PRELIMINARY ECONOMIC ASSESSMENT FOR THE HAVERI GOLD-COPPER DEPOSIT, FINLAND

Prepared For

Palmex Mineral AB

Report Prepared by

srk consulting

SRK Consulting (Sweden) AB
SE471

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EXECUTIVE SUMMARY
PRELIMINARY ECONOMIC ASSESSMENT FOR THE HAVERI GOLD-COPPER DEPOSIT, FINLAND

1 EXECUTIVE SUMMARY

1.1 INTRODUCTION

This report comprises a preliminary economic assessment (PEA) of the Haveri Project (Haveri or the Project) in southern Finland. It has been prepared by SRK Consulting (Sweden) AB (SRK) on behalf of Palmex Mineral AB (Palmex). For Mineral Resource reporting this PEA uses the definitions and guidelines provided by the 2010 Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards on Mineral Resources and Reserves (CIM Definition Standards).

The Haveri Project is an advanced exploration project comprising the Haveri, Peltosaari, Tombstone, and Sankari-Ansomäki gold-copper ("Au-Cu") deposits, which are collectively referred to as the Haveri project ("Haveri", or the "Project"). It is located 1 km from the town of Viljakkala in Länsi-Suomen lääni County, southern Finland. Palmex Mineral AB (Palmex) currently holds a 100% of the Project.

Historic mining occurred at Haveri in the mid-19th century, and again between 1942 and 1962, utilising a combination of open pit and underground methods. Mining between 1942 and 1962 produced a total of 1.5 Mt of material with an average grade of 3.5 g/t Au and 0.5% Cu.

SRK has prepared an independent Mineral Resource estimate ("MRE") for the Project and has used this as a basis for a conceptual mining study. In addition, SRK has reviewed all other technical work completed on the Project both historically and by the Company and its contractors and consultants. This review has been undertaken to a sufficient level to enable SRK to present its own opinions on the Project and to derive an audited NPV.

The Project is at a conceptual stage but it is currently envisaged that production will be from open pit methods through conventional drill and blast, with trucking to a dedicated on-site processing facility. Whilst other mining and processing scenarios were considered as part of this PEA and are discussed briefly, only the base case is discussed in detail for the purposes of this report.

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 the CIM Definition Standards.

1.2 GEOLOGY, DATA QUALITY AND RESOURCE ESTIMATION

The Haveri Project is located within a sequence of meta-volcanic and meta-sedimentary units that constitute part of the 100 km long Tampere Schist Belt. This Belt occurs within the Svecofennian Domain of Proterozoic age (approximately 1.9 Ga) and forms part of a much larger sequence of Mid-Proterozoic supracrustal rocks which occur in the southern part of Finland. The local stratigraphy upwards from the lowermost unit comprises mafic meta-lavas, lava breccia, tuffs and tuffites, and meta-sediments (turbidites metamorphosed to mica schists). Amphibolite facies metamorphism and intense deformation have modified these units considerably.

The main mineralisation type at Haveri comprises sulphide veins from a few millimetres to tens of centimetres wide to semi-massive zones 10 m thick and 50 m long. The mineralisation chiefly comprises pyrrhotite-chalcopyrite and magnetite patches, and pyrrhotite-chalcopyrite veins and vein networks. The massive to semi-massive type grades into a disseminated type with no obvious change in the mineral assemblage, except for the decrease in the relative volume of gold-bearing minerals and amphibole (Eilu, 2012).

Currently there is no one confirmed deposit type, however, the latest study by Eilu (2012) favours an origin as a re-mobilized VMS.

A considerable amount of exploration has been undertaken at Haveri, over numerous campaigns and by several operators. This data has been reviewed by SRK and used as a basis for estimating Mineral Resources. Whilst the quality of this data is considered to be uncertain in some cases, SRK has accounted for this through appropriate Mineral Resource classification, which is discussed in further detail below.

A 2 m composite file was used in a statistical and geostatistical study (inclusive of a quantitative kriging neighbourhood analysis - QKNA) that resulted in ordinary kriging (OK) being selected as the interpolation method. The interpolation used an elliptical search following the predominant dip and dip direction of the geological domains. The results of the variography and the QKNA were utilised to determine the most appropriate search parameters.

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

1.3 MINERAL RESOURCES

Table ES 1 below presents SRKs' Mineral Resource statement for the Haveri deposit. A pit optimisation exercise was carried out based on assumed operating costs, slope angles, mining recoveries and revenue assumptions derived from SRK's experience and was used to constrain the Mineral Resource statement to that material which SRK considers has reasonable prospect for eventual economic extraction.

The statement has been classified in accordance with the CIM Definitions by the QP, Lucy Roberts (MAusIMM(CP)), who is an independent consultant with no relationship to the Company. It has an effective date of 30 July 2014.

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

SRK is not aware of any environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues that would preclude the report of the Mineral Resource presented here.

Table ES 1: Mineral Resource statement (reported above a marginal cut-off grade of 0.45 g/t Au Equivalent and within the Whittle shell)

Category Inferred	Tonnes (Mt)	Au (g/t)	Cu (%)	AuEq (g/t)	Au (MOz)
Peltosari	4	0.84	0.07	0.93	0.12
Haveri	54	0.84	0.07	0.93	1.45
Total	58	0.84	0.07	0.93	1.56

(1) The effective date of the Mineral Resource statement is 30 July 2014.

*(2) The Mineral Resource reported for Haveri was constrained within a Lerchs-Grossman pit shell defined by a marginal cut-off-grade of 0.45 g/t AuEq, a metal price for copper USD7850 / t and metal price for gold USD1510 / oz; overall slope angles of 47°; a mining recovery of 97%; a mining dilution of 5%; mining costs of USD3.5/tonne, process operating costs (inclusive of G&A costs) of USD15/tonne; transport costs of USD0.28/tonne*km; and mineral royalties of 0.15%. The gold price assumed is higher than that assumed to derive a production schedule and the NPV presented in this report and was chosen so as to encapsulate all material with potential to be exploited at some point in the reasonable future.*

*(3) Gold Equivalent (AuEq) (g/t) = 0.994456*Au (g/t) + 1.288622*Cu (%)*

1.4 MINING METHODS

SRK has evaluated the potential to mine the deposit using both open pit and underground mining methods and reviewed the available geotechnical and hydrogeological information to determine suitable slope angles and hydraulic radius. Commercial pit optimisation software was then applied to the geological block model to determine the potential optimal pit boundary for economic analysis. SRK has produced a preliminary production schedule and estimated the mining costs. Several alternative production scenarios (including underground methods) were considered at a high level. On the basis of the findings from this analysis, a “base case” was selected which considers a conventional approach to open pit mining using an excavator-truck configuration from two pits (Haveri and Peltosari), at a run of mine production rate of 1.8 Mtpa, with processing at a dedicated on-site process facility. SRK has assumed owner-operator for mining operations.

Based on an assessment of core photographs, SRK has assumed an overall slope angle for the open pits of 47°. As the data is limited, only one geotechnical zone was considered during this study and there is currently insufficient data to separately characterise the hangingwall waste, footwall waste rock and orebody. Further data capture will facilitate this.

SRK used the Whittle 4X pit optimisation software to determine the economic pit limits for the Inferred Resources in the base case option. The nested pit shells produced by Whittle are graphically presented below with the highlighted option indicating the selected pit shell.

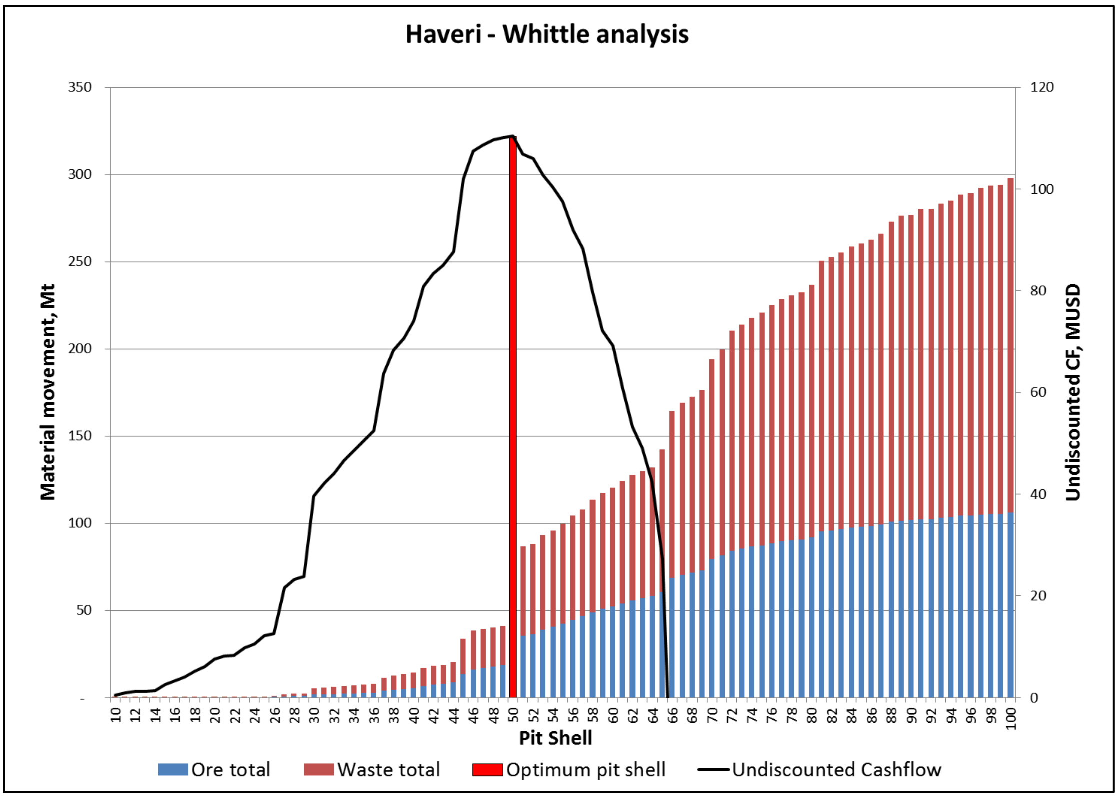

Figure ES 1: Pit optimisation results (Source: SRK, 2014)

SRK notes that the maximum undiscounted cash flow is achieved by shell 50 with the gold price 1160 USD/oz and copper price 5550 USD/t and SRK selected this shell for the further analysis. The selected pit shell is projected onto an aerial photograph in Figure ES 2 below.

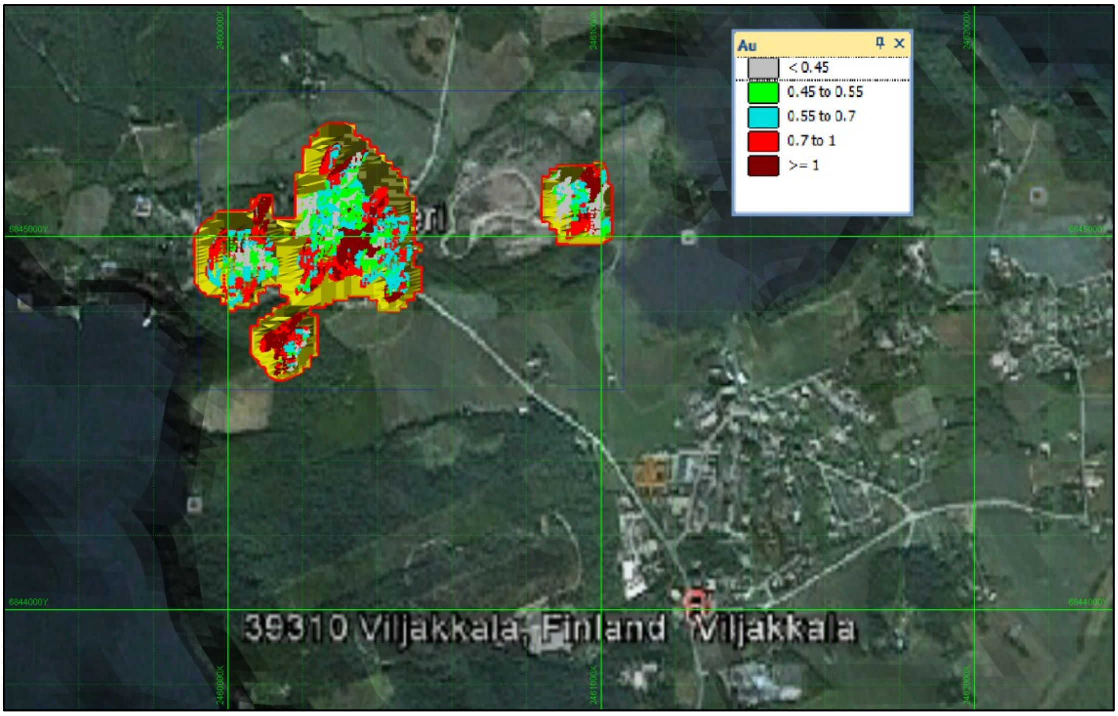

Figure ES 2: Selected pit shell with block model showing grade distrbution. Aerial photo as background (Source: SRK 2014)

The mine plan as presented in the summary cashflow model below is based on a production rate of 1.8 Mtpa which generates an overall mine life of 11 years. SRK considered a mining sequence based on average strip ratio in the final optimum pit shell – no pushbacks were selected.

SRK split the mineralised material into three categories using gold equivalent grade ("Au EQ"). The formula to calculate the equivalent gold grade:

(Au EQ) (g/t) = 0.994456 x Au (g/t) + 1.288622 x Cu (%)

The three categories are based on cut-off grade calculations as follows:

- High-grade: Au EQ > 0.70 g/t.
- Low-grade: 0.55 g/t < Au EQ < 0.7 g/t.
- Mineralised Waste: Au EQ < 0.55 g/t.

It has been assumed that the Mineralised Waste will be stockpiled for possible processing, should an increase in future gold price warrant this. This material is not included in the production schedule.

Equipment requirements have been determined using the following methods:

- 261 workings days per year and 16 working hours per day;
- truck and excavator requirements based on productivities and cycle times;
- 3 m^3 capacity excavators and 24 t articulated trucks have been assumed for rock mass movement
- drilling requirements based on 5 m benches with 115 mm blasthole drills for the ore and 10 m benches with 152 mm blasthole drills for the waste;
- ancillary equipment assumptions based on material movement and primary fleet requirements;

The mine equipment requirements and the mobile and auxiliary equipment requirements are shown on an annual basis in Section 16.6.

