resource Statement generated by MRG has an effective date of 27 October 2010.

Table 14-18: MRG Mineral Resource Statement

DEPOSIT	CLASSIFICATION	TONNES (Mt)	Ni-Total %	Ni-AC %	Fe %	Ni-AC kt	MAGNETITE Fe ktonnes
Sundsberget	Measured	-	-	-		-	
	Indicated	-	-	-		-	
	Inferred	185.7	0.176	0.104	5.95	193	7 071

MRG reported the Mineral Resource entirely as Inferred, rather than Measured or Indicated due to the uncertainty regarding the geometry of the internal barren pyroxenite zones. SRK has reviewed these internal zones and believes that the interpretation undertaken by MRG has enabled a reasonable and continuous three dimensional solid to be created. The internal zones modelled by MRG represent approximately 12% of the total tonnage within the optimised Whittle pit shell and SRK considers that the clear geological continuity of the mineralised Serpentinite unit and the quality of the data supports an Indicated classification.

The total Mineral Resource tonnage has also increased from 185.7 Mt to 269.7 Mt. This has changed due to the SRK Whittle optimisation using an $11/lb Ni price compared to the $9/lb Ni price used by MRG. Very minor changes in mineralisation wireframes also contributed minimal tonnage differences.

15 MINERAL RESERVE ESTIMATES

Not applicable.

16 MINING METHODS

Introduction

The Company currently envisages that the Rönnbäcknäset, Vinberget and Sundsberget deposits will all be exploited using open pit mining techniques. It is intended to use conventional mining methods including drilling, blasting, loading with hydraulic shovels and hauling with off highway dump trucks.

During the course of this study, SRK has completed a number of mining engineering activities including:

- the development of Whittle input models for all three deposits;
- the optimisation of the three deposits using updated costs and processing parameters;
- the generation of bench tonnages and grades for scheduling purposes; and
- a review of the resulting mining schedules.

16.1 Mine Optimisation, Design and Scheduling

16.1.1 Optimisation Parameters

Block Models

The geological block models used for the Rönnbäcknäset and Vinberget optimisations are those developed by SRK in April 2010 while that used for the Sundsberget deposit is the one produced by MRG in October 2010 and validated by SRK in 2011.

These models have been modified to include generic waste and air blocks and the required optimisation fields have been added to each. This work was completed in Datamine before Whittle MOD and PAR files were exported, ready for the optimisation process.

The models are all on different frameworks and as there is a reasonable geographic distance between the deposits, it was therefore decided to treat each deposit as a separate entity for the purposes of the optimisation. Although it is unlikely to make a significant difference, combining these three models in a multi-mine scenario in due course would add a further level of completeness to the study.

The table below shows the general statistics for the three Whittle models used. There is a discrepancy in the cobalt grades for the Sundsberget model in that they are not reported in the same units as those for the other models. However, cobalt has not been considered in the optimisation process, therefore this discrepancy does not need to be considered. The Sundsberget grid is also different to that used for the other deposits, but again this is not an issue for this study.

Table 16-1: Whittle Model Statistics

	Rönnbäcknäset	Vinberget	Sundsberget
Model Origin			
X Origin	1,479,250	1,483,600	9,000
Y Origin	7,267,000	7,262,200	8,500
Z Origin	0	250	-200
Block Dimensions			
X	50	25	50
Y	50	25	100
Z	10	10	10
No of Blocks			
X	55	50	30
Y	60	50	20
Z	75	100	75
Nickel Content[1]	530,840	79,139	322,631
Max Ni-Ac Grade	0.17%	0.1805%	0.1392%
Min Ni-Ac Grade	0.013%	0.0709%	0.0625%
Cobalt Content[2]	16,609	3,774	9,470,481
Max Co Grade	0.054%	0.0085%	71.594
Min Co Grade	0.001%	0.0026%	5

Geotechnical Parameters

Geotechnical parameters have been derived by SRK and differ for each pit. There are also no footwall/hangingwall variations. The overall slope angles which have been used are:

Rönnbäcksnäset: 48^0
Vinberget: 50^0
Sundsberget: 49^0

[1] *The Nickel grades are expressed in percent and represent the nickel in the sulphide phase only. Nickel content is therefore in the same units as the nickel bearing rock, in this case tonnes.*
[2] *The cobalt grades for the Rönnbäcksnäset and Vinberget deposits are expressed in percent, in the same fashion as the nickel grades. The cobalt grades in the Sundsberget model are expressed in parts per million.*

Mining Parameters

The following mining parameters have been used in the optimisation process.

Base Mining Cost (at Reference Block - RFBK):	USD1.35/t mined
Incremental mining cost above RFBK:	USD0.05/t/10 m
Incremental mining cost below RFBK:	USD0.07/t/10 m
Mining Recovery:	95%
Mining Dilution:	2.5%

SRK was provided with an expected range of mining costs for the operation of between USD1.72/t and USD1.96/t. This range was based on the assumption that a contract mining fleet would be utilised and matches SRK's expectation based on experience with similar operations in the region.

Given the proximity of the deposits, and the fact that the same mining method will be used for all three, it was considered appropriate to use the same base mining cost for all three deposits.

Processing Parameters

Although other elements are present in the geological models, only nickel has been considered in the optimisations. Cobalt and the other elements should be considered in future optimisation work if more metallurgical information becomes available.

The following processing parameters have been used for the optimisation process.

Ni Processing Recovery:	80%
Processing Cost:	USD4.96/t processed
Transport Cost:	USD0.10/t processed
General & Administration Cost:	USD0.40/t processed
Concentrate Grade:	25%

Economic Parameters

The following economic parameters have been used in the optimisation process.

Nickel Price:	USD9/lb
Discount Rate:	0%
Government Royalty:	0.05%
Private Royalty:	0.15%
Smelter Cost:	USD1.14/lb
Smelter Recovery	100%

Optimisation Constraints

No geographic restrictions have been placed on any of the optimisations. The Sundsberget deposit runs under Lake Gardiken in the south and it has been assumed that a pit wall dam will be constructed in this area to allow mining of this material. Although not included in this optimisation process, the cost of construction of the pit wall dam has been included in the

TEM.

16.1.2 Optimisation Results

Introduction

For each deposit, two sets of optimisations were conducted, each with a different nickel price. Both were then transferred into Datamine for the generation of bench tonnages and grades for scheduling.

Whittle results

Given the preliminary nature of the study, no subjective selection of Whittle shells has been undertaken. The shells chosen for further analysis are the ones that returned the highest pit value; naturally, this occurs where the Revenue Factor (RF) = 1. Future optimisation should consider other shells along the various optimisation curves, and take into consideration incremental stripping ratios and mining costs.

The Whittle results for the three depositsare shown below in Figure 16-1 to Figure 16-3 and the shells chosen for each deposit are:

Rönnbäcksnäset:	Shell 32	RF=1
Vinberget:	Shell 33	RF=1
Sundsberget:	Shell 27	RF=1.22

The optimum shell for Sundsberget is shell 27 which has a RF of 1.22. The reason this shell is optimal is because all of the RF from 0.96 to 1.22 produce the same shell. In other words, the shell at RF=1 is the same shell as that produced at RF=1.22.

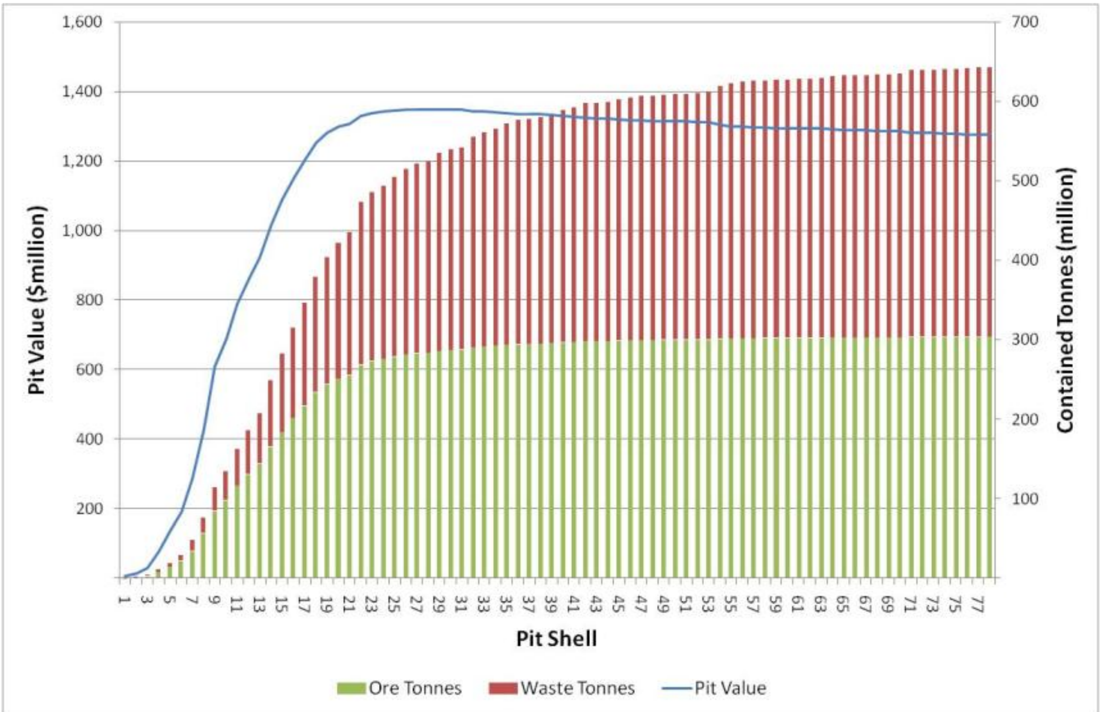

Figure 16-1: Rönnbäcksnäset Whittle Optimisation Results

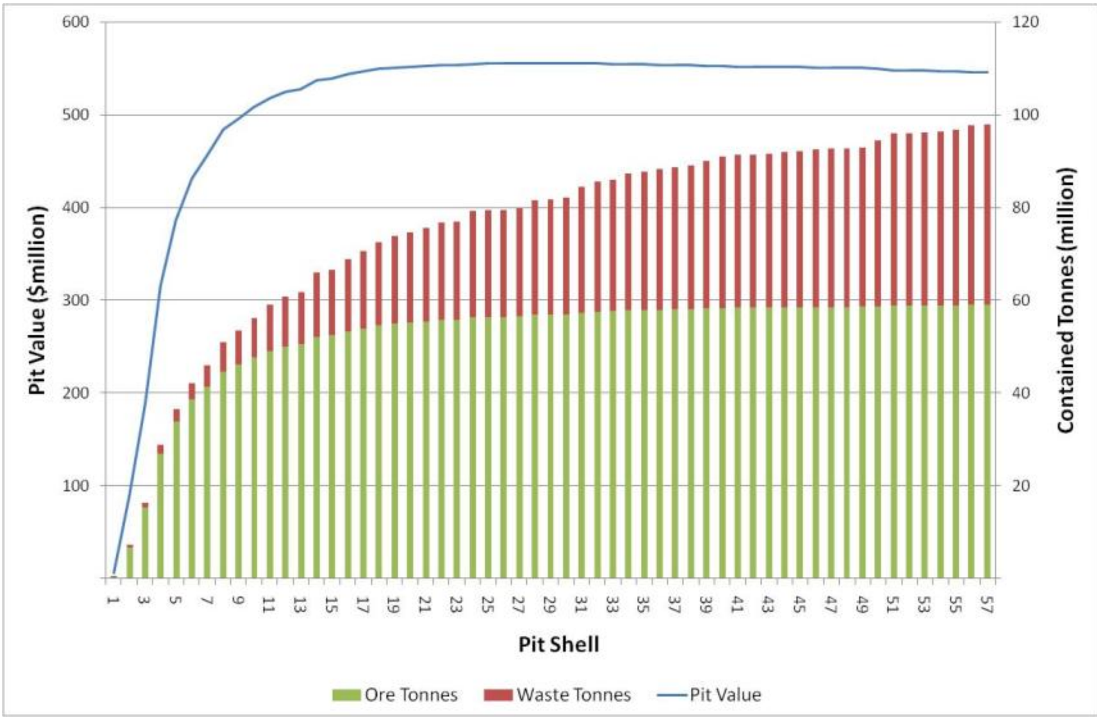

Figure 16-2: Vinberget Whittle Optimisation Results

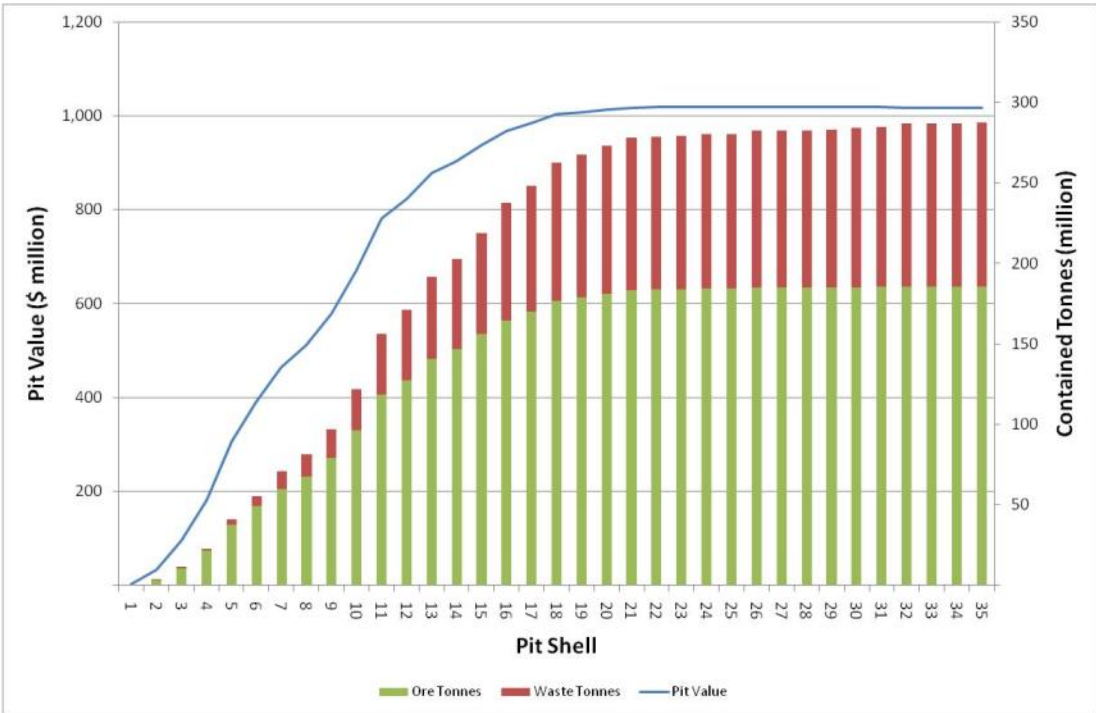

Figure 16-3: Sundsberget Whittle Optimisation Results

16.1.3 Pit Tonnages and Grades

Cut-off Grade Calculation

SRK generated pit tonnages and grades for scheduling purposes for each of the three deposits. It has been assumed that the Rönnbäcksnäset deposit will be mined in two phases with the deposit divided along the north-south line at easting 1,481,000. The other two deposits (Vinberget and Sundsberget) have been scheduled as though they are mined in a single stage, that is, without cutbacks.

Two cut-off grades are relevant to most open pit operations. These are the breakeven cut-off grade and the incremental (or marginal) cut-off grade. The breakeven cut-off grade takes into account all mining and processing costs and represents the minimum grade of material that should be mined in order to pay for all mining and processing activities, including the mining of waste material to access the material in question. The material which is above this grade is referred to as ore, or sometimes high grade ore.

The incremental cut-off grade represents the minimum grade of material that should be mined in order to pay for processing costs only. This assumes that the material has been mined as waste in order to reach higher grade material below it. The material is now at surface and as the higher grade material has paid for the mining costs, the eventual processing / stockpiling strategy can be reassessed. If the grade is higher than the incremental cut-off grade then it is economic to process. Typically, it would be stockpiled and treated in periods where there is insufficient higher grade material being mined, or at the end of the mine life when all of the higher grade material has been depleted. The material with a grade above the incremental cut-off grade but below the breakeven cut-off grade is generally referred to as low grade or marginal material.

Cut-off grades were calculated for each of the deposits, based on the whittle results and input parameters. Table 16-2 below shows the calculated cut-off grades for each of the deposits.

Table 16-2: Cut-off grades for each deposit

	Rönnbäcksnäset	Vinberget	Sundsberget
Breakeven Cut-off Grade (Ni-AC %)	0.0651%	0.0578%	0.0607%
Incremental Cut-off Grade (Ni-AC %)	0.0395%	0.0395%	0.0395%

The incremental cut-off grades are the same for all three deposits. This is expected as all deposits are using the same processing plant, concentrate transport system and smelter. In reality, this cut-off grade would be slightly different for each deposit, once transport costs for ore to the processing plant are taken into account, however the applied level of detail is deemed sufficient for the purposes of this study.

Tonnage and Grade Reporting

The selected Whittle shells were imported into Datamine and the tonnages and grade of material above the respective cut-offs were derived. Table 16-3 below shows these results for each pit. Vinberget and Sundsberget do not have any low grade material within the pit shells.

Table 16-3: Scheduled Tonnages and Grades

	Rönnbäcksnäset	Vinberget	Sundsberget
Ore Tonnes	268.9 mt	57.1 mt	178.9 mt
Ore Ni Grade	0.1063%	0.133%	0.104%
Ore Co Grade	0.0032%	0.006%	0.0034%
Marginal Tonnes	28.7 mt		
Marginal Ni Grade	0.0537%		
Marginal Co Grade	0.0027%		
Waste Tonnes	217.9 mt	23.2 mt	95.6 mt
Total Tonnes	515.5 mt	80.3 mt	274.5 mt
Total Volume[3]	187.8 m BCM	29.5 m BCM	96.2 m BCM

The total Rönnbäcken project contains 504.9 Mt of material at a mean grade of 0.109% Ni-AC and 0.0035% Co above the breakeven cut-off and within the optimum pit shells. In addition, there are 28.7 Mt of marginal material and 336.7 Mt of waste.

The above tonnages have been further split into bench tonnages and grades, with Rönnbäcksnäset also being divided into two phases (as discussed above) and provided to Rolf Ritzén of Ritzén Consult for scheduling purposes.

16.1.4 Mining Schedule

The mining schedules were developed by Ritzén Consult and have been reviewed by SRK. The final schedule received was entitled *Project Rönnbäcken Production Schedule March 2011.xlsx*.

While there is a slight discrepancy between the total material mined in the schedule and that reported in the bench tonnages provided by SRK, the difference is less than 1% and not material. The mining schedule has considered low grade material in the Rönnbäcksnäset pit as ore. Consideration should be given to the adoption of suitable cut-off grade and stockpiling strategies in the future.

[3] *BCM: Banked Cubic Metres – a measure of the volume of in-situ rock in cubic metres.*

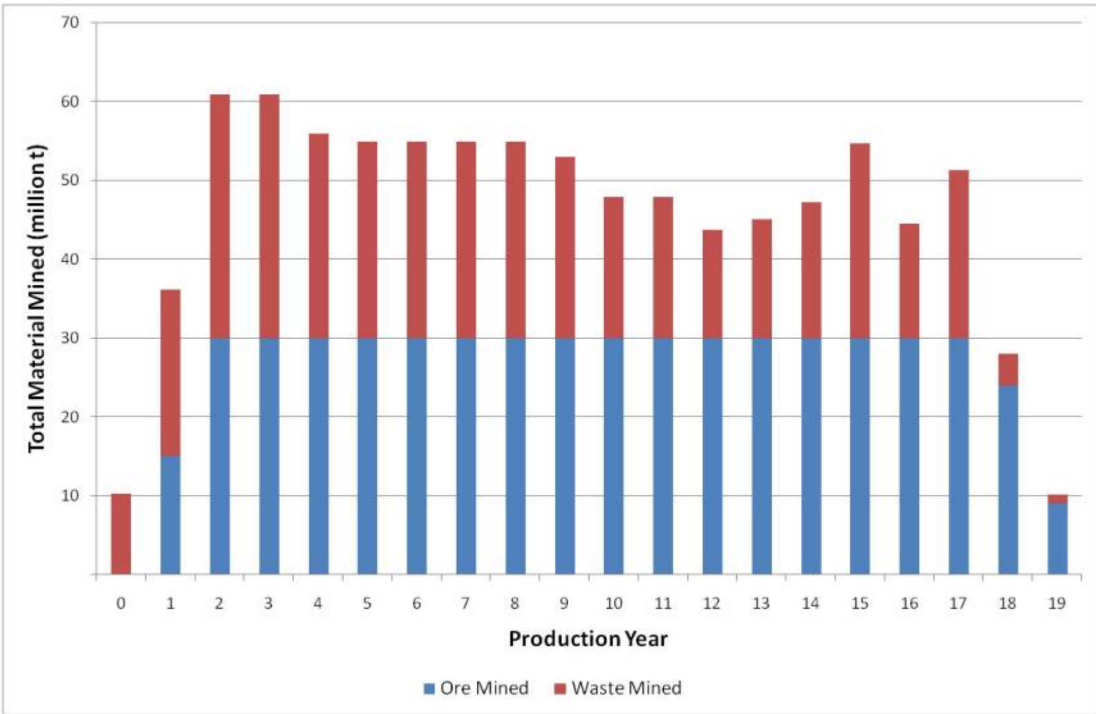

Figure 16-4: Total Material Mined

Figure 16-4 shows mining commencing in Year 0 with 7.1 Mt of pre-stripping occurring at Rönnbäcksnäset and a further 500,000 t of pre-stripping at Vinberget. Some 2.7 Mt are also scheduled to be stripped from Sundsberget in 0, although mining of this deposit then ceases until Year 5. SRK understands that the Company intends to use this material for road and pad construction.

Ore production commences in Year 1 at both Rönnbäcksnäset and Vinberget and full ore production is achieved in Year 2. Marginal material from Rönnbäcksnäset has been stockpiled during the first six years of operation. This allows the higher grade material from Vinberget to be preferentially treated, increasing the feed grade in the early years and increasing the NPV of the project. Although additional costs will be incurred at the end of the project due to the re-handling of the marginal material, this is offset by the increased grade during the early years.

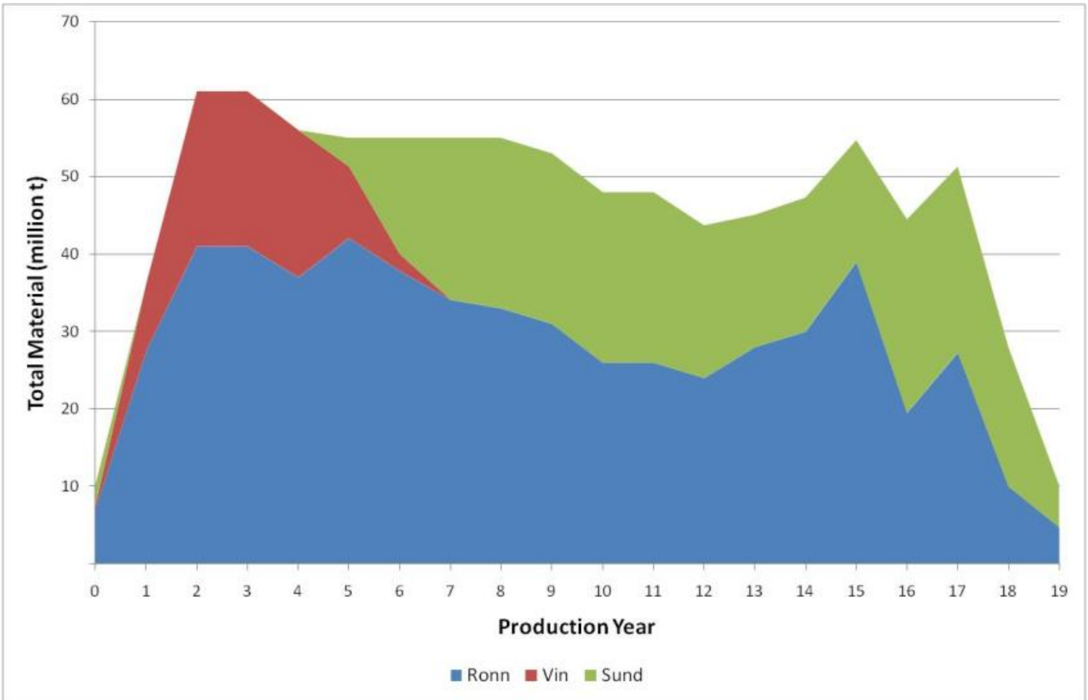

Figure 16-5: Total Material Mined by Pit

The Sundsberget deposit is recommenced in Year 5 and is in full production by the time the Vinberget deposit is depleted. This is a reasonable strategy and ensures consistent supply of ore to the processing plant.

The maximum mining rate is achieved in Year 2 and is sustained at similar levels until Year 8. The ramp up period is aggressive, but should be obtainable assuming:

the contractor is established and has personnel and equipment that can be moved from another operation.

site infrastructure can be constructed in time to meet utilisation demand.

access roads, particularly to Vinberget, can be established in time.

The second part of the schedule sees a drop in material moved from Year 9 to Year 12, but a substantial increase over the following three years. This difference will need to be addressed and removed in future work, either through the use of additional cutbacks in Rönnbäcksnäset and Sundsberget, or by refinement of the use of contract mining equipment as it is unlikely that contractors would be willing to commit additional resources to the project for such a short period of time. Despite this, the schedule is in SRK's opinion reasonable and achievable.

Ore supply is fixed at 30 Mtpa from Year 2 onwards (Figure 16-6). Stockpiles have been used to improve the feed grade in the first several years of operation by deferring the processing of low grade material until the end of the mine life. This approach allows high grade ore to be preferentially fed and is a sound mining practice.

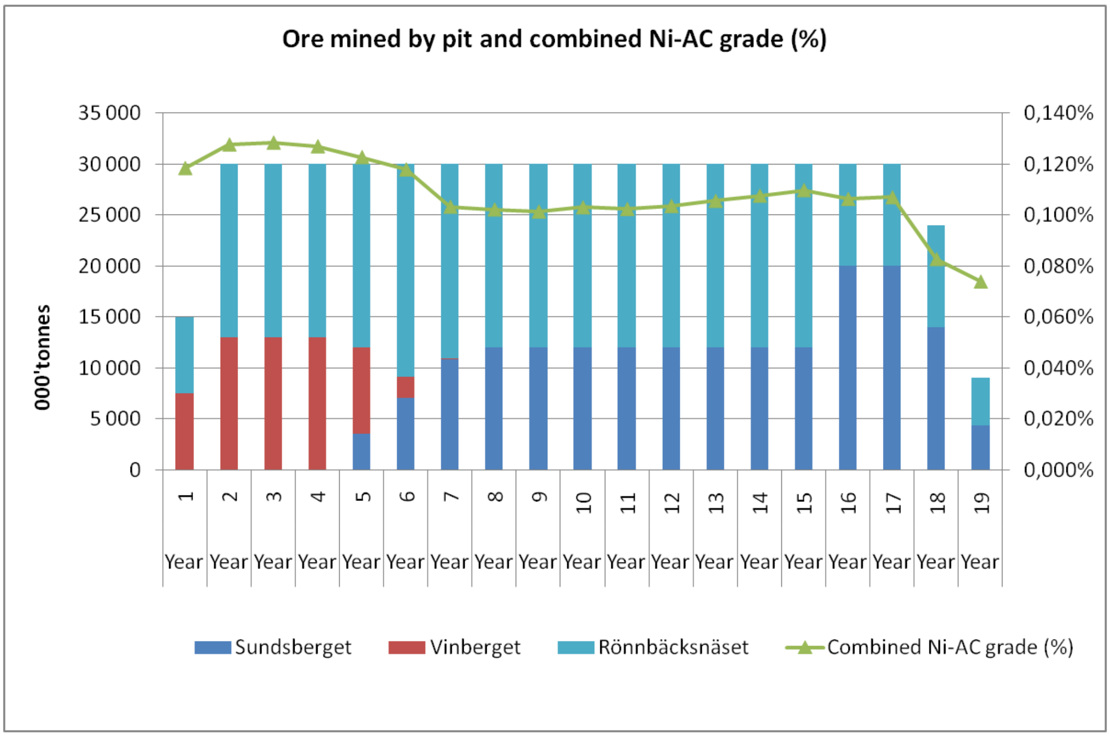

Figure 16-6: Tonnes of ore mined by pit and combined Ni-AC grade

16.1.5 Mining Equipment Selection

Rolf Ritzen of Ritzen Consult has prepared a mining equipment list on behalf of the Company. The base assumption for the TEM is that a contract workforce will be used for the mining operations. This would seem to be a logical choice given the large number of mining contractors and experience in contract mining in Sweden. However, given the relatively long mine life, and the large volume of material to be mined, SRK has recommended to the Company that consideration also be given to converting to an owner-mining operation as the feasibility studies are progressed.

Another possibility would be to grant a 5 year + 1 year mining contract. This will allow the contractor a reasonable amount of time to recoup any initial capital outlay and will also allow the client to consider options for owner mining.

Initial studies suggest that the loading units will be 700 t hydraulic shovels with 34 m^3 buckets. These would be matched with 225 t haul trucks. These are a reasonable combination for the volume of material scheduled in each period.

Allowances have been made for auxiliary equipment including wheel loaders, track dozers, graders water trucks and cranes. At this stage, no indication of specific numbers of these machines has been provided, however, given the assumption of a mining contractor, this is not critical at this stage of the study.

SRK has noted some minor absences in the mining equipment list, particularly lighting plants. In general, however, the proposed size and type of mining equipment would appear to be adequate for this style of operation and as such, no adjustments to the assumptions proposed by the Company have been made in the TEM.

16.1.6 Mine Workforce

A manning schedule has been provided and has been incorporated into the TEM. This workforce schedule relies on the assumption of contract mining, which SRK considers to be reasonable.

16.1.7 Operating Costs

Table 16-4 shows the unit costs which were provided to SRK for the various mining activities. These costs have been derived from first principles by Ritzen Consult, who has compared these with other operations in Sweden. Although SRK considers these operating costs to be low, they are within the expected range and well within the tolerances of the study parameters.

Table 16-4: Unit operating costs for the mining operation

	Unit Cost / t (SEK)	Unit Cost / t (USD)
Drilling	1.19	0.149
Blasting	1.98	0.248
Loading	1.01	0.126
Hauling	2.55	0.32
Personnel	2.53	0.32
Auxiliary Equipment	0.99	0.124
Mine Services	0.90	0.11
Administration	0.28	0.035
Maintenance	0.52	0.065
Contractor Margin (15%)	1.79	0.224
Total	13.74	1.72

16.1.8 Capital Requirements

Assuming that mining will be undertaken by a contractor, much of the capital for the mining operation is removed. However, there are still a number of items of mining infrastructure which will need to be provided by the owner, including:

- office facilities;
- warehouse facilities (possibly included in processing if mining contractor provides their own warehousing for mobile equipment components);
- workshop facility (possibly supplied by mining contractor – although the contract rates would be increased);
- communications facilities*;
- lunch/change out facilities;
- explosives magazines*;
- water storage facilities;
- fuel and lube storage facilities (should the contractor supply their own fuel and lube facility, the client will still require a small fuel store for light vehicles and other client owned ancillary equipment)*;
- emergency response centre/equipment*;
- first aid post*; and

- power and water supply.

Those aspects marked with an asterisk (*) do not appear to have been considered by the Company in developing mining capital costs assumptions. Whilst SRK recognises that provision for these items should be included, the cost of these is not considered material and no adjustment to the TEM has been made for the purposes of this study. Notwithstanding this, SRK recommends that these costs are included in the next phase of study.

16.1.9 Other Considerations

The Sundsberget deposit partially underlies Lake Gardiken at its southern end. In order to extract this material, it will be necessary to construct a dyke or retaining wall around the edge of the pit prior to excavation. While technically feasible given the shallow depth of the lake in this area, this exercise will need to be carefully planned, scheduled and budgeted for in future assessments of the project.

16.1.10 Conclusion

From a technical perspective the Rönnbäcken project appears to be feasible and achievable, in the context of the information available and the level of study. The project is proposed to be a standard contract operated truck/shovel operation. Given the developed mining culture and industry in Sweden and the nature and size of the deposit, from a technical mining perspective, SRK does not foresee any major issues that would prevent the operation from commencing.

16.1.11 Recommendations

Based on the results of this study, SRK has recommended to the Company that the following aspects of the operation be studied further as the Project is further assessed:

A conventional truck/shovel mining operation has been proposed for the operation. Other mining methods may be applicable for the operation and a review of these should be completed.

The three deposits have been optimised using separate stand-alone models. This was done because of the close proximity of the deposits. Although it is unlikely to make a significant difference in the overall result, reoptimising these deposits using a multi-mine configuration would add an additional level of completeness to the project.

Although present in the geological models, cobalt and other elements have not been included in the optimisation process. These elements should be included in future optimisations as additional metallurgical test work is completed.

Separate ore transport costs have not been considered for each pit during the optimisation process. This should be examined and included in future optimisations to reflect the different ore haulage distances. The most economical method to haul ore from Vinberget should also be considered. This may include smaller haul trucks or even road trucks hauling from a stockpile at the pit edge.

A reduction in waste stripping occurs in Years 10 to 14 of the schedule, but then the stripping increases dramatically over the following three years. If possible, the use of cutbacks should be examined in more detail to try to remove this inconsistency, and bring waste stripping forward, in order to balance out the mining fleet.

Contract mining has been assumed for the operation. While this is common practice in Sweden, given the life and size of the operation, consideration should be given to implementing an owner mining operation after an initial term of contract mining. Allowances for contract management and client technical services personnel also need to be made in the proposed manning numbers.

Although some ancillary equipment has been included, there are some absences. These will need to be included in future work.

Consideration needs to be given to the source, production and amount of sheeting material which will be required for road maintenance, given the length of proposed roads on the site, and the climate of the region.

16.2 Geotechnical Design Criteria

16.2.1 Slope Angle Estimation

For this preliminary economic assessment, indicative overall slope angles on a lithology basis were estimated using rock mass classification ratings derived from photologging of selected borehole core carried out by Gary Dempers of Dempers and Seymour Pty Ltd (D&S) and reviewed by SRK.

A total of 35 boreholes (7,649 m) were logged to provide input to produce Laubscher Mining Rock Mass Ratings (MRMR) for the three deposits; 14 for Rönnbäcksnäset (North and South; 2,603 m), 9 for Sundsberget (2,743 m) and 12 for Vinberget (2,303 m). The MRMR classification system used the fracture frequency, intact rock strength and the joint condition to calculate values for each interval. The calculated MRMR values are adjusted to account for the potential effects of mining and exposure. MRMR adjustments were applied as follows:

- Weathering – 1.0
- Stress – 1.0
- Orientation – 0.9
- Blasting – 0.95

The unadjusted and adjusted MRMR value for each lithology at each deposit was calculated using mean, lower and upper quartile values for fracture frequency, intact rock strength and joint condition. The Haines-Terbrugge empirical slope design chart, which related adjusted MRMR to slope angle and slope height for specified factors of safety, was used to estimate slope angles. A maximum vertical slope height per lithology of 160 m and a nominal factor of safety of 1.2 was used.