1.5 RECOVERY METHODS

The conceptual flowsheet design for Haveri is based on the following two process aims:

- The production of a marketable copper concentrate; and
- The production of the bulk of the tailings essentially devoid of sulphide and arsenic.

The flowsheet will therefore produce three process streams, as follows:

- A "clean" copper concentrate;
- A bulk sulphide flotation concentrate, containing the remaining sulphides. This concentrate would be cyanide leached for gold recovery, and the tailings, following cyanide detoxification, stored in a lined and capped Tailings Storage Facility ("TSF"), to prevent the generation of acid and/or toxic metal containing effluent; and
- A "clean", i.e. essentially free of sulphides and heavy metals, tailings stream for disposal in the primary TSF.

This flowsheet is identical in concept to that recently proposed by SRK for the Kopsa project in central Finland, for material of a similar Au but slightly higher Cu grade. It is also similar in concept to the flowsheet used by another operating mine in the region. In this case, the only valuable metal is gold, and so both the sulphide concentrate and the flotation (and gravity) tailings are cyanide leached – in separate circuits. Both tailings are detoxified following cyanidation, after which the sulphide tailings are stored in a lined facility, and the main flotation tailings stream is stored in a paddock facility.

SRK notes that no metallurgical testwork has yet been undertaken in support of the production of this PEA; the metallurgical parameters developed for the PEA were based on a relatively limited amount of historical data. In addition, virtually no specific engineering was conducted with regard to the process plant design, and the process plant capital and operating costs subsequently generated are very high level estimates.

1.6 TAILINGS AND WASTE MANAGEMENT

SRK has undertaken an options assessment to identify suitable storage method for the approximately 19.8 Mt of tailings material produced at the plant over the LoM.

Based upon the assessment requirements, four potential TSF outlines were mapped within the selected 5km range, utilising the natural land contours to maximise the available storage capacity while minimising embankment fill requirements. The relative locations of location options are shown in Figure ES 3 below:

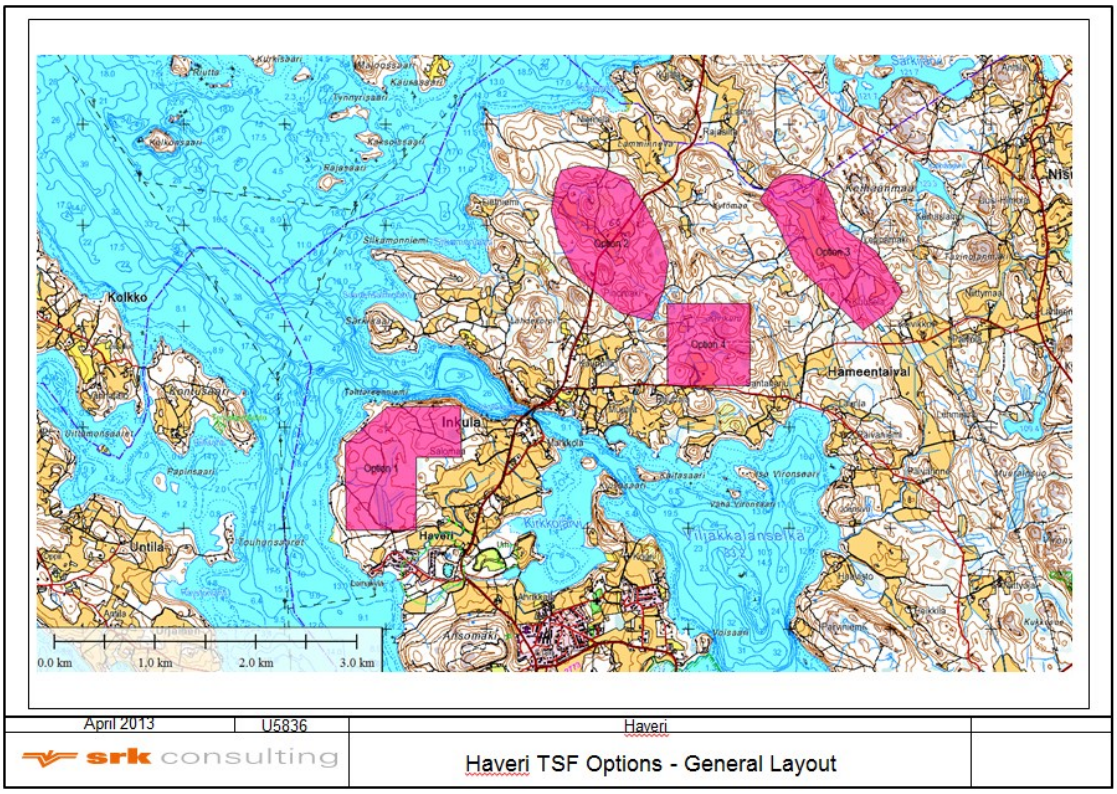

Figure ES 3: TSF location options

To take into account the environmental and social impacts of each proposed TSF development options, multi criteria analysis was undertaken, in which all sites were ranked based upon specific criterion. On the basis of this analysis, Option 3 was selected primarily due to the relatively isolated nature of the site, the minimal impacts related to dust, noise and visual disturbance and the low operating and capital costs relative to the other sites considered.

1.7 PROJECT INFRASTRUCTURE

Given the location of the project, and the fact that it hosted a historic mining and processing operation, albeit of a small scale, SRK has assumed that site access and the provision of electrical power and water to the project site will be relatively straightforward.

The project's electrical power requirements are likely to be of the order of 6-8 MW. The project's make-up (i.e. net) water requirements are likely to be of the order of 1 Mm^3/annum (equivalent to 2,800 m^3/day or 30 l/s).

1.8 HYDROGEOLOGY & GEOCHEMISTRY

Haveri is located in a peninsular of rolling hills, with a strong glacial influence that has imparted a NW-SE direction that is apparent in the shape of the lakes and hills. Preliminary catchment delineation suggests that the current pit is located on a watershed divide between two sub-catchments that drain to the nearby lakes Kyrösjärvi (to the west) and Viljakkalanselkä (to the east).

Two main groundwater zones were considered during this preliminary study, the glacial overburden and the deeper more competent bedrock. Historical groundwater elevations from nearby to the mine indicate a westerly groundwater flow direction suggesting that groundwater may be discharging into the lake.

The till is likely to play a relatively minor role in saturated groundwater flow at Haveri due to its limited thickness relative to the other rock types present. The preliminary conceptualisation indicates that the majority of groundwater inflow may originate from zones of intense fracturing within the bedrock. Hydraulic parameters have been based on values for similar hydrogeological units published in the public domain and previous SRK studies in Finland, with consideration to the preliminary conceptual model.

The range of potential groundwater and surface water inflows into Haveri is estimated by SRK to be between 5 and 32 L/s, with simple modelling of the recent pit lake recovery suggesting that actual inflows may be towards the lower range of this estimate. Surface water inflows contribute an average of 1/s of to this total, with highest surface inflow in April (3.6L/s) corresponding to the April snowmelt and lowest from November to March (0L/s) due to the below freezing temperature.

The cone of depression caused by any dewatering of this mine may extend to between 1 and 3.km away from the mine. The Haveri Class I groundwater area (which the project borders) will likely be impacted by dewatering. This groundwater area is not currently in use, apparently due to elevated Manganese concentrations.

Predicted groundwater inflows can likely be managed to ensure a "dry" pit floor though simple sump pumping, although this will need to be reviewed at PFS level in conjuncture with geotechnical requirements for slope stability.

As a preliminary assessment of the potential wastes, based on the limited available data, the waste rock material may be directly classified as non-inert. This is based on the total sulphur concentration in the waste rock which is expected to be of the order of 0.85 %, as assessed against the EU mine waste directive. Notably, the majority of the waste rock mined could potentially exceed the both the PIMA threshold for As, Co, Cu and Ni. In addition, The existing tailings dam has an historical ARDML issue in the form of metal leaching to the surrounding environment and it is highly probable that the new tailings will also be acid generating due to the potential sulphide minerals present. Several metals (Co, Cu, Fe, Ni, Zn) are still currently leaching from the historical tailings.

Further geochemical assessment will clarify this, but for the purpose of this PEA the waste rock and tailings material is regarded as potentially acid generating and that therefore treatment of the water coming from the waste rock and pits and containment facilities for the tailings will be required. In addition the mine dewatering could potentially result in the need for likely requirement to treat both the old tailings facility and any drainage waters. Alternatively, if reprocessing of tailings is an economical alternative, it might also have positive environmental impact in connection to the project.

No predictions have been undertaken for flows after closure or flows from the TSF.

1.9 ENVIRONMENTAL & SOCIAL MANAGEMENT

The Project area is currently used mainly for agriculture, forestry, tourism and housing.

Haveri is located on a relatively small peninsular in Kyrösjärvi lake and potential mining would take place near water. Kirkkojärvi and Viljakkalanselkä bays of Lake Kyrösjärvi have been impacted by Haveri mine historically and lake sediments are showing recovery. Copper has been the most important pollutant and the impact was largest soon after closure, from the mid-60´s to the mid-70’s. Haveri groundwater protection area would have to be decommissioned, if mining would take place again. This resource is not utilised currently, but it still has the official groundwater protection status.

There is a Natura 2000 object (according to Birds Directive 2009/147/EC) in the Viljakkalanselkä bay. It is not in the immediate vicinity of the potential mine, but potential impacts cannot be discounted completely given current information.

The envisaged open pits lie close to permanently inhabited Haveri and Viljakkala communities and numerous households are likely to be impacted by dust, noise and vibrations. Relocation of number of households is probably required. A suitable safety buffer zone around the mine can be defined, when more detailed information of the operations is available and dust, noise and vibration can be modelled.

Limited space and location near water are part of the Project’s key challenges. Both pollution and safety issues need special attention when mining operations near both community and water front are being planned.

The possibilities to undertake environmental and social impact assessments so that the same studies support both permitting and regional planning processes should be investigated.

1.10 CAPITAL AND OPERATING COSTS

1.10.1 OPERATING COSTS

Summary

An overview of SRK’s estimated operating costs for the major costs centres is presented in Table ES 2 and illustrated in Figure ES 4 over the Project life of mine. No overall operating cost contingency has been assumed, however contingency (and G&A) is included in processing operating costs.

Table ES 2: Overview of operating costs by major cost centre

	USD/t moved	USD/t processed	Percentage of total
Mining	3.2	6.7	29%
Processing	7.3	15.0	64%
Tailings	0.2	0.5	2%
Environmental & Closure	0.6	1.3	5%
Total	**11.3**	**23.5**	**29%**

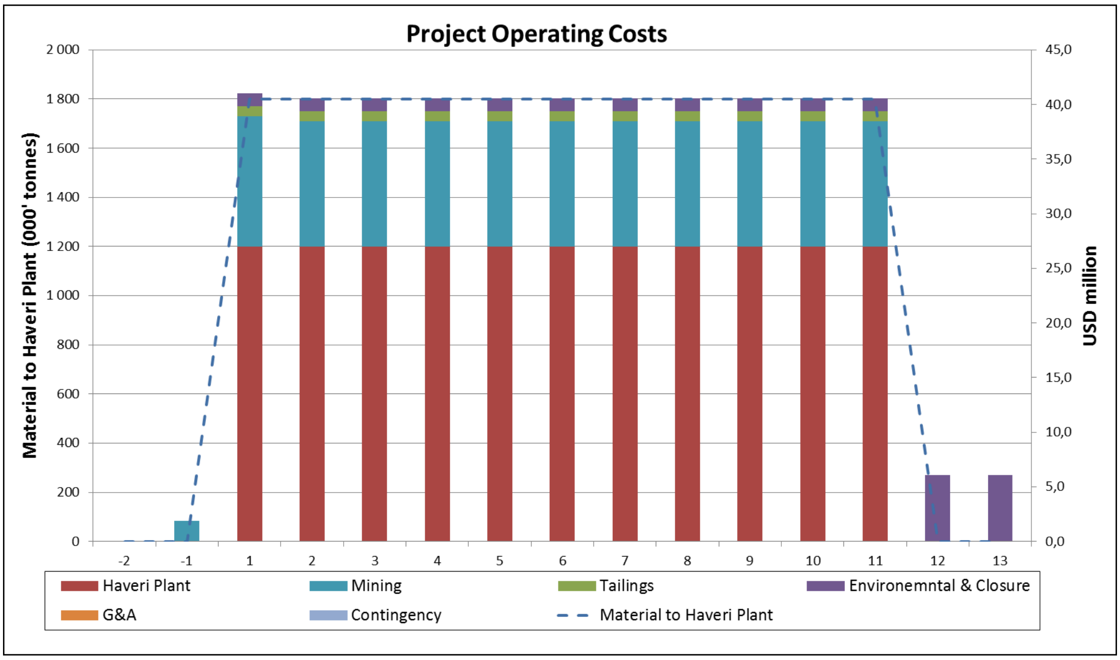

Figure ES 4: Summary of operating costs over the life of mine (Source:SRK, 2014)

Mining

SRK's estimated mining operating costs are presented below. These estimated costs are based on the selected mining production schedule (1.8 Mtpa) and corresponding equipment usage. Increasing costs with pit depth are accounted for as is the cost of re-handling material from stockpiles into haul trucks.

Table ES 3: Mine operating costs

Mining Cost Centre	USD / tonne total material
Drilling	0.04
Blasting	0.24
Loading	0.25
Hauling_In pit	0.34
Stockpile Excavation	0.11
Haulage_Mine to plant	0.00
Mobile Mining Equipment	0.27
Auxiliary Equipment	0.11
Labour	1.63
Mine Facilities & Other (incl. grade control)	0.18
Total Mining	3.16

Processing

Table ES 4 presents the assumed operating costs for processing Haveri. These costs include a provision for general administration (G&A) and contingency.

Table ES 4: Process operating costs

Processing Cost Centre	USD / tonne
Crushing, Grinding & Flotation	14.0
Cyanidation	1.00
Total Processing (includes contingency and G&A)	15.0

Tailings

SRK's estimated operating cost for the disposal of tailings is USD 0.5 per tonne of material processed. There is currently no separate provision for the treatment of the high-sulphide tails and it is assumed that this material will be blended with tailings from the bulk sulphide concentrate for deposition in the tailings facility. Further work will be required to confirm that this is alternative is acceptable to the permitting authorities.

Environmental, Rehabilitation & Closure

SRK';s estimate of the operating costs for environmental aspects and closure amount to USD 1.3 per tonne of material processed, or USD 25 million over the life of mine. The major costs items in this figure comprise USD 12 million for closure of the tailings and waste rock dump facilities and USD 13 million for water treatment.

Treatment Charges and Refining Costs

In addition to the costs presented in Table ES 2 above, the following treatment charges and refining costs (TCRC's) have been assumed.

Table ES 5: Treatment Charges and Refining Costs

TCRC's	(Unit)	Cost
Cu Treatment Charge	**(USD/t)**	63
Cu Refining Charge	**(USD/lb)**	0.063
Au Refining Charge	**(USD/oz)**	5.0

1.10.2 CAPITAL COSTS

The capital costs estimated as part of this study have been derived by SRK and are discussed in detail elsewhere in this report. The following section presents a summary of these costs, which total USD 92.2 million. SRK notes the following:

- Contingencies of 10% have been applied to all capital costs;
- Working capital has been assumed at 20% of operating costs incurred during the first year of production;
- No provision has been made for sustaining capital, which for the purposes of this study is accounted for in operating cost provisions.
- In general (with the exception of tailings construction), capital costs have been profiled with 80% of expenditure occurring in the first two years preceding production, and the remaining 20% occurring in the first year of production.

Figure ES 5 gives a breakdown of the envisaged capital expenditure over the life of mine and split between the major cost centres, including contingency and working capital.

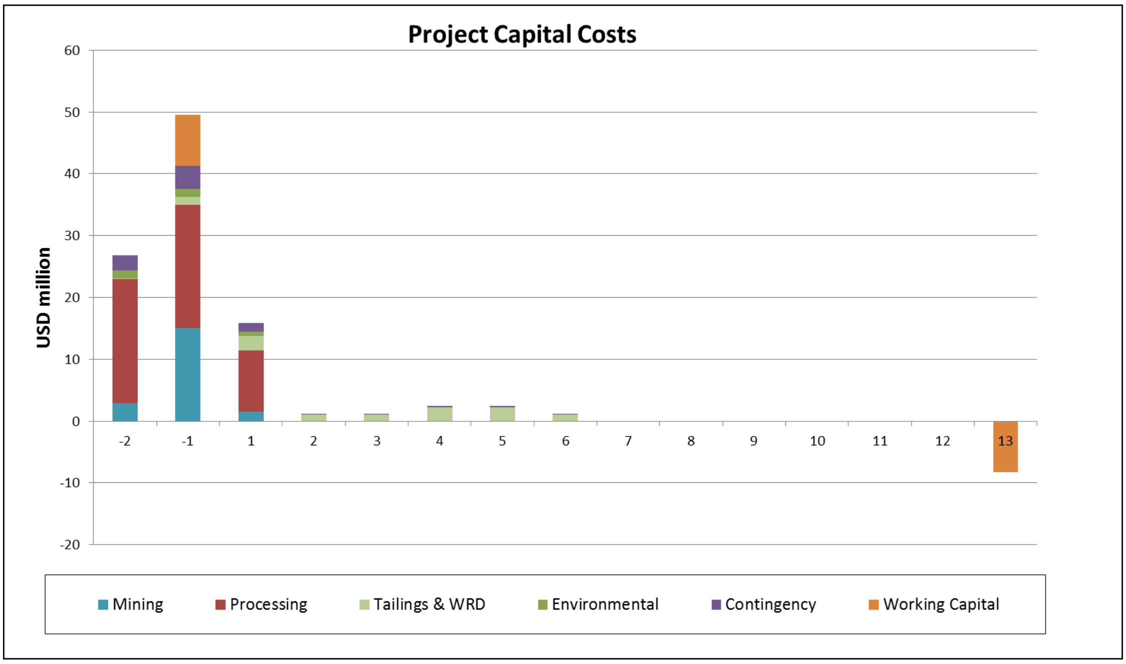

Figure ES 5: Capital cost assumptions

Table ES 6 below presents SRKs' capital cost assumptions, inclusive of a high-level breakdown under the major costs centres.

Table ES 6: Capital cost assumptions

Description	Value (USD million)
Mining	
Mine Facilities & Haulage Dispatch System	7.0
Haul Roads	0.3
Mobile Mining Equipment	10.0
Auxiliary Equipment	2.1
Total Mining	**19.4**
Processing	
Process plant (incl. EPCM & contingency)	50.0
Total Processing	**50.0**
Tailings & WRD	
Tailings construction costs	9.9
WRD Construction (incl. ground prep & liner)	1.2
Total Tailings & WRD	**11.1**
Environmental	
Water Management Facilities	1.5
Water Treatment Plants	1.6
Land purchase	0.3
Total Environment	3.3
Contingency (10%)	8.4
Total	**92.2**

1.11 ECONOMIC ANALYSIS

SRK has constructed a technical economic model (TEM) to derive a post-tax Net Present Value (NPV) for the Project which is based solely on Inferred Mineral Resources only and is therefore preliminary in nature. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves, which would also require the support of a pre-feasibility level study. There is no certainty that the reserves development, production, and economic forecasts on which this Preliminary Assessment is based will be realised.

The model is based on production from a main open pit at Haveri and a satellite pit at Peltosaari, with on-site crushing, grinding and flotation for production of a marketable copper sulphide concentrate and smelted gold doré through conventional flotation, cyanide leaching and Carbon-in-Pulp (CIP) / Carbon-in-Leach (CIL).

For the purposes of this study, the material contained within the selected pit shells has been mined at a constant production rate of 1.8Mtpa over the life of mine.

SRKs' TEM is constructed in post-tax and pre-finance form and assumes:

- a US Dollar (USD) valuation currency, with any Euro (EUR) derived costs being converted at a EUR:USD exchange rate of 1:0.75;
- a base case discount rate of 8%;
- the TEM is in real 2014 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 cashflow starts in year -2;
- working capital based on 25% of the operating costs from the first year of production;
- depreciation on a 20% declining balance basis; and
- corporate tax rate of 24.5%.

The TEM considers the revenue and cost implications of both a marketable copper sulphide concentrate and smelted gold doré. The following commodity price assumptions have been used:

- Copper USD 6,500 / tonne
- Gold USD 1,300 / troy ounce

A summary of the combined mass movement of material is presented below. It is assumed that marginal material is processed along with run of mine material.

Table ES 7: Summary of movement of material from the open pit

Mining	Unit	Value
ROM	(tonnes '000)	14 180
Marginal Material	(tonnes '000)	5 620
Waste Rock	(tonnes '000)	19 160
Glacial Ovb	(tonnes '000)	1 740
Total Material Mined	(tonnes '000)	40 700
Strip ratio	(w:o)	1.1
Life of mine	(years)	11
Grade Cu	(%)	0.08%
Grade Au	(g/t)	0.90

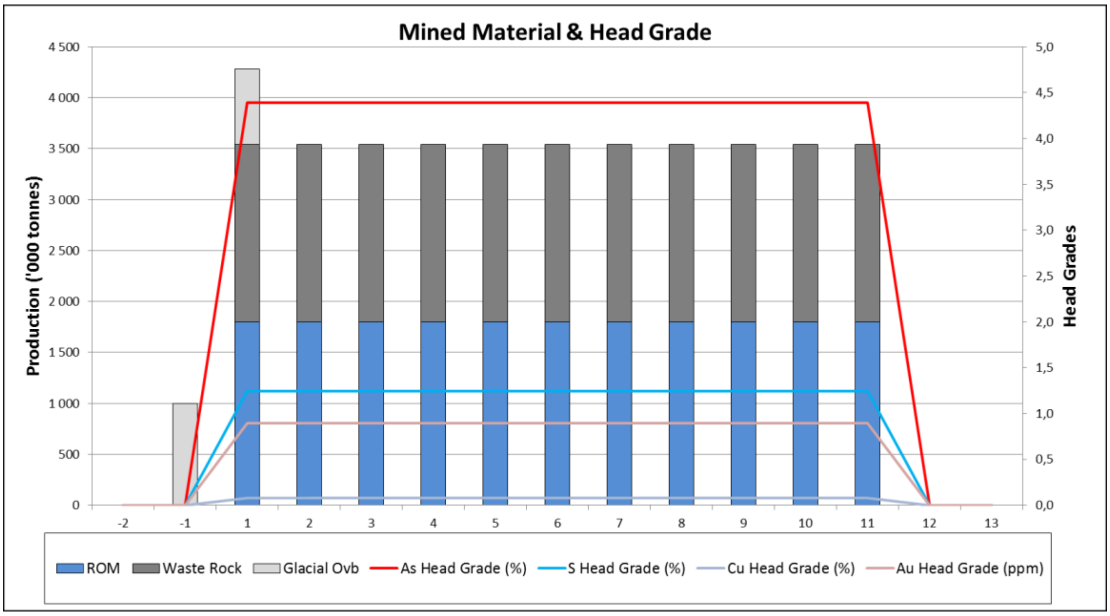

Figure ES 6: Summary of mass movement of material (Source: SRK, 2014)

Process recovery, concentrate grade and smelting and refining assumptions are presented in the two tables below.

Table ES 8: Process Design Criteria

Item		Unit	Value
RoM Production		tpa	1,800,000
Flotation Feed Grade	Cu	%	0.09
	Au	g/t	1.00
	S	%	1.24
Copper Concentrate		tpa	3,900
	Cu Rec	%	60.0
	Au Rec	%	20.0
	Cu	%	25.0
	Au	g/t	92.6
Sulphide Concentrate		tpa	45,000
	Au Rec	%	60.0
	Au	g/t	24.0
Cyanidation Recovery	Au	%	95.0
Recovery to Doré	Au	%	57.0
Overall Recovery	Cu	%	60.0
	Au	%	77.0

Table ES 9: Smelting and Refining assumptions

Item	Unit	Value
Copper Concentrate Losses & Deductions		
Cu Payable	(%)	95.0
Cu unit deduction	(%)	1.0
Au unit deduction	(g/t)	1
Leach Doré		
Au Payable	(%)	99.5

SRK notes that no penalties have been assumed for contained arsenic. For the purposes of this study, it is assumed that these costs are non-material and will be covered by the deduction, treatment and refining charges.

Figure ES 7 below provides an overview of forecast net revenue for Cu and Au over the life of mine.

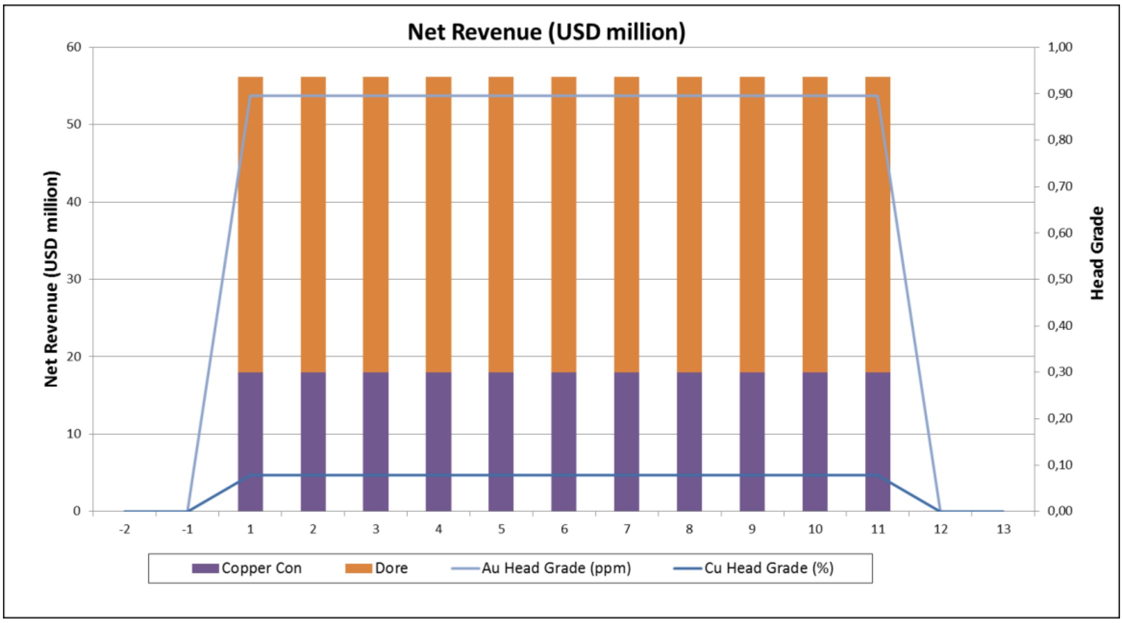

Figure ES 7: Contribution to net revenue of copper concentrate and Au doré (net of TCRC's, losses and deductions). (Source:SRK, 2014)

Forecast annualised net post-tax, pre-finance cashflow is summarised in Figure ES 8, Table ES 10 and Table ES 11 below.

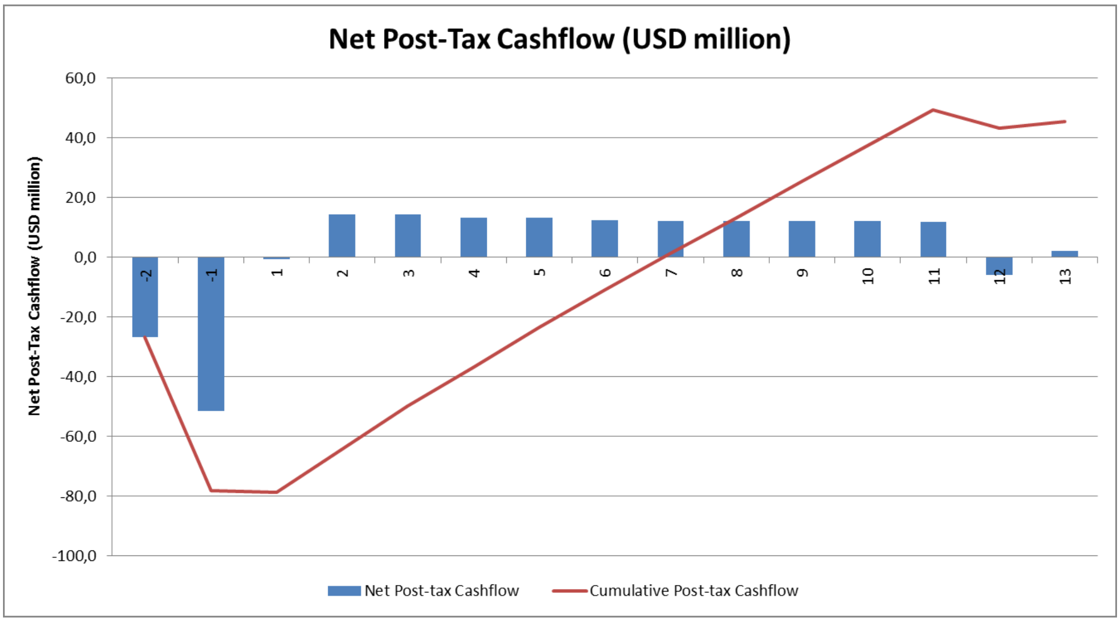

Figure ES 8: Annual and cumulative net post-tax cashflow. (Source: SRK, 2014)

Table ES 10: Summary undiscounted net-post tax cashflow

Description	Units	Total
Gross Revenue	(USDM)	622
Operating costs / t ROM	(USD/t)	23
Capital costs	(USDM)	92
Net pre-tax cashflow (undiscounted)	(USDM)	65
Net post-tax cashflow (undiscounted)	(USDM)	46
Payback period	(years)	6.5

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 and based on the inputs and assumptions already presented. All figures are presented in real terms.