The results of this evaluation are presented in Table 16-5 for the mean and lower quartile values only. Given the preliminary nature of the study and the method by which the slope angles have been estimated, SRK considers it inappropriate to consider the use of slope angles derived from the upper quartile of the data set. The minor lithologies, such as the conglomerate, are not shown as they do not make up a significant proportion of the slope. SRK has reviewed the methodology used to estimate slope angles and considers this to be to internationally accepted standards and appropriate for a preliminary economic assessment.

Table 16-5: Slope angles by lithology for each deposit

Rönnbäcksnäset North						
Rock Type	Percentile	RMR	MRMR	Slope Angle	Rock Mass Classification	Swedish Lithology Nomenclature
Serpentinite	Lower Quartile	39	34	47	Poor	Serpentinit, Ni rik; Serpentinit, pyroxenforande; Serpentinit, lag haltig
	Mean	44	37	49	Fair	
Sediments	Lower Quartile	48	41	51	Fair	Fyllit
	Mean	50	43	52	Fair	
Mafic	Lower Quartile	50	43	52	Fair	Basisk intrusion
	Mean	52	44	52	Fair	
Chlor/Seds	Lower Quartile	42	36	48	Fair	Kloritiskt Fyllit
	Mean	49	42	51	Fair	

Rönnbäcksnäset South						
Rock Type	Percentile	RMR	MRMR	Slope Angle	Rock Mass Classification	Swedish Lithology Nomenclature
Serpentinite	Lower Quartile	34	29	44	Poor	Serpentinit, Ni rik; Serpentinit, pyroxenforande; Serpentinit, lag haltig
	Mean	40	34	47	Poor	
Sediments	Lower Quartile	38	32	46	Poor	Fyllit
	Mean	44	38	49	Fair	
Mafic	Lower Quartile	50	43	52	Fair	Basisk intrusion
	Mean	52	44	52	Fair	
Chlor/Seds	Lower Quartile	34	29	44	Poor	Kloritiskt Fyllit
	Mean	45	38	49	Fair	

Vinberget						
Rock Type	Percentile	RMR	MRMR	Slope Angle	Rock Mass Classification	Swedish Lithology Nomenclature
Serpentinite	Lower Quartile	40	35	47	Poor	Serpentinit, Ni rik; Serpentinit, pyroxenforande; Serpentinit, lag haltig
	Mean	44	38	49	Fair	
Sediments	Lower Quartile	39	34	47	Poor	Fyllit
	Mean	45	39	49	Fair	

Sundsberget						
Rock Type	Percentile	RMR	MRMR	Slope Angle	Rock Mass Classification	Swedish Lithology Nomenclature
Serpentinite	Lower Quartile	42	36	48	Fair	Serpentinit, Ni rik; Serpentinit, pyroxenforande; Serpentinit, lag haltig
	Mean	46	40	50	Fair	
Sediments	Lower Quartile	42	36	48	Fair	Fyllit
	Mean	49	42	51	Fair	
Mafic	Lower Quartile	46	39	50	Fair	Basisk intrusion
	Mean	49	42	51	Fair	
Chlor/Seds	Lower Quartile	45	38	49	Fair	Kloritiskt Fyllit
	Mean	49	42	51	Fair	

16.2.2 Recommended Slope Angles for Pit Optimisation

In order to use these individual lithology slope angles to develop an overall slope angle for pit optimisation, SRK overlaid Whittle shell profiles on geology sections and constructed pit slopes from the floor of the Whittle shell to the crest using the individual lithology slope angles. This is illustrated in Figure 16-7 for a cross section through the Rönnbäcken deposit. The results for all cross sections analysed in this way are presented in Table 16-6.

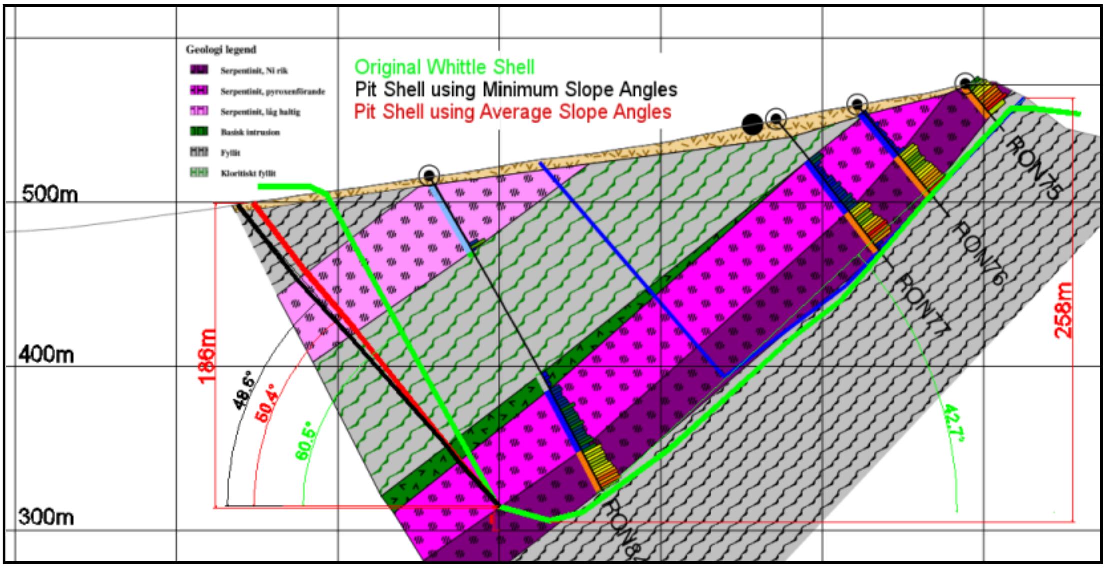

Figure 16-7: Whittle shell and new pit slopes (average and min) overlaid

Table 16-6: Updated highwall and footwall pit slope angles

Pit	Section	Base of Pit (AOD) (m)	Depth (m)	Highwall Whittle Angle - HW (°)	HW Angle (°) Lith Average	HW Angle (°) Lith Minimum
Ron North	400	315	185	60.5	50.4	48.6
	900	350	130	46.8	51.1	49.4
	1100	350	135	54.9	51.2	49.4
Ron South	200E	215	235	49.6	50.7	48.7
	200W	190	230	55.6	49.4	47.4
Vin	700	320	255	54.6	51.4	50.2
	550	455	155	44.1	52	51
Sund	2600	230	165	48.1	50.6	49.1
	3200	100	305	50.4	50.2	48.6

Pit	Section	Base of Pit (AOD) (m)	Depth (m)	Footwall Whittle Angle - FW (°)	FW Angle (°) Lith Average	FW Angle (°) Lith Minimum
Ron North	400	315	260	43	43	43
	900	350	190	40	40	40
	1100	350	180	36	36	36
Ron South	200E	215	225	27	27	27
	200W	190	210	29	29	29
Vin	700	320	295	52.5	51.3	50.8
	550	455	165	35.1	52	51
Sund	2600	230	180	20	20	20
	3200	100	365	33	33	33

SRK notes that for Rönnbäcksnäset and Sundsberget the footwall dip is relatively shallow and the pit walls follow the dip of the orebody. The dip of the Vinberget deposit is steeper and consequently the angle of the footwall slope is also steeper. Using the data gathered from the individual section analysis there appears to be little difference to overall slope angle whether using minimum or average slope angles. However, the influence of groundwater and the close proximity of Lake Gardiken remains to be evaluated with regard to stability of the pit walls. As such, SRK has chosen to use the lower bound slope angles for the PEA. The recommended maximum overall footwall and hangingwall slope angles for pit optimisation are:

- Rönnbäcksnäset – 48°
- Vinberget – 50°
- Sundsberget – 49°

Furthermore detailed geotechnical investigations comprising the drilling of specific orientated geotechnical boreholes together with field and laboratory rock testing and detailed structural and hydrogeological interpretation will be required to advance the project to the next phase. This will allow slope angles to be optimised and bench/berm configurations to be designed, taking into consideration the interaction of rock structure and groundwater with the pit slopes.

16.3 Waste Rock Management

The preliminary waste rock dump design and location has been carried out on behalf of the Company by Ritzen Consult. The details of the waste rock dumps (WRD) are limited in the documents reviewed by SRK. The Company has indicated that the WRD will have an average height of 50 to 70 m and a slope angle of 1:5. The proposed locations of the WRD in proximity of each of the proposed pits is considered by SRK to be reasonable.

The acidic rock drainage (ARD) potential from the waste rock dumps is discussed in Section 18 below.

17 RECOVERY METHODS

17.1.1 Metallurgical Testwork

Historical Testwork

Previously published reviews of the historical metallurgical testwork conducted on various samples from the Rönnbäcken area have indicated that nickel can be recovered into commercially acceptable sulphide concentrates at nickel recoveries between 67 and 73% and at concentrate grades of 26 to 34% Ni. Typically, a primary grind of P_{80} 44 µm was required.

Initial bench scale studies were performed in the early 1970s by the Royal Institute of Technology in Stockholm. Standard flotation tests were performed on three different samples from the Rönnbäcken area and the nickel concentrates produced contained 31% Ni to 47% Ni, 1.5% Co to 2.8% Co, 4 g/t Au to 8 g/t Au, and some minor platinum group metals (PGM) at recoveries reported to be 80% of the sulphide nickel.

In 1974, Boliden carried out test mining in the Rönnbäcken area and conducted pilot flotation testing. Nickel concentrates containing 26% Ni to 34% Ni, 1.5% Co, 5 g/t Au, and 2 g/t combined PGM at a recovery of 67% to 73% were produced. In addition, grinding was tested using both a rod mill – pebble mill and autogenous (AG) mill – pebble mill circuits. The best metallurgical results were obtained using AG – pebble mill circuit with a P_{80} of 44 µm. The total energy consumption using two-stage fully AG grinding was reported to be approximately 25 kWh/t.

The suitability of autogenous grinding was confirmed by industrial scale testing at a Boliden concentrator and demonstrated that Vinberget ore media were very competent, comparing favourably with ore from the Aitik copper mine which utilises the large diameter AG mill – pebble mill grinding circuit.

The gold grades in the concentrate from initial testwork were assumed to be the result of contamination and not representative of gold in concentrate from the Project.

Minpro Tests 2007

In 2007, standard bench scale flotation tests were performed at Minpro AB (Minpro) laboratories on a sample taken used in the earlier Boliden investigations. Despite the earlier work, two coarser primary grind sizes, P_{80} 80 and 60 µm were investigated.

Minpro reported total nickel (Ni_T) rather than only sulphide nickel (Ni_S) in its report. Even at the low head grade of 0.10% Ni_S , concentrate grades as high as 25% Ni were achieved. The best tests resulted in a recovery of sulphide nickel to the rougher concentrate of 90%. After two stages of cleaning in an open circuit test, a concentrate grading 18% Ni was produced at approximately 77% recovery (estimated due to assays reported as Ni_T).

Outotec Research Centre (ORC) Tests – ORC Phase 1 Testing 2008

In 2008, five 25 kg samples from the drilling campaign were sent by the Company to the Outotec Research Centre (ORC) in Finland. The samples were collected from two drillholes at Vinberget (VIN29 and VIN30) and one drillhole at Rönnbäcksnäset (RON53). The samples provided were half core that had been crushed to -2 mm. In addition, a reference sample was provided from the Boliden test pit near Vinberget which had previously been used in the Minpro tests.

The range of nickel analyses of the five samples was 0.104% to 0.153% Ni_S and 0.182% to 0.202% Ni_T, using bromine methanol (BM) assay methods for determination of sulphide nickel and four acid digestion for total nickel. The samples were also assayed for Ni_S content by AC method at Labtium, Finland, and by for Ni_T content by both four acid digestion at ALS, Vancouver, and aqua regia digestion at Labtium, Finland,. The assay methods compared reasonably well. The comparative results are presented in Table 17-1.

Table 17-1: Assay Method Comparison

Sample	Sulphide Ni		Total Ni		
	BM %Ni_S	AC %Ni_S	ORC %Ni_T	Four Acid %Ni_T	Aqua Regia %Ni_T
VIN29 (40-62m)	0.13	0.1325	0.190	0.190	0.1747
VIN30 (6-28m)	0.133	0.1334	0.189	0.1985	0.1771
VIN30 (184-206m)	0.153	0.161	0.187	0.1858	0.1757
RON53 (44-66m)	0.104	0.0953	0.182	0.1688	0.1653
RON53 (72-94m)	0.131	0.1438	0.202	0.1852	0.1915

During this phase of testing, ORC conducted a total of 14 standard rougher flotation tests using primary grinds of approximately P_{80} 80, 50 and 40 µm to determine the best grind size. The reagents used were potassium amyl xanthate (PAX) as collector, a standard dispersant, Dowfroth 250, and sulphuric acid for pH control.

The results from the first phase ORC testing could be summarized as follows:

Sulphide nickel recovery to rougher concentrate ranged from 75% to 85%.

Improved results were achieved at the finer grind sizes.

The two assaying methods, ammonium citrate and bromine methanol, provided similar results.

The rougher concentrate typically contained approximately 1% Ni_s and contained many liberated gangue minerals.

Sedimentation of solids in the tailings was slow but manageable.

In addition to this preliminary work, standard grinding tests on two individual core samples, one each from Vinberget and Rönnbäcksnäset, gave Bond ball mill work index values of 17.5 kWh/t and 16.4 kWh/t, respectively. These results characterize the ore as medium-hard in a typical ball milling size range, with Rönnbäcksnäset slightly softer than Vinberget.

ORC Phase 2 Testing 2009

Phase 2 of the ORC test program commenced in March 2009 and finished in July 2009. The objective of this testing was to produce higher grade concentrates in laboratory scale batch flotation tests while improving operating costs.

Two composite samples representing the Vinberget and Rönnbäcksnäset deposits were prepared by the Company and sent to ORC. The composite sample assays are detailed in Table 17-2 below.

Table 17-2: Composite sample assays from ORC Phase 2 Testing 2009

	$\%Ni_S$	$\%Co_S$	$\%Ni_T$	$\%Co_T$	$\%Fe_T$	MgO
Vinberget	0.118	0.006	0.177	0.009	5.36	35.6
Rönnbäcksnäset	0.117	0.002	0.189	0.009	5.31	34.8

This test work focused on standard flotation tests using a finer grind. Initial tests were conducted at a P_{80} 50µm, while in the later stages the grind size was varied between P_{80} 38µm and 31µm. The reagent additions were modified throughout the testing using the following general scheme:

- PAX as a collector;
- Dowfroth 250 as a frother;
- H_2SO_4 as a pH modifier;
- Carboxymethyl Cellulose (CMC) as a dispersant or MgO depressant (predominantly for Viberget); and
- a second standard dispersant (predominantly for Rönnbäcksnäset ore).

A total of 18 rougher flotation tests and 14 cleaner flotation tests were conducted on the two composite samples and were limited to open circuit batch tests.

The finer grind sizes, P_{80} 38 µm and 31 µm, produced much better results than coarser grinding. Concentrate grades of between 25% and 35% were produced at overall sulphide nickel recoveries of 50% to 60%. Typical rougher recoveries at the finer grind were 77% to 83%, and typical cleaner recoveries were 66% to 70%. The results for the Vinberget sample were generally slightly better than those for the Rönnbäcksnäset composite at lower concentrate grades, however, at a grade of 28% Ni, the recovery from both composites was similar.

Following this testwork Outotec simulated closed circuit metallurgical performance in a commercial plant using HSC Chemistry®, steady state simulation software, by using the kinetic information from the laboratory results for the Vinberget ore only. The results were validated against the open circuit results. Laboratory locked cycle tests were not performed but the company made the decision to verify the simulations by closed circuit minipilot testing (see Phase 4 testing below). Based on the simulation work, after four stages of cleaning, ORC predicted that a cleaner concentrate would contain 28% Ni at 74.5% recovery and approximately 1.0% Co.

Other potential payable metals include gold, silver, platinum, and palladium. There are only minor quantities of these metals in the ore, so it is unlikely that these will contribute much revenue and, therefore, they have not generally been assayed for in the test work. The estimated recoveries based on the very limited data available from one test at ORC are 20% recovery of Au and Ag, and 35% recovery of Pt and Pd

ORC Phase 3 Testing 2009-2010

The phase 3 testing at ORC, was performed in November 2009 through to March 2010 to investigate the effects of a coarser primary grind prior to rougher flotation followed by cleaning incorporating concentrate regrinding. The primary grind was P_{80} 50 to 60 µm. In summary batch flotation gave a recovery of 65% at 25% nickel grade. Overall the results of these batch flotation tests indicated an improvement in metallurgical performance and, using the simulation methodology previously described, the predicted performance in a commercial plant was 78% nickel recovery at a 28% concentrate grade.

GTK Phase 4 Mini pilot plant testing 2010

In March 2010, phase 4 mini pilot plant testing was performed at the Geological Survey of Finland Mineral Processing laboratories (GTK) in Finland using a 50 : 50 blend of samples of Vinberget and Rönnbäcksnäset ores. The composite used in the phase 1 to 3 tests was exhausted in early 2010 and a second composite blend (Comp 2) of 50:50 Vinberget and Rönnbäcksnäset ore was made.

The Comp 2 analysis was 0.128 % Ni_S and 0.203 % N_T, 0.004% Co_S and 0.011 % Co_T, and 0.074% S.

Laboratory tests were performed on Comp 2 to compare the flotation response with Comp 1. In general the nickel recovery was below that achieved with Comp 1 by approximately 10%. No specific reason was identified for this effect.

In the mini-pilot tests 1300 kg of sample was used in six 10-hour tests, over a six day period. Typically, the feed rate was 21 to 22 kg/hour.

The grinding circuit product size was P_{80} 45 to 60 µm over the testing period. A number of circuit configurations and reagent regimes were tested and the best results were achieved with four stages of cleaning (Figure 16-1). The best results in terms of nickel recovery were 80% at a grade of 22.3 % Ni, with 75% recovery achieved at a 25.8% Ni grade.

It would be expected that the results in a larger pilot plant, incorporating full stream recycle would improve these results.

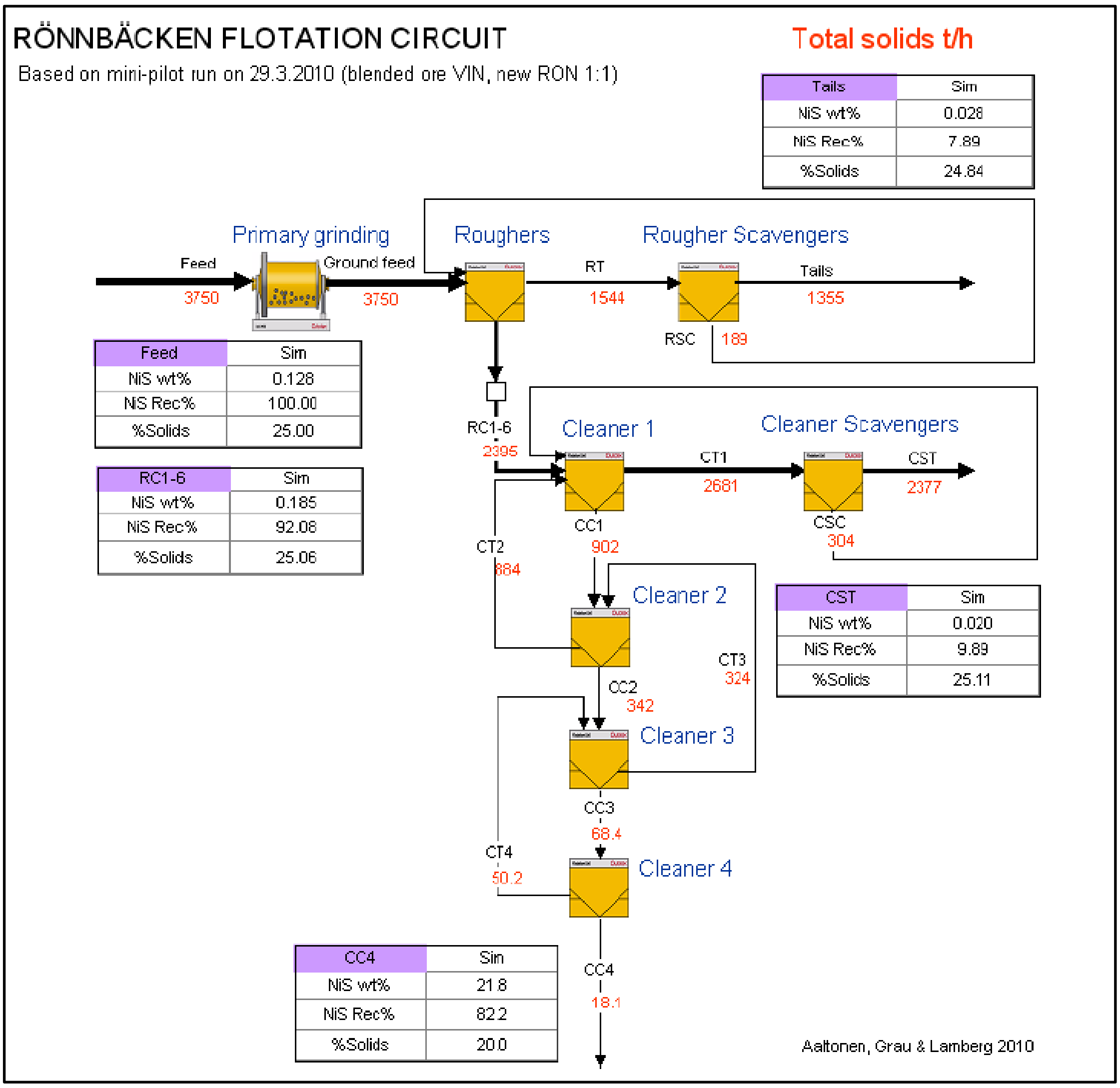

Figure 17-1: Rönnbäcken flotation circuit

Figure 17-2 below demonstrates the appearance of the final concentrate as produced during the minipilot campaign.

Figure 17-2: Final concentrate as produced during the GTK minipilot campaign

ORC Phase 4 testing of Sundsberget

Preliminary laboratory testing of ore from Sundsberget was performed in 2010. Test results indicated a similar metallurgical response to that achieved with the blends of Vinberget and Rönnbäcksnäset ores and consequently for this evaluation SRK considers it reasonable to apply similar nickel recovery and grade predictions to Sundsberget ore.

Further more extensive testing of blends containing this ore type is planned.

ORC Phase 4 Testing 2010

Further laboratory testing of the Comp 2 blend of Vinberget and Rönnbäcksnäset were performed in 2010 to investigate the effects of alternative reagent regimes and lower reagent dosages and to try and reduce the MgO content of the concentrate. In general terms, further improvements in the nickel grade – recovery relationship were achieved with up to 28% nickel grade achieved at recoveries equivalent to 80% in closed circuit, as demonstrated in scaling up from laboratory to minipilot during the minipilot campaign. This indicates that there is scope to use different reagents and reduce the reagent dosages without adversely affecting metallurgical performance.

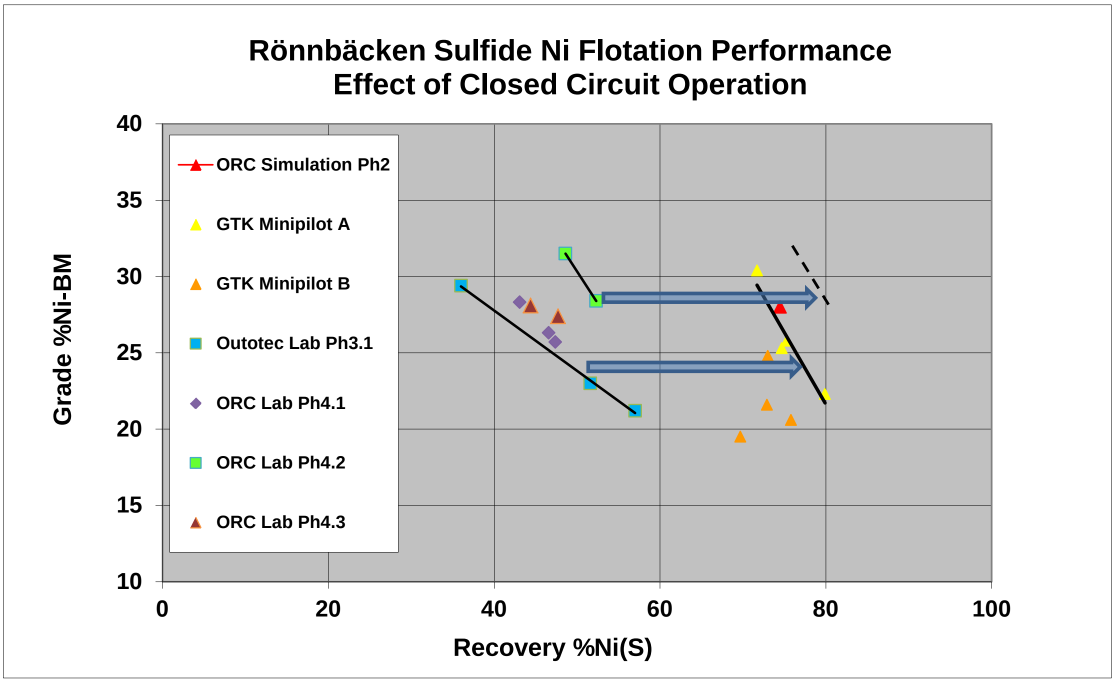

Figure 17-3: Sulphide Ni flotation performance

Magnetite recovery from flotation tailings

Preliminary testwork to investigate the recovery of a magnetite concentrate from the Rönnbäcken ores was performed in late 2010 and early 2011. In order to simplify the initial tests and to avoid complications with the sulphide flotation, preliminary magnetic separation tests were performed on head samples of both Vinberget and Rönnbäcksnäset ores. Consequently, the levels of nickel, cobalt and sulphur in the magnetite concentrate are likely to be higher than would be expected from the testing or treatment of flotation tailings. The head samples typically contained 5 to 6% Fe and most of the recoverable iron was in the form of magnetite.

A total of 27 tests were performed and focused on low intensity magnetic separation (LIMS), using roughing and up to three stages of cleaning followed by desliming and reverse flotation to reduce impurity levels. Previously published results indicated that concentrate grades up to 62.4% Fe at 70% magnetite recovery were possible (test BMS24).flowsheet is presented in Figure 17-4. Ultrafine grinding of the concentrate to nominally P_{80} 20 µm was required to liberate impurities. The sulphur level was acceptable but other impurities, SiO_2, MgO and Cr_2O_3, were slightly high and further testwork was required to improve the concentrate quality.

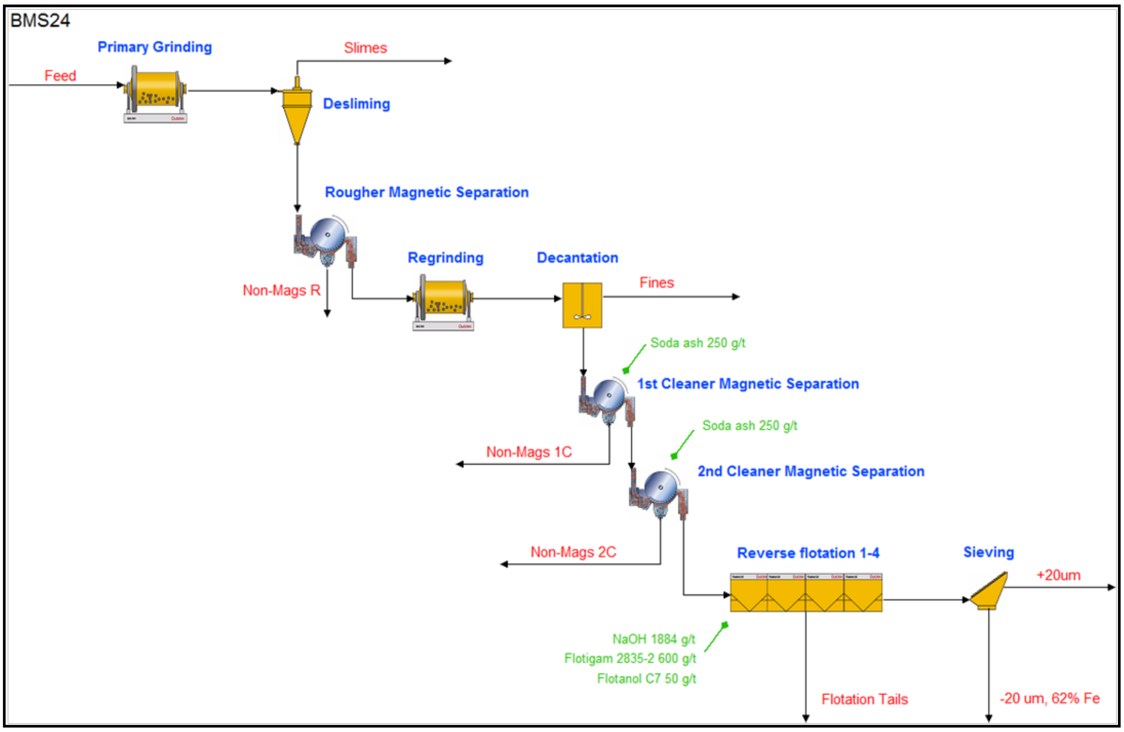

Figure 17-4: Flowsheet –Magnetite Circuit - Test BMS24

(courtesy of Outotec)

Further metallurgical testwork was performed by Outotec at GTK's facilities in Finland in October 2011. Thirteen batch tests were performed on the nickel flotation tailings from the mini-pilot plant work performed in March 2010 to evaluate magnetite recovery using flowsheets including desliming, magnetic separation, concentrate regrinding, reverse flotation and final product classification.

The initial test (BMS101) on the mini-pilot plant tailings, using the BMS24 flowsheet, confirmed the results achieved previously, 61.6% magnetite recovery at a grade of 63.5% Fe.

Further tests demonstrated that reverse flotation was not necessary and could be replaced by magnetic separation. Flotation reagents were eliminated but the amount of dispersant was high and was potentially a significant operating cost.

The feed desliming losses represented between 10 and 12% of the contained magnetite and testing without this process stage indicated that there was no detrimental effect of the magnetite metallurgy. In addition concentrate cleaning prior to regrinding was found to be beneficial.

Further testing with reduced amounts of dispersant demonstrated that an addition of 200g/tonne was beneficial during concentrate regrinding. The final flowsheet adopted for batch optimisation and confirmatory testing is given in Figure 17-5. Test BMS115 achieved a magnetite recovery of 90.2% at a grade of 66.2% Fe in open circuit batch tests. This test was repeated in BMS116 and confirmed the result.

The significant test results are presented in Table 17-3.

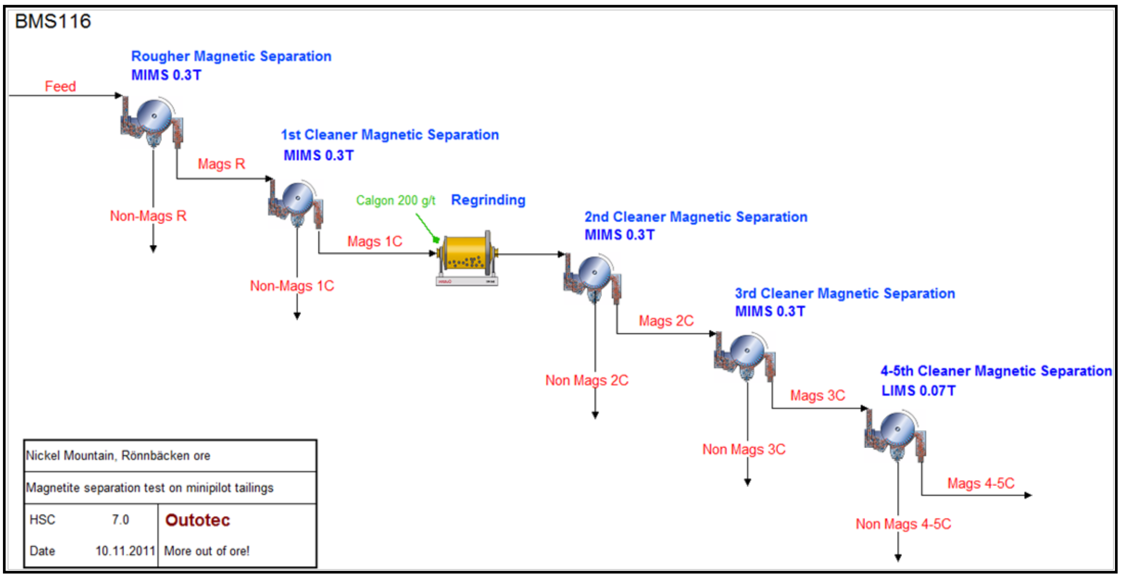

Figure 17-5: Flowsheet – Final Magnetite Circuit –Test BMS115/116

(courtesy of Outotec)

Table 17-3: Final Magnetite Recovery Test Results

Test No.	Description	Mass	% Fe	Magnetite (Satmagan)		Concentrate analyses (XRF)			
		%	(XRF)	%	% Recovery	% Cr	% Mg	% Ni	% Si
BMS 113	800 g/t dispersant	5.94	65.1	82.4	90.6	2.15	2.93	0.57	1.66
BMS 114	No Dispersant	5.37	66.1	83.5	82.2	2.17	2.52	0.57	1.41
BMS 115	200 g/t dispersant	5.86	66.2	85.5	90.3	2.18	2.51	0.58	1.40
BMS 116	200 g/t dispersant	5.88	66.0	84.7	89.5	2.16	2.56	0.56	1.44

The chromium content of the concentrate, approximately 2.2%, is slightly above that normally required but is still acceptable in a saleable magnetite product. Analysis has indicated that this element is closely associated with the magnetite and is probably contained in the mineral lattice. Consequently it may be difficult to reduce significantly. The other impurities, including phosphorus are acceptable.

Future Metallurgical Testwork

Additional laboratory testing, mini pilot testing and full scale pilot testing is planned for 2011 and 2012. This testing is required to:

- finalise the optimum flowsheet for blends of the three main ore types, Vinberget, Rönnbäcksnäset and Sunsberget;
- further investigate the reduction of penalty elements, particularly MgO and As, in the nickel concentrate;
- improve metallurgical performance in terms of both nickel concentrate grade and nickel recovery, and to confirm the extrapolated metallurgical performance for the three ore types;
- finalise the reagent regime and consumption figures;

- finalise the circuit and metallurgical performance of the magnetite recovery circuit including magnetite concentrate regrind particle size;
- test a wide range of samples from across the different ore bodies (variability testing);
- finalise the plant design parameters;
- provide final design data for equipment such as regrinding, thickeners and filters; and
- concentrate transportable moisture determinations.