In summary, for the base case, at a Cu price of USD 6 500/tonne and Au price of USD 1 300 / troy ounce, and an 8% discount rate the project has a **post-tax, pre-finance NPV of USD -1.4 million (IRR 8%)** for production of both a copper concentrate and Au doré.

Table ES 11: Summary Annual Cash Flow

SE471 Haveri PEA			Year														
Summary Annual Cashflow	**Units**	**Total**	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
CASHFLOW																	
Mining																	
ROM	**(000' tonnes)**	**19 800**	0	0	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	0	0
Waste Rock	**(000' tonnes)**	**19 160**	0	0	1 742	1 742	1 742	1 742	1 742	1 742	1 742	1 742	1 742	1 742	1 742	0	0
Glacial Ovb	**(000' tonnes)**	**1 739**	0	1 000	739	0	0	0	0	0	0	0	0	0	0	0	0
Total Material Mined	**(000' tonnes)**	**40 699**	0	1 000	4 280	3 542	3 542	3 542	3 542	3 542	3 542	3 542	3 542	3 542	3 542	0	0
Stripping Ratio (waste / ROM)	**(w:o)**	**1,06**	0,00	0,00	1,38	0,97	0,97	0,97	0,97	0,97	0,97	0,97	0,97	0,97	0,97	0,00	0,00
Processing																	
Material to Plant	**(000' tonnes)**	**19 800**	0	0	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	0	0
Au Head Grade (ppm)	**(ppm Au)**	**0,90**	0,00	0,00	0,90	0,90	0,90	0,90	0,90	0,90	0,90	0,90	0,90	0,90	0,90	0,00	0,00
Cu Head Grade (%)	**(% Cu)**	**0,08**	0,00	0,00	0,08	0,08	0,08	0,08	0,08	0,08	0,08	0,08	0,08	0,08	0,08	0,00	0,00
Copper Concentrate Product	**(tonnes)**	**37 372**	0	0	3 397	3 397	3 397	3 397	3 397	3 397	3 397	3 397	3 397	3 397	3 397	0	0
Dore - Au	**(oz)**	**323 245**	0	0	29 386	29 386	29 386	29 386	29 386	29 386	29 386	29 386	29 386	29 386	29 386	0	0
Revenue																	
Gross Revenue																	
Copper Con	**(M USD)**	202	0	0	18	18	18	18	18	18	18	18	18	18	18	0	0
Dore	**(M USD)**	420	0	0	38	38	38	38	38	38	38	38	38	38	38	0	0
Total	**(M USD)**	622	0	0	57	57	57	57	57	57	57	57	57	57	57	0	0
Net Revenue																	
Copper Con	**(M USD)**	198	0	0	18	18	18	18	18	18	18	18	18	18	18	0	0
Dore	**(M USD)**	420	0	0	38	38	38	38	38	38	38	38	38	38	38	0	0
Total	**(M USD)**	618	0	0	56	56	56	56	56	56	56	56	56	56	56	0	0
Operating Costs																	
Mining	**(M USD)**	**128,7**	0,0	1,9	11,9	11,5	11,5	11,5	11,5	11,5	11,5	11,5	11,5	11,5	11,5	0,0	0,0
Processing	**(M USD)**	**297,0**	27,0	27,0	27,0	27,0	27,0	27,0	27,0	27,0	27,0	27,0	27,0	0,0	0,0	0,0	0,0
Tailings	**(M USD)**	**9,6**	0,0	0,0	0,9	0,9	0,9	0,9	0,9	0,9	0,9	0,9	0,9	0,9	0,9	0,0	0,0
Environemntal & Closure	**(M USD)**	**25,4**	0,0	0,0	1,2	1,2	1,2	1,2	1,2	1,2	1,2	1,2	1,2	1,2	1,2	6,0	6,0
G&A	**(M USD)**	**0,0**	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Contingency	**(M USD)**	**0,0**	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Total Operating Costs	**(M USD)**	**460,8**	0,0	1,9	41,0	40,6	40,6	40,6	40,6	40,6	40,6	40,6	40,6	40,6	40,6	6,0	6,0
Unit Operating Costs	**(USD / oz AuEq)**	**963**	0	0	943	933	933	933	933	933	933	933	933	933	933	0	0
Capital Costs																	
Mining	**(M USD)**	19,4	2,9	15,0	1,5	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Processing	**(M USD)**	50,0	20,0	20,0	10,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Tailings & WRD	**(M USD)**	11,1	0,0	1,1	2,2	1,1	1,1	2,2	2,2	1,1	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Environmental	**(M USD)**	3,3	1,3	1,3	0,7	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Contingency	**(M USD)**	8,4	2,4	3,7	1,4	0,1	0,1	0,2	0,2	0,1	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Working Capital	**(M USD)**	0,0	0,0	8,2	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	-8,2
Total	**(M USD)**	92,2	26,7	49,4	15,8	1,2	1,2	2,4	2,4	1,2	0,0	0,0	0,0	0,0	0,0	0,0	-8,2
Cashflow																	
Net Pre-tax Cashflow	**(M USD)**	**65,1**	-26,7	-51,3	-0,6	14,4	14,4	13,2	13,2	14,4	15,6	15,6	15,6	15,6	15,6	-6,0	2,2
Cumulative Pre-tax Cashflow	**(M USD)**	**0,0**	-26,7	-78,0	-78,7	-64,3	-49,9	-36,7	-23,5	-9,1	6,5	22,1	37,7	53,4	69,0	62,9	65,1
Corporation tax	**(M USD)**	**-19,6**	0,0	0,0	0,0	0,0	0,0	0,0	0,0	-1,9	-3,4	-3,5	-3,5	-3,6	-3,6	0,0	0,0
Net Post-tax Cashflow	**(M USD)**	**45,5**	-26,7	-51,3	-0,6	14,4	14,4	13,2	13,2	12,5	12,2	12,1	12,1	12,0	12,0	-6,0	2,2

Figure ES 9 shows the varying NPV for varying single parameter sensitivities at an 8% discount rate for revenue, operating costs, capital costs and EUR:USD exchange rate.

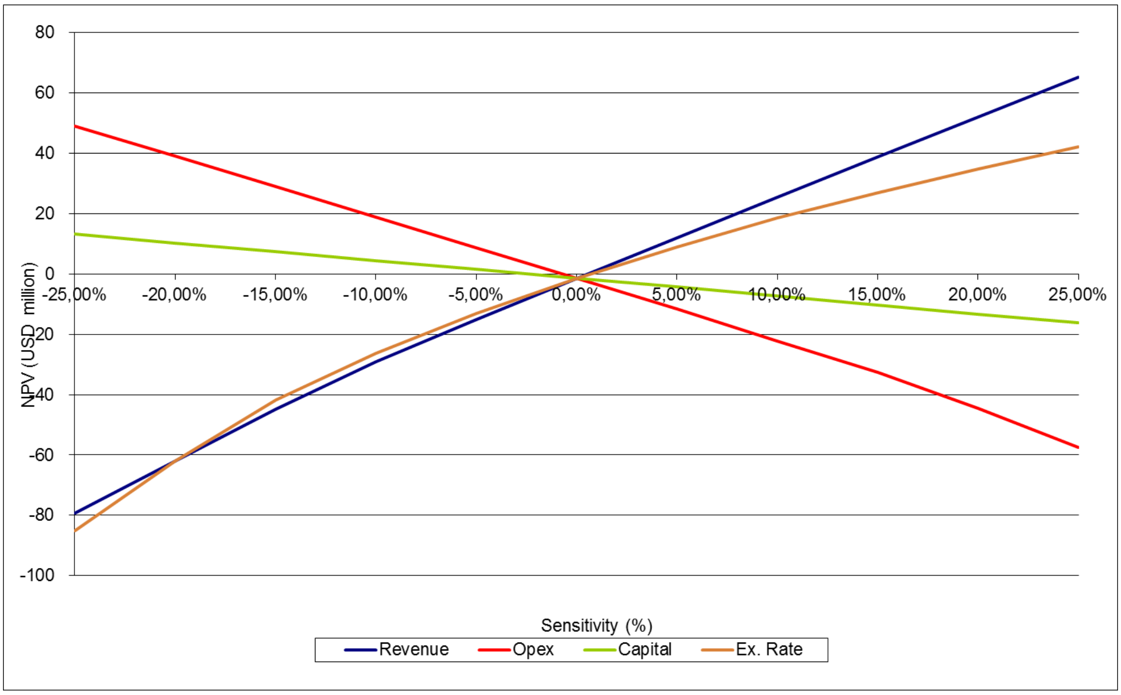

Figure ES 9: Single parameter sensitivity post-tax, pre-finance NPV at 8% discount rate. (Source:SRK, 2014)

SRK notes that the Project is most sensitive to changes in commodity price and least sensitive to changes in capital cost. For illustrative purposes, a summary table of production physicals, costs, revenue and cashflow is presented in the below, for three different gold price scenarios; 1 100 USD/oz (low), 1 300 USD/oz (base case) and 1 500 USD/oz (high).

Table ES 12: Summary results for three gold price scenarios

	Unit	Low	Base Case	High
Gold price scenario	**USD / oz**	**1 100**	**1 300**	**1 500**
Cut-off grade ROM	**(g/t Au)**	0,7	0,7	0,7
Cut-off grade Marginal Material	**(g/t Au)**	0,55	0,55	0,55
ROM	**(000' tonnes)**	19 800	19 800	19 800
Waste Rock (incl. overburden)	**(000' tonnes)**	20 899	20 899	20 899
Material to Plant	**(000' tonnes)**	19 800	19 800	19 800
Au Head Grade (ppm)	**(g/t Au)**	0,90	0,90	0,90
Cu Head Grade (%)	**(% Cu)**	0,08	0,08	0,08
Dore Au produced	**(oz)**	323 245	323 245	323 245
Dore Au produced	**(kg)**	10 053	10 053	10 053
Copper Concentrate produced	**(tonnes)**	37 372	37 372	37 372
Overall Au Recovery	**(%)**	77%	77%	77%
Overall Cu recovery	**(%)**	60%	60%	60%
Total Gross Revenue (Dore & Copper Con)	**(USD million)**	579	579	579
Total deductions (TCRC's & losses)	**(USD million)**	4	4	4
Total Net Revenue (Dore & Copper Con)	**(USD million)**	575	575	575
Operating Costs				
Mining	**(USD/t)**	6,5	6,5	6,5
Processing (incl. G&A, transport)	**(USD/t)**	15,0	15,0	15,0
Tailings	**(USD/t)**	0,5	0,5	0,5
Environmental & Closure	**(USD/t)**	1,3	1,3	1,3
Total	**(USD/t)**	23,3	23,3	23,3
Unit operating cost / oz AuEq	**(USD/oz AuEq)**	956	956	956
Capital Costs	**(USD million)**	92	92	92
Net Pre-tax Cashflow	**(USD million)**	-22	65	152
Corporation tax (24,5%)	**(USD million)**	0	20	-41
Net Post-tax Cashflow	**(USD million)**	-22	45	111
NPV (post tax, 8% WACC)	**(USD million)**	-41	-1	36

1.12 INTERPRETATION AND CONCLUSIONS

The Project would appear to be marginal to sub-economic given current cost, technical and long-term commodity price assumptions. This is largely due to low average Au and Cu grades and the nature of the mineralisation, which does not appear to be conducive to selective mining of higher grade portions. Re-sampling and improved geological understanding may improve confidence in the data supporting the current Mineral Resource estimate, which should improve the Mineral Resource category and may facilitate higher grade zones to be better defined.

SRK notes that other operators in the Nordic Region are trialling Optical and/or X-ray transmission (XRT) sorting technologies, which could conceptually be applied to run of mine material prior to conventional milling and flotation. Whilst this technology is still relatively unproven in the mining industry, successful use of this may help to improve the project economics and may be worth investigating further.

Given the extensive amount of available historic data, certain focused technical studies on key areas of opportunity could be carried out at relatively low cost. These studies should be undertaken prior to any significant additional expenditure and critically assessed in the financial model to gauge impact on the overall Project viability, prior to committing to any significant additional expenditure on the Project. A phased 12 to 18 month budget of USD 409 750 has been proposed by SRK for this, based on the recommendations presented in this report.

1.13 RECOMMENDATIONS

Based on the findings of this PEA, SRK makes the following recommendations:

- The locating of the original historic assays and the comparison of these with the GMH re-assayed values in order to help with verification of the historic data;
- Review, assay and re-logging of the GMH drill core at the GTK archive in Loppi;
- Complete re-assaying of the coarse reject material from the Northern Lion drilling along with an appropriate QAQC programme;
- Subsequent to the re-sampling, the systematic re-logging of drill sections with a view to developing a detailed structural interpretation and improving the understanding of the geological controls on mineralisation;
- The modelling of the Ag mineralisation and the inclusion of this in the Mineral Resource statement to enable this to be considered as a by-product in the cashflow model;
- The collection of representative metallurgical samples;
- New metallurgical laboratory scale test-work to assess how to produce a higher grade concentrate (in order to reduce freight costs) whilst maintaining a high processing recovery and suppressing contamination (arsenic in the copper concentrate; copper in the gold concentrate);
- Optical / XRT sorting test-work;
- Detailed analysis of the geotechnical domains to determine appropriate slope angles which comply with pit design standards;
- Detailed analysis of hydrogeological and hydrological factors and the impact on dewatering and the design of water management systems;
- Commencement and/or continuation of discussions with the owners of existing third party processing facilities to determine whether the sale or toll treatment of crushed Haveri ROM is possible and if so, what terms may be reasonable to assume for the purposes of comparison during further phases of study;

- Further work to improve the accuracy of cost estimates; and
- An assessment of whether the cost of developing the infrastructure can be shared with the regional authorities.

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PRELIMINARY ECONOMIC ASSESSMENT FOR THE HAVERI GOLD-COPPER DEPOSIT, FINLAND

2 INTRODUCTION

The Haveri Project is an advanced exploration project comprising the Haveri, Peltosaari, Tombstone, and Sankari-Ansomäki gold-copper ("Au-Cu") deposits, which are collectively referred to as the Haveri project ("Haveri", or the "Project"). It is located 1 km from the town of Viljakkala in the county of Länsi-Suomen lääni in southern Finland. Palmex Mineral AB (Palmex) currently holds a 100% of the Project.

Historic mining occurred at Haveri in the mid-19th century, and again between 1942 and 1962, utilising a combination of open pit and underground methods. Mining between 1942 and 1962 produced a total of 1.5 Mt of material with an average grade of 3.5 g/t Au and 0.5% Cu.

This report comprises a preliminary economic assessment ("PEA") of the Project and has been prepared by SRK Consulting (Sweden) AB ("SRK") on behalf of Palmex. This Mineral Resource estimates presented here are reported according to the definitions and guidelines of the 2010 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards on Mineral Resources and Reserves ("CIM Definition Standards") which is an international reporting code recognised by the Committee for Mineral Reserves International Reporting Standards ("CRIRSCO").

SRK has prepared an independent Mineral Resource estimate ("MRE") for the Project and has used this as a basis for a conceptual mining study. In addition, SRK has reviewed all other technical work completed on the Project both historically and by the Company and its contractors and consultants to a sufficient level to enable SRK to present its own opinions on the Project and to derive an audited NPV.

The Project is at a conceptual stage but it is currently envisaged that production will be from open pit methods through conventional drill and blast, with trucking to a dedicated on-site processing facility. Whilst other mining and processing scenarios have been considered in the production of this PEA and are discussed briefly, only the base case is discussed in detail for the purposes of this report.

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 the CIM Definition Standards.

The details of the various contributing authors and their respective areas of technical responsibility are presented in Table 2-1. For the purposes of this report, the following persons act as QP: Johan Bradley and Dr Mike Armitage. QP Johan Bradley visited the property on February 04 and 05, 2014.

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

Contributing Author	Area of technical responsibility	Sections of this report
Johan Bradley	Geology and Technical Economic Model	Sections 2 to 12 (inclusive)
Lucy Roberts	Resource Estimation	Section 14
Maxim Lesonen	Mine Optimisation, Design and Scheduling	Section 16
David Saiang	Geotechnical assumptions	Section 16.2
John Willis	Process Metallurgy, Infrastructure, Markets and Concentrate Transport	Sections 13; 17 & 18
Jamie Spiers	Tailings Dam Design and Waste Rock Dumps	Section 20.3
Päivi Picken	Acid Rock Drainage and Metal Leaching	Section 20.3
Tony Rex	Hydrology and Water Management	Section 20.1
Päivi Picken	Environmental, Permitting and Social Impacts	Section 20
Dr Mike Armitage	Peer Review	overall

2.1 Basis of Technical Report

This report is based on information collected by SRK during a site visit performed February 04/05, 2014 and on additional information provided by Palmex and their consultants, AB Scandinavian GeoPool Ltd ("Geopool"). Additional information was obtained from the public domain. Specifically, this technical report is based on the following sources of information:

- Discussions with Palmex and Geopool personnel;
- Inspection of the Haveri Project area, including outcrop and drill core;
- Assay results from validation samples collected by SRK;
- Review of exploration data collected by Palmex; and
- Additional information from public domain sources.

2.2 Declaration

SRK's opinion contained herein and effective 30 July 2014, is based on information collected by SRK throughout the course of this mandate. The opinions stated reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

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

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

3 RELIANCE ON OTHER EXPERTS

Sections 4 to 10 of this report are to some degree extracts from the Company's existing technical reports, including the 2008 technical report by Maptek Pty Ltd ("Maptek") on behalf of Lappland Goldminers AB ("Reed, 2008"). 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 this, SRK has conducted face to face meetings with those consultants responsible for certain technical aspects of the Project to enable it to take responsibility for the assumptions given here.

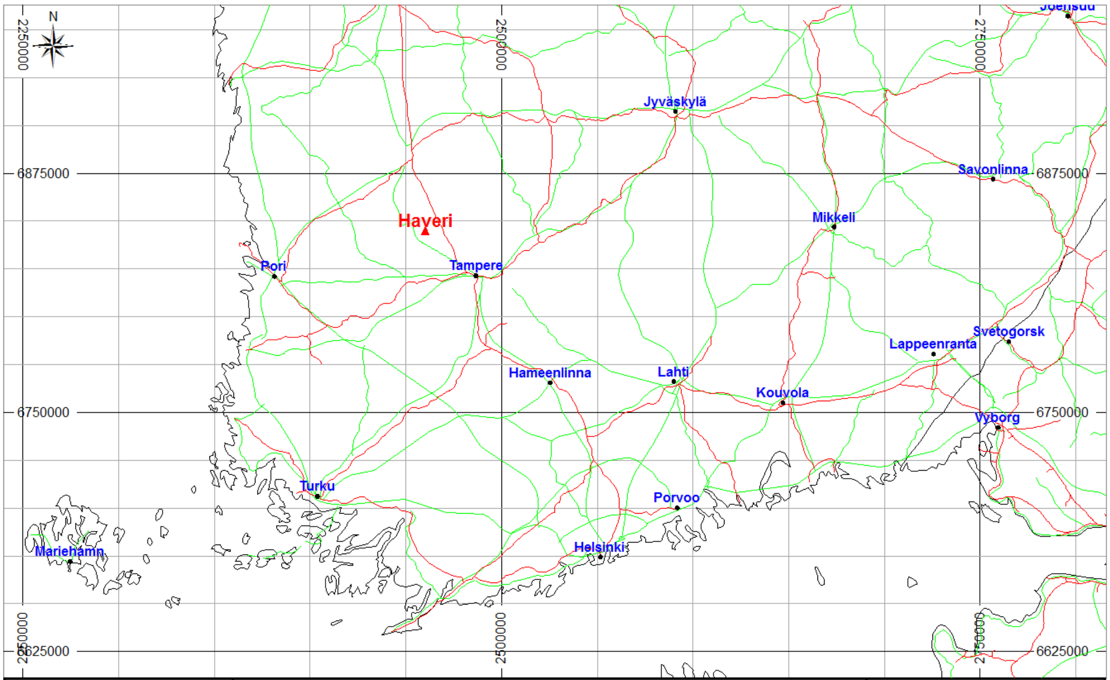
SRK has confirmed that the Mineral Resources reported herein are within the exploration claim boundaries and that the exploration and mining leases presented by the Company reflect the information in the public domain. SRK has not, however, conducted any legal due diligence on the ownership of the exploration permits or exploitation concessions themselves and has relied upon the Company's legal advisor (Thomas Myrdal, Partner at Hamilton Advokatbyrå), who presented its' opinion of the Company's legal tenure in a letter dated 15 May 2014, which SRK has reviewed.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Description

The Haveri Project is a gold-copper advanced exploration project located 1 km from the town of Viljakkala in Länsi-Suomen lääni (County), southern Finland (Figure 4-1). The Project is located 35 km northwest of regional centre of Tampere, and 220 km northwest of the Finnish capital Helsinki. The deposit is centred around Finnish National Coordinate System (Kartasto Koordinaatti Järjestelmä (KKJ) No.2): X: 2460000; Y: 6843500, European coordinate system (EUREF-FIN / ETRS89): X: 301500; Y: 6846000, and latitude and longitude: N 61.7, E 23.25.

The coordinates quoted in this report are from the KKJ no.2 system.



Plan	Haveri Au-Cu Project, Finland	Dec 2013
srk consulting	Haveri Project location in Finland. Major towns, roads (green) and railways (red) shown.	Scale 1:2000000

Figure 4-1: Location of the Haveri project in Finland (Source: SRK, 2014)

The Haveri Project is divided into different areas based on historic deposit names, as shown on Figure 4-2. This PEA focuses on the Haveri area (including Haveri mine, Haveri North and Tombstone) and the Peltosaari area. The Casino Bay, Sankari and Ansomäki areas have not been sufficiently explored to support the declaration of Mineral Resources.

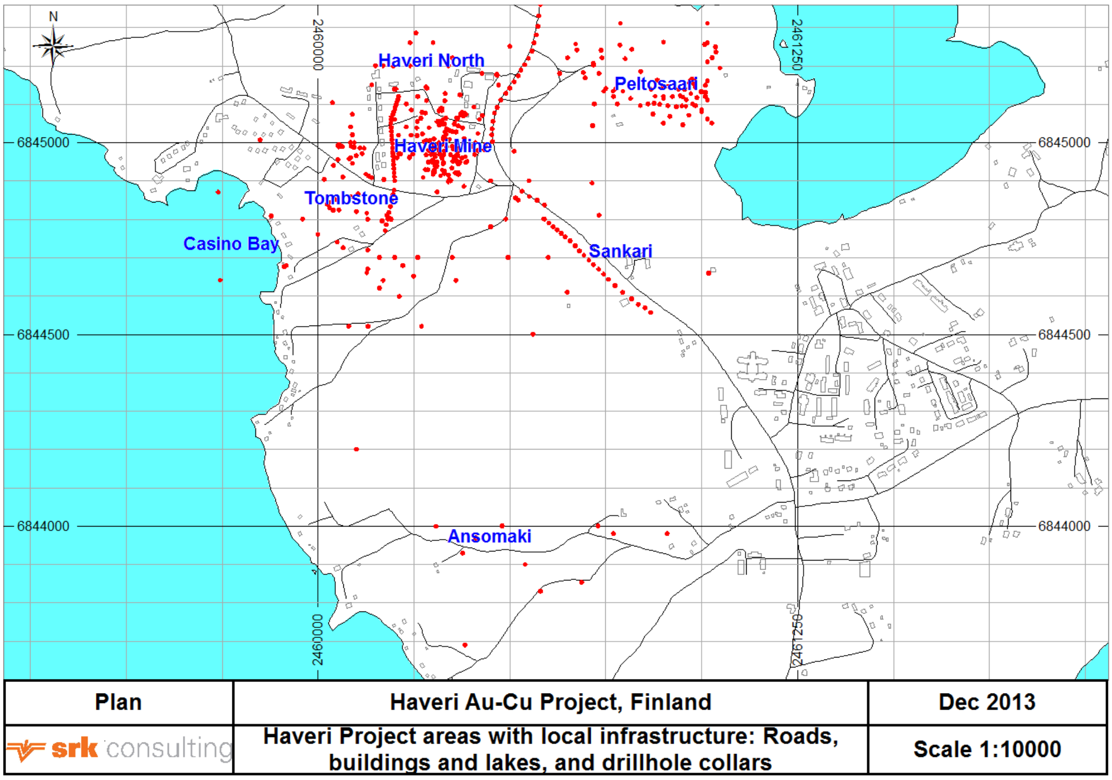


Figure 4-2: Haveri Project areas with drillhole collars and local infrastructure (Source: SRK, 2014)

4.2 Property Ownership

The Haveri Project area is currently under application for a 'reservation notification' by Palmex Mining Oy (Palmex Finnish subsidiary) since August 2013. The notification area is shown in Figure 4-3. The description of 'reservation notification' below is from the Finnish mining inspectorate (TUKES) Mining Act 2011:

'For the purpose of preparing an application for an exploration permit, an applicant may reserve an area for himself by submitting notification to the mining authority about the matter (reservation notification). A privilege based on reservation notification is valid once the reservation notification has been submitted in accordance with the provisions laid down in section 44 (of the Finnish Mining Act) and no impediment exists, as specified in this Act, to approval of the reservation. The validity of the privilege shall expire when the decision made by the mining authority on the basis of the reservation notification (reservation decision) expires or is cancelled.'

After the moratorium of previous owners Lappland Goldminers' exploration claims expires, between February and June 2015, Palmex will be able to apply for an exploration claim.

In addition to the Haveri reservation notification, Palmex Mineral AB also own the exploration claim 'Osara', which is adjacent to Haveri towards the west.

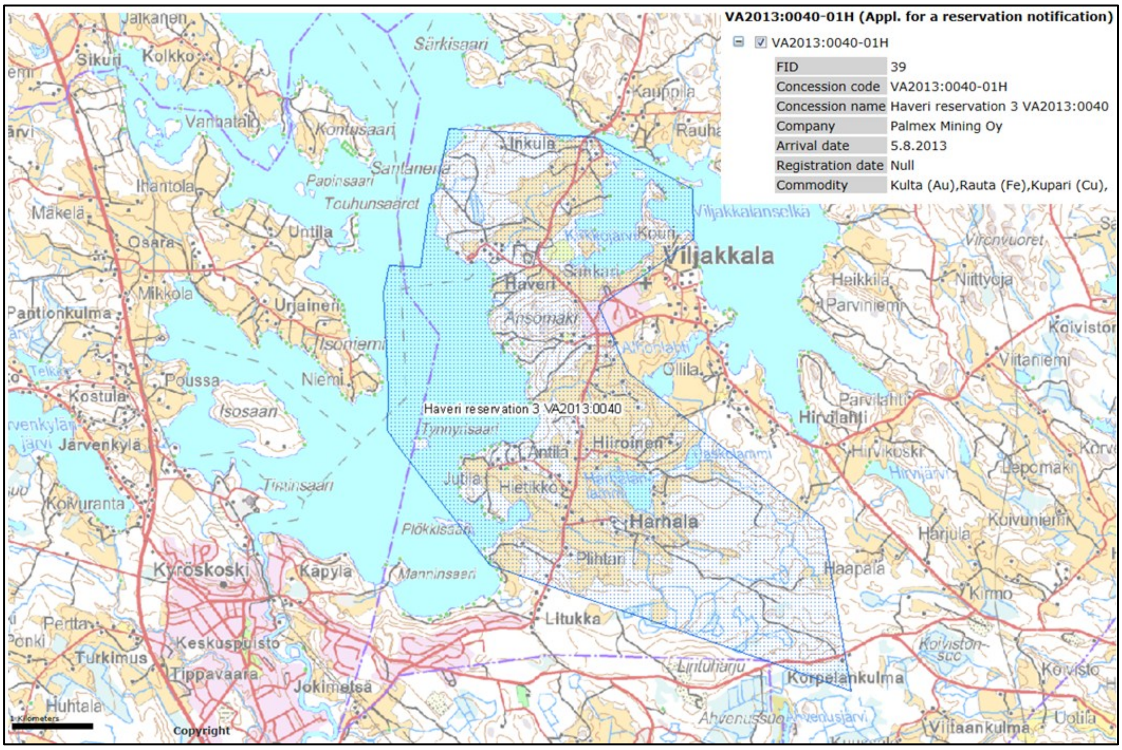


Figure 4-3: Haveri Reservation Notification 3 (Source: Tukes 2013)