Testing should include pilot scale testing of the circuit using blends of the three main ore types.

Comminution testwork will be required to confirm the application of autogenous grinding and for sizing of the crushing and grinding circuits. The requirements should be discussed with the major equipment suppliers and should include internationally recognised testing methods and if possible large scale grinding tests using the different ore types.

The magnetite concentrate produced in the batch tests was very fine, nominally minus 20 µm, and further testwork is required to optimise this parameter and to establish if it will be necessary to pelletize the product.

Concentrates from pilot plant test work may also be used to produce larger amounts of concentrate for negotiation with smelters for off-take agreements and for the transportable moisture determinations.

17.1.2 Mineralogy

Mineralogical work has been a mix of qualitative and quantitative work, consisting of optical microscopy examinations, and SEM analysis of the various minerals observed, together with Mineral Liberation Analysis (MLA) of feeds, and selected concentrates and tails. The work has been based on composite samples from each major deposit comprising the resource.

The mineralogical assay of magnetite was approximately 6%. This figure is used in magnetite recovery studies.

Vinberget Sample

The predominant mineral in the Vinberget sample is antigorite, but significant amounts of chlorite are also present suggesting that altered peridotite or pyroxenite rock types exist in addition to serpentinites. Diopside was also found, which was not present in samples studied earlier.

Pentlandite, $(Ni,Fe)_9S_8$, is the main Ni-sulphide, but some heazlewoodite, Ni_3S_2, is also present in the Vinberget sample. Heazlewoodite occurs mainly as lamellae in pentlandite. Based on optical observations the pentlandite to heazlewoodite ratio is approximately 3 to 1. Antigorite contains trace amounts of nickel, at an average of 0.1% Ni. Magnetite and chromite also contain low amounts of nickel.

The total nickel content of the Vinberget sample was assayed as 0.177% Ni. Sulphide nickel was measured as 0.118% Ni, using the BM method. This indicates that 67% of the nickel is in sulphide form, and 33% is in non-sulphides.

Optical microscopy examinations showed that 50% of the nickel sulphides were liberated at

<45 μm, 65% at <38 μm, and 90% at <20 μm. MLA showed that 91% liberation is achieved at a 38 μm grind. This indicated that a primary grind of approximately P_{80} 45 μm was required to achieve the necessary liberation and that regrinding will be required in the flotation cleaner circuits to produce acceptable Ni concentrate grades. The predominant unliberated particles were identified as binaries with serpentine (antigorite).

Rönnbäcksnäset Sample

In the Rönnbäcksnäset sample, antigorite was found to predominate, but with heazlewoodite as the dominant sulphide. The grain size was found to vary mainly from 10 to 100 μm, while very small grains of Co-pentlandite and maucherite were encountered together with heazlewoodite.

Heazlewoodite is the main Ni-sulphide, while only a couple of pentlandite grains were found and, in each case, they were Co-rich. Heazlewoodite was found to occur mainly as locked grains with antigorite and, to a lesser degree, with magnetite. No pyrite or pyrrhotite was noted.

The total nickel content of the Rönnbäcksnäset sample was assayed as 0.189% Ni, with sulphide nickel measured at 0.117% Ni, using the BM method. This indicates that 62% of the nickel is in sulphide form, and 38% is in non-sulphides.

Again the optical microscopy work indicates that a relatively fine grind is required for good liberation. MLA work shows 89% liberation at a grind of 39 μm. Both methods indicate that relatively high grade concentrates can be produced.

Sundsberget Sample

The Sunsberget sample consists mainly of the serpentine mineral, antigorite. The remainder of the sample is mainly composed of diopside, chlorite, magnetite, chromite and magnesite. Tiny amounts of carbonates, olivine, chromium-bearing magnetite, Ni-sulphides and maucherite are also present. Heazlewoodite is the main nickel sulphide, containing approximately 54% of the total nickel. Pentlandite is the secondary nickel sulphide containing approximately 11% of the total nickel. The remaining nickel is contained in magnetite (19%) and silicates.

The total nickel content of the Sundsberget sample was assayed as 0.190% Ni, with sulphide nickel measured at 0.112% Ni, using the BM method. This indicates that 59% of the nickel is in sulphide form.

Iron is present as magnetite (66%) and serpentine (23%). The MgO% is nearly 36%.

As with the Vinberget and Rönnbäcksnäset samples, the nickel grain size was found to vary mainly from 10 to 100 μm.

17.1.3 Concentrate Quality

Nickel Concentrate

The quality of concentrates, in terms of nickel content, produced from the various testing programmes has been high. Results from minipilot campaign are shown in Figure 17-6, Figure 17-7 and Figure 17-8. It should be noted that this concentrate would be considered fairly unique amongst Ni concentrates as it has a high Ni content and very low Fe content, owing to the high percentage of Ni contained in heazlewoodite (Ni_3S_2).

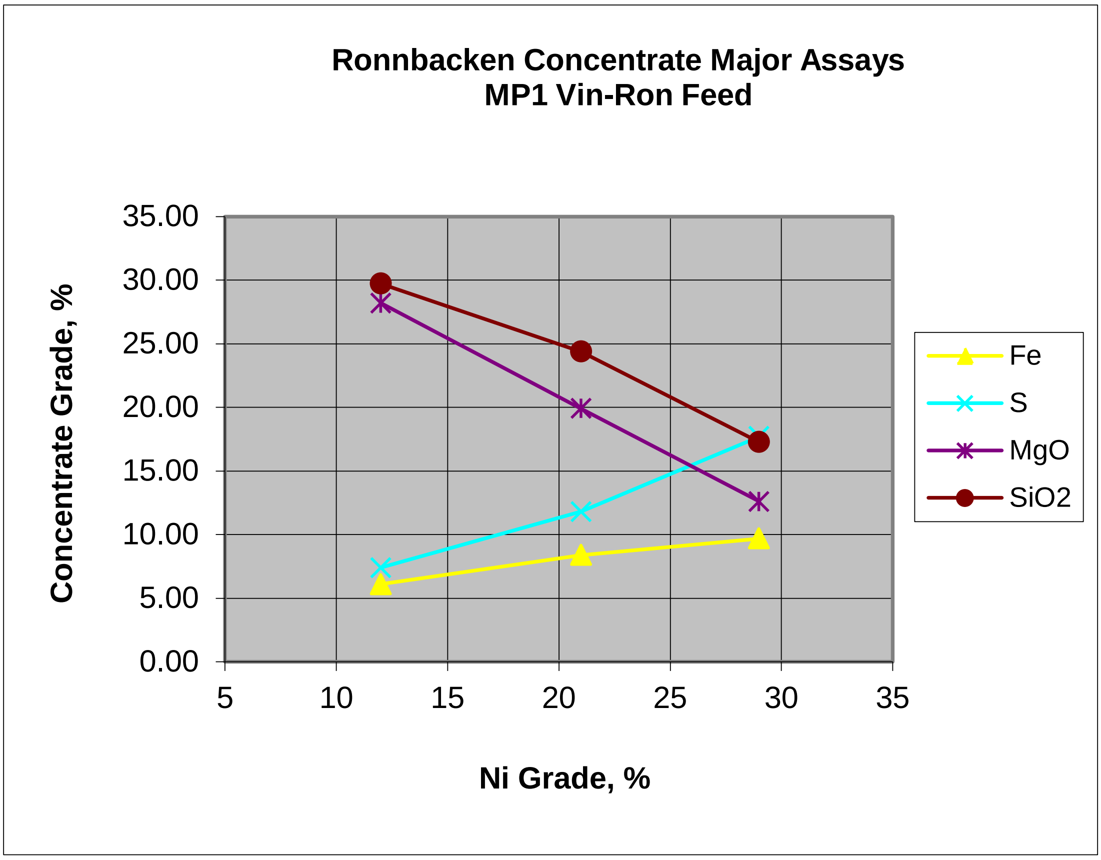

Figure 17-6: Major Assays in Concentrate from Minipilot Testing

Preliminary terms have been received from a number of smelters around the world. From these terms and as noted in previous reviews, the main concern will be with the high content of MgO in the concentrate, which can attract penalties, depending on the smelter. Based on the results to date it is likely that the concentrate would be penalized by some smelters.

Minor elements include Co, Al, Cu, Cr and Ca. The concentrate is unusually rich in cobalt compared to typical Ni concentrates. Trace elements of concern include arsenic and antimony.

The As content may attract a penalty depending upon the smelter treating the concentrate. It may be possible that the attractiveness of the high Ni:Fe ratio will offset potential disadvantages from MgO, SiO_2, and As. Impurities such as Pb and Zn are often associated with sulphide nickel deposits. At Ronnbacken these elements do not appear to be in sufficiently high concentration to require special treatment at downstream facilities. However, levels of Hg at <10 ppm in concentrate have been found, which may require removal downstream. This potential issue can be resolved by way of mercury removal technology, which is commercially available.

Other elements, such as Sb and Mn, have been noted at levels <500 ppm which should not incur significant commercial penalties and Cr has been noted at levels <1000 ppm which should also be manageable at downstream smelters.

Further testwork is planned to investigate the possibility of reducing these impurity levels.

Precious and platinum group metals are present at close to payable levels, depending on the grade of concentrate produced.

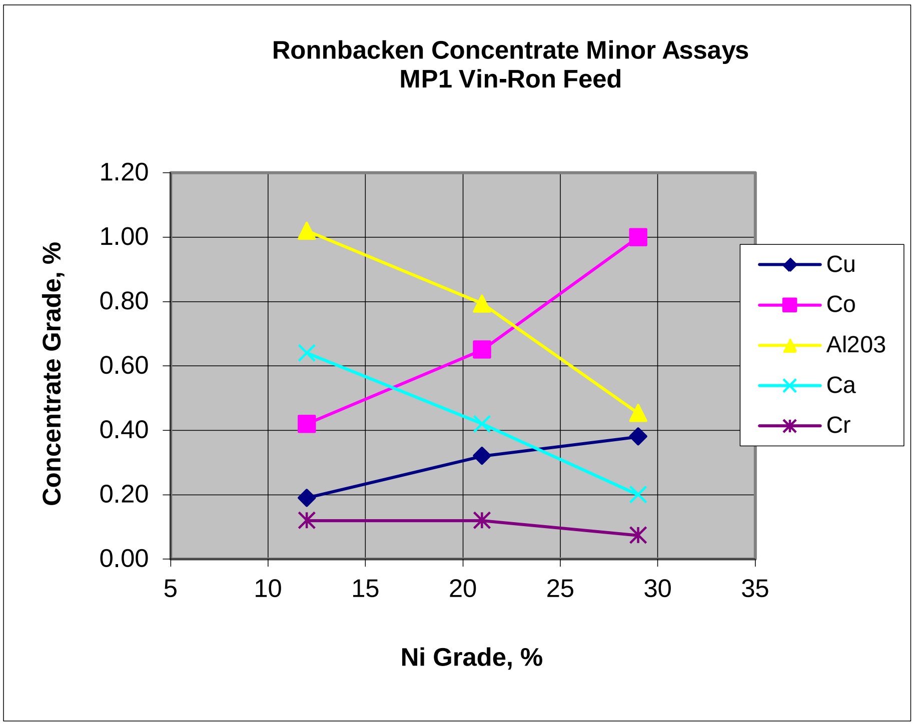

Figure 17-7: Minor Concentrate Assays from Minipilot Testing

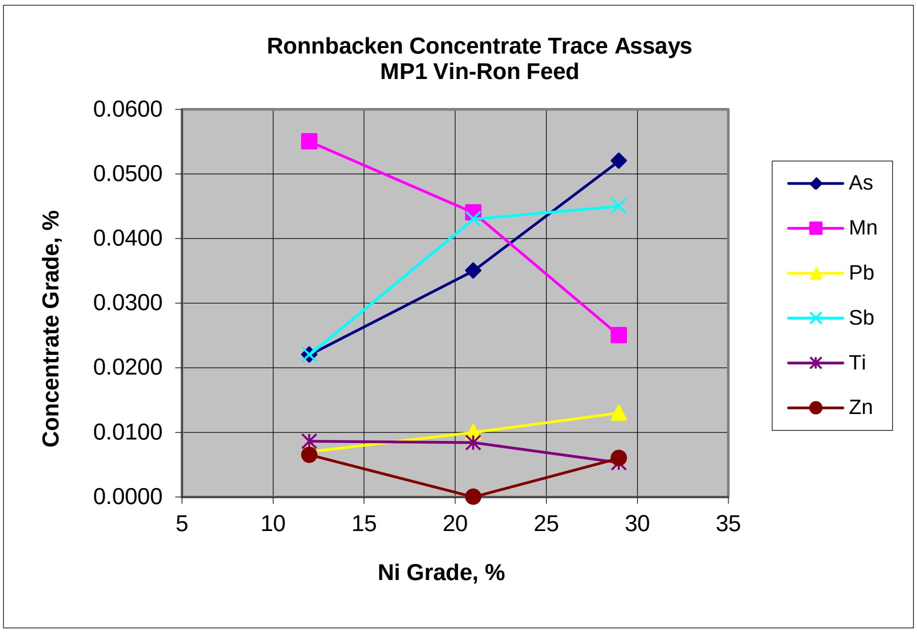

Figure 17-8: Concentrate Trace Element Assays from Minipilot Testing

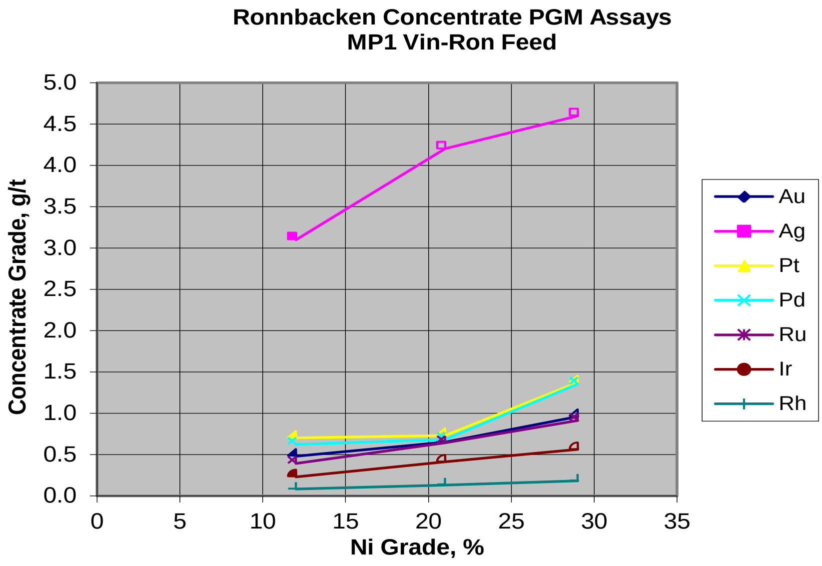

Figure 17-9: Concentrate PM/PGM Assays from Minipilot Testing

Magnetite concentrate

At the time of the report detailed magnetite concentrate analyses were not available. The XRF analysis of the final concentrate from test BBMS116 is presented in Table 17-4. The main impurity concern is chromium.

Table 17-4: Magnetite concentrate analysis (XRF)

	% w/w			% w/w
SiO2	3.08		Bi	0.004
TiO2	0.070		Te	0.000
Al2O3	0.24		Y	0.0000
Cr2O3	3.15		Nb	0.0000
V2O3	0.031		Mo	0.038
FeO	84.9		Sn	0.002
MnO	0.479		W	0.001
MgO	4.25		Cl	0.000
CaO	0.169		Th	0.0058
Rb2O	0.012		U	0.011
SrO	0.0000		Cs	0.004
BaO	0.005		La	0.003
Na2O	0.00		Ce	0.001
K2O	0.001		Ta	0.000
ZrO2	0.000		LOI	2.78
P2O5	0.024		Ga	0.0004
CO2			Si	1.44
OxSumm	97.40		Ti	0.042
Cu	0.008		Cr	2.16
Ni	0.56		V	0.021
Co	0.049		Fe	66.0
Zn	0.038		Mn	0.371
Pb	0.000		Mg	2.56
Ag	0.004		Ca	0.120
S	0.002		Ba	0.005
As	0.001		C	
Sb	0.010		Satmagan	84.74

17.1.4 Metallurgical Performance

In general, based on the results of the batch flotation and the mini pilot tests, together with the ORC simulations, the metallurgical performance of nickel in a commercial plant should be around 80% sulphide nickel recovery at 28% nickel grade. The metallurgical performance of the magnetite recovery circuit will be 90% magnetite recovery to a concentrate containing 66% Fe.

17.1.5 *Mineral Processing*

The currently assumed Rönnbäcken flow sheet consists of crushing, grinding, flotation, and dewatering steps typical of Scandinavian concentrator operations. The conceptual concentrator design, together with capital and operating cost estimates, has been prepared by Outotec AB (Sweden). The mill will have a capacity of 30 Mtpa or 3,750 tph, and will produce approximately 95,000 tpa of nickel concentrate at 28% Ni, and 1.6 Mtpa of magnetite concentrate at 66% Fe.

The mining schedule indicates that a blend of ore will be produced from the three ore deposits, predominantly Vinberget and Rönnbäcksnäset in the first four to five years and Rönnbäcksnäset and Sundsberget in subsequent years. The Vinberget ore has a slightly higher Ni grade than the other two ore bodies and consequently the plant feed grade will be approximately 0.116% Ni in years 1 to 5 and approximately 0.103% Ni in later years when Rönnbäcksnäset and Sundsberget ore is processed.

The plant will be split into two parallel streams. In each stream ore will be blended and crushed to less than 300 mm in a gyratory crusher and sent to a coarse ore stockpile. Ore will be fed from the stockpile by apron feeders onto belt conveyors to feed the two parallel grinding circuits each comprising a primary autogenous grinding mill (AG) and two secondary pebble mills. The pebble mills are fed directly from the discharge of the primary mills and pebbles will be extracted automatically from the AG mills as required. The concentrator will operate 8,000 hours per year (91.3% availability) at a feed rate of nominally 2,000 tph to each grinding line.

The grinding circuit product will be classified using hydrocyclones. Hydrocyclone underflow will gravitate back to the pebble mills and overflow will report by gravity to the rougher conditioning prior to flotation.

A dispersant will be added directly to the grinding mills, along with collector. Water will be further added to the conditioners to reach an optimal pulp density of approximately 30% solids. Sulphuric acid will be used to modify the pH. Additional reagents will be added to the flotation circuit as required.

The conditioned slurry will be pumped to rougher flotation consisting of several 500 m^3 flotation cells. Flotation tailings will be pumped to the tailings paste thickeners, located at the tailings pond site.

The rougher concentrate, approximately 5% to 10% by weight of the feed, will be pumped to the concentrate regrind circuit from where it will be pumped to a four stage cleaner flotation circuit consisting of 200 m^3 , 100 m^3, 50 m^3 and 10 m^3 flotation cells.

A final nickel concentrate grading approximately 28% Ni will be produced.

The final concentrate will be pumped to two concentrate thickeners from where it will be pumped to a single concentrate holding tank. The concentrate will be dewatered in two pressure filters. The filter cake will report to a concentrate loading system for bulk shipping by truck.

Tailings from the flotation circuit will be pumped to the magnetite recovery circuit. Magnetite recovery will be accomplished using low intensity magnetic separation (LIMS). Based on the testwork performed the magnetite rougher concentrate would be cleaned in additional magnetic separators prior to regrinding to nominally 80% passing 20 µm. The reground magnetite concentrate will be cleaned by further magnetic separation and the final concentrate dewatered in conventional thickeners and filters. A dispersant will be added to the regrind mill to assist concentrate cleaning. The final magnetite concentrate will contain 66% Fe. The requirement for concentrate drying or peletizing will be assessed during the PFS.

Final tailings, from the magnetite recovery circuit, will be pumped to a paste plant for creating paste for disposal at the tailings dam. The water will be mainly reclaimed in the thickeners and recycled back to the plant, primarily to the grinding circuit.

SRK considers the flowsheet to be a conventional flotation concentrator utilising the accepted Scandinavian autogenous style grinding circuit configuration. The use of large flotation cells has been shown to maintain or improve metallurgical performance, while reducing maintenance and power costs.

17.1.6 Processing Plant Location and Layout

The processing plant site location has been selected to be in close proximity to the Rönnbäcksnäset deposit and the planned tailings management facility (TMF).

The plant layout has been chosen to utilize the natural geography and topography of the area. The proposed location is suitable for the size of the plant and the proposed layout provides for a conventional flow of material from the crusher to the milling and gravity section, to the flotation plant, and finally to the concentrate handling and tailings handling parts of the plant.

The preliminary location selected is on Rönnbäcksnäset to the northwest of the Rönnbäcksnäset proposed pit, as illustrated in Figure 17-10.

An initial plant layout has been prepared by Outotec AB and is shown in Figure 17-11 and Figure 17-12.

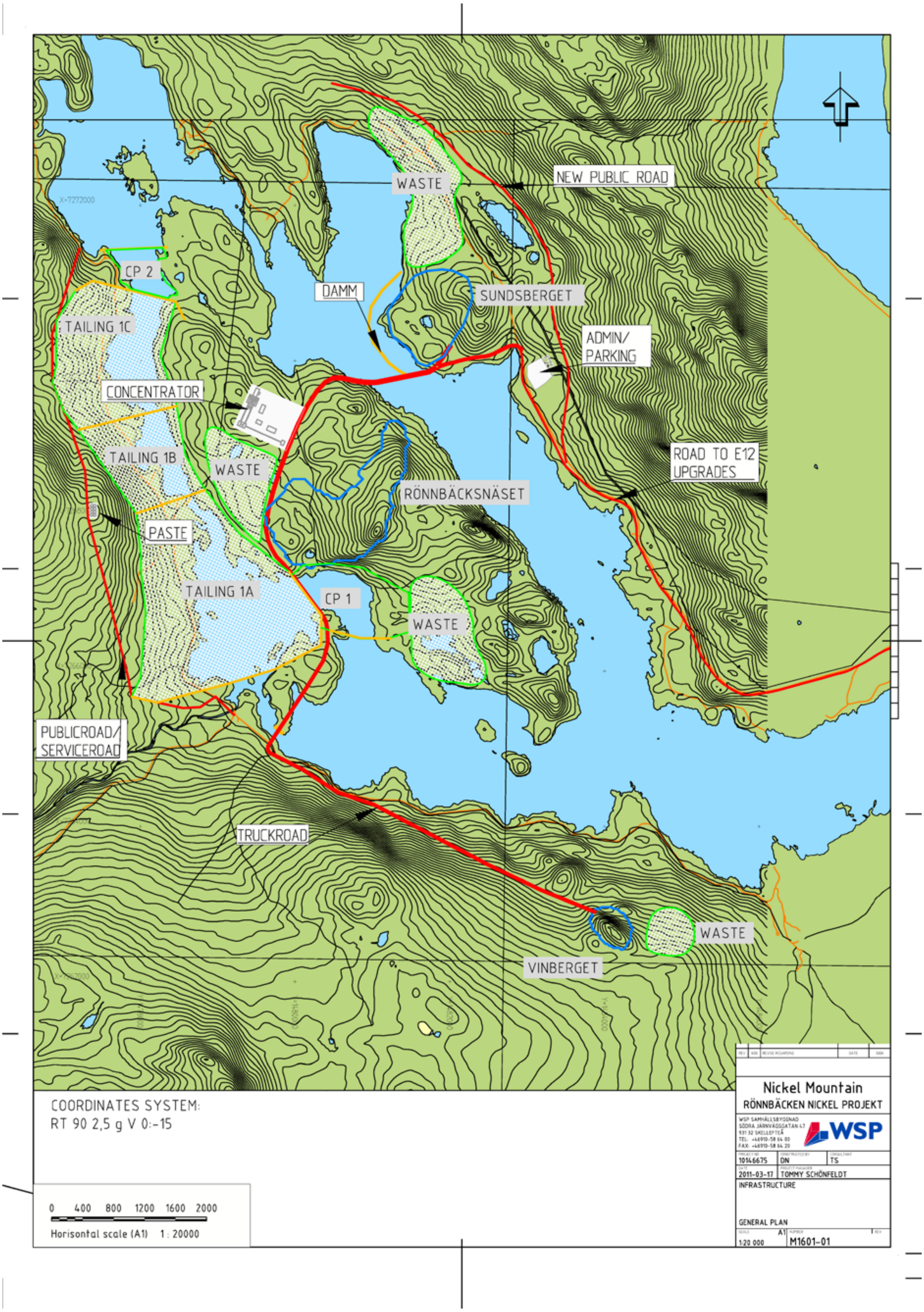

Figure 17-10: Proposed Site Layout

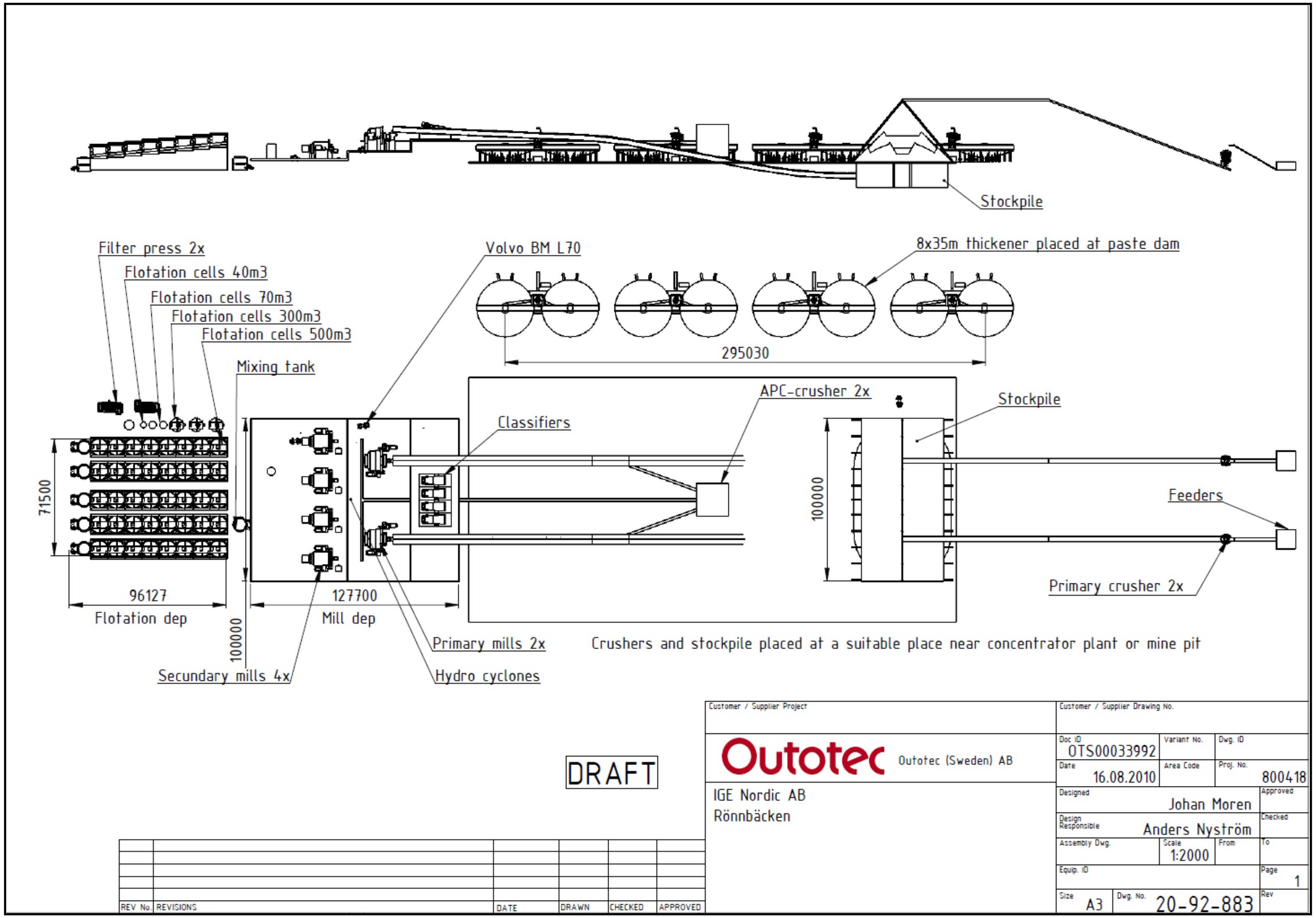

Figure 17-11: Process Plant Layout (Outotec Sweden AB)

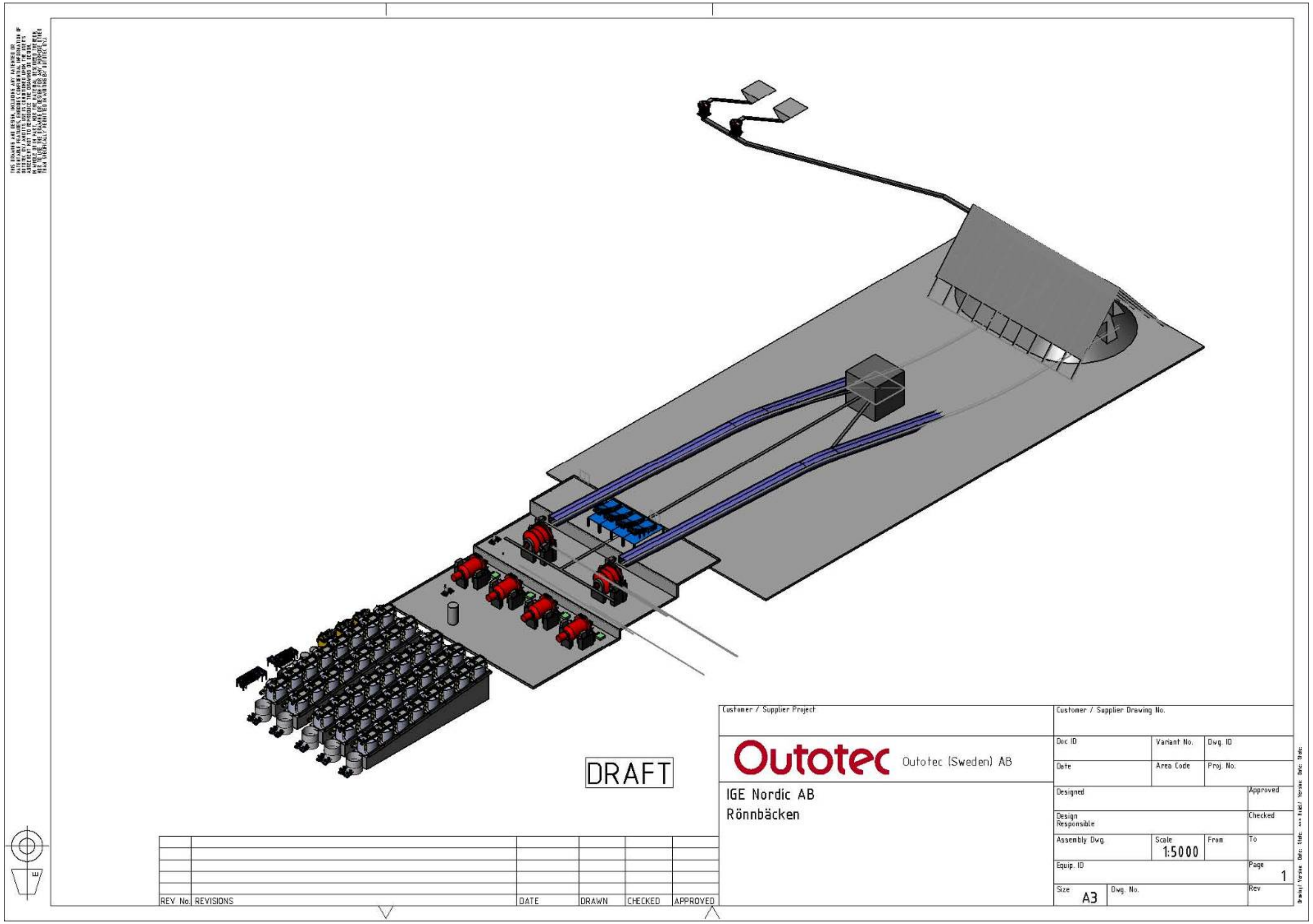

Figure 17-12: Process Plant Layout, Oblique View (Outotec Sweden AB)

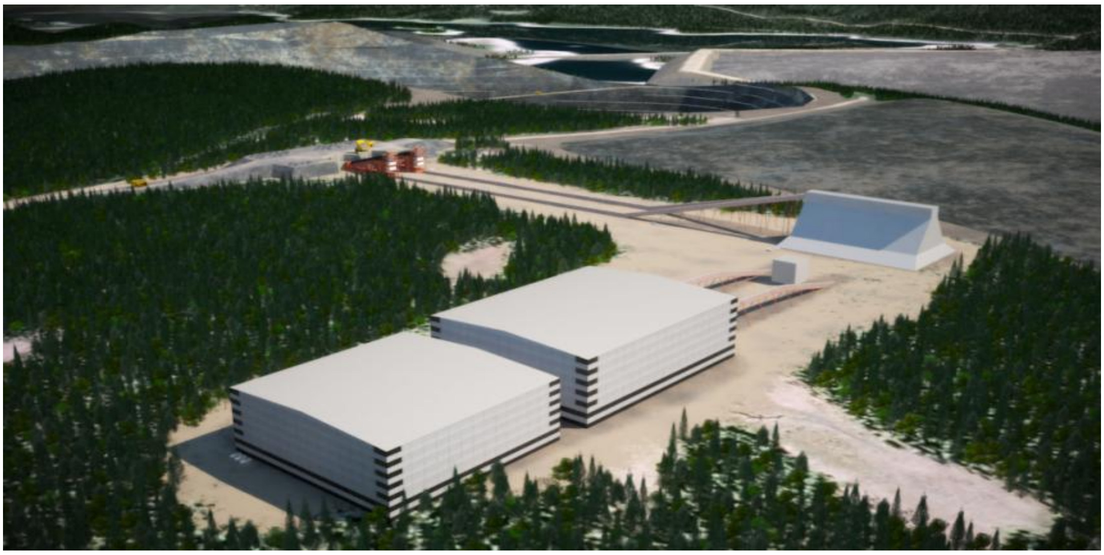

Figure 17-13: Visualisation of Process Plant, stockpile, crusher and in the background the wast rock, tailings and Rönnbäcksnäset open pit, looking south-east

Figure 17-14: Visualisation as above but looking towards north-west, crushers in front and the stockpile and concentrator plant in background

17.1.7 Infrastructure

A preliminary assessment of the external infrastructure, power supply, transport links, shipping facilities has been made and in SRK's opinion there are no major issues that would affect project development.

17.1.8 Capital Costs

The process plant, TMF (excluding dam construction) and associated infrastructure capital costs, based on the treatment of 30 Mtpa of Run of Mine (RoM) ore, have been estimated by Outotec Sweden AB, to a scoping study level of accuracy, using budget quotations for major equipment and in-house data base costs for other items, are presented in Table 17-5. A 25% contingency has been applied to all general costs, with 20% applied to specific equipment items for which quotes were received. SRK considers these costs to be acceptable based on the information available and will be refined during the PFS.

A nominal capital cost 87 M USD has been included for the magnetite recovery section of the plant based on information supplied by Outotec. No details of the make-up of this figure are available at the time of the report. SRK consider this cost to be acceptable based on the limited information available and will be refined during the PFS.

Table 17-5: Process Plant Capital Costs

Description	**Units**	**Total**
Processing directs		
Civil	**(USDM)**	87
Primary crusher	**(USDM)**	21
Stockpile	**(USDM)**	16
Ore feed	**(USDM)**	8
Grinding area	**(USDM)**	153
Flotation area	**(USDM)**	35
Dewatering area	**(USDM)**	4
Reagent area	**(USDM)**	6
Tailings area	**(USDM)**	63
Piping	**(USDM)**	39
Electrical	**(USDM)**	113
Erection	**(USDM)**	58
Air and hydraulics	**(USDM)**	6
Sub-total	**(USDM)**	**608**
Indirects		
Commissioning and training	**(USDM)**	3
Indirects: Freight, First Fills, Spares, Ins	**(USDM)**	20
Project Management	**(USDM)**	22
Design	**(USDM)**	52
Owner's Costs	**(USDM)**	46
Sub-total	**(USDM)**	**142**
Contingency		
Contingency - Equipment (20%)	**(USDM)**	36
Contingency - General (25%)	**(USDM)**	176
Sub-total	**(USDM)**	**212**
Total capital costs_Flotation concentrator	**(USDM)**	**962**
Total capital costs_Magnetite concentrator	**(USDM)**	**87**

17.1.9 Operating Costs

The processing operating costs for the plant, based on the treatment of 30 Mtpa of RoM ore, have been estimated by Outotec Sweden AB. SRK concurs with this estimate. The operating cost summary is presented in Table 17-6.

The estimate power consumption for the concentrator is 1190 GW for a 30 Mtpa RoM feed. This corresponds to approximately 39.7 kWh/tonne Rom. The unit power cost included in the TEM is 0.38 SEK/kWh.

Table 17-6: Processing Operating Costs (30 Mtpa)

Description	Units	
Total variable costs	(USD / t)	3.95
Total fixed costs	(USD / t)	0.33
Total tailings op costs	(USD / t)	0.14
Total Processing Op Costs (Ni-Co & Fe)	(USD / t)	4.41

17.1.10 Concentrate Grade

Most nickel concentrates are predominantly pentlandite and range in nickel content from 5% to 15% nickel. The mineralogy of the Rönnbäcken ores allows nickel concentrate grades to be relatively high grade in comparison with other operations (Figure 17-15). This nickel concentrate grade is achievable through conventional grinding and flotation. Most sulphide nickel ores (Sudbury, Norilsk, Jinchuan) contain a mixture of pentlandite and iron sulphides - typically pyrite and pyrrhotite - which form finely textured mineral aggregates that can be difficult to liberate. Pentlandite needs to be separated from gangue, pyrite and nickel containing pyrrhotite which limits the typical nickel grade of concentrates to typically 7% to 15%.

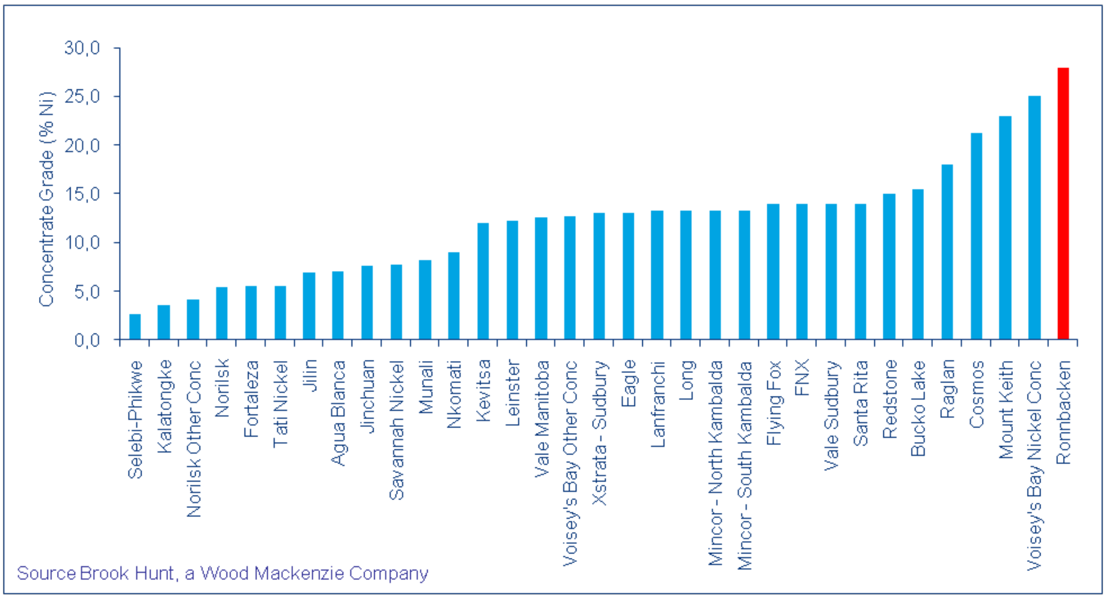

Figure 17-15: Comparison of Rönnbäcken concentrate grade with other operations

The Rönnbäcken serpentinites are distinct from other well-known nickel sulphide ores, due to the presence of nickel-rich sulphide minerals, specifically; heazlewoodite (71-75% nickel), pentlandite of 35-44% nickel (typically 32-35% nickel) and minor amounts of millerite (61-65% nickel). In addition, iron sulphides, such as pyrite and pyrrhotite, are almost completely absent.

18 PROJECT INFRASTRUCTURE

Overview

The technical and cost assumption for Project infrastructure were prepared for the Company by Tommy Schönfeldt of WSP Environmental, Skellefteå and summarized in the spreadsheet "*Nickel Mountain kalkyl infrastruktur rev 20110208".* In developing the technical and cost assumptions for the Project, SRK understands that WSP has drawn on this recent experience and also information provided to WSP by the Company, particularly with regard to exising public roads.

As part of this work, WSP prepared a high-level design for roads, buildings, electrical infrastructure. Costs associated with contractor barracks were assumed by WSP to be part of the Owner's Cost and as such were excluded from design and cost estimation.

Site layout is illustrated in Figure 17-10 above, and a summary of infrastructure capital cost estimates are presented in Table 18-1 below.

Table 18-2 presents a summary of shared infrastructure between mining operations at each of the three proposed pits.

Table 18-1: Infrastructure capital cost estimate prepared by WSP, for roads, buildings, electrical the industrial pad and Sundsberget pit dam wall

Infrastructure Item	Estimated Capital Cost (SEKM)
Public Roads	109
Mine Roads	256
Temporary Roads	13
Industrial Area	33
Sundsberget Pit Wall	189
Fence	7
Buildings	389.5
Electrical	68
Total	1 064.5
Total USD Equivalent ***(SEK:USD = 8)***	**133.1**

Contingencies of between 20% and 25 % have been applied to infrasture capital costs. SRK consider these assumptions to be reasonable and have incorporated these into the TEM.

WSP has not assumed any costs for transport of mine waste material and overburden for construction of infrastructure. Unit mining operating costs for haulage to the waste dump have been applied to the total tonnage of waste rock mined at the operation and these costs are assumed to cover transport to each area of construction as and when required. SRK notes that costs of waste material transport for construction may be higher than the cost of transport to the waste rock dump, given necessary selection and compaction of material. In general however, this may be a reasonable assumption for this level of study and provided that sufficient material of the desired quality can be sourced from on-site mining activities.

Table 18-2: Shared Infrastructure Summary

	Rönnbäcksnäset	Vinberget	Sundsberget
Mill	✓	✓	✓
TMF	✓	✓	✓
Rönnbäcksnäset WRD	✓		
Vinberget WRD		✓	
Sundsberget WRD			✓
South access road – site	✓	✓	
North access road – site	✓	✓	✓
Central administration	✓	✓	✓
Raw & excess water pipelines	✓	✓	✓
Dewatering facility	✓	✓	✓
Product loading station	✓	✓	✓
Product transport infrastructure	✓	✓	✓

Public Roads

During operation, nickel concentrate production is expected to average around 90 ktpa. In accordance with currently regulations, a 40 t payload is assumed, which implies between six and 8 round-trips per day.

Road traffic related to transport of consumables, chemicals (including acid) and spare parts to the mine is assumed to be in the range to 14 to 20 trucks per day. Transport of fuel for is expected to be in the range one to two trucks per day.

In total, the Company estimates that between 21 to 30 trucks per day may be required to access the Project site, although for certain periods during the construction years, the volume of traffic to and from site may be greater.

At present, no consideration has been given to transport of waste material for construction either from or to site.

Access to the Project site is currently along the east side of Lake Gardiken, via a narrow public gravel road. The road connects to highway E12 at its northern and southern end. From the proposed Rönnbäcken industrial area to this local road the distance to highway E12 is 14 km in a southerly direction and 31 km in a northerly direction.

To meet the demands of heavy traffic during mining and construction the road going south, must be upgrade to meet year around two way road standard “BK1”. The gravel road going north, must also be upgraded to meet year around personal commuting traffic and small truck transports from the airport and local business in Tärnaby/Hemavan. The Swedish Transport Administration (Trafikverket Region Nord) typically organises construction and upgrading of public roads. The costs for this however, are typically shared between the Company and the State after negotiations. Estimated costs for upgrading public roads are estimated at SEK109M.

Part of the existing gravel road connection to highway E12 passes over the Ajaure hydropower dam. The dam wall is built to meet BK1 standard. In addition, the dam was reinforced during to support heavy traffic during construction of the nearby Gejmån hydropower dam. The Company has not allowed for any provision for upgrades to the public road at the Ajaure hydropower dam.

During operation, the volume of heavy traffic is estimated at between 17 and 25 trucks per day, including concentrate transport and mine/plant comsumables.

Mine Roads

A road profile with a lane width of 21 m and 2 x 5 m dikes has been adopted, with waste rock of an average height of 3.0 m, with a surface of 30 cm crushed rock at 0/40 mm. The road embankment across lake Gardiken between Sundsberget and Rönnbäcksnäset includes a number of culverts, designed to withstand a flow rate of 325 m^3/s.

Industrial Area Pad

The area of the industrial pad at the concentrator and workshops is assumed to be 15 ha, of which 7.5 ha will be tarmac paved. The area for the changing rooms and offices is assumed to be 2.5 ha, of which 1 ha is assumed to be tarmac paved.

WSP has assumed that 50% of the total costs associated with establishing the industrial pad are included as capital costs in the construction of the concentrator.

Sundsberget Pit Wall Dam

The current life of mine (LoM) plan assumes that mining at the south end of Sundsberget will be carried out within an area currently covered within and below the seasonal variation of Lake Gardiken. As such, construction of a retaining dam wall will be necessary. The construction of the dam may require a cofferdam prior to initiate the construction of the dam, depending on the level of Lake Gardiken during the construction period.

It is assumed that the dam itself will compose a core of sheet piling, with an embankment of waste rock and overburden. The dam has an average height of about 27 m and a length of approximately 2 km. The Company has estimated the cost of construction of this dam to be USD24M. SRK notes that this cost will be strongly dependent on prevailing ground conditions and that this capital cost may increase following further ground investigation.

Buildings

WSP has assumed a total work force, including contractors, of between 250 and 300 people. Buildings include provision for dressing room, office, restaurant, mechanical and electrical workshop, truck workshops, heated and cold storage, a sewage treatment plant and recycling facilities are also included.

Capital cost assumptions include construction, soil and foundation engineering, electrical, plumbing and HVAC installations.

Electrical Power Supply

WSP has considered a complete electrical power grid system for mining and the processing plant, assuming a power demand of 160 MW. Specifically, capital cost assumptions include the incoming line and receiving main station 220 V (four transformers), all electrical grid and substations to the drive systems for crushers, mills, flotation, tailings pumping, fresh water and dewatering for the Project open pits.

18.1 Tailings Management

The current mine plan for the three pits estimates a total ore production of 528 Mt which will essentially become tailings once processed. Should a magnetite concentrate also be produced at the operation, the total tonnage of tailings may be closer to 500 Mt. Figure 17-1 provides a conceptual illustration of the Project layout during operation.

Figure 18-1: Conceptual layout of the Project during operation

The tailings will be subject to a thickening process to produce a bulk dry density estimated at 1550 kg/m3 once deposited. This density will therefore produce a tailings volume of up to 340 Mm3 that will be stored in the tailings impoundment. The proposed design for the TMF is to construct a cluster of three cells that will require the construction of four dams located west of the Rönnbäcksnäset island in Lake Gardiken (Figure 18-2).

Deposition of the tailings will be achieved by spigotting the tailings over the TMF to maximise the storage capacity. Two clarification ponds will be constructed at both ends of the TMF and are identified as CP-1 and CP-2 in Figure 18-2.

The two internal tailings dams for cell 1B will have an estimated average height of 25 m and lengths of 700 and 1,100 m. The south dam for cell 1A will have a dam with an average height estimated at 40 m and a length of 2,800 m. The dam at the north end of Cell 1C will be about 800 m long and have an average height of 55 m. The two internal dams will eventually be covered by tailings once Cell 1C is in operation.

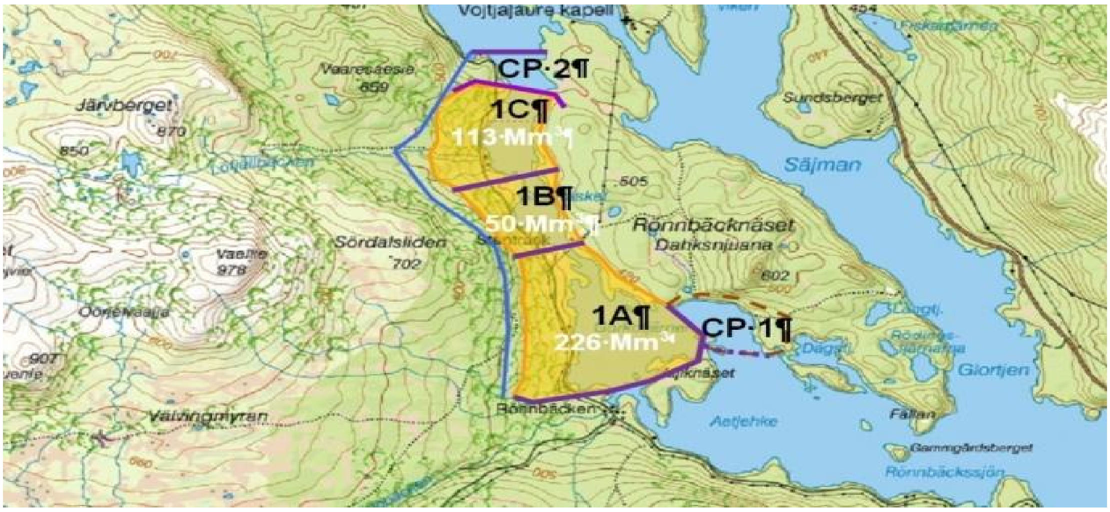

Figure 18-2: Proposed design layout of the Rönnbäcken TMF

The proposed dam design consists of a pervious rockfill dam that will not restrict the movement of pore water between the deposited tailings and the dam. It is designed to retain the tailings particles and act as a drain. The dams will include a filter zone on the tailings side for restricting the movement of fine particles with the groundwater flow. The current approach is to import filter material from a source located roughly 30 km from the mine site. Figure 18-3 below presents a cross-section of the proposed dam design.

Irrespective of siting and mitigation measures, there will be a surplus of water in and on the impounded tailings. The runoff of this water will either flow through the drainage system or the decanting outlet or seep through the dam. Some of that water will reach the downstream lake and river system. To minimize the transport of particles via surface runoff, the decanted water will pass a clarification pond where most of the suspended particles settle.

Pore water from the tailings will eventually be released into the lake via the dam drainage system due to lake level fluctuations, excess pore water from the tailings deposition and from precipitation. The quantity and quality of that water is difficult to estimate at this stage because of limited information. There are plans however to investigate the concentrations of particles and potential contaminants in these effluent waters as well as the annual loads of them to the river system. Additionally, seepage analysis through the dams and geochemical testing such as leaching tests are scheduled for the subsequent phase of the project. This will be an important background for the coming EIA study that accompanies the permit applications.

During the construction period of the tailings impoundment, there will be periods when the external, regulated water level will be higher than the surface of the tailings immediately close to the dam. Lake water will infiltrate the tailings impoundment during that period and come in contact with the tailings in that situation. The adjustment of the water level inside the impoundment would control the hydraulic gradient across the dam, but it will not prevent the water inside the impoundment to come in contact with the tailings. The potential for contamination will be dependent on the properties of the tailings. As mentioned above, it is difficult at this stage to estimate the level of contamination as additional characterisation is required before making predictions. The possible transport of suspended solids and the risk of contaminated water from the tailings should therefore be assessed and the consequences of mitigating measures on the operation costs be evaluated.

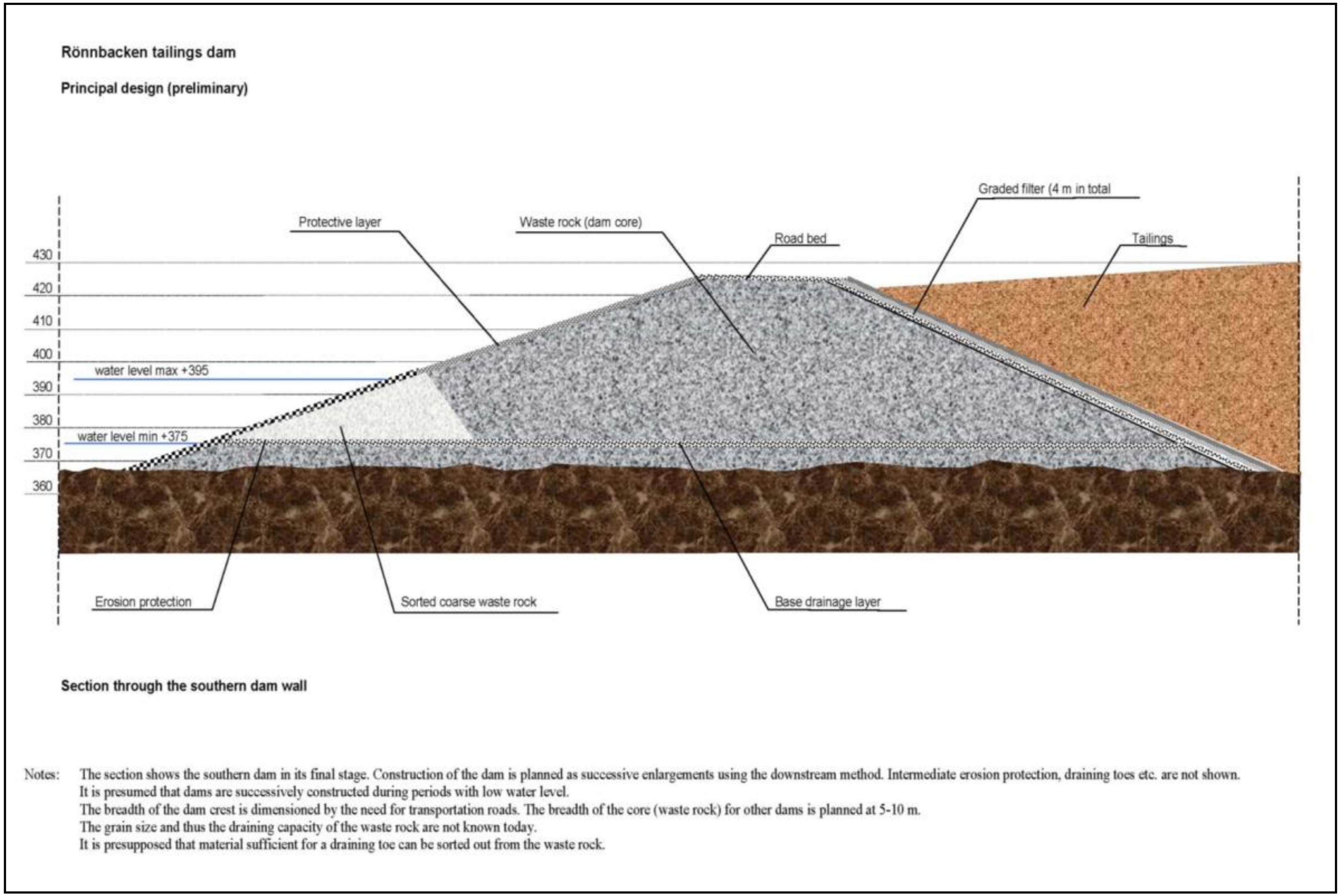

Figure 18-3: Section of the proposed Rönnbäcken TMF dam construction design

The two clarification ponds will have to be designed as water retaining dams in order to be able to control the water turn over rate and the internal water level and to retain suspended solids from surface runoff from the deposited tailings. They will also require a water barrier such as a liner or a low permeability barrier within the dam and potentially some grouting in the bedrock. These dams will have to be designed to cover the 20 m high water fluctuation in the lake plus the minimum water column along the dam alignments. As with the other dams, cofferdams may be required for construction at least for some parts of the dam lengths. The construction of such dams will also require the inclusion of a low permeability component to retain the water. Construction control will also be greater for such dams because of their water retaining requirements. Information provided by the Company’s design team indicates that the cost to construct the two clarification ponds would be included in the cost to construct the tailings dam. SRK estimates that construction cost for the clarification ponds would typically be in order of USD15M or more depending on the site conditions. This capital cost provision has been included in the TEM.

The details of the tailings dams and their construction are at present limited. The construction of the dams will require cofferdams to isolate the foundation elements from the lake water. Cofferdams are not mentioned in the reviewed documents. The dam and cofferdam materials could represents a combined volume in the order of 30 Mm^3 based on the schematic details provided for review.

Capital Cost Estimate for TMF Construction

The capital costs provided by the Company estimates tailings distribution system at USD92M (see Table 17-5 above), construction costs for tailings dam and clarification ponds of USD107M and an additional USD6M for the excavation and stockpiling of the overburden material from beneath the dams.

In the context of the level of detail provided, SRK estimates the cost to construct the tailings dams at about USD150M, excluding the distribution system for the tailings and excluding construction costs for the clarification ponds.SRK’s cost assumptions for this have been incorporated into the TEM and profiled by applying 20% in the pre-production years and following this, 10% bi-annually until Year 16.

SRK notes that the Company has assumed that the costs of placing waste rock at the dams for dam construction will incur the same cost as hauling the waste rock to the dumps. Hauling waste rock to the dump is usually limited to loading, hauling and dumping. Dam construction will usually require selecting the material and involves spreading and compacting the material once hauled and dumped. Travel conditions (road width, turning area, slopes, surface conditions, etc) and distances will also contribute to increase the unit cost of waste rock material as dam material. Dam construction will require greater efforts and thus present a higher comparable cost than hauling waste rock material to the waste rock dumps.

Sub-aerial deposition of thickened tailings has been the selected process as a measure to increase the density of the tailings once deposited. Thicken tailings has the benefit of reducing the storage requirements, the time for consolidation and segregation of the tailings particles. Total operating cost estimates of USD0.14/t, for preparation and pumping of tailings were prepared by Outotec. SRK considers these costs to be reasonable.

Recirculation of the pore water from the TMF could introduce accumulation of contaminants over time. The accumulated contaminant will eventually reach the lake water given the pervious nature of the tailings dams. High contaminant levels could require treatment at the clarification ponds, thus potentially adding to operating and/or closure costs. For instance, a water treatment could eventually be required for removing those contaminants.

The location of the TMF is inside a fresh water lake that was created by a hydro-electric dam. Although the lake is man-made, it remains that it is clean water that currently supports a fish population which is another issue that will need to be addressed. This fact indicates that the investment costs should include costs for compensation for loss of values of fishing, as discussed in later sections of this report.

Summary and SRK Comments

In summary, SRK considers that the construction costs for the tailings dams and the clarification ponds have been underestimated. The cost estimate does not itemise separately the dams for the clarification ponds and the cofferdams that would be required for dam construction. Additionally, the cost estimate is based on the assumption that waste rock material used for dam construction will have the same cost as waste rock disposal in the dumps. Accordingly, for the purpose of the TEM presented in the report, SRK has adjusted the capital cost estimate for dam construction (USD165M) to include a 15 MUSD provision for clarification pond construction and USD150M for tailings dam construction.

Operating cost assumptions for tailings management are considered to be appropriate.

Further investigation is required to confirm the acid generating and metal leach potential of the tailings. This presents a particular area of risk given the location of the TMF in Lake Gardiken and a permeable dam wall.

SRK notes that the Swedish authority responsible for granting the Project an exploitation concession, the Mining Inspectorate, has issued the required permits and/or approvals for the Project to proceed to the next phase of permitting based on the environmental impact assessments (MKB) submitted with the exploitation concession applications for Rönnbäcksnäset and Vinberget.

18.2 Acid Rock Drainage and Metal Leaching (ARDML)

Background

Preliminary acid rock drainage and metal leaching studies of tailings and waste rock material at Rönnbäcken have been carried out by Tom Lundgren of Ambiental Ltd.

It is stated in the Ambiental (2011) memorandum that:

‘*The process of permitting the mining of the Rönnbäcken ore deposits according to the Environmental Act will to a large extent be based on evaluations of the risk that waste deposits from the operation will generate acid leachates that contaminate the groundwater or the downstream surface waters with trace elements’. These evaluations are made at three stages where the first stage results in a classification of the waste in “inert” or “not inert waste”. This is conducted with respect to the potential to generate acid and to hold potentially harmful substances. If classified as “inert” the waste is judged not to need any more detailed investigations with regard to its environmental properties.*

At this stage, the “acid generating” aspect is based entirely on the concentration of sulfur. If it is lower than 0.1 % by weight it is judged not to be able to produce acid irrespective of its *content of neutralizing agents. If the percentage of sulfur is higher than 1 %, special studies have to be carried out in order to show if and how much acid will be produced and what the quality is of the resulting leachate in terms of dissolved, potentially hazardous elements. If the sulfur concentration is between 0.1 and 1.0 %, the internal ability of the waste to neutralize the acid shall be considered. This is accomplished by a standardized procedure called acid base accounting (ABA) where the potential to produce acid is calculated from the sulfur concentration and the neutralizing capacity is measured by titration with acid. The procedure is defined and described in the standard prEN 15875. If the ratio between the neutralizing potential and the acid potential in the sample is higher than 3 (and the sulfur content < 1%), the waste is classified as "inert".*

If the waste is not classified as "inert", a permit for disposal of the waste (Environmental Protection Act) must be based on a prognosis of the amount of acid and potentially hazardous elements that will be produced from the deposit. Such a prognosis should be based on kinetic tests such as column leaching tests and humidity cell tests'.

The exploration surveys of the Rönnbäcken ore deposits have shown that the sulfur content of the waste rock is variable, between 0.38 and 0.02%, but that the tailings is well below 0.1 %. In conjunction with the ABA assessment the tailings will be classified as "inert" waste and it is stated that no kinetic tests are needed for this waste type. However, results from the analysis of the tailings decant water indicates high levels of sulfate in solution, >1,300 mg/L, that could imply the potential oxidation of any sulfides present yet from the metallurgical testing the ore has a high buffering pH, that requires pH correction during the processing of the ore. This pH correction was undertaken with sulfuric acid, reported at 12 kg/t of ore, and this addition could explain the excess of sulfate ions observed. In more recent process tests, this consumption of sulfuric acid has been reduced to less than 2 kg/t of ore. SRK would therefore recommend that further analysis of the new tailings decant water is undertaken to confirm the material's 'inert' classification.

For the other waste rocks it is less clear as to their classification due to the minimal sampling and characterisation undertaken to date.

The samples

The sampling procedure is also described in Ambiental (2011) as follows. Key aspects are underlined:

'The *samples used for the ABA-tests are master (composite) samples collected from rock cores taken at the exploration drillings for the Vinberget and Rönnbäcksnäset nickel deposits. These cores were taken to delineate the ore bodies. Hence, they were taken in the mineralized zone (ore) and in close vicinity to the ore (waste rock). Therefore these rock cores do not represent* ***<u>the whole rock mass that will be excavated if the ore body will be exploited</u>****. The extent of the waste rock volume will be defined at a later stage of the project when supplementary studies of the ARD potential have to be carried out.*

The program for sampling was established with the objective to compose a set of samples that are representative for those parts of the waste rock types that were found in the exploration rock cores with the highest sulfur content of the two ore deposits Vinberget and Rönnbäcksnäset. Based on core log data the exploration geologists identified four main types of rock types that are ***<u>present in the immediate vicinity of the ore bodies</u>*** *at Vinberget and Rönnbäcksnäset. They were named "Mafic" (felsic-mafic metavulcanite, pyroxenite), "Sed"* *(phyllitic schist of the regular type for the area), "G-Sed" (graphite schist) and "K-sed" (chlorite schist).* ***The exploration geologists made a selection of cores that represent the four "high-sulfur" main types of waste rock at both exploration areas and from these cores 10 subsamples were randomly selected for each of the four master samples representing the respective waste rock type.*** *The master samples were delivered to the Swedish Geotechnical Institute for acid-base-accounting according to the standard prEN 15875.*

The above extract describes the sample selection procedure for Rönnbäcken. It is clear that the assessment is still in the early stages of development and further work needs to be completed to define the waste rock.

Compilation of the acid-base accounting results

The results of the acid-base-accounting were reported by the Swedish Geotechnical Institute. Table 18-3 is a compilation of that report, which also includes acid-base-accounting on a sample of tailings that was retrieved from a mini-pilot test on thickening of tailings from the Vinberget and Rönnbäcksnäset ore resources.

The two samples having a neutralization potential ratio (NPR) that is lower than the stipulated value 3 for "inert" waste are marked with red.

Table 18-3: Compilation of the acid-base-accounting on the sample of tailings and the four master samples of waste rock

Sample	S-content (%)	AP (mole H+/kg)	NP (mole H+/kg)	NNP (mole H+/kg)	NPR
Thickened tailings	0.056	0.035	1.00	0.96	28
Mafic waste rock "Mafic"	0,31	0.19	0.28	0.089	1.5
Sedimentary waste rock "Sed"	0,32	0.20	1.85	1.65	9.4
Sedimentary waste rock "G-Sed"	0,38	0.24	0.41	0.17	1.7
Sedimentary waste rock "K-sed"	0,02	0.015	0.85	0.83	56

Conclusions

As shown in Table 18-3, the tailings have a very low content of sulfur and a relatively high neutralization capacity. The neutralization potential ratio (NPR) of 28 is high enough to ensure that ARD will not be generated from this waste. The same is valid for the rock waste type called "K-sed" which must be designated as having a very low potential to form acid. However, the sulfur content of the remaining three "worst case" waste rock types is substantially higher, in the order of 0.3-0.4 %.

The worst case sulfur concentrations are considered to be low or moderately high but high enough to be subjected to further evaluation according to the regulation. One of these, the ordinary schist ("Sed") has a high enough neutralization potential to be classified as "inert waste". In order to be prepared for possible ARD problems, the other two waste rock types should be subject to further studies using kinetic tests.

As was emphasized in the introduction, this study is preliminary and indicative. Further sampling has to be accomplished of both tailings and waste rock when more specified information is available regarding the waste materials that actually will be generated. A specific programme should be developed in order to achieve a high degree of representation of the samples that are chosen for future studies.

Recommendations

On the basis of the studies undertaken and rerpot by Ambiental (2011), SRK would recommend the following work to be undertaken:

- A full programme of sampling and testing. The sampling program should take advantage of existing exploration drillhole assay data available in order to determine the variability of sulfur within the waste rock units. In addition, sampling should be undertaken from geotechnical drilling in the waste rock in order to fully characterise the bulk of the waste. Sample testing should include the following:
  - Whole Rock Assay (through multi acid digestion or similar);
  - Acid Base Accounting (in accordance with prEN15873);
  - two stage deionised water leach testing (in accordance with EN12457-3);
  - Net Acid Generation (NAG) testing (in accordance with EGI, 2002);
  - Mineralogical Assessment; and
  - humidity cell testwork (if deemed necessary).
- Kinetic characterisation testing of any materials subsequently not classified as inert to determine their long term stability.

18.3 Water Management

18.3.1 Hydrological Setting

The project area is located in low mountain terrain with an average precipitation of around 750 mm/year and up to 2.7 mm/day or 97 mm/month for July when the most precipitation occurs (FAO, 2011). Rainfall events are mainly frontal or orographic. The annual snow-melt, which generally occurs in April or May, is also an important consideration when sizing surface water control infrastructure.

No site specific groundwater data has yet been collected at Rönnbäcken and so the current hydrogeological interpretation of the site relies on the current geological understanding, publically available data and SRK's experience in similar deposits in Scandinavia and other areas of the world. Further work will be required at the next stage of investigation in order to better define the hydrogeological understanding and make an assessment of the potential implications on pit water management, water supply and water and tailings dams.

The geology of the Rönnbäcken deposits and surrounding area are discussed in detail in previous sections of this report. Broadly, geology at Rönnbäcken comprises mainly greenschist and amphibolite metamorphic facies, phyllites, metavolcanics, metasediments and serpentinised ultramafic rocks, in which the nickel sulphides are hosted. Matrix permeability in the majority of these rocks is likely to be on the whole low and therefore groundwater flow in the bedrock will be almost exclusively limited to fractures associated with faults and jointing, which in turn is likely to be mainly structurally controlled although lithological differences will have an influence on the nature of fracture properties and development.

The basement rocks in the area belong to two significant nappe complexes, formed during a continental collision. As in the rest of the Caledonides, the bedrock of this region is therefore structurally complex, with a significant amount of large-scale folding and faulting. Brittle structures have the potential to be highly transmissive and it is the nature and distribution of local and regional brittle structures that is likely to be the dominant controlling factor for groundwater flow in the area. The basement rock is overlain by a generally thin (generally <5 m but <20 m in some isolated locations) cover of overburden sediments. The permeability of these sediments is likely to be variable, but they are unlikely to be significant in terms of groundwater inflows to mining operations due to their limited thickness. Hydraulic properties of the overburden and bedrock formations will require further investigation at the next stage of investigation including structural interpretation, as discussed in below.