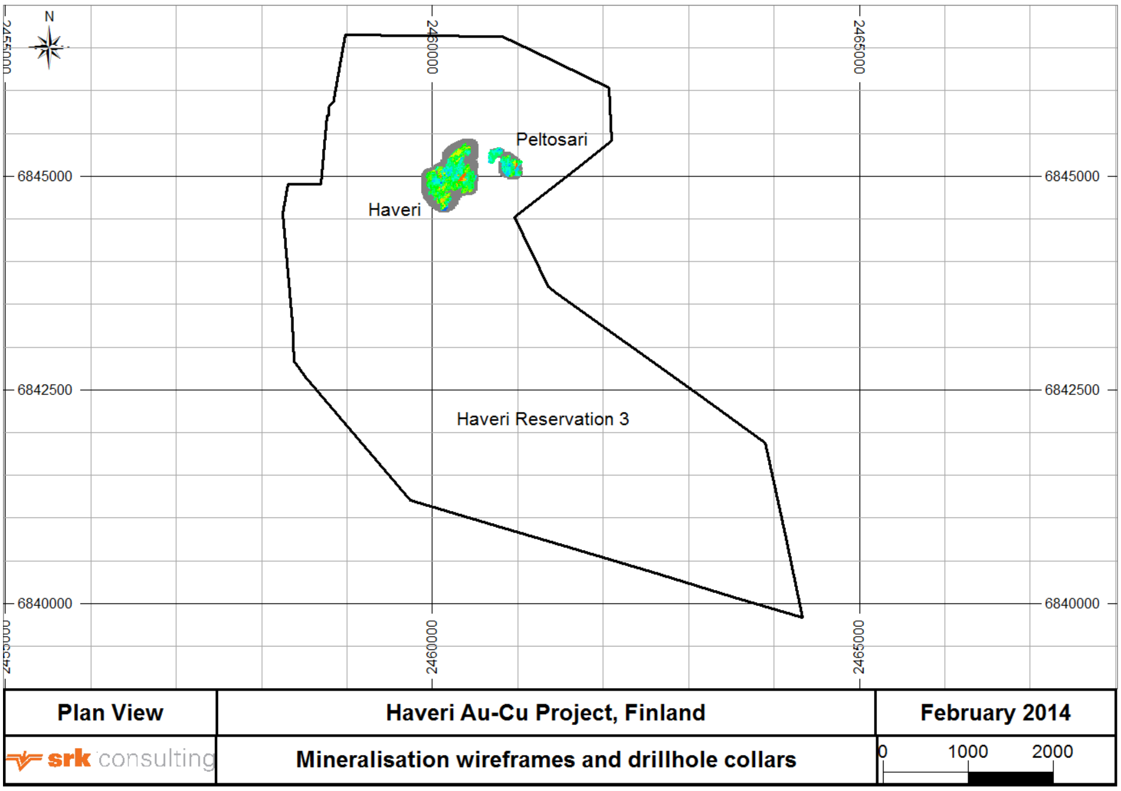


Figure 4-4: Haveri Reservation Notification 3 with Resource Model and Resource Pit Shell (Source: SRK, 2014)

4.3 Additional Permits, Royalties and Payments

SRK is not aware of any known environmental liabilities, royalties, back-in rights, payments or other encumbrances to which the property is subject. All the payments for damage compensations are also up to date as far as SRK is aware.

4.4 Surface Rights

Under the current reservation notification, Palmex does not have surface rights to perform any exploration or sampling without consent of the land owners. An exploration claim entitles the holder (individual or company) to carry out exploration activities in the claim area with or without the consent of the landowner. The claimant must, however, compensate the landowner in full for any permanent or temporary damage or inconvenience caused by the exploration activities inside or outside the claim area. The claimant shall also act in compliance with environmental legislation and other laws and regulations. Additional permitting is required prior to commencement of mining operations.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

The historic Haveri mine, which lies roughly at the geographic centre of the Projectarea, is located 1 km by road from the town of Viljakkala. The Project can be accessed by sealed roads to the larger town centres, and then via secondary gravelled roads.

As is common with much of Finland, the existing service infrastructure is excellent. The claim area is 35 km by road from the nearest railhead at Tampere, which in turn is well connected to Helsinki. The nearest commercial airports are at Tampere, Turku (175 km by road to the southwest) and Helsinki, (230 km by road to the southeast). All these airports have international and domestic flights daily.

Field work in the area involving geochemical sampling and geological mapping is restricted to the Finnish summer (May to November), while drilling and geophysical surveying is typically performed during winter (January to April) to minimise ground damage. Exploration drilling activities can however be carried out year-round.

5.2 Physiography and Climate

The local topography comprises rolling hills, with a strong glacial influence that has left a predominant northwest-southeast orientation that is apparent in the shape of the lakes and hills. The area lies at 100 masl on gently undulating shallow dipping hill slopes, some 10 m above the level of the surrounding lakes.

Haveri and its surrounding region belong to the temperate coniferous-mixed forest zone (Taiga/Boreal) with a climate described, according to the Köppen Climate Classification, as continental (micro-thermal) fully humid with mild summers. Winters (November-April) are cold and wet with an average temperature of -3°C. During the temperate summer period (June-August) the temperature averages 10°C.

5.3 Local Resources and Infrastructure

The principal land use in the area is forestry. All social and industrial needs and services such as accommodation, provisions, supplies, and communications are readily available and are of high standard, typical of the modern industrial democracy that is Finland. The national power grid extends throughout the region.

6 HISTORY

6.1 Discovery and Early Exploration

The Project area has been explored since the 1730s. Initial exploration was focussed on iron mineralisation found close to the main Haveri mine open pit, along with a sulphide outcrop at Peltosaari. Sporadic exploration continued until 1836, which resulted in the mining of iron in the mid-19th century.

6.2 Historic Exploration

The ownership of the property has changed hands several times since post-1900 exploration began in 1935, as shown in Table 6-1. An outline description of historic exploration conducted by each company is shown below.

Table 6-1: Historic property ownership and operators since 1935

Property Owner	Property Operator	Period of Ownership
Vuokseniska Oy	Vuokseniska Oy	1935 – 1962
Outokumpu Finn Mines Oy	Outokumpu Finn Mines Oy	1962 – 1986
Glenmore Highlands Inc	Glenmore Highlands Inc.	1996 – 2002
Vision Gate Ventures Ltd	Mountain Glen Mining Inc.	2002
Mountain Province Diamonds Inc	Northern Lion Gold Corp.	2003 – 2004
Northern Lion Gold Corp.	Northern Lion Gold Corp.	2004 – 2007
Lappland Goldminers AB	Lappland Goldminers AB	2007 – 2013
Palmex Mineral AB	Palmex Mineral AB	2013 – present

6.2.1 Vuokseniska Oy (1935 – 1962)

Between 1935 and 1962, Vuokseniska Oy ("Vuokseniska") completed exploration work including diamond drilling, geological mapping, magnetic and electromagnetic ground surveys. This exploration resulted in the delineation of gold and copper mineralisation, which was subsequently mined from 1942 to 1962.

Limited details are available regarding the exploration protocols during this time. Diamond drilling was conducted using 22 mm core, initially from surface and later from underground drifts when mining began. From drill core found by subsequent owners, whole core sampling was conducted at least in part. Small remnant intervals were left in core boxes in order to assist future explorers with geological logging. There are no details regarding core recovery. The assaying method is not documented; however, a 0.2 g/t Au lower detection limit for many samples is evident from assay database. Sample lengths vary due to geological contacts, however, 91% of samples are between 0.5 – 1.5 m (29% are 1 m exactly).

Limited information is available regarding the sample preparation, analyses and security of Vuokseniska samples. The data in the compiled database was derived from documents obtained from the National Archives of Finland at Mikkeli and other publicly accessible sources.

6.2.2 Outokumpu Oy (1962 – 1986)

Outokumpu Oy (Outokumpu) drilled 13 diamond holes in the 1970s for a total 1,921 m, mostly targeting electromagnetic geophysical anomalies. From 1980 to 1983, Outokumpu conducted auger sampling of the Vuokseniska mine tailings material. A total of 1,201 samples, at 1 m spacing, from 165 holes in the tailings were taken and analysed for Au, Co, Zn, Ni, Co, Pb, As, Fe and S.

In addition to the drilling outlined above, the Outokumpu exploration comprised:

- Geological mapping;
- Airborne and ground magnetic, electromagnetic and gravity surveys;
- A study of the genesis of the deposit, including whole-rock geochemical analyses, S-isotope studies, evaluation of the geological evolution and regional metamorphism of the area (Mäkelä, K. 1980); and
- A high-level assessment of the amount of Au and Co in the tailings of the mine, including auger drilling, geochemical analyses, pilot enrichment and leaching of the tailings material. (Kokkola, M. 1986).

Limited information is available regarding the procedures for sample preparation, analyses and security used by Outokumpu. The data in the compiled database was derived from documents obtained from the National Archives of Finland at Mikkeli and other publicly accessible sources. Sample lengths vary due to sampling having been carried out on the basis of geological contacts, however, 69% of samples are between 1.5 – 2.5 m (45% are 2 m exactly).

6.2.3 Glenmore Highlands Inc. (1996 – 2002)

Glenmore Highlands Inc ("GMH") conducted the majority of its exploration in the Peltosaari area, totalling 6,168 m of diamond drilling (42 mm core). An additional 1,579 m of diamond drilling was completed in the Haveri Mine area. GMH also completed 1,052 m of RC drilling in the Haveri Mine area and 1,024 m of percussion (Rotary Air Blast, "RAB") drilling in the Peltosaari area.

In addition to the drilling outlined above, the GMH exploration comprised:

- Dewatering the mine for channel sampling in the underground workings;
- Trench sampling;
- Ground and airborne electromagnetic, self-potential and magnetic geophysical surveys;
- Re-assaying of more than 5,000 pulp samples of historic diamond drill samples;
- Geological mapping of the area; and
- Till geochemistry on a 50 m sample grid.

Limited information is available regarding the sample preparation, analyses and security of GMH samples. The assaying method is not documented; however, a 0.1 g/t Au lower detection limit is evident from many samples in the assay database. Sample lengths vary due to geological contacts, however, 97% of samples are between 0.5 – 1.5 m (50% are 1 m exactly).

The exploration data was not surrendered to the Finnish mining authority (Tukes) after relinquishment of the exploration claim, as is required by Tukes.

A helicopter-borne multi-frequency, multi-coil electromagnetic survey ("DIGHEM") was flown by Fugro in 1996. The result of the survey is shown in Figure 6-1, with surveys also covering the Ansömaki and Osara areas. The highest electromagnetic anomalies are pink and correlate reasonably well to the known mineralisation.

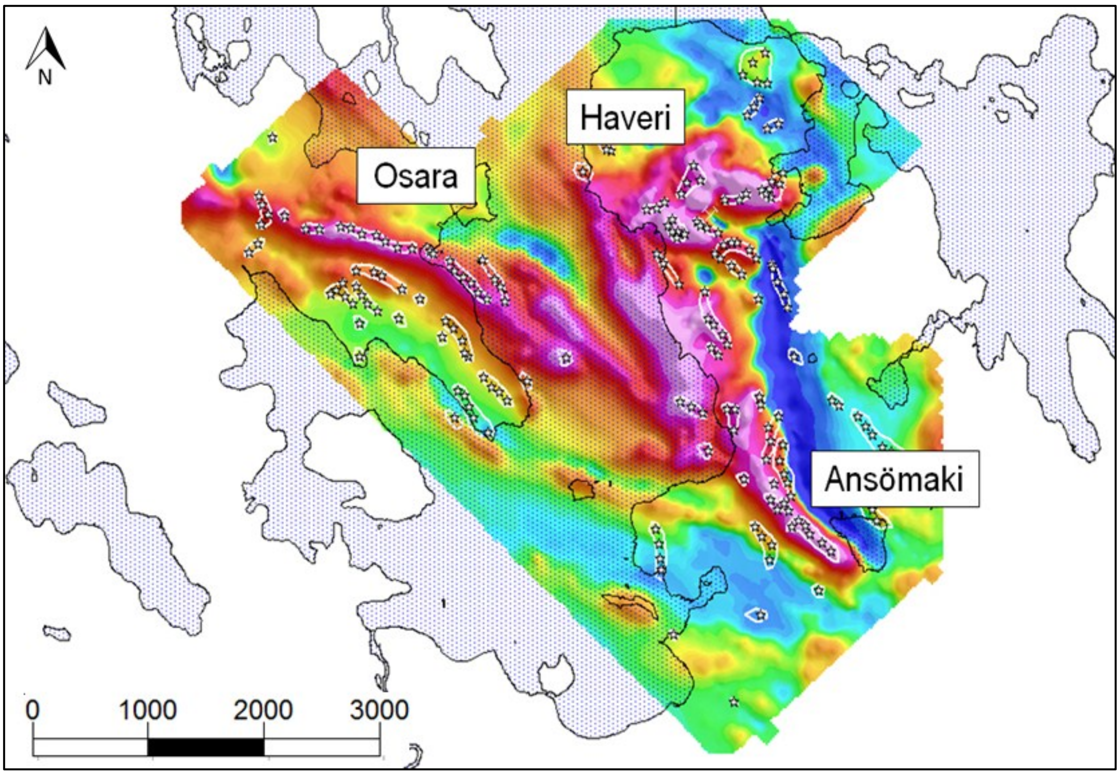


Figure 6-1: DIGHEM survey results for the Haveri, Ansömaki and adjacent Osara areas. Scale in metres (Source: Fugro, 1996).

Re-assays

It is stated in the Maptek report (Reed, 2008) that 5,000 historic assay pulps were re-assayed by GMH in an attempt to verify the historic data. SRK found approximately 4,500 samples in the GMH database relating to Vuokseniska holes along with additional samples relating to drillholes without collars, and channel samples. The historic database appears to only contain original Vuokseniska assays where no re-assays were taken. It is therefore not currently possible to compare the historic and re-assayed values.

6.2.4 Northern Lion Gold Corporation (2003 – 2007)

Northern Lion Inc ("NL") completed a total of 20,887 m of diamond drilling of which 2,294 m was drilled in the Peltosaari area, 4,016 m in the Ansomäki-Sankari area and the remainder in the Haveri Mine area. In addition, 236 m of percussion drilling was completed, of which 106 m was drilled in the Haveri Mine area and the remainder in the Ansomäki area.

In addition to the drilling outlined above, the NL exploration comprised:

- Till geochemical surveys;
- Induced polarization ("IP") and horizontal loop electromagnetic ("HLEM") surveys; and
- Trenching and channel sampling.

Geophysical Surveys

An IP survey was conducted in 2003 by SJ Geophysics Ltd, and is shown in Figure 6-2. The red and pink areas are associated with high IP readings. The analysis of the IP survey concluded that chargeability anomalies generally correlate with disseminated and stringer sulphides. Many previously unknown IP anomalies were delineated in the Haveri Project area and may correspond to areas of mineralisation. The large IP anomaly to the north (labelled 'Untested Anomaly' in Figure 6-2) is within mafic volcanic units, which has been partly sampled by percussion drillholes along the road which cut across it, as does the chargeability anomaly at Haveri North. Both of these chargeability targets were outlined for further drill testing subsequent to this IP survey.

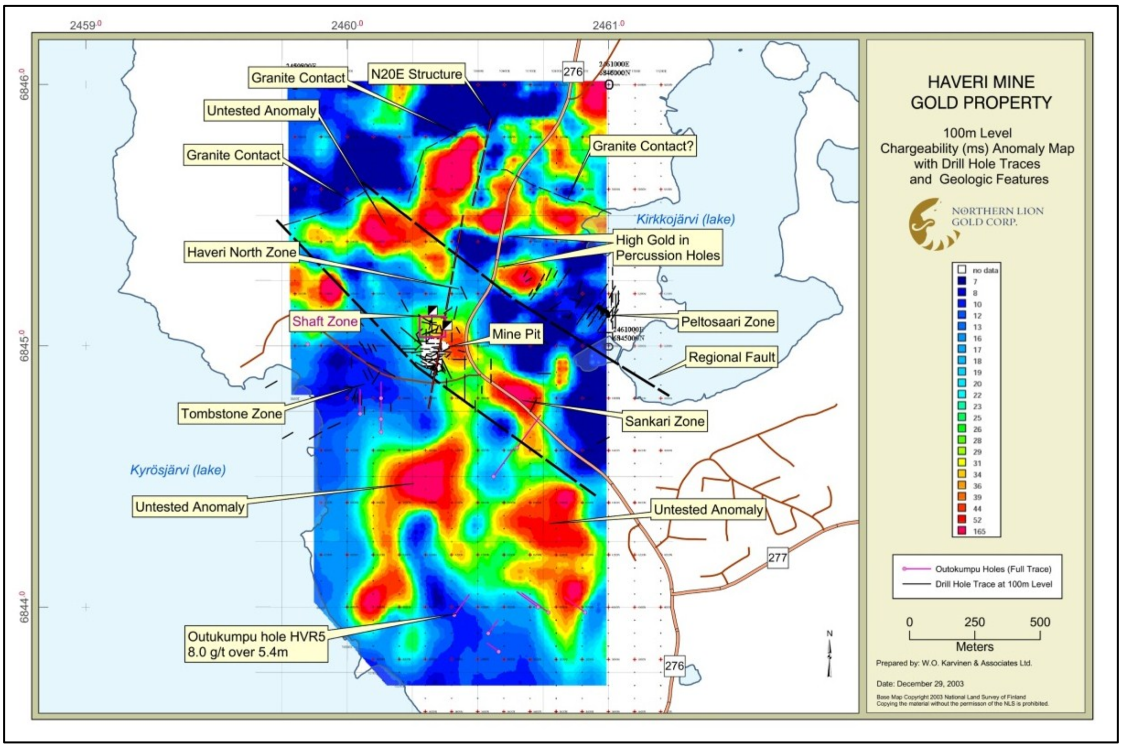


Figure 6-2: IP survey results and interpretation (Source: Northern Lion, 2003)

The HLEM survey undertaken by JVX Ltd in 2004 was conducted over the Haveri, Ansömaki and Eronen (south of Ansömaki) areas. The results from the Haveri area HLEM survey are shown in Figure 6-3. The red and pink areas are associated with high IP readings. The survey confirmed anomalies in the Tombstone and Haveri North areas. These results show good correlation to the regional-scale aeromagnetic results produced by the GTK, along with previous EM surveys.

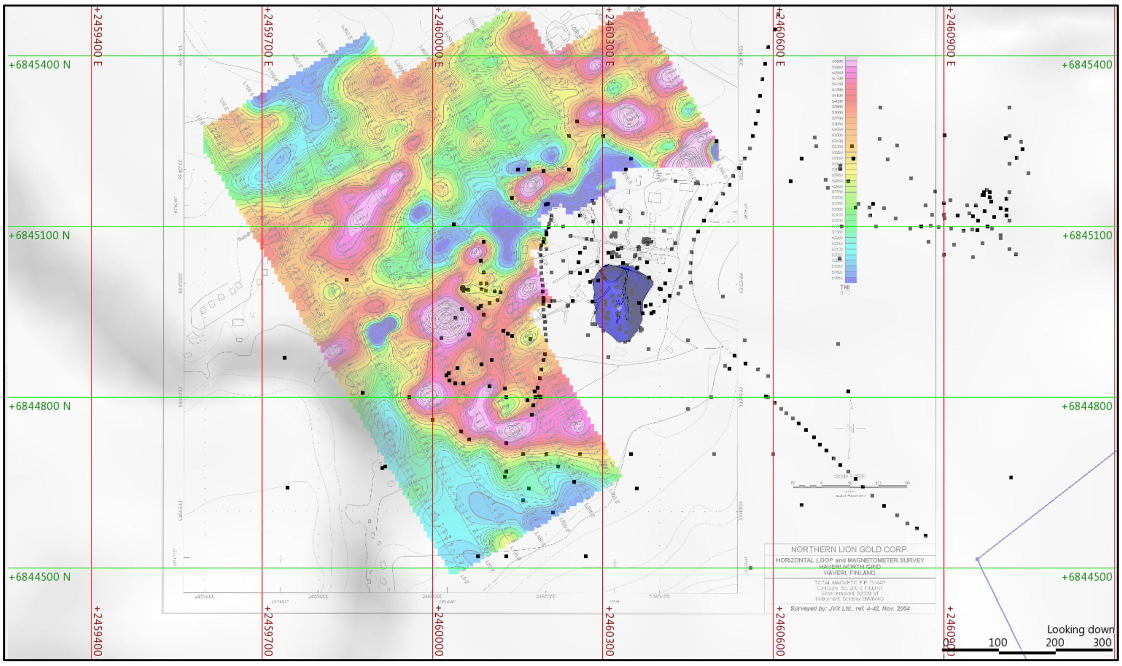


Figure 6-3: HLEM survey results for Haveri (Tombstone and Haveri North) along with drillhole collars (black dots) and historic open pit (blue) (Source: SRK, 2014)

NL Sample Preparation, Analyses and Security

Sample Preparation and Security

All drill core was transported twice daily by NL personnel to a secure logging and sampling facility located approximately 2 km to the southeast of the Haveri Project area. After the core was logged and the sample intervals marked, it was then cut by diamond saw. One half of the core was then placed in a plastic bag with a waterproof, numbered sample tag, which was sealed with a tamper-proof security tie, and the balance was retained in the core box for future reference. The sample ID was also written on the sample bag. The bagged samples were placed in a palletized wooden container that was secured by steel strapping and tamper-proof metal seals in preparation for transportation by truck to one of the laboratories operated by the GTK (now known as Labtium) or by air to the Activation Laboratories Ltd. ("Actlabs"), Ancaster, Canada. Sampled core and rejects from the GTK are stored in a locked and supervised building on a farm located about 2.5 km to the north of the logging facility in Haveri.

Sample lengths vary due to sampling having been based on geological contacts, however, 90% of samples are between 1 – 1.5 m (>60% are 1.5 m exactly).

Analyses

For gold analyses, the method employed by the GTK was 705A, which used a 50 g sample for lead-fire assay followed by flame atomization, atomic absorption spectrometry finish. For all samples above the detection limited of 100 ppm Au, a gravimetric analysis (704G) was performed. Actlabs used a similar method, 1A2, but with a 30 g sample. For all samples above the detection limited of 3 ppm Au, a gravimetric analysis (1A3) was performed. The detection limits for all four methods are shown in Table 6-2 and Table 6-3.

For full (62 element) geochemical analyses, an Ultratrace 2 method was used at Actlabs, which uses an Aqua Regia digest and analyses by ICP or ICP/MS. The detection limits for each element are shown in Table 6-4. It should be noted that this package is designed to analyse for trace element levels. The Cu analyses are therefore restricted to 10,000 ppm (or 1%), which has affected 45 of >14,000 assays (less than 0.5% of the NL assays). It is not clear which geochemical package was used at the GTK laboratory, and there are no codes in the database to differentiate the two datasets.

Table 6-2: GTK methods 705A/704G detection limits

Element	Method	Lower limit [ppm]	Upper limit [ppm]
Gold	705A (fire assay)	0.02	100
Gold	704G (gravimetric)	2	10,000

Table 6-3: Actlabs methods 1A2/1A3 detection limits

Element	Method	Lower limit [ppm]	Upper limit [ppm]
Gold	1A2 (Fire Assay)	0.05	3
Gold	1A3 (gravimetric)	0.03	1,000

Table 6-4: Actlabs method Ultratrace 2 detection limits (ppm unless specified)

Element	Lower limit [ppm]	Upper limit [ppm]	Reported By
Ag*	0.002	50	ICP/MS
Al*	0.01%	10%	ICP/MS
As*	0.1	10,000	ICP/MS
Au*	5 ppb	10,000 ppb	ICP/MS
B*	1	5,000	ICP/MS
Ba*	0.5	6,000	ICP/MS
Be*	0.1	1,000	ICP/MS
Bi	0.02	2,000	ICP/MS
Ca*	0.01%	50%	ICP/MS
Cd	0.01	-	ICP/MS
Ce*	0.01	10,000	ICP/MS
Co	0.1	5,000	ICP/MS
Cr*	0.5	5,000	ICP/MS
Cs*	0.02	-	ICP/MS
Cu	0.01	10,000	ICP/MS
Dy	0.1	-	ICP/MS
Er	0.1	-	ICP/MS
Eu*	0.1	-	ICP/MS
Fe*	0.01%	50%	ICP/MS
Ga*	0.02	500	ICP/MS
Ge*	0.1	500	ICP/MS
Gd	0.1	-	ICP/MS
Hf*	0.1	500	ICP/MS
Ho	0.1	-	ICP/MS
In	0.02	-	ICP/MS
K*	0.01%	5%	ICP/MS
La*	0.5	1,000	ICP/MS
Li	0.1	-	ICP/MS
Lu*	0.1	100	ICP/MS
Mg*	0.01%	10%	ICP/MS
Mn*	1	10,000	ICP/MS
Mo	0.01	10,000	ICP/MS
Na*	0.00%	5%	ICP/MS
Nb*	0.1	500	ICP/MS
Nd*	0.02	-	ICP/MS
Ni*	0.1	10,000	ICP/MS

Element	Lower limit [ppm]	Upper limit [ppm]	Reported By
P	0.00%	10%	ICP
Pb*	0.01	10,000	ICP/MS
Pr	0.1	-	ICP/MS
Rb*	0.1	500	ICP/MS
Re	0.001	100	ICP/MS
S	0.00%	20%	ICP
Sb	0.02	500	ICP/MS
Sc	0.1	-	IPC/MS
Se*	0.1	10,000	ICP/MS
Sm*	0.1	100	ICP/MS
Sn*	0.05	200	ICP/MS
Sr*	0.5	1,000	ICP/MS
Ta*	0.05	50	ICP/MS
Tb*	0.1	100	ICP/MS
Te	0.02	500	ICP/MS
Th*	0.1	200	ICP/MS
Ti*	0.01%	20%	ICP
Tl*	0.02	500	ICP/MS
Tm	0.1	-	IPC/MS
U*	0.1	10,000	ICP/MS
V*	1	1,000	ICP/MS
W*	0.1	200	ICP/MS
Y*	0.01	-	ICP/MS
Yb*	0.1	200	ICP/MS
Zn*	0.1	5,000	ICP/MS
Zr*	0.1	5,000	ICP/MS

**Note: May not be total. Unaltered silicates and resistate minerals may not be dissolved*

Accreditation

The GTK laboratory (now Labtium Ltd) is an accredited testing laboratory. The accreditation according to ISO/IEC 17025 was received originally in 1994 from the Finnish Accreditation Service FINAS at the The Centre for Metrology and Accreditation ("MIKES"). The accreditation code of Labtium is FINAS T025. Labtium is continuously participating in independent, international proficiency tests in the mineral sector run by e.g. Geostats Pty Ltd, Australia and the GeoPT sponsored by the International Association of Geoanalysts (IAG). In addition Labtium participates in a proficiency test for Canadian accredited mineral testing laboratories (CANMET PTP-MAL).

Actlabs' Quality Assurance System is accredited to international quality standards through the International Organization for Standardization /International Electrotechnical Commission (ISO/IEC) 17025 (ISO/IEC 17025 includes ISO 9001 and ISO 9002 specifications) with CAN-P-1579 (Mineral Analysis) for specific registered tests by the SCC. The accreditation program includes on-going audits which verify the QA system and all applicable registered test methods.

Quality Assurance and Quality Control (QAQC) Methodologies

In order to maintain a quality assurance/quality control ("QAQC") program, NL and the laboratories involved employed a system of certified reference material ("CRM") and duplicates in most batches of samples. The standards were employed by NL at a rate of at least 1 per 27 samples. The CRMs were purchased from Rocklabs Ltd, Auckland, New Zealand, and carried the designations SH13, SJ10 and SQ18 (prefix "S" refers to sulphide matrix). The round-robin certified values and the 95% confidence intervals for each CRM are shown in Table 6-5. The samples are intended as reference material for the determination of Au. No Cu CRM was used throughout the Northern Lion drilling campaigns. It is unclear if duplicates were inserted by Northern Lion or analysed as part of the Labs' internal QAQC protocol.

Actlab has analysed a total of 3659 assays and GTK has analysed 1540 assays.