Some initial surface water catchment analysis has been undertaken by the Company as part of initial options studies for the location of the proposed TMF. This study showed catchment areas for surface water courses in the vicinity of the proposed deposits to be generally small (<50 km^2). However, the steep topography and relatively impermeable basement in many areas may lead to rapid run-off during snow-melt or rainfall events leading to temporarily high (flashy) surface water flows.

18.3.2 Pit water management

The deposits at Rönnbäcken are adjacent to, or surrounded by in the case of Rönnbäcksnäset, Lake Gardiken which has formed behind the Gardiken hydro-electric power station dam, approximately 20 km to the east. Furthermore, it is understood that all three proposed pits will progress below the level of the lake. Thus the lake will provide a constant source of water for pit inflows (constant head boundary) should a permeable pathway exist between the pit and the lake. Although the permeability of the unfractured rockmass surrounding the proposed pits is likely to be low, discrete fractures provide the potential for locally significant inflows to the pit, which could be sudden and impact on mining operations.

The role of groundwater on pit slope stability is as yet unknown.

Significant further investigation is therefore required in order to:

- estimate groundwater inflows to the pits, including the potential for a hydraulic pathway between the lake and the pits, and to derive a cost effective solution for control of these inflows;
- and evaluate potential pore water pressures in the pit slopes and the feasibility of any depressurisation required.

Such an investigation would require structural analysis and interpretation as well as an investigative hydrogeological field programme, ideally including discrete interval testing such as double packer or spinner tests. These structural and hydrogeological investigations would be best undertaken in tandem with the geotechnical field programme for reasons of cost effectiveness and data sharing.

Site-specific climatological data will be required as the project progresses as the mountainous terrain may mean that some climatic variables differ between site and the nearest long-term weather station at Hemavan. It is therefore recommendable that at least one weather station be installed on-site as soon as practically possible. A surface water study will be required in order to evaluate surface water inflows to the pit and to size sump equipment and pit perimeter bunding or ditches.

The bedrock aquifer may provide yields of groundwater sufficient to supply the mine's potable water supply demand, although this would require further investigation.

18.3.3 Mineral Processing

The make-up water requirement is expected to be up to 500 m^3/h. Make-up water will be sourced from Lake Gardiken, the total storage capacity of which is 875 Mm^3. Abstraction of groundwater to meet water supply demand is not feasible due to the relatively low permeability of the bedrock. Water supply is therefore more of a permitting issue than a technical issue. However, a seasonal site water balance will be required in order to define accurately the make-up water requirement.

18.3.4 Infrastructure

Flood lines will need to be derived at the next stage of investigation to help with siting of mine infrastructure. A stormwater drain system will need to be designed for surface water protection of infrastructure, especially asphalted areas and roads.

Sediment control structures will need to be sized and located according to predicted discharge of site run-off.

Seepage from waste dumps, tailings storage and any water retention dams will also need to be assessed.

18.3.5 Environment

Baseline monitoring of streamflows, lake levels and surface water quality will need to be commenced as the project moves to the next development phase.

A network of groundwater monitoring boreholes for baseline monitoring of groundwater levels and quality will need to be installed. As with the hydrogeological investigations, discussed in Section 18.3 above, this might be best done in conjunction with geotechnical drilling, if possible.

The potential impacts of mining on groundwater and surface waters will be assessed upon completion of surface water and groundwater investigations and in conjunction with baseline monitoring data. For this reason, it is essential that baseline water monitoring be commenced well before any engineering feasibility study analysis is required.

18.3.6 Risks and Opportunities

No site specific data relating to the water environment has been collected to date. A high degree of uncertainty therefore surrounds water management requirements and risk at present. This applies especially to the risk of significant hydraulic connection between the proposed pits and Lake Gardiken. Therefore potential costs for water management, particularly dewatering of the pits, may be significantly underestimated.

Further hydrological investigation will be required at the next stage of investigation to define:

- pit inflows from groundwater and surface water;

- piezometric surfaces in the pit slopes and optimised solutions for reduction of slope pore water pressures if required;
- surface water protection and management requirements for mine infrastructure i.e. diversions, bunds, culverts, settling ponds etc.;
- seasonal site water balance;
- seepage from waste dumps and tailings storage facility including environmental impact assessment; and
- recovery of water levels after mine closure.

Recording of groundwater strikes and other anecdotal drilling information during initial resource drilling can provide useful data for initial desk-based groundwater studies. This will allow the development of a more targeted groundwater investigation programme, thus saving costs in the long-run. Some significant cost savings might also be derived from the development of an integrated hydrogeological and geotechnical field campaign.

19 MARKET STUDIES AND CONTRACTS

19.1 Nickel Market Review

The metal in concentrate production schedule for Rönnbäcken is shown in Figure 19-1.

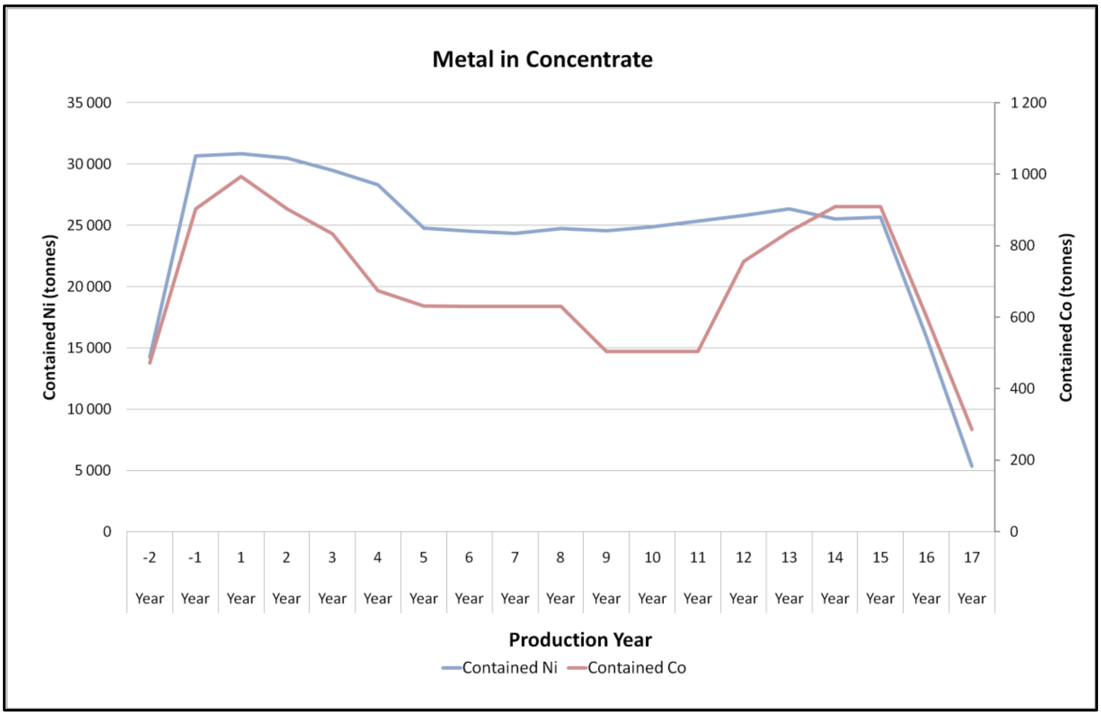

Figure 19-1 Metal in concentrate production schedule for Rönnbäcken

The Company intends to market the Rönnbäcken concentrate to nickel smelters globally. To place this in context, a brief overview of the nickel sulphide concentrate markets follows.

The dominant source of nickel throughout the last century has been sulphide concentrates, from sources such as Vale's (formerly Inco) operations in the well-known Sudbury Basin and at Thompson, Manitoba, from Xstrata`s (formerly Falconbridge) Sudbury operations, and from Norilsk's huge complex in Siberia and its other sites in the Kola peninsula of northwestern Russia. The Finnish operation, Outokumpu (now Norilsk-owned), brought additional mining, smelting and refining capacity on-stream, as did Western Mining Corporation (WMC), now BHP Billiton's Nickel West, at its Kalgoorlie smelter and Freemantle refinery. In China, Jinchuan has steadily increased capacity through phased expansions, as has Jilin. Notably, all of these operations were originally integrated; sourcing nickel from their own mine operations. However, all of these are now mature operations suffering from declining mine resources and grades, difficult or expensive mine expansion, and increasing costs; the latter is especially true of western operations such as those in Canada. In order to minimise unit costs and maximise both plant capacity utilisation and extra revenue generation, these operations have, for a number of years, sought outside sources of concentrates to smelt and refine. An additional part of the costs saving realized through treatment of third party concentrates comes from deferral of investment in new mine exploration/discovery and development, and the corresponding avoidance of associated risk, all of which can be very significant, especially in fluctuating metal markets.

The growth of world nickel demand in the last decade, indeed of demand in all commodities, has been driven to a large extent by China, and to a lesser extent by India. Forecasts of supply-demand balances in the early to mid-2000s predicted a massive shortfall in supply. This resulted in the start-up of many small sulphide mining operations around the world, especially in Australia. Most of these were too small to support smelting and refining facilities on their own. However, a natural synergy between these new, small sources of nickel, and the underutilisation of capacity at existing sulphide smelters/refiners was quickly recognised. The result has been the growth of trade in the sale of nickel concentrates from a relatively small portion of world nickel supply (less than 5%) to an estimated 15% of currently nickel supply. The net effect is that currently, there exists a very real, competitive marketplace for the sale of nickel concentrates produced from smaller operations.

The other type of ore from which nickel is produced is lateritic, or oxide (non-sulphide) ores. They are treated either by hydrometallurgical processes (Murrin-Murrin, Goro) or via reducing pyrometallurgical techniques (Koniambo), depending on the specific type of laterite ore sourced. To date these resources have proved difficult to exploit with numerous technical problems resulting in higher capital and operating costs than originally envisaged. Relatively high nickel prices will probably be required to support these projects in the long-term. Despite this, as sulphide resources continue to decline, it is likely that these types of ores will be a significant source of future nickel production. Sulphide operations, whether integrated or not, will benefit from this situation. They have the advantages of long established smelting/refining operations which have little requirement for new capital, and very low technical risk. In summary, they will continue to have available capacity for third party concentrates and hence will be able to take advantage of the higher metal prices. As such, small nickel sulphide concentrate producers, selling to existing smelters/refiners will have an excellent opportunity in the current and future marketplace for supplying concentrates to integrated operations facing declining supply from captive sources.

A key part of the synergy between independent concentrate producers and smelter/refiners is defined in the smelting-refining marketing contracts that exist between these parties. First, unlike the copper concentrate market, there is no standard form or structure of contract, and there is no transparent marketplace. The contract terms are negotiated individually and tend to be confidential. There are differences between the capabilities of each smelter and each refinery which may be reflected in the terms offered. These differences may show up in the percentage of payable metal (Ni, Cu, Co, Au, Ag or PGM) the smelter/refiner is prepared to pay for. The grade of the concentrate will be important for more than just the obvious calculation of gross metal value. The level of minor elements such as As, Hg, Sb, F and Cl can affect the smelter/ refiner's ability to safely handle the material. Levels of MgO in the range of 4% - 8% are likely to trigger extra costs at the smelter; these may be passed back to the concentrate supplier. Blending a concentrate, containing higher than normal levels of impurities, with other concentrates typically reduces any processing problems, allowing for treatment. Use of removal systems, as for mercury, can facilitate successful treatment. However, there may be thresholds for these or other elements, above which a smelter may not be prepared to accept. A marketing contract needs to recognise all of these issues and outline how they will be dealt with, especially during the start-up period when quality control of the final concentrate is likely to be less stable. Contracts can be relatively simple, or quite complicated. The format will affect the level of risk and return to each of the two parties as prices vary over time. Some contracts provide for price participation wherein the smelter/refiner receives a share of the higher returns at higher metal prices. Most contracts are for life of mine, but others have an expiration date. While the latter allows for a better reflection of changing markets over time, it creates an unpredictable risk for each party. Other items, such as transport of concentrate from the concentrator to the smelter may absorb a significant portion of the metal value of the concentrate, especially at low prices and/or with low payable metal grades. On the other hand, some contracts provide a transport credit to the supplier. As a general rule, risk (such as price risk) is passed on to the concentrate supplier, who needs to ensure his operation`s viability during periods of low metal prices. The flexible format of contracts allows for tailoring each contract to the needs of both parties and the particular analysis of the concentrate. Ultimately, the result is one of informed negotiation. While the lack of transparency in nickel concentrate markets may be disconcerting, a body of knowledge has developed over the years within the industry which ensures a fair and competitive result.

The market for custom concentrates is comprised of the following traditional, established smelter/refiners of Vale (Sudbury, Canada), Xstrata (Falconbridge, Canada), and Norilsk (Harjavalta, Finland), and the more recent participants of Jinchuan (China), Votorantim (Fortaleza, Brazil) and Jilin (China). Vale has announced the future shutdown of smelting at Thompson, Manitoba, thus removing one potential smelter from the list. It is likely that Vale will use their new hydrometallurgical facility at Argentia, Newfoundland, Canada, to process custom concentrates, thus replacing Thompson as a custom operation. However little is known about the requirements or limits in terms of quantity and quality of third party concentrate which Argentia will be prepared to handle. There is a reasonable understanding of the capabilities of the traditional smelter/refiners. Sulphide smelting employs two basic processes, flash smelting and roast/reduction. It is likely that with either process, blending of Rönnbäcken concentrate will be required in order to handle its low natural fuel level (low Fe, S), high MgO level, and possibly to reduce As, Sb or Hg levels. Flash smelters may be less sensitive to As than roast-reduction smelters (Xstrata). The volume and nature of the other concentrates being handled by a smelter at a given time will affect the extent to which blending of Rönnbäcken, or other adjustments to processing, will be necessary. Note that the accuracy and applicability of these comments, and those following, are heavily influenced by the current and forecast concentrate supply-demand situation (quantity and quality) at each smelter/refiner, a situation which is in a constant state of change.

With the pending shutdown of Vale’s Thompson operation, only Xstrata’s Falconbridge smelter will be using roast–reduction. This process provides excellent metal recoveries, particularly for cobalt, but has high power costs. It can recover all payable metals – Ni, Cu, Co, Au, Ag or PGM, subject to the usual minimum deductions, but it is sensitive to high MgO, and As. The smelter is located just outside Sudbury, Ontario, requiring rail (or road) transport from a port such as Quebec City or Montreal, Quebec. The facility has a well-established group experienced in receiving, sampling, handling, blending and smelting custom concentrates along with their own captive concentrates, which operates under a business philosophy of including the treatment of custom materials. With the expansion of concentrate production from the Raglan mine in Canada, plus potential concentrate from the Kabanga mine in Tanzania, Xstrata’s smelter is well-positioned in terms of feed for the future.

Vale`s Sudbury smelter is also experienced in all aspects of custom concentrate treatment. It uses its own proprietary flash smelting process requiring oxygen, generated on-site. Ni, Cu and Au, Ag or PGM recoveries are competitive, but cobalt recoveries are lower than most of the other facilities. As for the Xstrata smelter, labour costs are high. Despite this, and the 600 km inland transport from Quebec City (the smelter is located just on the opposite side of Sudbury from the Falconbridge smelter), Vale has, in the past, succeeded in acquiring custom concentrates against competition from Xstrata and Norilsk (Harjavalta). Custom concentrates have allowed deferral of capital investment in Vale`s Sudbury mines - capital which is now being put back into those mines. With this investment, but with Voisey`s Bay concentrates being sent to Argentia by 2013 and Thompson concentrate being sent to Sudbury, the future net demand for custom concentrate at Vale's Sudbury smelter needs to be determined.

In Newfoundland, Vale is constructing a hydro-metallurgical facility with a capacity of 60,000 tpa Ni, to treat concentrate from Voisey's Bay. The Rönnbäcken concentrate, with its high nickel grade compared to that of the Voisey's Bay concentrate, and low PGM content, could be an attractive alternative feed at Argentia, as hydrometallurgical processing does not have the capability to refine PGM. The facility would, however, need to be modified to accommodate concentrates with higher impurity levels, including MgO.The Norilsk smelter at Harjavalta has been treating third party custom concentrates longer than all others, and is well equipped to do so. Having no nearby captive ore sources, it is effectively a fully custom smelter. It uses the Outokumpu double flash furnace for both smelting and converting (the DON process). It employs a slag cleaning unit to improve metal recoveries, especially that of cobalt. Very competitive for Ni and Co, the smelter is less so for Cu and Au, Ag or PGM`s. However, it is close to port facilities, and is the closest smelter to Rönnbäcken, advantageous in terms of transportation costs and work-in-process inventory considerations. The concentrate feed situation for Harjavalta has improved over the last year with the recent progress at Mirabela's Santa Rita mine (Norilsk receives 50% of their production for a period of 5 years) and the ramp–up of expanded production at Norilsk's Nkomati mine in South Africa. In addition, the re-start of production at some of its Australian mines will provide Norilsk with further concentrate for its smelters.

The Fortaleza smelter in Brazil treats concentrate from Mirabela's new Santa Rita mine, in addition to its own and locally–sourced feeds. Fortaleza uses the Outokumpu DON process, with the mattes shipped to and refined at Harjavalta. There is potential to provide additional concentrate to make up any shortfall. The smelter is limited in its capacity to accept higher MgO levels. Fortaleza`s total concentrate capacity is limited, and could treat only part of Rönnbäcken's output.

China offers a good potential for off-take as a number of nickel concentrates from Australia and Spain are currently smelted there. The major nickel producer, Jinchuan, operates several smelters in China. At its largest smelting facility in Gansu, Jinchuan has capability of processing 350,000 tpa concentrate using a modified WMC smelter design, which itself was based on the early Outokumpu flash process. Metal recovery capability is competitive. Jinchuan has offered terms which have been very favourable to the supplier at times and has successfully sourced feed globally from third parties in Australia, Spain and Zambia. It has publicly stated that it will expand its smelting/refining capacity to meet China's needs, but cannot source adequate nickel supplies in China, thus suggesting it might wish to absorb all of the output of an external supplier such as Rönnbäcken. Jinchuan's facilities have the capability to take MgO-bearing feeds.

Jilin has recently doubled the capacity of its Ausmelt smelter with capacity of 200,000 tpa of concentrates to produce 15,000 tpa of nickel in nickel-copper matte. Some capacity for third party feeds may thus exist.

The grade of Rönnbäcken concentrate needs discussion in light of the smelting/refining processors in the marketplace. The likely need for blending has already been mentioned. Most striking is the high nickel content at 28%. This ranks the concentrate grade as one of the highest available in the nickel business, captive or custom (Figure 19-2).

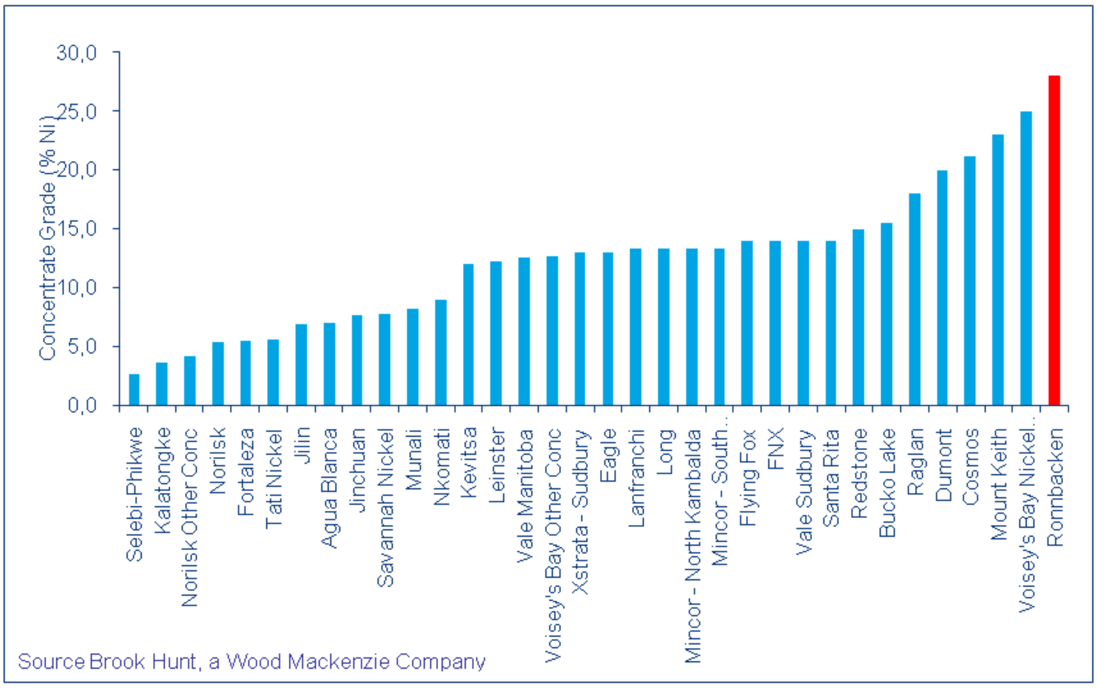

Figure 19-2: Comparison of nickel grade in nickel concentrates

Secondly, with a relatively low Fe content at 9-10%, the Ni/Fe ratio is very high. In the smelting process the iron (Fe) creates slag which causes nickel losses. However, with a low Fe content, the slag quantities and hence nickel (and cobalt) losses should also be very low. This indicates that Rönnbäcken concentrate could realise the highest possible nickel accountabilities. Cobalt accountabilities should also be very good. The high grade, of course, also helps reduce the costs of transportation, handling and smelting on a unit cost basis (USD/lb nickel). The precious metal content, although low, may be sufficient to attract some accountability and revenue, particularly at higher prices. Considering the structure of typical custom concentrate contracts, the high grades indicate that this concentrate should be able to carry costs and charges over the whole range of foreseeable prices, and be able to survive the price lows while generating excellent returns to both parties at higher prices. The extent to which the grade advantage is offset by the need to blend or handle penalty elements must be determined through discussion and negotiation with each potential smelter/refinery.

The planned production rate should provide Rönnbäcken with some flexibility in negotiations and help in attracting competitive terms. Each of the four larger smelter/refineries has the capacity to handle all the production. However, as a hedge against strikes or other disruptions to production, Rönnbäcken could, for example, split the tonnage between two or more smelters. The split volume would still be sufficient to be attractive to a smelter/refinery.

In summary, a significant market has developed over the last 20 years for custom concentrate, however, long-term nickel sulphide smelter capacity availability is expected to be less than originally anticipated as a result of a combination of the anticipated shutdown of the Vale Thompson smelter (Manitoba), and increased internal concentrate production by Xstrata Nickel and Norilsk Nickel. In addition, there are competing projects which could add to the supply of custom concentrates in the marketplace in the medium-term. Nevertheless, with its very high grade, long life and significant tonnage, Rönnbäcken concentrate should be competitive in the custom concentrate marketplace.. High grades allow a concentrate supplier to better carry extra costs of processing, if any, such as for penalty elements, while surviving periods of low prices. Returns to the supplier of the concentrate are subject to confirmation through final and binding negotiation with the smelter/refineries.

19.2 Magnetite Market Review

19.2.1 Steel demand and production

Iron is a key ingredient in steel making, and steel represents almost 95% of all iron metal consumed per year. For this reason, the conditions of the steel industry need to be examined to understand the demand for iron ore. In the three decades until the year 2000, world steel demand had grown only very slowly, at rates not exceeding 2% per year. During the period 1990-2000, the annual rate of increase was less than 1%. As seen from figure 1 (which shows crude steel production), there was a clear break in trend in 2001, which marked the beginning of China's dominance of world developments in iron and steel. From that year until 2007, world crude steel production grew at an average annual rate of just below 7%, with China accounting for close to 70% of the total increase. In 2010, China is estimated to account for 46% of world steel production.

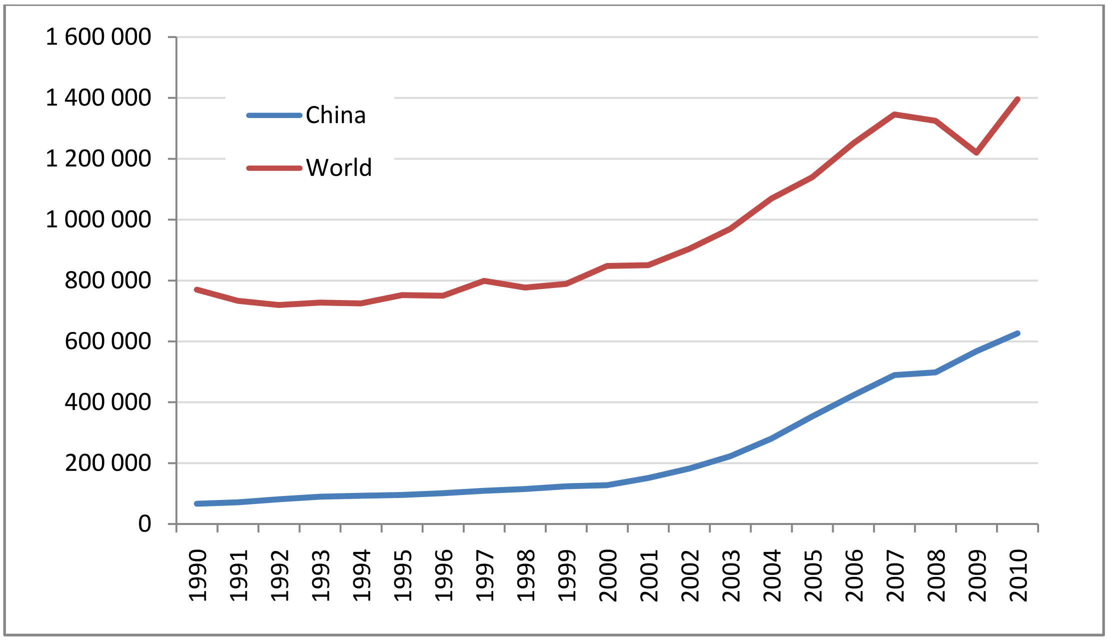

Figure 19-3: World and Chinese crude steel production, Mt (World Steel Association)

During the period of five years, from 2001, steel use in China grew by about 25% per year. This period of intensive construction came to an end around 2006 after which more normal, though still relatively high, growth prevailed. Since most of China's own iron ore deposits are low-grade and relatively small, China has to a large extent depended on imports to feed its steel industry. In 2009, China accounted for two-thirds of global imports of iron ore.

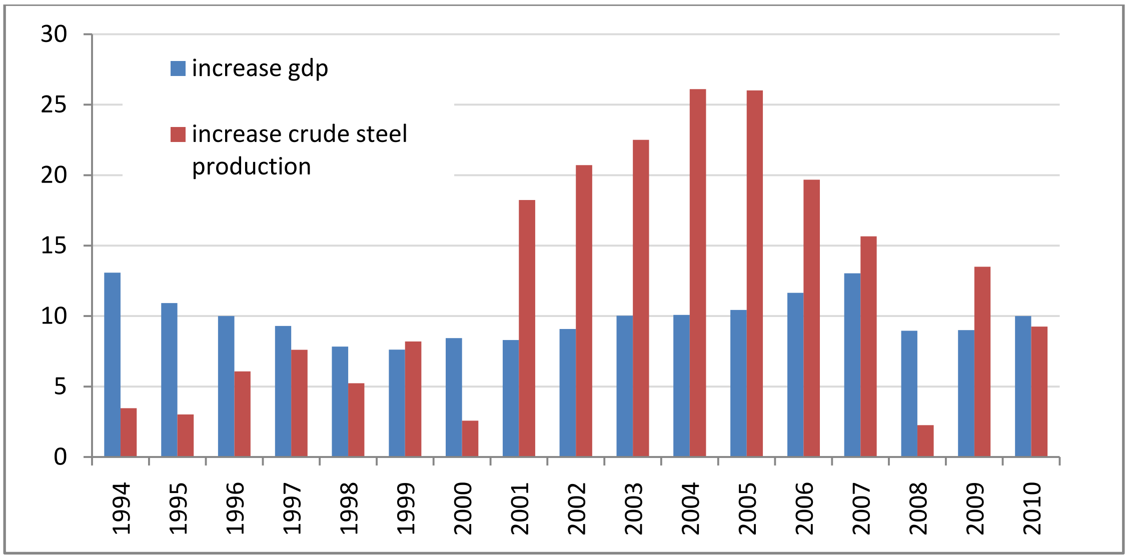

Figure 19-4: Annual increase in Chinese GDP and crude steel output, 1994-2010, per cent (China National Bureau of Statistics, World Steel Association)

19.2.2 Steel market going forwards

The deep recession of 2008 /2009 resulted in a sharp downturn in world crude steel production by 1.5% and 7%, respectively, as seen in Figure 3. In 2010, world production rose by more than 14%, initially due to a dynamic rebound in China. Steel output in the rest of the world declined by 21% in 2009, but surged back by 18% in 2010, as inventories were rebuilt in the first half of the year. By mid-2010, the sovereign debt crises in Europe and a slow-down in investment in China undermined market confidence, with a resultant slow down in crude steel production. By early 2011, a steady expansion of global steel production was again under-way.

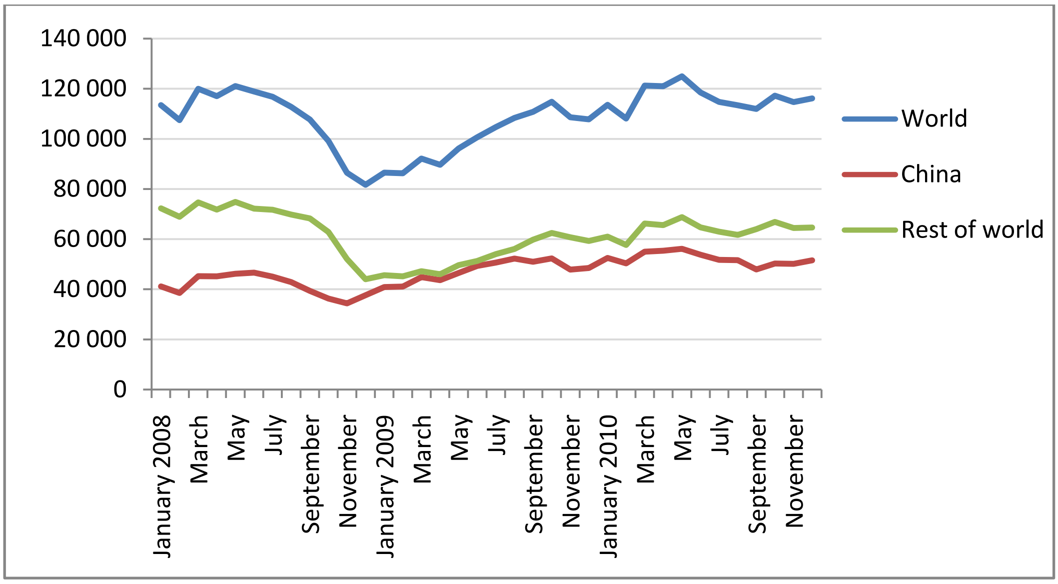

Figure 19-5: Monthly crude steel production, Mt

In China, output of finished steel products has been increasing to meet robust domestic demand. Apparent consumption of finished steel products in the first eleven months of 2010 increased by 12.7% over the same period in the previous year. The Chinese economy rebounded faster than most observers thought possible, due to greater private consumption. The need for increased capacity in the Chinese consumer goods industry is demonstrated by the rise in the rate of inflation and the strong growth in demand for consumer durables and household capital goods.

19.3 Outlook for steel demand and production

By the end of 2010, global steel demand and production had reached pre-crisis levels, with production of 1,396 million tonnes recorded, a level 3.4% higher than 2007. By 2020, global steel production will have reached 2,036 million tonnes, a growth rate of 3.8% per annum. While this growth rate is lower than that experienced in the period 2000-2007, the annual increase in absolute volume terms from 2010 to 2020 is only slightly lower at 64 million tonnes, compared to 71 million tonnes during the period 2000-2007. Steel production is forecast to rise a further 268 million tonnes to 2,304 million tonnes between 2020 and 2025, a growth rate of only 2.5%. This is reflective of the changing patterns of growth assumed for China.

Growth in China is assumed to be redirected towards consumption, with a consequent fall in capital investment and, therefore, the growth of steel demand. This trend is expected to steepen after 2020, when scrap availability catches up with steel production. A reorientation of China's growth towards increased consumption – a development that appears to be borne out by post-crisis economic developments – will have a significant effect on the growth of steel production to 5.5% per annum unto 2020, compared to the 21.2% per annum growth rate experienced in the period 2000-2007. Beyond 2020, growth in China would be expected to slow down further to 4-5%, as the population achieves higher living standards and productivity gains become more difficult to realize. Given its weighting in the steel industry, growth in global steel demand would also be expected to decline, although the extent of that decline depends on the balance of two main factors: whether India and other larger developing countries will emulate Chinese growth strategies and rates, and the influence of policy measures intended to reduce the use of fossil energy, some of which could exert downward pressure on steel demand.

Steel production from the OECD countries is expected to grow very slowly at less than 1%, given the constraints of restrictive monetary policies made necessary by large debt overhangs. The rest of Asia will experience robust growth of 5-6% per annum in steel production, based on continued increase in intra-regional trade, with the growth slowing somewhat after 2020; and steel production in developing countries outside of Asia is forecast to continue to grow reasonably well, mainly based on continued growth in exports.

The iron ore market

The rapid growth in Chinese steel production has been by far the most important factor for world iron ore demand. Since most of China's own iron ore deposits are low grade and relatively small (estimates of the average grade of China's mines vary from 20-30% Fe), China has to a large extent depended on imports to feed its steel industry, as seen from Figure 4. In 2009, China accounted for two thirds of global imports of iron ore.

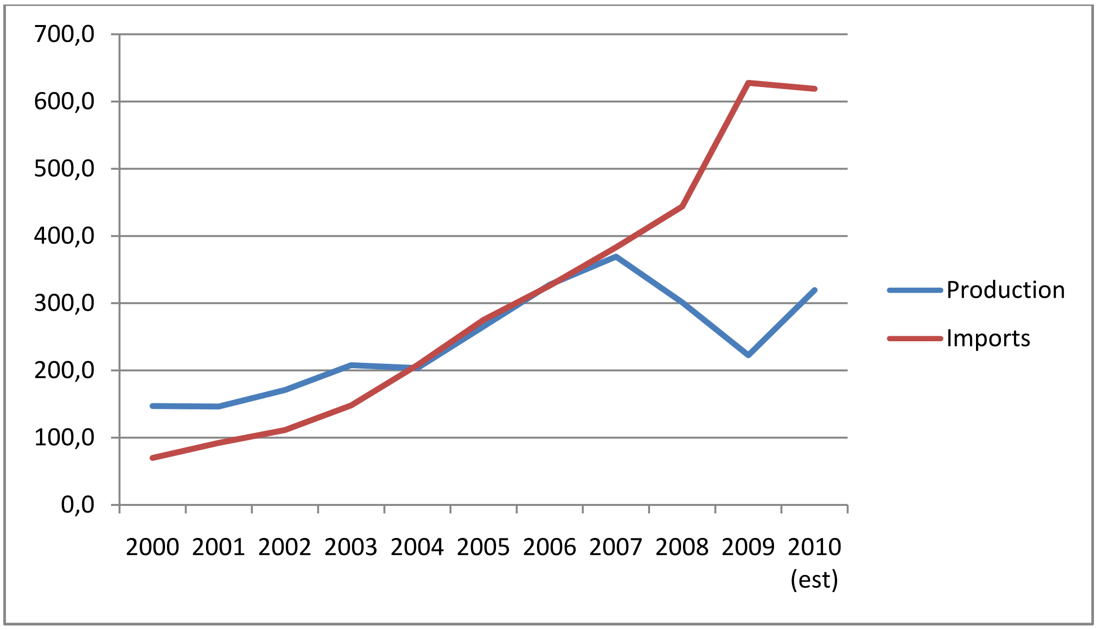

Figure 19-6: Chinese iron ore production and imports, 2000-2010, Mt

Note: Production has been converted from Chinese data for total weight to 63 % Fe content in order to make the figures comparable to imports.

More recently in 2010, as iron ore prices rose, increased domestic production met most of the rising need for iron ore, while imports were roughly stable, as seen in Figure 5.

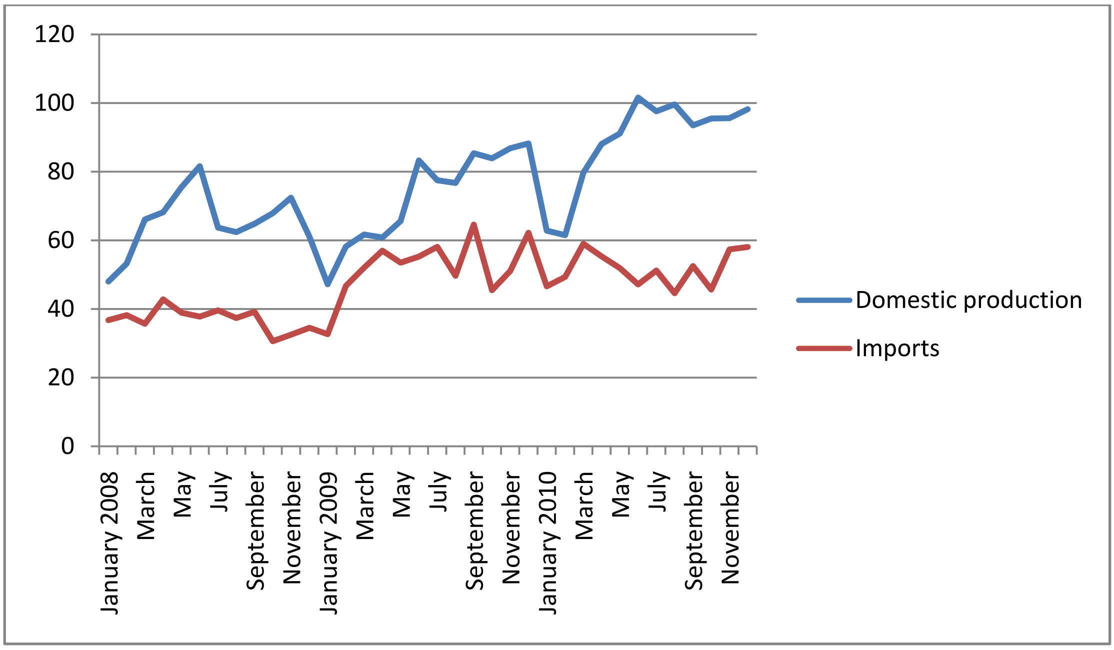

Figure 19-7: Iron ore (gross volume) and imports in China, Mt (Source: China Metallurgical Newsletter)

19.3.1 The demand for iron ore

During the period 2000 to 2007, iron ore use increased by 7.9 % annually, that is, by a percentage point more than crude steel production. The reason was twofold: first, since crude steel production was increasing faster than steel use had grown in the past, the amount of scrap available for recycling grew at a slower rate than production; second, most of the increase took place in China, where there was little scrap available. An annual increase by 4.8% per annum is forecast until 2020, resulting in total global demand of 2,984 million tonnes that year, or an addition of 1,117 million tonnes. The rate of growth is one percentage point more than that of crude steel production, reflecting continued limitations on the availability of scrap.

Beyond 2020, scrap supply would be expected to catch up and the rate of growth in demand would therefore decline gradually. From 2020 to 2025 growth in iron ore demand is expected to keep step with the 2.5% annual increase in crude steel production. Iron ore demand will reach 3.376 million tonnes in 2025, as shown in the forecast in Figure 19-8.

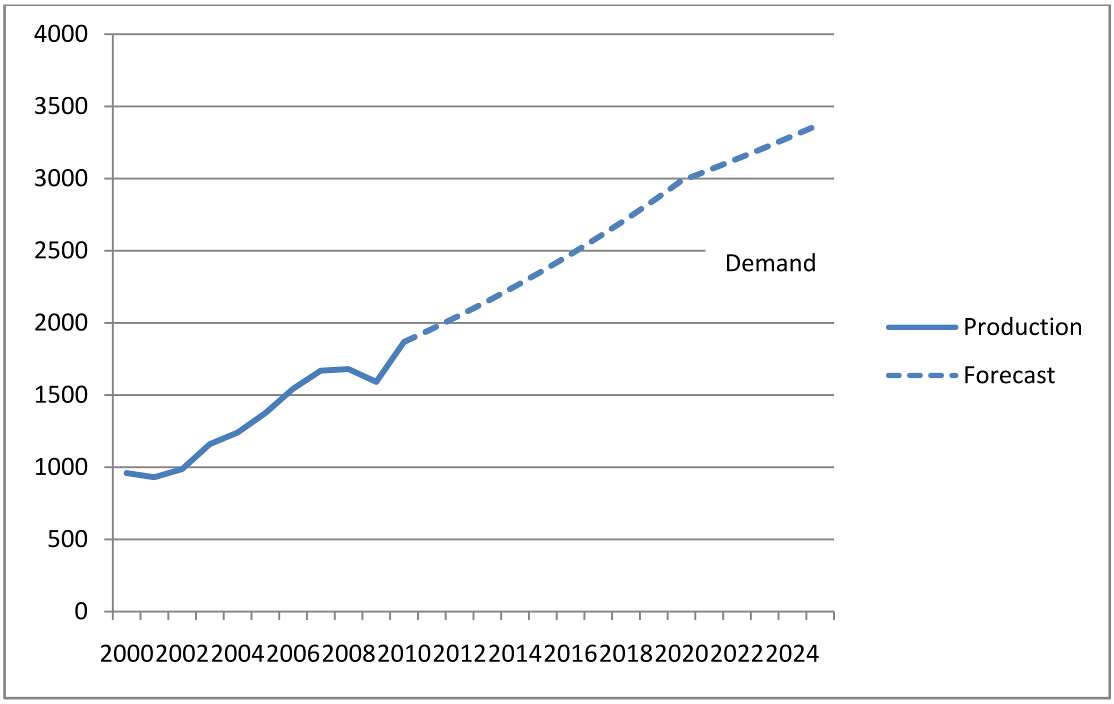

Figure 19-8: World iron ore demand 2000-2025, Mt

19.3.2 The supply of iron ore

The world's three largest producing companies - Vale, Rio Tinto and BHP Billiton - account for about a third of total world iron ore output and over 70% of the world seaborne trade market.

These three producers will be able to deliver much of the additional supply needed for the markets, with smaller producers in the Pilbara region of Western Australia, adding a significant amount. Based on port expansion plans, about 400 million tonnes per year of iron ore mining capacity could be added in Western Australia by 2020. Planned expansions in Brazil, by Vale and other companies, have the potential to add similar amounts. The balance of additional production capacity would have to come from other parts of the world.

While there are several projects at the planning or early development stage that could become very large, few of them will enter production during the next five years, and they will probably take considerable time to scale-up production. This means that existing producers will be hard pushed to get all the needed capacity on-stream.

A greater challenge to meeting world market needs for iron ore is the expected decline of iron ore production in China. Chinese iron ore production is expected to decline from the 2010 figure of 319 million tonnes to 160-240 million tonnes in 2020. Given the relatively small reserves, low quality, high impurity levels and high operating costs (today ranging from US$70-140 /tonne), this level of production could prove to be a challenge. When spot prices hit US$90-100/tonne in late-2008, half of the Chinese iron ore industry operated at a loss, and was subsidized either directly or indirectly. While new additions to capacity have taken place since then, it is worth noting that the domestic price has never fallen below US$95/ton. Moreover, there are at least three factors that have a larger impact on Chinese iron ore miners' costs than on those of their competitors:

Wage inflation is faster in China than in most other countries;

The low grade of Chinese ore deposits make their costs of exploitation disproportionately sensitive to increases in energy prices; and

The Chinese Yuan is likely to continue its appreciation; all of the costs of China's ore producers are Yuan denominated, while those of its competitors contain a significant dollar denominated element, in the form of freight rates.

Finally, freight rates are expected to remain low for several years to come, which means that imported ore is less disadvantaged on the Chinese market.

Accordingly, smaller new entrants to the international industry should be able to fit their extra production into the market without problems.

19.3.3 Price forecast

Over the next couple of years, prices will remain high, but from 2013 onwards, they will gradually decline towards a plateau below the present level. The latter will be determined by the cost of Chinese production, meaning the level at which a large portion of Chinese mines are ensured of staying in operation.

From 2018 to 2020, the situation will resemble that of the 1980s and 1990s, when production increased in an orderly fashion and iron ore price movements were moderate. However, price spikes may occur temporarily, particularly if producers encounter unexpected problems with capacity additions.

From 2020 to 2025, prices could be expected to decline gradually as a function of productivity improvements, as demand growth slows down. However, increases in cost due to significantly higher energy prices and/or rising extraction costs due to the need to mine lower grade ore bodies could set a floor for this decline.

19.3.4 The period 2015-2020

For the period after 2015, prices need to be high enough to encourageinvestment in new capacity to meet growing demand, thus making it possible for most existing producers to survive. Since major producers alone will not be able to meet market needs, and given the fact that there are not enough large low-cost projects under way, much of the new capacity will need to be developed from marginal deposits (from the point of ore characteristics or infrastructure costs). Market analysts consider the production cost for these marginal mines as being the best basis for a long-term price forecast.

A more relevant basis could be derived from Chinese mines. A large part of the Chinese industry has had to close over the past couple of years, no longer protected by high freight rates. While freight rates have decreased, production costs have increased. Only the strong recovery of the Chinese steel industry and the consequent high prices for iron ore have saved the remainder of China's domestic producers, enabling them to raise production in 2010.

From events of the past couple of years, it is evident that domestic iron ore production in China would decline significantly if prices were to fall below US$100/tonne, hence prices will have to remain high enough to maintain a significant part of Chinese production in operation. Below such a price, much of the Chinese industry would be forced to shut down; lost production which producers outside of China would be hard-pressed to replace.

19.3.5 The period 2020-2025

Beyond 2020, it is believed that factors exercising downward pressure on the prices, including rising productivity, measures to reduce fossil fuel use and the exhaustion of closure possibilities in China, will dominate.

19.3.6 Price forecast for Rönnbäcken magnetite concentrate

The pricing mechanism for iron ore has changed fundamentally. From being an annual benchmark price agreement with adjustments for any penalties or premiums based on the concentrate specifications, prices are now spot prices where the premium is market-driven. The premium for higher iron content under the new pricing mechanism has been increased compared to the earlier annual agreements.

In the long-term iron ore price forecast for Nickel Mountain, it is assumed that Nickel Mountain would receive a price of US$105-119/tonne fob Norway in 2015, declining to US$90-104/tonne fob Norway in 2025 for the Rönnbäcken magnetite concentrate.

In spot market pricing, there was in August 2011 a direct and proportionate increase/decrease of the price paid within 58-62% Fe content, ie the price for a 60% Fe product was the spot price for 62% with a deduction of 2/62 of the spot price of that specific date.

The size of the range of Fe content and of the adjustment to price for each percentage point variance in grade will vary considerably over time depending on the general conditions of the steel market. In times of increasing and strong steel demand, and hence a corresponding need for increased production and productivity in the blast furnace, there may even be a premium paid for higher iron content. In such times, 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 - a difficult, time consuming and costly procedure. A blast furnace is a highly capital intensive unit carrying considerable capital costs which need to be covered irrespective of whether the blast furnace is operating or not. In such difficult times, the steel company will attempt to decrease its production/productivity by feeding lower grade iron-bearing materials; this not only reduces costs (which is of prime importance when steel demand, and hence, steel prices are low), but also reduces output.

Given the difficulty of forecasting the detailed cycles of steel demand and output 10-20 years ahead, RMG has presented a qualitative discussion of the market factors which will influence price, rather than to quantify the effects so far into the future. RMG has assumed that the patterns of variations in price in past cycles will be the basis for price fluctuations in the future.

The iron ore price and the increase for each additional per cent unit of Fe has decreased considerably during the months of August and September but RMG believes that there is no reason to change its *long term* assumptions so far.

Table 19-1: Price forecast for Rönnbäcken magnetite concentrate

	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
HIGH PRICE ASSUMPTION																
CFR China, 62 % Fe, $/ton, dry	**127**	150	150	150	150	140	138	133	130	130	130	130	130	130	130	130
Premium 65 % Fe, $/ton	**18**	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18
Freight Norway-China $/ton, dry	**30**	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30
FOB price Norway, high assumption	**115**	**138**	**138**	**138**	**138**	**128**	**126**	**121**	**118**	**118**	**118**	**118**	**118**	**118**	**118**	**118**
LOW PRICE ASSUMPTION																
CFR China, 62 % Fe, $/ton, dry	**127**	150	150	150	150	130	125	120	110	110	110	110	110	110	110	110
Premium 65 % Fe, $/ton	**9**	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9
Freight Norway-China $/ton, dry	**30**	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30
FOB price Norway, low assumption	**106**	**129**	**129**	**129**	**129**	**109**	**104**	**99**	**89**	**89**	**89**	**89**	**89**	**89**	**89**	**89**
Midpoint 65% FOB, $/ton, dry	**111**	**134**	**134**	**134**	**134**	**119**	**115**	**110**	**104**	**104**	**104**	**104**	**104**	**104**	**104**	**104**
Midpoint 62% FOB, $/ton, dry	**97**	**120**	**120**	**120**	**120**	**105**	**101**	**96**	**90**	**90**	**90**	**90**	**90**	**90**	**90**	**90**

This summary is based on a study by Raw Materials Group, Stockholm dated March 2011 and an up-date of that report dated September 2011, both made at the request of Nickel Mountain.

19.4 Concentrate transport

During operation, nickel concentrate production is expected to average around 93,000dry metric tonnes per annum. In accordance with currently regulations, a 40 tonnes payload is assumed, which implies between 6 and 8 round-trips per day.

The concentrate is intended to be shipped in bulk to international customers. The location of smelters sourcing custom concentrates is illustrated in Figure 19-9.

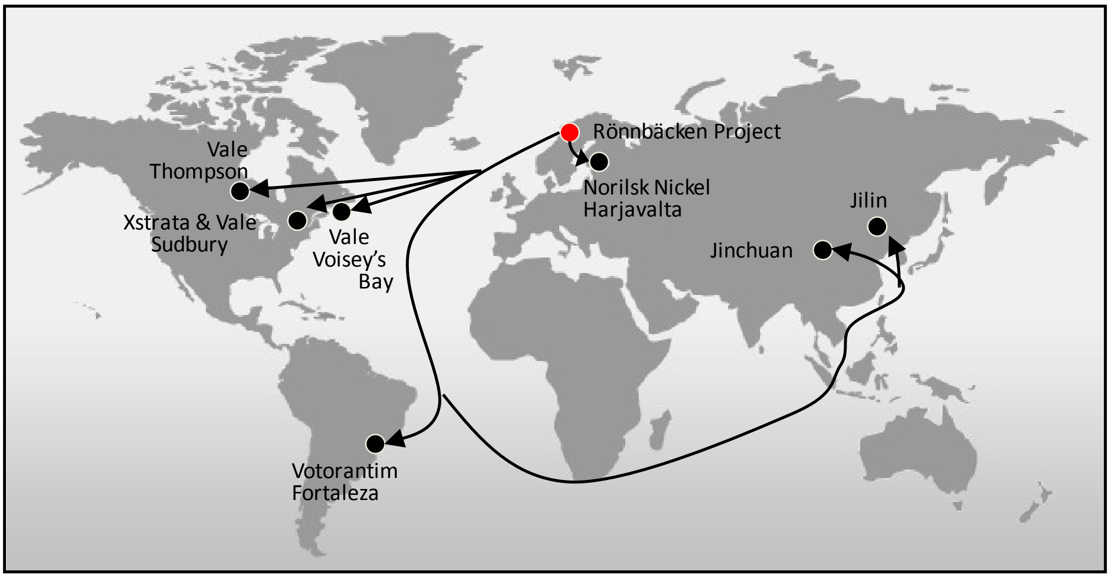

Figure 19-9: The locations of smelters sourcing custom concentrates

The Project site is located some 31 km away from highway E12 via public gravel roads to the north and 14 km via public gravel roads to the south. Highway E12 provides good access to the Port of Mo i Rana and Mosjoen in Norway at the Atlantic coast, and the Swedish ports of

Umeå and Skelleftehamn on the Gulf of Bothnia. The distance from site to the port at Mo i Rana is 166 km and Umeå and Skelleftehamn ports are 355 km and 347 km respectively. These ports support year round shipping, although only Mo i Rana is ice free year around.

With export shipments likely to international customers, Mo i Rana is the most favourable port. Mo i Rana has experience from bulk export, although mainly iron concentrates, and can allow up to 80,000 t vessels. An indicative loading cost of USD5/t has been used, which is in line with other, similar, projects.

Indicative bids for the land transport from Rönnbäcken to Mo i Rana have been received from four sources ranging between SEK150 and SEK175/t. These cost indications are based on a payload of 40 tonnes/truck. The governing regulations stipulate a maximum weight of 60 tonnes and length of 25.25 m.

A second alternative for bulk export is to use the port at Mosjoen, close to Mo i Rana, where the authorities are expanding the existing port as well as building a new port for industrial purposes, including mineral handling.

Both these two ports are well located, especially if the proposed railway between Umeå and Mo i Rana becomes a reality sometime in the future.

Regarding sea freight costs there are different patterns for nickel concentrates and iron concentrates. In Europe, nickel concentrates are traditionally sold on a CIF basis while iron concentrates are sold on an FOB basis. The implication is that for nickel concentrates the seller has to absorb the sea freight costs.

For Rönnbäcken an average sea freight cost of USD44/t has been applied.

Regarding the magnetite iron concentrate the only export alternative is Mo i Rana for time being as Mosjoen lacks permit for 60 tonnes trucks with a length of 25.25 m. The land transport cost per tonne for the magnetite iron concentrate is anticipated to be the same as for nickel concentrates, see above. However an annual production of 1.7 Mt p.a. magnetite iron concentrate implies 130-140 truck round-trips per day, which means a truck passing on the road every fifth to sixth minute including trucks with nickel concentrate. Road authorities have been notified on this intense traffic situation for comments.

Regarding sea transport of the magnetite iron concentrate, the market destination will depend on the final quality produced. It is initially envisaged to ship the magnetite iron concentrate as pelletizing feed to either the Middle East or China, which would require large shipments in 100,000 to 120,000 tonnes vessels to minimize the freight costs. In Mo i Rana there are two alternatives under investigation to expand the port to accommodate vessels in the size of 100,000 to 120,000 tonnes. To load such larger vessels, new loading equipment and indoor storage facilities are likely to be required and will be the subject of further studies.

At this stage a loading cost of US$5/tonne has been assumed, same as for nickel concentrates.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1.1 Introduction

This section highlights the key issues identified in the Environmental Assessment reports (Miljökonsekvensbeskrivning, MKB) for Rönnbäcksnäset and Vinberget and the draft version of the Sundsberget MKB, which was made available to SRK.

SRK's comments on the status of these issues is given along with an indication of whether any of the issues are considered material to the project and how the issues are being managed.

20.1.2 Swedish regulatory requirements

The following regulatory requirements apply before mining operations can be started in Sweden.

Exploration Permits: Under the Minerals Act (1991:45) exploration licenses are issued by the Mining Inspectorate of Sweden (in Swedish: Bergsstaten), giving the holder a preferential right to an exploitation concession for the area and minerals covered by the exploration permit at a later stage. An exploration permit furthermore secures the holder access to land for exploration work that does not damage the environment or the land use.

Exploitation Concessions: Exploitation of a property for concession minerals requires an exploitation concession under the Minerals Act, which is issued by the Mining Inspectorate of Sweden. An exploitation concession is granted if there is a probability for economic exploitation of the deposit and if the site is considered appropriate from a mining and environmental point of view. The latter requirement calls for an Environmental Impact Assessment (EIA) to be included in the application, and if relevant also an assessment on the impact on reindeer herding. An exploitation concession normally secures the holder the right to carry out mineral exploitation for a 25 year period. A decision by the Mining Inspectorate upon a concession application may be appealed to the Government.

Environmental Permits: In addition to an exploitation concession, mining activities require an environmental permit under the Swedish Environmental Code (1998:808, in Swedish: Miljöbalken), which is issued by the Environmental Court (in Swedish: Miljödomstolen). The permit will define the conditions for the design, building, operation and closure of a mining installation. The permit application must be supported by a comprehensive EIA, which includes formal consultations with stakeholders. Decisions by the Environmental Court may (with leave to appeal) be appealed to the Environmental Court of Appeal and further to the Supreme Court. Construction activities within water areas (such as tailings dam, clarification pond), requires special considerations in the application for an environmental permit. One such consideration is the right of disposition of the water, which the Company must have before the application is submitted. Right of disposition of the water is normally obtained by acquisition of the land where the water works will take place or through an easement granted either by the landowner or by an authority.

In addition to the above mentioned permits, mining activities require an agreement with the landowner(s) or a decision by the Mining Inspectorate regarding designation of land above ground to be used for the activities.

A building permit is also needed under the terms of the Planning and Building Act (1987:10). The permit is issued by the local authority.

The process of permitting a mining operation in Sweden is graphically illustrated in Figure 20-1 below.

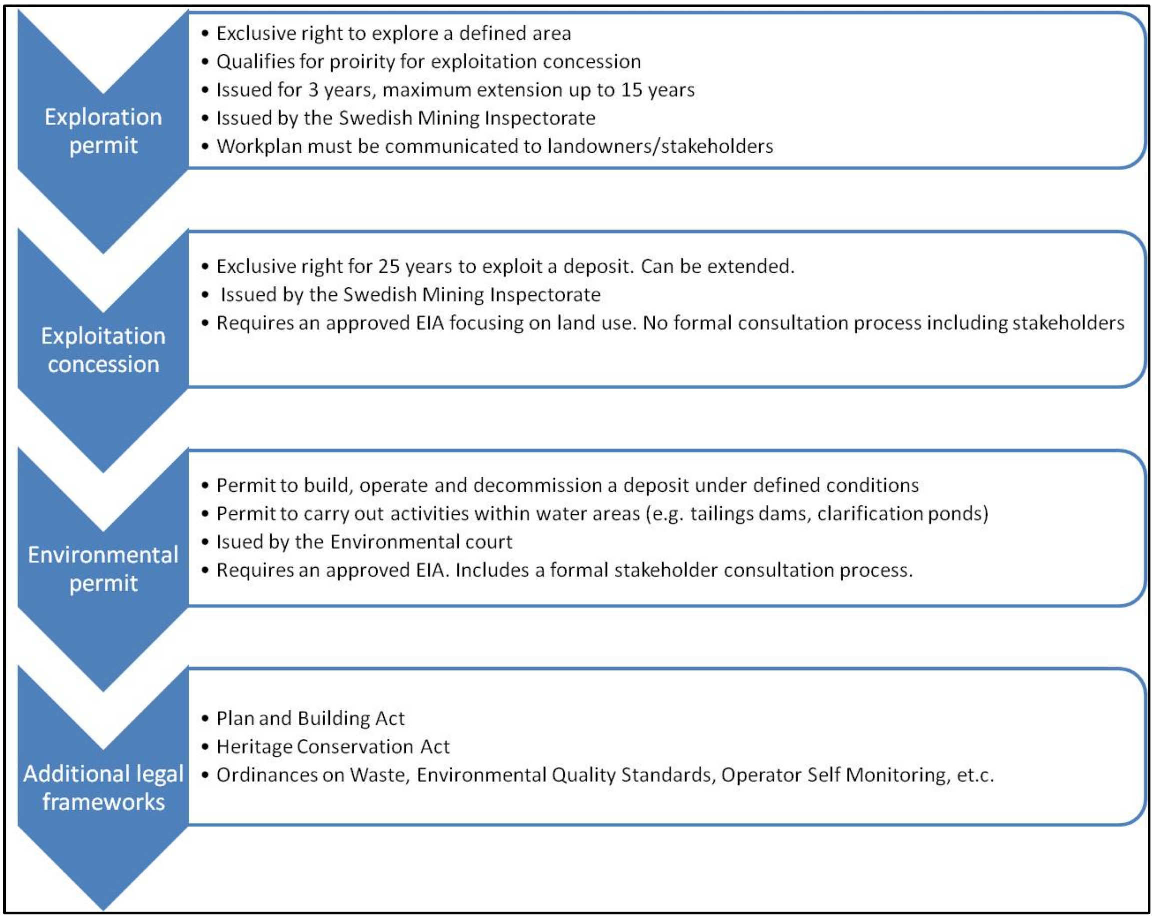

Figure 20-1: Swedish Mine Permitting Process

20.1.3 Permit Status for the Rönnbäcken Nickel Project

Key milestones concerning the Project deposits include:

- Full biophysical, physical, socio-economic baseline commenced in summer 2009 and are ongoing.
- Exploitation concessions granted by the Mines Inspector (Bergsstaten) for the Vinberget and Rönnbäcksnäset deposits in June, 2010 following submission of the first MKB for these projects.
- Designation of the Project as being of National Interest for mineral extraction by the Swedish Geological Survey in June, 2010, strengthening the company's position ahead of applying for the Environmental Permit.
- Application for exploitation concession for Sundsberget is underway with the first MKB currently being drafted. The Company anticipate filing by the end of 2011;
- The preparation of the second MKB for the application of an environmental permit began in August, 2010 by an initial consultation with the County Administrative Board on the content of the MKB.

- The Project presupposes the construction of a tailings dam including clarification pond, and access road banks within a natural water area. To carry out such activities within water areas requires a “water permit” in accordance with the Environmental code. The required right to apply remains to be secured by acquiring the area in question from the landowner or to receive an easement to the area from the Landowner.

A summary of the permit status is given in Table 20-1 below.

Table 20-1: Rönnbäcken permit status as of November 2011

Permit Status	Rönnbäcksnäset	Vinberget	Sundsberget
Exploitation concession application	Approved June 2010	Approved June 2010	To be submitted by end 2011
Environmental permit application	To be submitted Q2 2013	To be submitted Q2 2013	To be submitted Q2 2013
Construction permit	Building permit following receipt of environmental permit		

20.1.4 Baseline Studies

Baseline studies are a series of surveys and studies to categorise the biophysical, infrastructure, social, socio-economic, and economic condition of an area prior the implementation of a proposed investment. This is to ensure all possible interactions of a project’s activities can be properly assessed on pre-existing conditions. A critical result of the baseline studies is to identify key indicators which will be used in monitoring programs to assess the processes of change in spatial, economic, social and environmental conditions caused by the project.

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Baseline studies were implemented in 2009 and are ongoing, in anticipation of developing the environmental permit application, planned to be submitted to the Environmental Court in 2012.

Table 19-2 lists the baseline categories as defined by EU legislation and the corresponding categories under Swedish legislation, what categories were investigated for the Rönnbäcken Nickel Project and which consultancy performed the works. . Figure 20-2 illustrates the geographical extent of the studies carried out to date in the Project area.

Table 20-2: Baseline Requirements and Project Status

EU EIA[4]	Swedish Environmental Code[5]	Nickel Mountain EIA Baseline categories	Consultancy
Population	People, incl heatlh	Human beings	
Flora and fauna	Animals & plants	Biodiversity	PelagiaMiljökonsult AB
Water	Water	Surface water	PelagiaMiljökonsult AB
Water	Water	Groundwater	Ambiental
Soil	Land	Soil	Ambiental
Air	Air	Air	
Climatic	Climate	Climate	Ambiental
Material assets[6]	Cultural environment	Cultural heritage	LK Konsult, 2009 & 2010
Landscape	Landscape	Landscape/ geography	Perbkonsult
Population	Cultural environment	Indigenous people	Hifab,
Population; Material Assets	Cultural environment	Macro economics	
Population; Material Assets	Cultural environment	Micro economics	
The inter-relationship between the above factors.	Management of land, water and the physical environment in general	Captured during impact assessments	Perbkonsult

[4] *EU EIA Dir Annex III paragraph 3*
[5] *Swedish Environmental Code (Eng lang vers), Chapter 6, Section 3*
[6] *Including the architectural and archaeological heritage;*

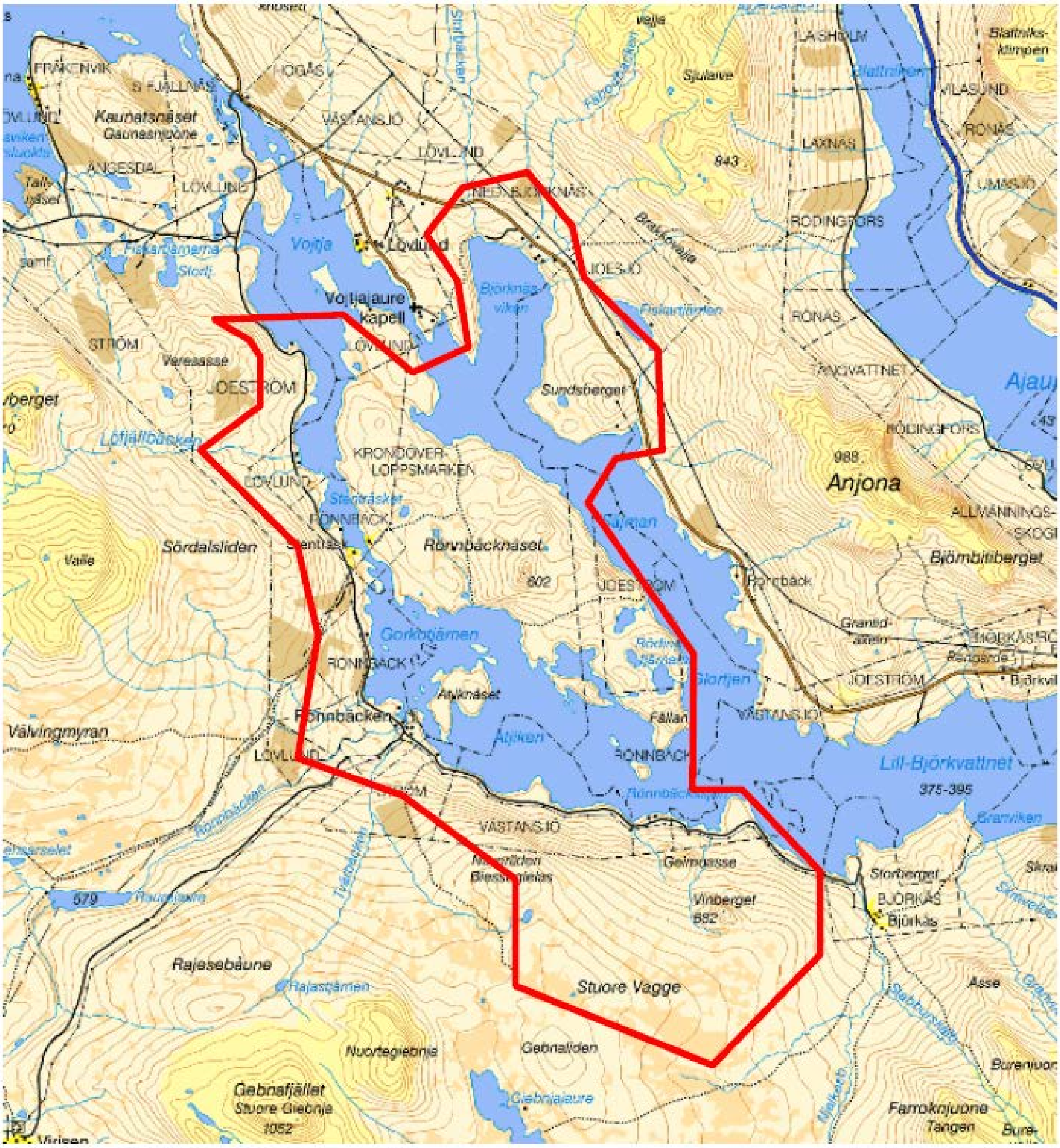

Figure 20-2: Geographical extent of the Project baseline studies

20.1.5 Status of land and water access rights

Whilst land and water rights are not an issue at this stage of the Project, it is critical to obtaining approval to mine to demonstrate confirmation of right of access to water and land. Water rights for those areas directly impacted by drawdown of water from the pits must be obtained prior to submission of the environmental permit application, whilst land access rights must be obtained before construction commences.

Demonstration of the right to water directly impacted by drawdown is required as a prerequisite for submittal of the environmental permit application to the Environmental Court. Demonstration of the land access agreements for all lands required for a project, including land within the exploitation concession and land required for project infrastructure if outside the exploitation concession is required before a permit can be validated.

As such, it is planned that water rights will be in place in advance of submitting the application for permit to mine in 2012.

20.1.6 Opposing Interests and Zero Alternative

A key part of assessing the impacts of a project is to consider other activities which could or would otherwise take place and the 'Zero Alternative', that is, should the project not go ahead.

Should the project not go ahead, all key biophysical, social and economic indicators will remain the same including maintaining the same trends. That is;

- the biophysical baseline condition will remain unchanged;
- reindeer herding will continue without externally driven change (to their grazing area);
- the risk to water resources and water catchment area will continue as it is today;
- social and recreational use of the area shall continue (berry and mushroom picking, snowmobile use, hunting)
- no significant alternative economic activity will be established;
- the economic and demographic decline of the area shall continue;
- the creation of new employment opportunities and (significant) contribution to the local economy shall not take place.

For the Project location, the following opposing interests have been identified and are listed below, together with comments on the nature of impacts and possible strategies to resolve conflicting issues:

- Reindeer husbandry and Sami culture - A discussion is presented in Section 20.1.8.
- Hydro-power industry – an intrest to cooperate as the project develops.
- Permanent and temporary residences – full impacts not yet fully understood. Options remain open as how best to compensate such stakeholders.
- Hunting and fishing – compensation to be offered to those directly impacted.
- Nature values – the project footprint impacts on some areas of High Nature Values, Class 1, which may be eligible for protective status, and Class 2, which may have Red Listed Species present, see Figure 20-3.
- Cultural values – no major values directly impacted by the Project;noise from operations may cause an indirect impact on part of the heritage value area.
- Outdoor recreational activities – some loss of recreational opportunity will result.

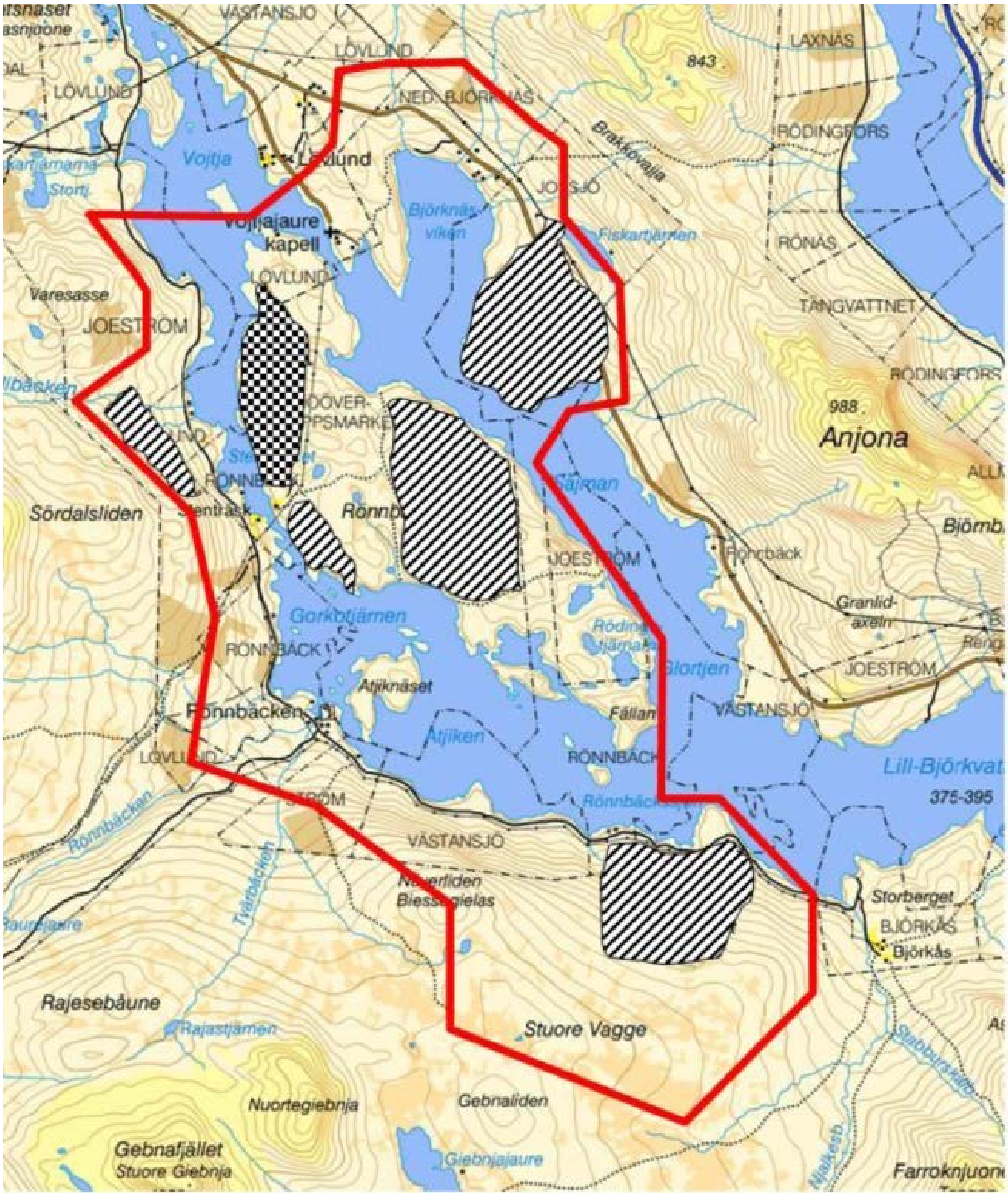

Figure 20-3: Checked or striped marking shows areas of High Nature Value, Class 1 or 2 respectively.

20.1.7 Socio-Economic Benefits

The following socio-economic benefits are expected to arise from the execution of the project:

- new employment will be created directly at the mine (direct employment);
- new employment will be created in the local economy (indirect employment);
- local economic activity will increase;
- taxes and other revenue for the public sector will increase;
- infrastructure such as roads and energy supply infrastructure will be improved;
- municipal services such as education, health care and other public services will improve;

- there will be improved international exposure of the region for other investors, including other mining companies;
- demographic and other social parameters will improve through the movement of workers and their families into the area;
- the availability of goods, services and operations in the region will improve; and
- tourism will long term (post mining) benefit from improved and increased housing and infrastructure in tourist centres Hemavan and Tärnaby.

According to the MKB, the impact of the operations on the social economy has been assessed to be positive. The population and the labour market have demonstrated a steep downward trend during recent decades. The Company has assumed that the planned operations will result in approximately 250 to 300 direct employment opportunities at full production and including contractors and sub-contractors. Additional indirect employment opportunities will be created via subcontractors and service industries in the surrounding communities. This in turn will lead to increased local economic activity and increased taxes and revenue for the public sector.

It is expected that improved infrastructure and demand for social services will also result from the establishment of the mine, and in turn will lead to increased interest from regional and/or international investors or various types. The establishment of the mine may result in an overall improvement in the population trend through employees and their families moving to the region.

20.1.8 Key environmental and social issues

The key environmental and social issues identified through a review of the Project MKB's and other available data are summarized below with the exception of mine closure and rehabilitation, which is discussed in Section 20.2. SRK's review of available documentation has not identified any further issues likely to be material to the project.

Water

The Project is not expected to lead to any reduction in the water quality of the area, due to integrating water protection measures in project design, such as diversion dikes, collection ditches, all runoff water within the mine and pits areas and any infiltration water into the pits being collected and pumped to the clarification pond, effectively preventing any potentially contaminated water from leaving the site.

The Rönnbäcken river, southeast of the Rönnbäcksnäset deposit is an important trout breeding river and as such will not be impacted and will retain its functionality. However, due to the presence of the TMF Rönnbäcken will need to be extended with a new outlet into Atjiken.

The placement of the TMF in Gorkotjärn-Stenträsk, which is a part of Gardiken reservoir reduces water storing capacity of the reservoir by some 4-5% (estimated at some 40 Mm3), resulting in a limited loss of revenue from hydro-power production. This loss of revenue is due to the reduced ability to keep water volume from summertime to wintertime, with a limited loss of power value (SEK 3M annually) for the producer as power in wintertime is priced higher than in summertime. However, the overall annual power production is not expected to be reduced. The Company have assumed a capital cost estimate of SEK 58M to compensate the hydro-power company for loss of capacity in the Gardiken reservoir. After decommissioning of the operations, however, it may be possible to return up to 30Mm3 of water storing capacity to the reservoir by connecting it to the water-filled open pits.

The placement of the TMF also results in the loss of bottom fauna and fish habitat. However, Gardiken is classified as a heavily modified water body under the EUs Water Framework Directive, with annual water level amplitude of some 20m. This means that the banks of the lake presents significant challenges for the natural bio-diversity to establish, notably bottom fauna.

The three pits and waste rock management facilities cause some loss of water catchment area with a corresponding loss of direct water runoff into Gardiken, which however is fully compensated by increased flows via other routes. Runoff from the waste rock management facilities and precipitation in the pits, along with groundwater infiltration, will be directed to the clarification pond for use in process plant.

Groundwater drawdown around the pits is estimated to extend some 50 m, although along fracture zones it may extend further. Sealing of such fracture zones by cement injection may be necessary and would limit such drawdown. Given the low pressure gradient and low hydraulic conductivity, impacts to groundwater are judged to be little.

Landscape

The operations will lead to significant changes to the existing landscape. Pit lakes and new topographic highs from the waste rock and tailings storage facilities will remain once operations cease. These need not be negative impacts, especially as acid rock drainage potentials will be mitigated through design of the facilities including how they are closed. With capping and re-vegetation these new topographic highs may return value to local stakeholders by replicating the pre-existing vegetation.

Figure 20-4: Visualisation of project layout following decommissioning

Employment & economy

As with many inland communities in the north of Sweden, Storuman Municipality has been experiencing depopulation over recent decades. This trend shows no sign of abating. Quality of life through rewarding employment opportunities is one of the key reasons for this trend.

The Project may halt or even partially reverse this trend through the employment creation when the operations are in full production.

This provides the Municipality of Storuman some 20 years to develop the business and commercial sectors and to diversify the economy, using the Project as an economic driver.

Reindeer husbandry

Rönnbäcken is located in an area which includes prime land for Vapstens reindeer husbandry all-year round. An impact assessment including limited social aspects was made in cooperation with Vapsten Sameby to better understand potential impacts and to identify potential mitigation measures. A key concern is whether the project will irreversibly impact on Vapsten Sameby's ability to continue reindeer husbandry practices in the future. Because of this Vapsten Sameby has objected to and is opposed to the establishment of the Project.

Parts of the Project area are classified as a 'Core Area' of National Interest for Reindeer Husbandry. A reindeer migration route passes Rönnbäcken, utilised twice per year. Consequently the Project may result in:

- reduced grazing land, including increased risk for predation due to reindeer being concentrated on a smaller grazing area;
- impact to the migration route;
- disturbance to reindeer through project activities;
- likely increase in man-hours to compensate for the changes to husbandry patterns;
- increased risk of traffic accidents involving reindeer;
- increased need for supplementary feed.
- Potential mitigation measures identified included:
- creating a specific communication channel created between the Company and Vapsten Sameby;
- knowledge sharing such that Company employees understand the specifics of reindeer husbandry how their work may impact upon it and for the Sameby members to learn about the proposed mining operations to better aid planning reindeer husbandry activities in the area;
- localisation and design of project infrastructure to reduce as much as feasible intrusion into grazing and migratory lands, and to integrate project infrastructure with the natural environment;
- construction of fences to prevent reindeer, in particular those unmarked, moving into neighbouring Samebyar;
- construction of a new migratory path around Rönnbäcken;
- protection and care of valuable lichen areas where in proximity to project infrastructure;
- timing of project activities to take into consideration particularly sensitive periods for reindeer husbandry;
- support for any additional labour or other resources required as a consequence of the project; and
- a rehabilitation and revegetation plan such that post-closure the Sameby can continue reindeer husbandry across the area.

The Company intends to continue the dialogue with the Vapsten Sameby to further refine mitigation measures with a view to ensure reindeer husbandry can continue effectively at the same time as mining operations take place.

The majority of impacts occur during construction when site preparation takes place and infrastructure, roads and transport corridors are built. This largely includes land clearance which has a profound impact on local biodiversity, and will cause change to the currently quiet and peaceful nature of the area.

During operations, sustained activities, such as heavy vehicle transport, blasting, excavation, transport and crushing of ore, deposition of waste rock and tailings, leads to a consistent level of impact including noise, dust, vibration, blast waves and regulated discharges to water and air and unregulated discharges to soil. These cause impacts based on intensity, frequency, duration and consequence of the activity.

For Rönnbäcknäset, Vinberget and Sundsberget such impacts can be mitigated or reduced to acceptable levels.

A small number of permanent and temporary residences and properties will need to be acquired purchased to mitigate any risk to people and/or property from blasting and mining activities.

Following cessation of mining activities, most impact cease and conditions will gradually return to the prior to Project circumstances. However, there will be residual impacts in the form of pits lakes and topographic highs formed by the waste rock storage facilities and tailings storage facility.

20.1.9 Conclusion

Impacts to the natural environment in the vicinity of the planned project occur mostly during construction, continuing to a lesser extent through operations. Once the mines closed and are rehabilitated, impacts largely cease.

Rönnbäcknäset, Vinberget and Sundsberget are all located in the same geographic, geo-political and biophysical area. By and large, they have the same type of topography, biodiversity, fall under the governmental jurisdiction from a National to Municipal level, and have very similar to identical social aspects.

The population density is low and aged. Hydro-power generation and forestry are the most important economic sector active in the area.

There are no areas afforded protected status. Species with vulnerable, near-threatened and threatened status are found in the region. Otters have not been reported.

The environmental impacts of the proposed project are not deemed significant and a range of measures are available to mitigate and/or reduce such impacts. Following cessation of operations, the area is expected to return to a prior-to-intervention state except for the presence of pit lakes and new topographic highs from the waste storage facilities and tailings storage facility, which shall be rehabilitated with vegetation.

The Project area includes areas of national interest for three purposes: Reindeer husbandry, valuable deposits for mineral supply, and outdoor activities. The Project area also approaches an area of national interest for cultural heritage values.

The Company’s activities, in particular land take, will impact these areas, except for valuable mineral deposits, for which the Project will contribute. Regarding the reindeer herding, such compensation measures will be negotiated with the Vapsten Sameby (Reindeer Village).

Social and economic impacts are largely positive particularly through new job creation, increased economy of the region and increased tax revenue to local authorities. Potential negative impacts stem from the having to relocate a number of dwellings in the area due to risk from blasting and other mining activities. Increased transport on roads, safety and disturbances from mining activities are other social impacts.

SRK Recommendations

The most significant recommendation is that the social, environmental and economic aspects of the project are continued to be assessed as the project development plan matures. This assessment should include all parts of the project, including the mines, process plant, waste streams, on- and off-site transport, etc. Feedback from any significant socio-economic and environmental issues identified incorporated into the design process. This includes the results of proactive, voluntary and formal stakeholder consultation.

Other, more specific recommendations are that the Company should:

- perform detailed impact assessment against a final project description (project freeze) per mine and collectively;
- assess the cumulative impacts of project aspects and with other projects in the area;
- review and test key project aspects, such as water balance, blasting, noise dust and vibration,
- detailed modelling of the water balance including how groundwater and surface water flow will be influenced by the project, detailed models of possible dispersion halos around the waste rock and tailings storage facilities,
- detailed modelling of airborne particulate matter transport.
- perform a micro- and macro-economic benefits assessment for the project as a whole;
- perform dedicated social impact assessment, focussing on local and regional impacts;
- design project wide environmental monitoring program including costing for water, air and climate, biodiversity; and
- design social impact and community liaison monitoring programmes for reindeer herding, community development and social cohesion in cooperation with relevant authorities.

20.2 Mine Closure

Implementation of the EU Directive relating to wastes from extractive industries (Directive 2006/21/EC) into Swedish law has resulted in the requirement for mine operators to submit a preliminary plan for closure with the environmental permit application (Section 20.1.2). This closure plan and the associated costs will be approved by the Environmental Court. The operator must then make provision for a financial guarantee to cover the reclamation costs should it not be able to fulfil its duties. The guarantee is required for the actual area of land affected and as such is linked in part to the LoM schedule. During operation, the actual disturbance will be reported to the authorities and the increase in the closure provision will be determined accordingly. If progressive rehabilitation is undertaken, the cost for this can be withdrawn from the bond upon acceptance by the regulatory authority appointed by the Environmental Court. The closure costs and associated bond will be reviewed when the closure plan is reviewed, at least once every three years.

For the purposes of the MKB supporting the Project mine permit application, there is no requirement to present a closure plan in any detail. As a consequence, the level of closure planning available for review by SRK is limited.

WRD Closure

Closure of the WRD will require stable slopes for the long term. It is often the practice to slope the dumps to the slope required for post-closure. It also enables progressive reclamation of the dumps as deposition is completed over portions of the dumps. Slope of 3H:1V are often adopted for final slopes for WRD.

The estimated closure cost has not been detailed in the documents reviewed by SRK. It is however stated that covering the WRD fully with a soil cover would have an excessive cost and this has consequently been rejected by the Company as an option. The proposed approach is limited to a discussion of optional concepts. It is important that the closure of the WRD be detailed in future studies for detailed costing.

TMF Closure

The details for the proposed closure of the TMF are limited to very broad concepts that could require considerable efforts and/or involve long term involvement. The closure plan should be developed using proven technologies and methods as it would facilitate the permitting process and improve the level of confidence in the cost estimates.

Summary

Despite a lack of closure planning detail available for review, SRK considers that the cost estimates proposed by the Company are of the correct order of magnitude. Technical and cost assumptions supporting the closure plan should be refined during the next level of study. Table 19-3 below lists the closure costs by area.

Table 20-3: Summary of closure costs by area

Area	Estimated Closure Cost (MSEK)
Open pit	-
TMF	425
WRD	
Overburden storage	75
Clarification Pond	
Clarification Pond Dyke	
Industrial Area	30
Process Water Pond	
Post-closure Monitoring	5
Total	535
Total USD Equivalent ***(SEK:USD = 8)***	**66.9**

21 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

The capital costs estimated as part of this study have been review by SRK and adjusted where appropriate to reflect SRK's views. These costs total USD 1 668M. SRK notes the following:

- contingencies of between 20% and 25% have been applied to capital cost associated with the process plant and infrastructure (roads, buildings and electrical);
- capital costs have been profiled with roughly 75% of expenditure occurring in pre-production years and the remaining 25% occurring in the first year of production; and
- no capital costs have been assumed for mining equipment.

Figure 21-1 illustrates a breakdown of the envisaged capital expenditure over the life of mine and split between the major cost centres. The total provision for sustaining capital over the LoM is USD 280M.

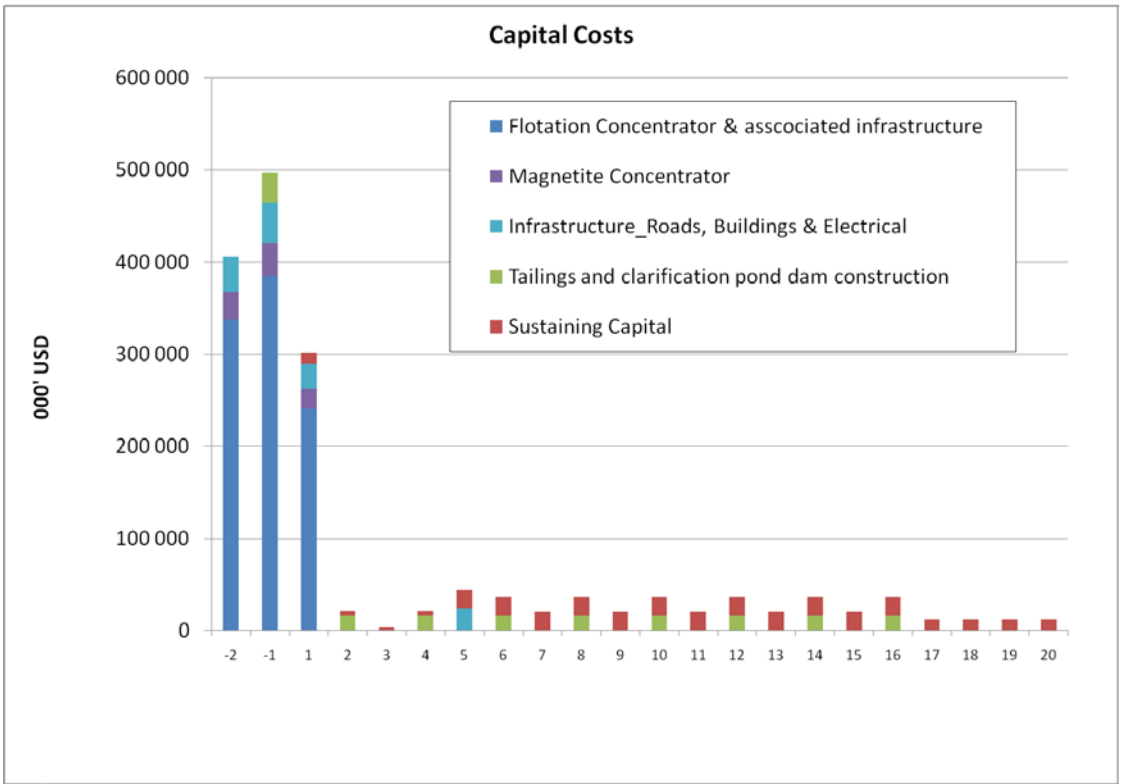

Figure 21-1: Summary of capital cost assumptions by major cost centre

Table 21-2 presents a summary of the capital cost assumptions the Project for start-up capital over Years -2 to 1 and also sustaining and remaining infrastructure capital requird over the remaining LoM.

Table 21-1: Capital cost assumptions

Description	Unit	Value
Flotation Concentrator	(USDM)	962
Magnetite Concentrator	(USDM)	87
Infrastructure	(USDM)	150
Working Capital	(USDM)	59
Start-up Capital[1]	**(USDM)**	**1 258**
Infrastructure (Ongoing)	(USDM)	156
Sustaining	(USDM)	254
Total	**(USDM)**	**1 668**

[1] Includes contingency of 23.5% based on 20% for quoted costs on major equipment items and 25% on general items.

Process plant

A breakdown of capital costs associated with the process plant is presented in Table 21-2 below.

Table 21-2: Process plant capital cost estimates

Description	Units	Total
Flotation Concentrator		
Processing directs	(USDM)	608
Indirects	(USDM)	142
Contingency (includes infrastructure contingency)	(USDM)	212
Magnetite Concentrator		
Magnetite Concentrator Total	(USDM)	87
Process plant capital costs	**(USDM)**	**1 050**

21.1.1 Infrastructure and tailings dam construction

A breakdown of capital costs associated with infrastructure and tailings dam construction is presented in Table 21-3 below.

Table 21-3: Infrastructure capital cost estimates

Description	Units	Total
Roads, buildings, electrical	(USDM)	133
Tailings & clarification construction costs	(USDM)	165
Compensation to hydro-power operator	(USDM)	7
Total	(USDM)	305

21.2 Operating Costs

The operating costs estimated as part of this study have been review by SRK and adjusted where appropriate to reflect SRK's views. An overview of operating costs is presented in Table 21-4 and illustrated in Figure 21-2 as net C1 cash costs over the life of mine.

Table 21-4: Operating costs overview

	USD/t moved	USD/t milled	USD/lb contained Ni	USD/lb payable Ni
Mining	1.79	3.10	1.61	1.73
Processing	2.89	5.03	2.61	2.80
General & Administration	0.22	0.38	0.20	0.21
On-going rehabilitation	0.07	0.13	0.07	0.07
Operating Cost at Mine Gate[1]	**4.97**	**8.64**	**4.48**	**4.82**
ConcentrateTransport				0.96
TC/RC's				1.20
By-product Credits				-3.42
C1 Cash Cost[2]				**3.55**

[1] Mine Gate operating costs per pound of nickel recovered to concentrate

[2] C1 costs include mining, processing, site admin, transportation, smelting and refining, net of byproduct credits.

The total unit operating costs amount to USD4.97/t of total material mined. The total cash cost is USD3.55/lb Ni, net of both cobalt in the sulphide concentrate and the magnetite concentrate. Net C1 cash costs are illustrated below in Figure 21-2 over the life of mine.

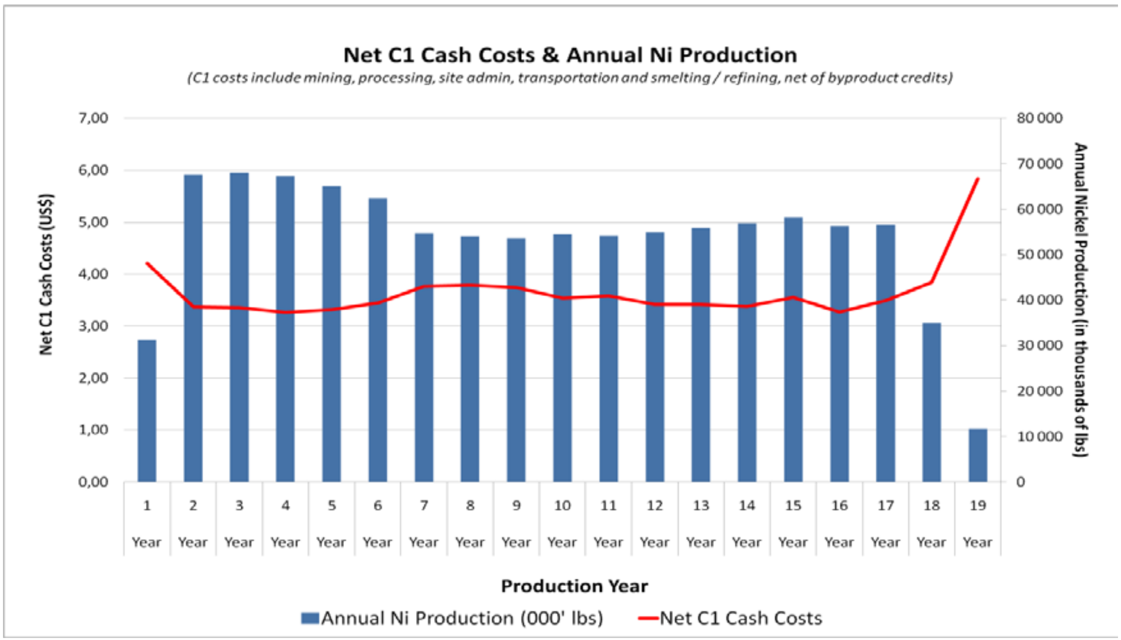

Figure 21-2: Net C1 cash costs over the LoM

21.2.1 Mining

Table 21-5 presents the unit mining costs for contract mining. The same unit cost has been applied to ore, waste and pre-stripping of overburden. Hence, varying unit costs with the increasing depth of the open pit are assumed to be accounted for in the unit costs presented below.

Table 21-5: Summary of unit mining costs for total material moved

Description	Units	Total
Drilling	(USD / t)	0.15
Charging / blasting	(USD / t)	0.24
Loading	(USD / t)	0.12
Hauling	(USD / t)	0.32
Personnel	(USD / t)	0.31
Auxiliary equipment	(USD / t)	0.12
Mine Services (incl. grade control drilling)	(USD / t)	0.11
Administration	(USD / t)	0.03
Maintenance	(USD / t)	0.06
Contractor margin (15%)	(USD / t)	0.22
Overburden stripping	(USD / t)	0.02
Stockpile re-handling costs	(USD / t)	0.01
Infrastructure power costs	(USD / t)	0.06
Mining cost USD / t total material mined	(USD / t)	1.79
Mining cost USD / lb contained Ni in con	(USD / lb Ni)	1.61

21.2.2 Process, tailings pumping and concentrate transport

Table 21-6 presents unit fixed and variable costs for the process plant and costs associated with thickening and pumping of tailings. SRK notes that the cost of power is assumed to be 0.38 SEK / kWh (0.048 USD / kWh).

Total refining, treatment and penalty charges are presented in Table 21-7 below.

Table 21-6: Processing, tailings and concentrate transport costs

Description	Units	
Flotation Circuit		
Total variable costs	(USD / t)	3,95
Total fixed costs	(USD / t)	0,35
Magnetite Circuit		
Total variable costs	(USD / t)	0,55
Total fixed costs	(USD / t)	0,04
Total tailings op costs	(USD / t)	0,14
Total Processing Op Costs / t ore	(USD / t)	5,03
Total Processing Op Costs / lb contained Ni in con	(USD / lb Ni)	2,61
Total Concentrate Transport / t ore	(USD / t)	1,72
Total Concentrate Transport / lb contained Ni in con	(USD / lb Ni)	0,89

Table 21-7: Refining, treatment and penalty charges

Item	Unit	Total
Total refining, treatment & penalty charges	(USD / t)	2.15

21.2.3 Other Operating Costs

Table 21-8 presents the total mineral royalty, G&A and total closure cost provision, including on-going rehabilitation. In addition, the Company has assumed an employment grant of USD8,250 per employee and per year for a total of three years at the start of operations. This amounts to a total revenue to the Project of USD3.4M and has been included in the TEM.

Table 21-8: Other Operating Costs

Description	Units	Total
G&A	(USDM)	200
G&A percentage of total operating costs	(%)	4%
Total closure cost & on-going rehabilitation	(USDM)	67
Royalty	(%)	0.20%

22 ECONOMIC ANALYSIS

22.1.1 Introduction

SRK has constructed a technical economic model (TEM) to derive a pre-tax Net Present Value (NPV) for the Rönnbäcken Nickel Project. The TEM is based on the technical assumptions developed by both the Company and from work undertaken by SRK, as commented on in the previous sections of this report. The Company has provided SRK with the processing physical parameters, refining/smelting charges and various assumptions for operating and capital costs. SRK has reviewed these assumptions and has adjusted these where appropriate to reflect the views as presented in previous sections of this report.

The economic analysis contained in this report is partially based on Inferred Mineral Resources, and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have mining and economic considerations applied to them and to be categorized as Mineral Reserves. There is no certainty that the reserves development, production, and economic forecasts on which this Preliminary Assessment is based will be realised.

22.1.2 Valuation Process

General Assumptions

The model is based on production from three open pit mines (Rönnbäcksnäset, Vinberget and Sundsberget), feeding a single process stream with a maximum annual throughput of 30Mtpa and housed within a single processing plant. The plant produces a nickel-cobalt flotation concentrate and magnetite concentrate from the flotation tailings. Concentrates are transported from site to port via road.

As part of the NI 43-101 process, SRK has constructed a pre-tax and pre-finance TEM and assumes:

- a US Dollar (USD) valuation currency, with any Swedish Krona (SEK) or Euro (EUR) derived costs being converted at a SEK:USD exchange rate of 8:1 and USD:EUR exchange rate of 1.125:1;
- a base case discount rate of 8%;
- the TEM is in real 2011 terms and no nominal model is presented;
- due to the uncertainty of when this project may be brought into production, the start of mining is assumed to be from 'Year 1' with two pre-production years ('Year -1' and 'Year -2') for the set up of basic mine infrastructure and access;
- discounting of cashflows starts in year -2;
- working capital based on 25% of the operating costs from the first year of production; and
- sustaining capital based on approximately 2% of site asset value.

The TEM considers the revenue and cost implications of both a nickel-cobalt sulphide flotation concentrate and a magnetite concentrate.

Commodity Price Assumptions

Table 22-1 below presents the nickel, cobalt and iron ore price forecast for the life of mine.

Commodity price forecast data was provided to SRK by the Company.

Table 22-1: Commodity Price Assumptions. Magnetite Iron Concentrate Prices 65% Fe FOB Mo i Rana (Norway).

Description	Units	2011
Ni	(USD/lb)	9.00
Co	(USD/lb)	15.00
Fe (Year 1)	(USD/t)	110
Fe (Year 2 & onwards)	(USD/t)	104

Mining Physical Assumptions

A summary of the combined mass movement of material from all three pits is presented in Table 22-2 below. A discussion of material movement by pit is presented in Section 16.1 above.

In order to maximise grade during the initial production years, marginal material from the Rönnbäcksnäset deposit has been stockpiled in the waste dump areas during the first six years of operation. The TEM assumes this material will be processed in the final three years of production.

Table 22-2: Scheduled Ore Tonnage and Grade

Description	Units	Total
Total ore mined	(000' tonnes)	528 030
Total waste mined	(000' tonnes)	379 943
Total mass mined (including overburden)	(000' tonnes)	918 273
Overall strip ratio	(w:o)	0.72
LoM	(years)	19
Ni-AC grade	(%)	0.109%
Co-AC grade	(%)	0.004%

Processing, smelting and refining

Presented below in Table 22-3 are the process recovery and concentrate grade assumptions as discussed in previous sections of this report, as well as the resulting concentrate tonnages for each. Figure 22-1 illustrates concentrate tonnage production for the flotation concentrate over the life of mine. Table 21-7 presents a unit refining, treatment and penalty charges.

Table 22-3: Process Recovery Assumptions

Description	Units	Total
LoM feed tonnage	('000 tonnes)	528 030
Plant through-put per day	(tonnes/day)	90 000
Flotation concentrate (Ni-Co)		
Ni recovery	(%)	80%
Ni concentrate Ni grade	(%)	28%
Co recovery	(%)	70%
Ni concentrate Co grade	(%)	0.90%
Contained Ni	(tonnes)	462 000
Contained Ni	(M lb)	1 018
Contained Co	(tonnes)	13 000
Contained Co	(M lb)	29
Nickel concentrate tonnage	(000' tonnes)	1 649
Magnetite concentrate		
Magnetite recovery	(%)	90%
Fe grade	(%)	60%
Magnetite concentrate tonnage	(000' tonnes)	29 000

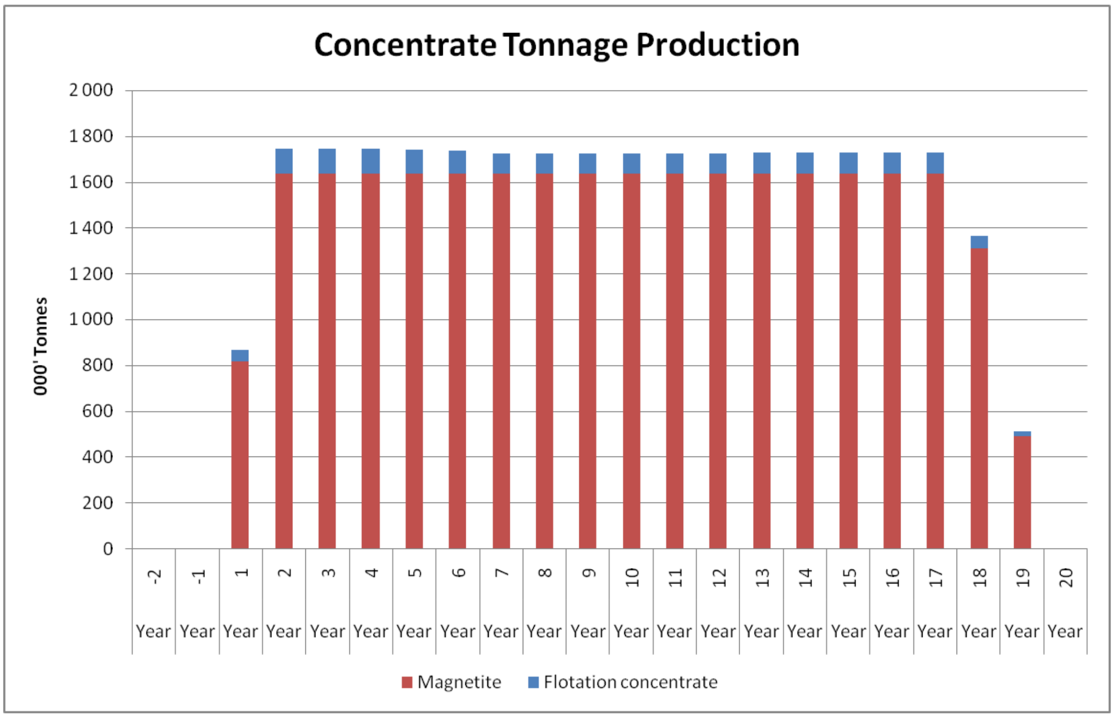

Figure 22-1: LoM flotation and magnetite concentrate tonnage production

22.1.3 Cash flow projections

A valuation of the Project has been derived based on the application of Discounted Cash Flow (DCF) techniques to the pre-tax, pre-finance cash flow based on the inputs and assumptions presented in this and previous sections of this report. All figures are presented in real terms.

In summary, at a Ni price of USD9/lb and an 8% discount rate the project has a pre-tax, pre-finance NPV of USD1045M for production of both a nickel-cobalt and magnetite concentrate. A summary of the results of the cash flow modelling and valuation are presented in Table 22-4 and Figure 22-2. A summary cashflow is presented in Table 22-5.

Table 22-4: DCF modelling and valuation (Ni price USD9 / lb)

Description	Units	Total
Gross Revenue	(USDM)	11 760
Operating costs / t total material	(USDM)	5.9
Capital costs	(USDM)	1 668
Net pre-tax cashflow	(USDM)	3 468
Payback period	(Production years)	5.5
Pre-tax pre-finance NPV (8%)	(USDM)	1 045
IRR	(%)	19.9%

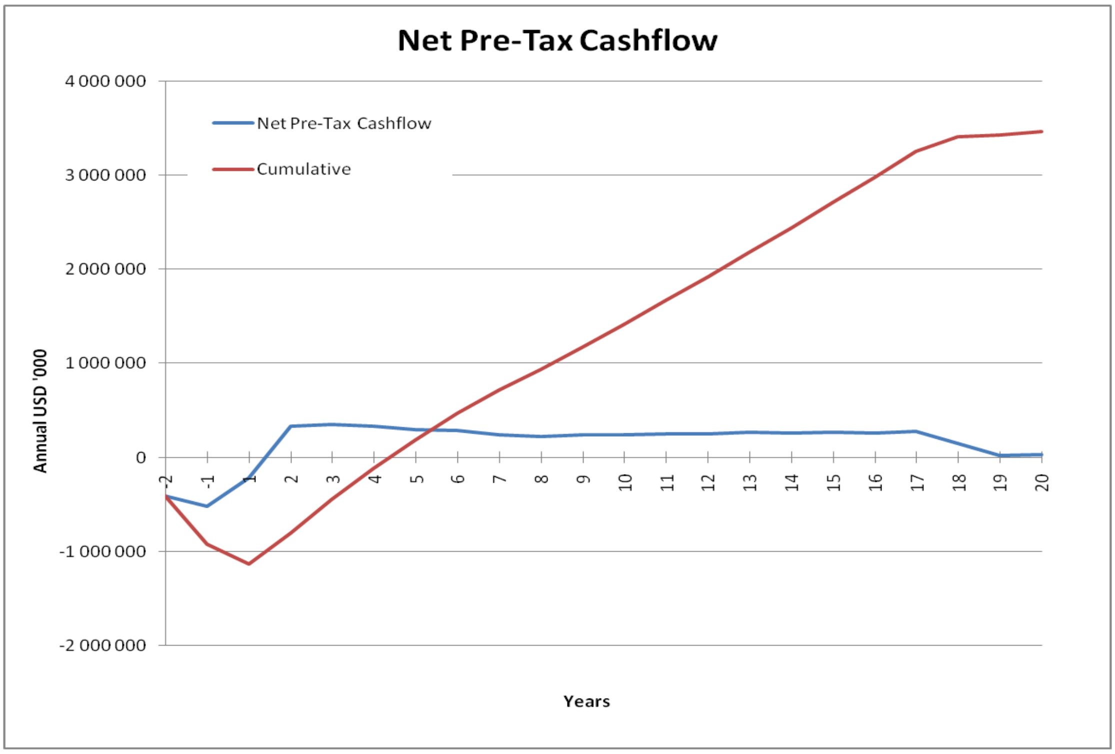

Figure 22-2: Net pre-tax cashflow for Ni-Co flotation and magnetite concentrate production

Table 22-5: Summary Cashflow

SE355 Rönnbäcken TEM	Assumptions	Units	Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20
PHYSICALS																									
Mining																									
Ore tonnes		(000' tonnes)	528 030	0	0	15 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	24 000	9 030	0
Ore grade_Ni AC		(%)	0,109%	0,000%	0,000%	0,119%	0,128%	0,129%	0,127%	0,123%	0,118%	0,103%	0,102%	0,101%	0,103%	0,102%	0,104%	0,106%	0,108%	0,110%	0,106%	0,107%	0,083%	0,074%	0,000%
Ore grade_Co		(%)	0,004%	0,000%	0,000%	0,005%	0,004%	0,005%	0,004%	0,004%	0,003%	0,003%	0,003%	0,003%	0,003%	0,002%	0,002%	0,002%	0,004%	0,004%	0,004%	0,004%	0,004%	0,005%	0,000%
Ore grade_Fe		(%)	4%	0%	0%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	0%
Mass waste		(000' tonnes)	379 943	0	0	21 200	31 000	31 000	26 000	25 000	25 000	25 000	25 000	23 000	18 000	18 000	13 715	15 085	17 300	24 700	14 500	21 293	4 000	1 150	0
Total mass		(000' tonnes)	907 973	0	0	36 200	61 000	61 000	56 000	55 000	55 000	55 000	55 000	53 000	48 000	48 000	43 715	45 085	47 300	54 700	44 500	51 293	28 000	10 180	0
Strip ratio		(w:o)	0,72	0,00	0,00	1,41	1,03	1,03	0,87	0,83	0,83	0,83	0,83	0,77	0,60	0,60	0,46	0,50	0,58	0,82	0,48	0,71	0,17	0,13	0,00
Overburden removed		(000' tonnes)	10 300	0	10 300	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing																									
Ore tonnes		(000' tonnes)	528 030	0	0	15 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	30 000	24 000	9 030	0
Ni-Co flotation circuit																									
Ni recovery		(%)	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%
Co recovery		(%)	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%
Ni grade in concentrate		(%)	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%
Co grade in concentrate		(%)	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%	0,9%
Contained Ni		(000' tonnes)	461,8	0,0	0,0	14,2	30,7	30,8	30,5	29,5	28,3	24,8	24,5	24,3	24,7	24,6	24,9	25,3	25,8	26,4	25,5	25,7	15,9	5,3	0,0
Contained Co		(000' tonnes)	13,1	0,0	0,0	0,5	0,9	1,0	0,9	0,8	0,7	0,6	0,6	0,6	0,6	0,5	0,5	0,5	0,8	0,8	0,9	0,9	0,6	0,3	0,0
Concentrate tonnage		(000' tonnes)	1 649,1	0,0	0,0	50,8	109,5	110,2	108,9	105,3	101,1	88,5	87,6	86,9	88,3	87,8	88,8	90,5	92,2	94,1	91,1	91,7	56,7	19,0	0,0
Magnetite circuit																									
Fe recovery		(%)	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Mass yield		(%)	5,5%	0,0%	0,0%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	5,5%	0,0%
Concentrate tonnes		(000' tonnes)	28 802	0	0	818	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 636	1 309	493	0
Concentrate grade		(%)	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%
REVENUE																									
Commodity price_Ni		(USD/lb)	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9
Commodity price_Co		(USD/lb)	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15
Commodity price_Fe		(USD/t)	105	119	115	110	104	104	104	104	104	104	104	104	104	104	104	104	104	104	104	104	104	104	104
Gross revenue_Ni		(MUSD)	8 521	0	0	262	566	569	563	544	522	457	453	449	456	453	459	468	477	486	471	474	293	98	0
Gross revenue_Co		(MUSD)	239	0	0	9	16	18	16	15	12	11	11	11	11	9	9	9	14	15	17	17	11	5	0
Gross revenue_Fe		(MUSD)	3 000	0	0	90	170	170	170	170	170	170	170	170	170	170	170	170	170	170	170	170	136	51	0
Total gross revenue		(MUSD)	11 760	0	0	361	752	757	749	729	705	639	634	631	638	633	638	647	660	672	658	661	440	155	0
Total charges (TCRC's)		(MUSD)	-1 137	0	0	-35	-76	-76	-75	-73	-69	-61	-60	-60	-61	-60	-61	-62	-64	-65	-63	-64	-40	-14	0
Total royalty		(MUSD)	-24	0	0	-1	-2	-2	-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	0	0
Net Revenue		(MUSD)	10 599	0	0	325	675	680	673	655	634	577	573	570	576	572	576	584	595	605	593	595	400	141	0
OPERATING COSTS																									
Mining		(MUSD)	-1 639	0	-18	-65	-108	-108	-99	-97	-97	-97	-97	-94	-85	-85	-78	-80	-84	-97	-79	-92	-56	-23	0
Processing		(MUSD)	-2 654	0	0	-82	-150	-150	-150	-150	-150	-150	-150	-150	-150	-150	-150	-150	-150	-150	-150	-150	-123	-55	0
Concentrate Transport		(MUSD)	-906	0	0	-26	-53	-53	-53	-52	-52	-51	-51	-51	-51	-51	-51	-51	-51	-52	-51	-51	-40	-15	0
G&A		(MUSD)	-200	0	0	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	0
Closure		(MUSD)	-67	0	0	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-3	-1
Employment grant total		(MUSD)	3	0	1	1	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL OPERATING EXPENDITURE		(MUSD)	-5 463	0	-17	-186	-323	-324	-315	-313	-313	-312	-312	-309	-300	-300	-293	-295	-299	-312	-294	-307	-232	-107	-1
TOTAL CASH COST NET OF BYPRODUCT CREDITS		(USD / lb Ni)	3,55	0,00	0,00	4,21	3,37	3,35	3,26	3,32	3,44	3,76	3,79	3,74	3,53	3,58	3,41	3,42	3,38	3,55	3,27	3,49	3,84	5,84	0,00
CAPITAL COSTS																									
Flotation Circuit (incl 22% contingency)		(MUSD)	-962	-337	-385	-241	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Magnetite Circuit		(MUSD)	-87	-30	-35	-22	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Infrastructure		(MUSD)	-305	-41	-80	-29	-17	0	-17	-24	-17	0	-17	0	-17	0	-17	0	-17	0	-17	0	0	0	0
Sustaining Capital		(MUSD)	-313	0	0	-12	-4	-4	-4	-20	-20	-20	-20	-20	-20	-20	-20	-20	-20	-20	-20	-12	-12	-12	-12
Working Capital		(MUSD)	0	0	0	-47	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	47
TOTAL CAPITAL EXPENDITURE		(MUSD)	-1 668	-408	-499	-350	-21	-4	-21	-44	-37	-20	-37	-20	-37	-20	-37	-20	-37	-20	-37	-12	-12	-12	34
CASHFLOW, PRE-TAX, PRE-FINANCE																									
Net Revenue		(MUSD)	10 599	0	0	325	675	680	673	655	634	577	573	570	576	572	576	584	595	605	593	595	400	141	0
Operating Expenditure		(MUSD)	-5 463	0	-17	-186	-323	-324	-315	-313	-313	-312	-312	-309	-300	-300	-293	-295	-299	-312	-294	-307	-232	-107	-1
Capital Expenditure		(MUSD)	-1 668	-408	-499	-350	-21	-4	-21	-44	-37	-20	-37	-20	-37	-20	-37	-20	-37	-20	-37	-12	-12	-12	34
Net Pre-Tax Cashflow		(MUSD)	3 468	-408	-516	-211	332	352	337	298	285	245	224	241	239	252	247	269	260	273	262	277	155	22	34
NPV	8%	(MUSD)	1 045																						
IRR	8%	(%)	19,9%																						

22.1.4 Project Sensitivities

Introduction

For illustrative purposes the following analysis presents the sensitivity of the Project under various scenarios.

Single Parameter

Figure 22-3 shows the varying NPV for varying single parameter sensitivities at an 8% discount rate for commodity price, operating costs, capital costs, SEK:USD exchange rate and Ni recovery. In addition, Table 22-6 presents the Project valuation sensitivity under various nickel price scenarios.

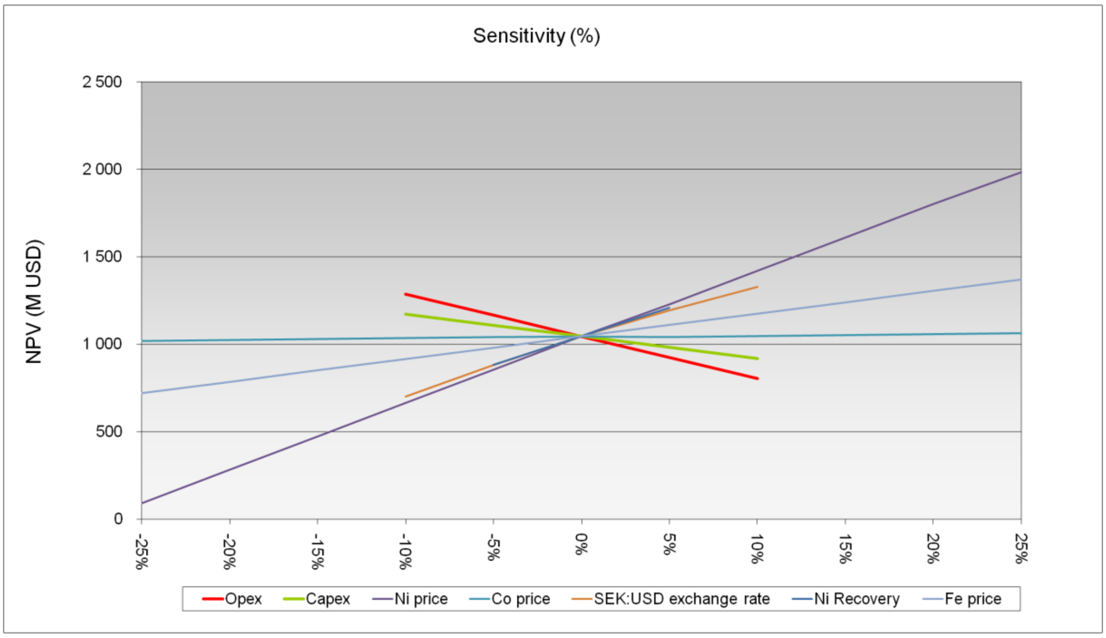

Figure 22-3: Single parameter sensitivity.

Table 22-6: Project valuation sensitivity under different nickel price scenarios

Nickel Price (USD / lb)								
Description	Unit	7	8	**9** **base case**	10	11	12	13
Net pre-tax cashflow	(MUSD)	1,577	2,522	3,467	4,393	5,338	6,264	7,208
NPV (@ 8% Discount Rate)	(MUSD)	195	620	1,045	1,461	1,885	2,301	2,726
IRR	(%)	10.5	15.4	19.9	24.0	27.9	31.6	35.2
Payback	(Production years)	7.5	5.4	4.4	3.8	3.3	3.0	2.7

Twin Parameter

Table 22-7 shows the sensitivity of the Project to simultaneous changes in two parameters, specifically; nickel recovery and discount rate, nickel price and discount rate, iron price and discount rate, cobalt price and discount rate, SEK:USD exchange rate and discount rate, operating cost and discount rate and capital costs and discount rate respectively.

Table 22-7: Twin parameter sensitivities

NPV (MUSD)		Ni Recovery Sensitivity				
		-4%	-2%	0%	2%	4%
	4%	1 733	1 829	1 924	2 019	2 114
	6%	1 269	1 347	1 426	1 504	1 582
Discount Rate	8%	914	980	1 045	1 111	1 176
	10%	641	696	752	807	862
	12%	429	476	523	570	617

NPV (MUSD)		Nickel Price Sensitivity				
		-20%	-10%	0%	10%	20%
	4%	813	1 369	1 924	2 467	3 022
	6%	510	968	1 426	1 873	2 331
Discount Rate	8%	281	663	1 045	1 419	1 801
	10%	107	429	752	1 067	1 390
	12%	(27)	248	523	792	1 067

NPV (MUSD)		Iron Price Sensitivity				
		-20%	-10%	0%	10%	20%
	4%	1 539	1 731	1 924	2 116	2 309
	6%	1 111	1 268	1 426	1 583	1 740
Discount Rate	8%	784	915	1 045	1 176	1 306
	10%	533	642	752	861	970
	12%	338	430	523	616	708

NPV (MUSD)		Cobalt Price Sensitivity				
		-20%	-10%	0%	10%	20%
	4%	1 893	1 908	1 924	1 927	1 943
	6%	1 400	1 413	1 426	1 428	1 441
Discount Rate	8%	1 024	1 034	1 045	1 047	1 058
	10%	734	743	752	754	763
	12%	508	515	523	525	532

NPV (MUSD)		SEK:USD Exchange Rate Sensitivity				
		-8%	-4%	0%	4%	8%
	4%	1 560	1 750	1 924	2 085	2 233
	6%	1 114	1 276	1 426	1 563	1 691
Discount Rate	8%	775	915	1 045	1 165	1 276
	10%	513	638	752	857	954
	12%	311	421	523	617	703

NPV (MUSD)		Operating Cost Sensitivity				
		-8%	-4%	0%	4%	8%
	4%	2 206	2 065	1 924	1 782	1 641
	6%	1 658	1 542	1 426	1 309	1 193
Discount Rate	8%	1 238	1 142	1 045	949	852
	10%	914	833	752	670	589
	12%	661	592	523	454	385

NPV (MUSD)		Capital Cost Sensitivity				
		-8%	-4%	0%	4%	8%
	4%	2 038	1 981	1 924	1 867	1 809
	6%	1 533	1 479	1 426	1 372	1 318
Discount Rate	8%	1 146	1 096	1 045	995	944
	10%	847	799	752	704	656
	12%	614	568	523	477	432

23 ADJACENT PROPERTIES

Not applicable.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data available concerning the Project.

25 INTERPRETATION AND CONCLUSIONS

SRK understands that the Company is proposing to undertake a pre-feasibility study commencing in Q1 2012, with completion expected by late Q1 2013. The budget for the study is some USD8.5M, excluding overheads. Details of these costs are presented in Table 25-1.

Subject to the results of the pre-feasibility study, the Company expects to commence a full feasibility study in Q2 2013 for completion towards the end of 2014.

Table 25-1: PFS Budget (*excluding overheads)

	Total PFS Budget*
Planning	$100,000
Geology & Resources	1,440,000
Mining	690,000
Processing	2,300,000
Infrastructure/Services	840,000
Waste Rock/Tailings Mgmt.	380,000
Water Management	430,000
Social/Environment	281,000
Manpower	20,000
Financial	510,000
Contingency	1,503,000
Total Project Budget	**$8,494,000**

A preliminary Project development schedule has been developed by the Company and suggests that production could begin during Q4 2016 (Figure 25-1).

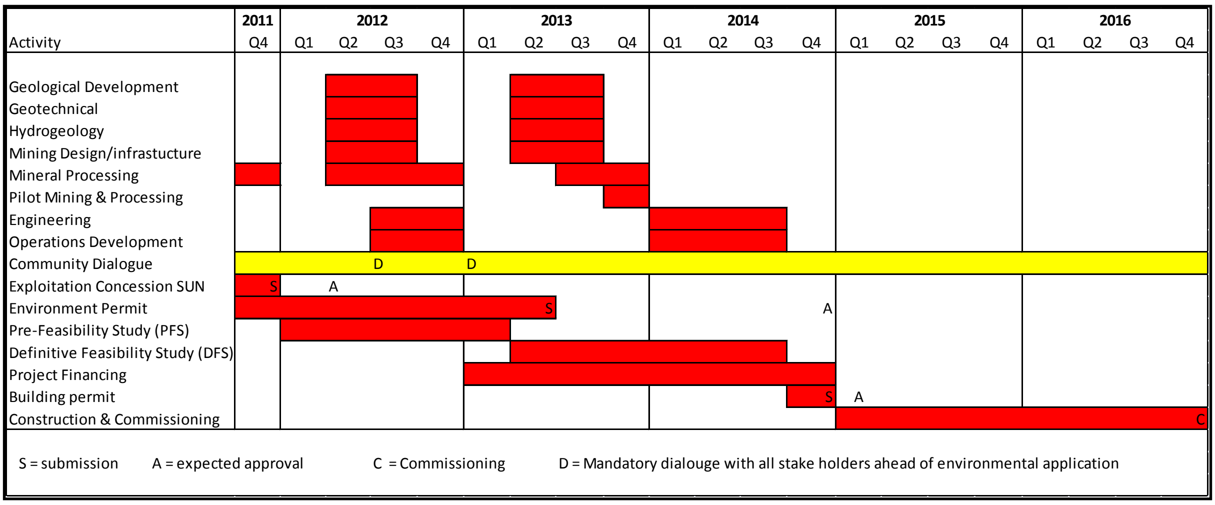

Figure 25-1: Preliminary Project Development Schedule, developed by the Company

This schedule assumes that the environmental permit application will be submitted during Q2 2013 after completion of the Pre-Feasibility Study, and that approval will take 6 quarters. The Feasibility Study will be conducted during this time, completing a significant amount of detailed engineering and initiating procurement for long lead items such as mills, crushers and transformers. Thereafter and subject to approval of all the necessary permitting, the Company expect construction to commence in Q1 2015 and to last for approximately two years.

SRK notes that achieving the construction timeframe will be dependent upon early ordering of long lead items and the delivery periods prevailing at that time. The development schedule will be re-assessed during the course of the Pre-Feasibility Study.

25.1 Risks and Opportunities

In undertaking the technical and economic appraisal of the Project, certain risks and opportunities relating to the development of the project have been identified, the most material of which are commented on below.

25.1.1 Risks

There are a number of risks inherent to the mining industry, including the stability of the markets, uncertainties related to Mineral Resource and Ore Reserve estimation, equipment and production performance. The specific risks SRK has identified relating to Rönnbacken are summarised below.

Deposition of tailings into an existing lake, the Gardiken hydroelectric reservoir, will involve technical challenges during design and construction of the tailings dam, given the seasonal oscillations of the lake water level. These lake level oscillations will also present challenges during operation and post closure, to ensure that any flushing of pore water within the permeable dam wall does not results in the transport of contaminants and excess suspended solids into the lake. Further, it is not clear at this stage how re-habilitation of the tailings area will be carried out and it is suspected by the Company that this may require new and innovative methods.

The edges of the Sundsberget and Rönnbäcksnäset open pits will lie close to the edge of the lake. Geotechnical and hydrogeological studies of the charater of the bedrock in these areas is at an early stage and hence water influx and effect on pit slope stability are as yet poorly understood.

There may be certain technical challenges with transporting a magnetite concentrate of the fineness considered as part of this study. It may be necessary to consider a pelletizing the magnetite concentrate, which could involve considerable additional capital cost.

Public roads between the mine site and port will be subject to intensive use for sulphide and magnetite concentrate transport. The Company have estimated that 130-140 truck round-trips will be required per, implying a truck passing every fifth to sixth minute. This level of public road use will require negotiation with local stakeholders and permitting authorities and also upgrade and continual maintanence of roads.

The mine road between the Sundsberget pit and the process plant on Rönnbäcksnäset Island, will pass over the lake at a narrow point, where the rate of water through-flow is highest. The road will be constructed using a series of culverts, which will be required to withstand seasonal flooding.

25.1.2 Opportunities

SRK consider there to be specific opportunity to improve project economics as follows:

Exploration drilling should be progressed to evaluate the potential that still exists within the existing exploration permit areas. In addition, the Rönnbäcksnäset and Sundsberget deposits are open at depth.

Metallurgical performance could be improved to produce a higher grade Ni concentrate, which may attract improved terms including payability for precious metals and platinum group metals.

As tailings design is at an early stage, optimisation could further reduce capital cost requirements.

26 RECOMMENDATIONS

Based on the review of available data and the results of the work carried out as part of this study, SRK recommends that work on the Project continue to pre-feasibility stage. Notably, SRK has recommeneded that the following work is undertaken as part of this study:-

- Targeted resource delineation drilling to improve confidence in some areas of the deposits
- Hydrogeological drilling around the pit and around the project site generally to characterise hydrogeological regimes. Data analysis and predictive modelling, where appropriate.
- Geotechnical drilling to provide additional information for the mine rock mass models, to select locations for surface infrastructure and to characterise stability of pit walls and tailings dam locations.
- Comminution drilling to provide large diameter core for comminution domaining characterisation, variability and autogenous grinding testwork.
- Lab-scale metallurgical work to optimise nickel and magnetite byproduct flowsheet development for designated ore blends and to characterise variability across the resource, including mineralogy.
- Further geometallurgical domaining of the resource, based on geochemistry, mineralogy and metallurgical testing to feed into mining and metallurgical planning activities.
- Scheduling and sequencing of construction (tailings dams, pit dams, infrastructure) relative to the mine plan to confirm availability of mobile mining equipment, material quantity/quality and appropriate costing.
- Completion and submission of the Sundsberget exploitation concession application
- Testwork to further characterise tailings and mine waste rock.
- Laboratory scale revegetation studies on mine waste products.
- Further work to define a decommissioning and closure plan.
- Continued collection of baseline data for the project environmental permit application
- Completion of the preliminary social impact assessment.
- Addition of multiple certified standards in the sample stream to improve QA/QC protocol.
- Further development of the mine block model, pit shells, mine production plan, operations and infrastructure requirements.
- Modelling and determination of the tailings dam design options.
- Minipiloting of nickel and byproduct flowsheets.
- Lab and mini-pilot scale dewatering and handling testwork to further characterise product preparation and transport.
- Engineering and preliminary design of mine and process plant with associated site and external infrastructure.
- Assessment of environmental impact of mining and mineral processing activities on the environment, for example water and air emissions, noise, vibration and dusting
- Further development of the health and safety risk register.

- Assessment of operations development requirements, such as human resources, operations management, information management.
- Further development of major contracted activities positions, for example mining, power, transport, land ownership/water rights.
- Further work on project development and product marketing strategies.

Certainly, in SRK’s opinion, the commissioning of a pre-feasibility study is justified by the potential of the Project and the timing and budgets proposed for this by the Company are reasonable given the work planned to be undertaken which includes the work listed above. Further, whilst the justification for a full feasibility study following this will be dependent upon the results of the pre-feasibility study, the preliminary budget proposed for this is of the correct order of magnitude for a Project of this nature and location.

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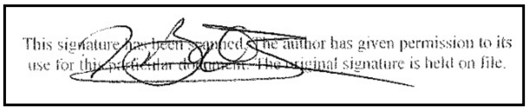
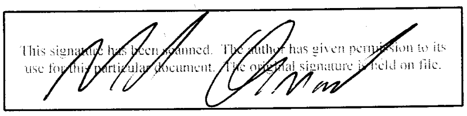
For and on behalf of SRK Consulting (UK) Limited



Johan Bradley,
Managing Director & Principal Geologist,
SRK Consulting (Sweden) AB



Mike Armitage,
Chairman & Corporate Consultant,
SRK Consulting (UK) Limited

CERTIFICATE

To Accompany the Report Entitled
"Preliminary Economic Assessment for the Rönnbäcken Nickel project, Sweden"
Dated December 2011

I, **Johan Bradley**, do hereby certify that:

1. I reside at Mässgatan 11, Ursviken, SE-93235, Sweden.

2. I am a graduate from the University of Oxford, UK, with an Honours BA. degree in Geology, awarded in 1996 and also have a Masters degree (MSc) in Mineral Deposit Evaluation, specialising in Mineral Exploration from the Royal School of Mines, Imperial College, University of London, UK, awarded in 1993. I have practised my profession continuously since 2000.

3. I am a Chartered Geologist (CGeol), Fellow of the Geological Society of London (FGS) and a member of the European Federation of Geologist (EurGeol).

4. I am a Senior Geologist with SRK Consulting (Sweden) AB, a firm of consulting mining engineers and also Managing Director.

5. I have experience with the evaluation of nickel sulphide prospects.

6. I am a Qualified Person for the purposes of NI 43-101, I am the main author of this report and I am responsible for the sections on geology and economic analysis.

7. I have visited the property most recently in February 2011 and once prior to this in March 2010.

8. I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this report.

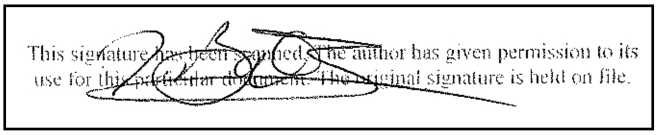
9. Neither I, nor any affiliated entity of mine, is at present under an arrangement or understanding, nor expects to become, an insider, associate, affiliated entity or employee of Nickel Mountain Resources AB, or any associated or affiliated entities.

10. Neither I, nor any affiliated entity of mine, own either directly or indirectly, nor expect to receive, any interest in the properties or securities of Nickel Mountain Resources AB, or any associated or affiliated companies.

11. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from Nickel Mountain Resources AB, or associated or affiliated companies.

12. I have read NI 43-101 and Form 43-101F1 and have prepared the technical report in compliance with these and in conformity with generally accepted International mining industry practices.

13. As of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.



Johan Bradley, MSc, FGS CGeol, EurGeol
5th December 2011

CERTIFICATE

To Accompany the Report Entitled
"Preliminary Economic Assessment for the Rönnbäcken Nickel project, Sweden"
Dated December 2011

I, **Mike Armitage,** do hereby certify that:

1. I reside at Maesaeson House, Peterston-Super-Ely, Vale of Glamorgan CF5 6NE, Wales, UK.

2. I am a graduate from the University of Wales, College Cardiff with an Honours BSc. degree in Mineral Exploitation, (Specializing in Mining Geology) awarded in 1983 and also have a PhD from Bristol University in Structural and Resource Geology awarded in 1993. I have practised my profession continuously since 1983.

3. I am a Member of the Institution of Materials Mining and Metallurgy and I am a Chartered Engineer.

4. I am a Principal Mining Geologist with SRK (UK) Ltd, a firm of consulting mining engineers and also Chairman.

5. I have experience with the evaluation of nickel sulphide prospects inclusive of resource estimation techniques.

6. I am a Qualified Person for the purposes of NI 43-101 and I am responsible for the review of all aspects of this report.

7. I have not visited the property.

8. I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this report.

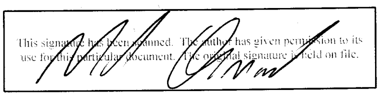
9. Neither I, nor any affiliated entity of mine, is at present under an arrangement or understanding, nor expects to become, an insider, associate, affiliated entity or employee of Nickel Mountain Resources AB, or any associated or affiliated entities.

10. Neither I, nor any affiliated entity of mine, own either directly or indirectly, nor expect to receive, any interest in the properties or securities of Nickel Mountain Resources AB, or any associated or affiliated companies.

11. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from Nickel Mountain Resources AB, or associated or affiliated companies.

12. I have read NI 43-101 and Form 43-101F1 and have prepared the technical report in compliance with these and in conformity with generally accepted International mining industry practices.

13. As of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.



Dr Mike Armitage, BSc, MIMMM, CEng
5th December 2011

CERTIFICATE

To Accompany the Report Entitled
"Preliminary Economic Assessment for the Rönnbäcken Nickel project, Sweden"
Dated December 2011

I, **Howard Baker**, do hereby certify that:

1. I work at SRK Consulting (UK) Ltd, 5th Floor, Churchill House, Churchill Way, Cardiff CF10 3HH, UK.

2. I am a graduate from the Oxford Brookes University, UK, with a degree in Applied Geology, awarded in 1994 and also a Masters degree (MSc) in Mineral Resources from Cardiff University, UK, awarded in 1995. I have practised my profession for a total of 13 years since my graduation from university.

3. I am a Member of the Australasian Institute of Mining and Metallurgy (AusIMM);

4. I am a Principal Mining Geologist with SRK Consulting (UK) Ltd, a firm of consulting mining engineers.

5. I have experience with the evaluation of nickel sulphide prospects inclusive of resource estimation techniques.

6. I am a Qualified Person for the purposes of NI 43-101, I am a co-author of the report and take overall responsibility for the Mineral Resource Estimate.

7. I have not visited the property.

8. I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this report.

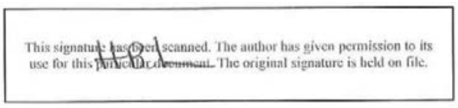
9. Neither I, nor any affiliated entity of mine, is at present under an arrangement or understanding, nor expects to become, an insider, associate, affiliated entity or employee of Nickel Mountain Resources AB, or any associated or affiliated entities.

10. Neither I, nor any affiliated entity of mine, own either directly or indirectly, nor expect to receive, any interest in the properties or securities of Nickel Mountain Resources AB, or any associated or affiliated companies.

11. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from Nickel Mountain Resources AB, or associated or affiliated companies.

12. I have read NI 43-101 and Form 43-101F1 and have prepared the technical report in compliance with these and in conformity with generally accepted International mining industry practices.

13. As of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.



Howard Baker. AusIMM, Principal Mining Geologist
5th December 2011

CERTIFICATE

To Accompany the Report Entitled
"Preliminary Economic Assessment for the Rönnbäcken Nickel project, Sweden"
Dated December 2011

I, **Dr. David Pattinson**, do hereby certify that:

1. I reside at Glenbrook, Llandow Village, Cowbridge, South Glamorgan, UK.

2. I am a graduate from the Birmingham University, UK, with a Bachelor's Degree in Minerals Engineering and a PhD in Fine Coal Processing. I have practiced my profession continuously since 1981.

3. I am a registered Chartered Engineer in the UK, and a Member of the UK Institute of Materials, Minerals & Mining.

4. I am a Principal Metallurgist with SRK Consulting (UK) Ltd, a firm of consulting mining engineers.

5. I have experience with the evaluation of nickel sulphide prospects.

6. I am a Qualified Person for the purposes of NI 43-101, I am a co-author of this report and responsible for metallurgy and mineral processing.

7. I have not visited the property.

8. I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this report.

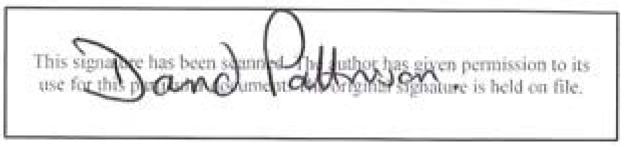
9. Neither I, nor any affiliated entity of mine, is at present under an arrangement or understanding, nor expects to become, an insider, associate, affiliated entity or employee of Nickel Mountain Resources AB, or any associated or affiliated entities.

10. Neither I, nor any affiliated entity of mine, own either directly or indirectly, nor expect to receive, any interest in the properties or securities of Nickel Mountain Resources AB, or any associated or affiliated companies.

11. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from Nickel Mountain Resources AB, or associated or affiliated companies.

12. I have read NI 43-101 and Form 43-101F1 and have prepared the technical report in compliance with these and in conformity with generally accepted International mining industry practices.

13. As of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.



Dr. David Pattinson. BSc, MIMMM, CEng
Principal Process/Metallurgical Engineer
5th December 2011