Table 6-5: Certified Reference Material (CRM) certified values

Rocklabs CRM	Recommended Au Concentation (g/t)	95% Confidence Interval (+/- g/t)	Upper Limit (+2s)	Lower Limit (-2s)
SH13	1.315	0.015	1.383	1.247
SJ10	2.643	0.028	2.763	2.523
SQ18	30.49	0.35	32.25	28.73

SRK QAQC Analysis

Certified Reference Material (CRM)/ Standards

Figure 6-4 through Figure 6-7 show the performance of the SH13 and SJ10 standards at the GTK as well as the Actlab laboratories for Northern Lion drilling. The SQ18 high grade standard was only inserted twice into the sample stream sent to GTK. In both cases it reported outside the lower limit at 21.09 and 25.3 g/t Au. There is no information available for either Actlabs' or GTK's internal QAQC performance.

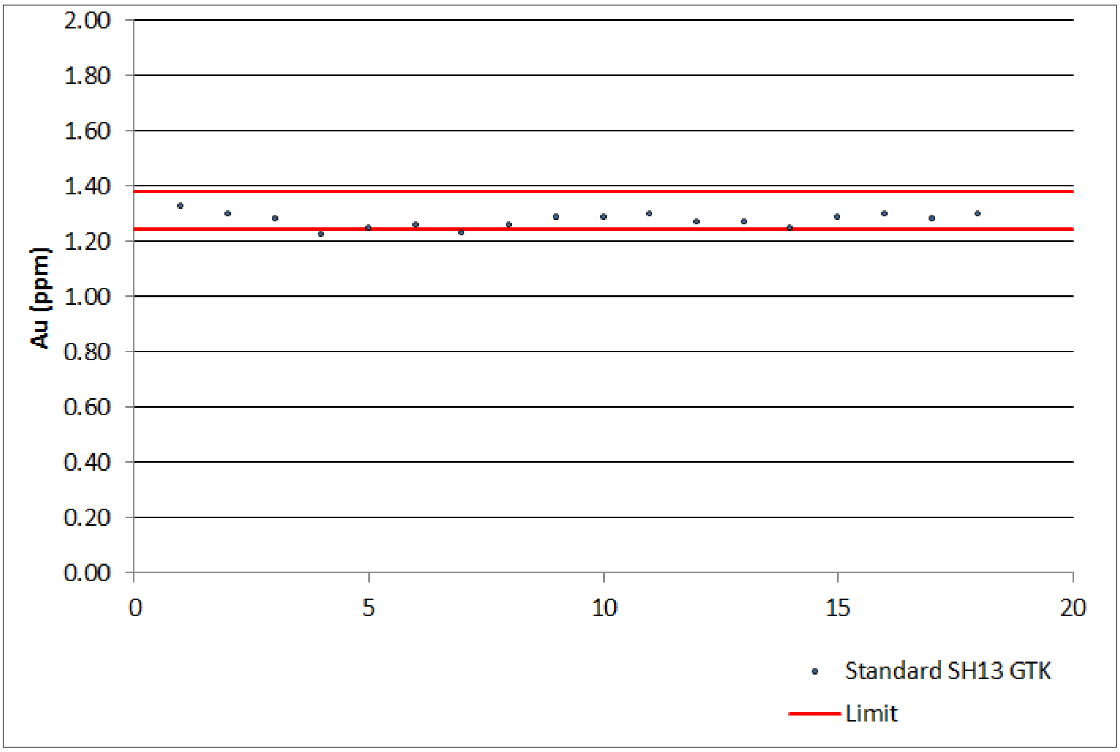


Figure 6-4: Northern Lion Standard SH13 performance at GTK lab

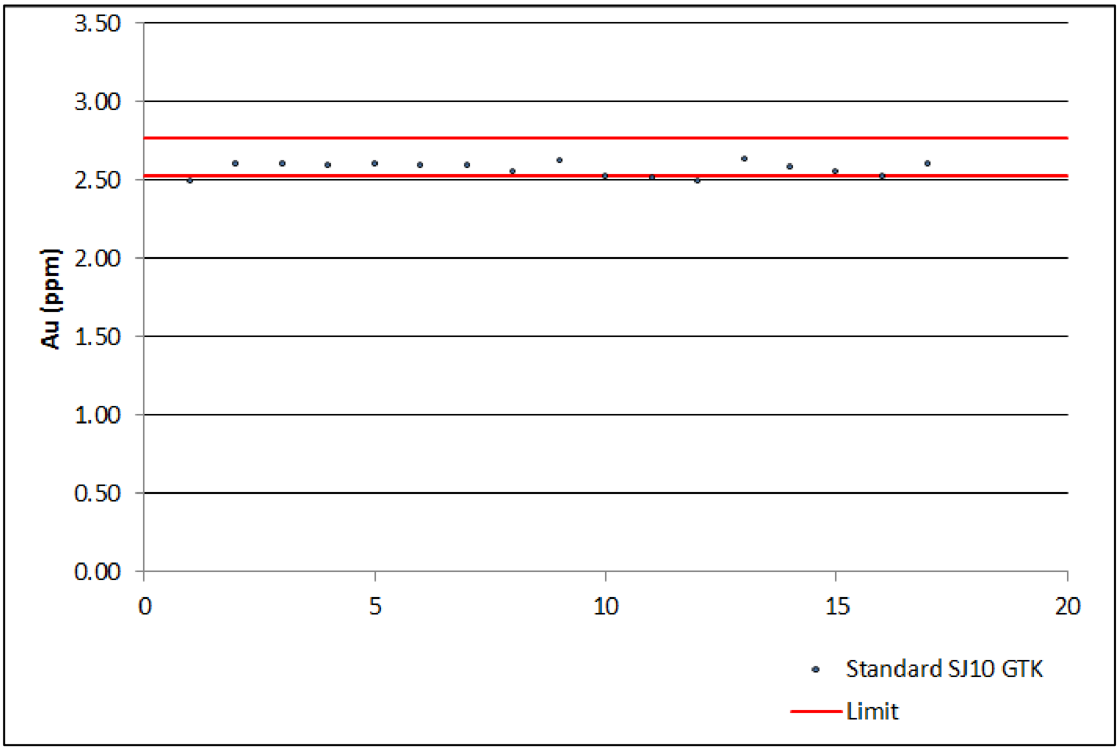


Figure 6-5: Northern Lion Standard SJ10 performance at GTK lab

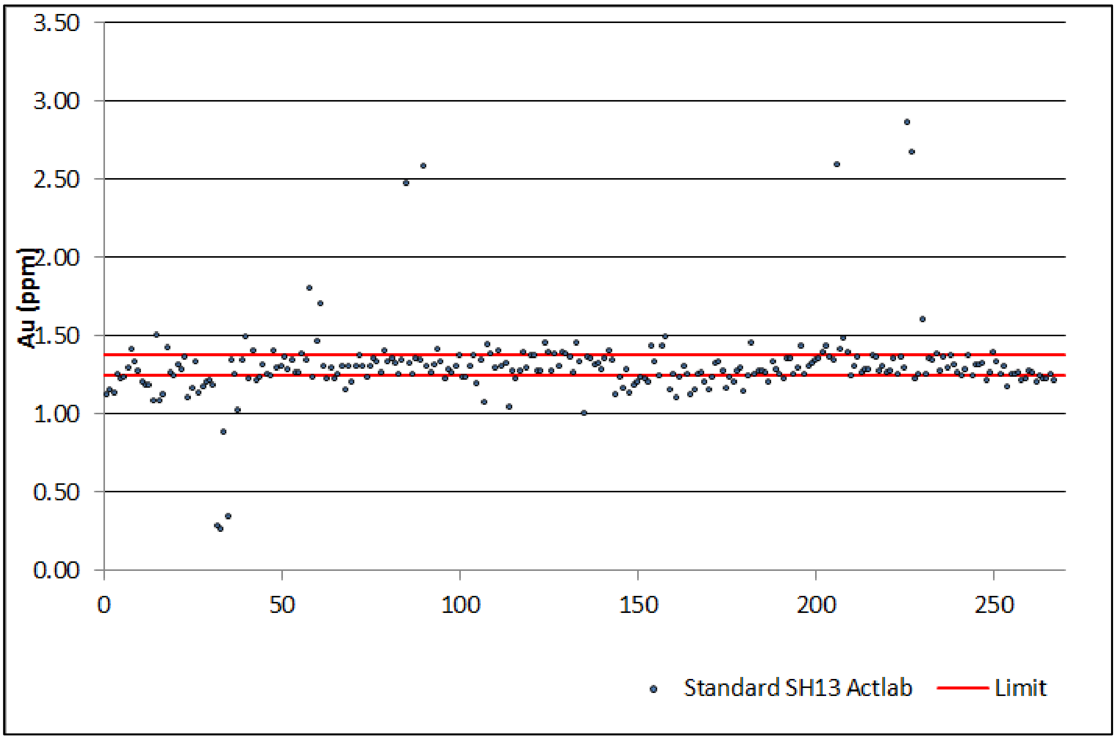


Figure 6-6: Northern Lion Standard SH13 performance at Actlab lab

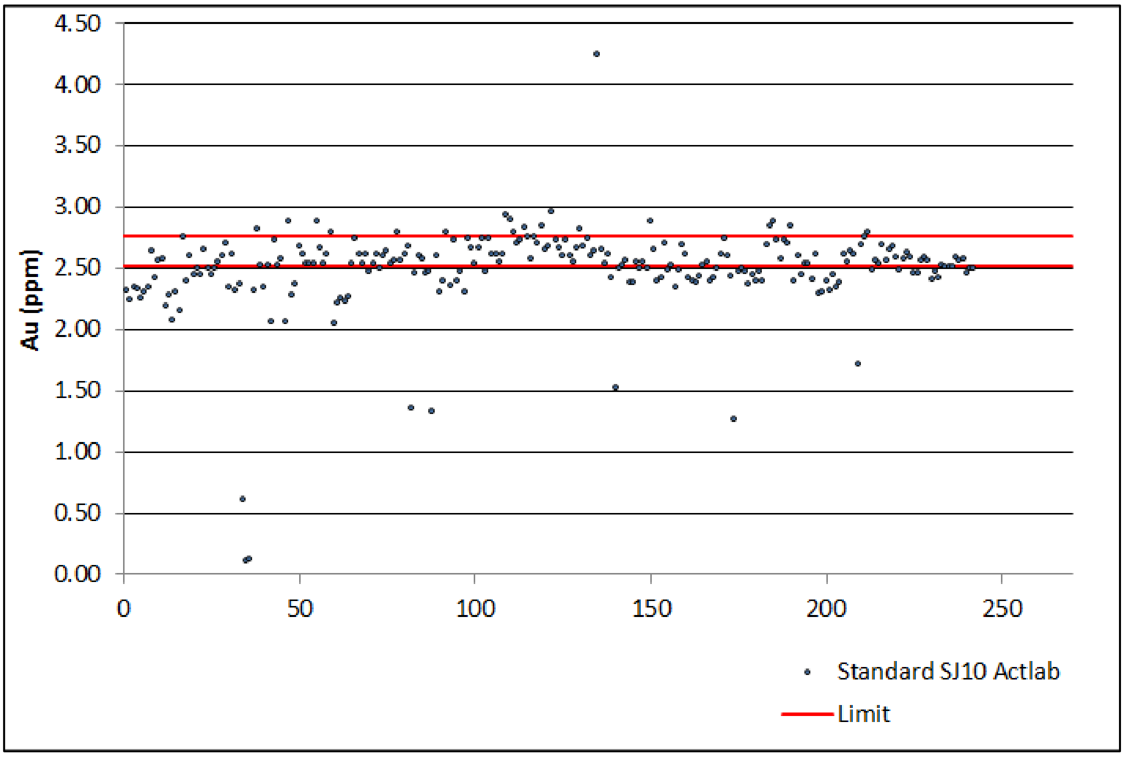


Figure 6-7: Northern Lion Standard SJ10 performance at Actlab lab

The Actlab performance for both standards is below industry standard with a significant number of gross outliers above and below acceptable limits. Both standards have periods of positive and negative bias. Overall, a slight negative bias can be noted for both standards at Actlab if the gross outliers are removed.

GTK's performance is within acceptable limits for both standards. A slight bias towards lower-than-actual reporting can be noted, but is not materially significant.

Duplicates

Actlab and GTK have assayed lab duplicates. GTK analysed a total of 61 duplicates during the four Northern Lion drilling campaigns the results of which were positive (Figure 6-8). Actlab analysed a total of 170 duplicates of which two have been omitted from the scatter plot due to one of the assays being above analytical limits. The result of the duplicate analysis from Actlab shows questionable repeatability especially up to 2 ppm Au (Figure 6-9).

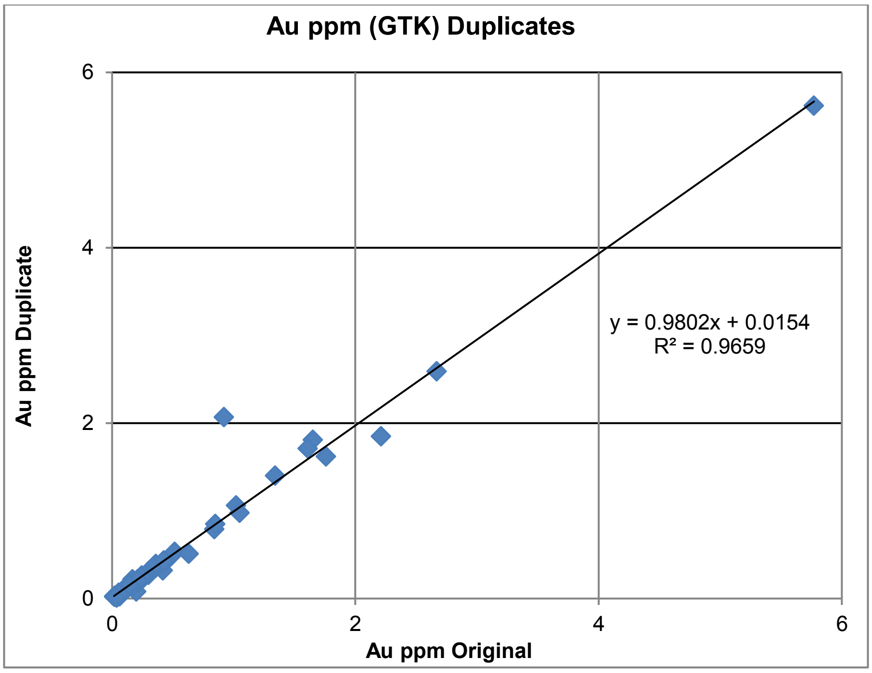


Figure 6-8: GTK lab duplicates. Two pairs (168/162 and 18/16.4) have been removed in order to display relevant grades at a deposit scale.

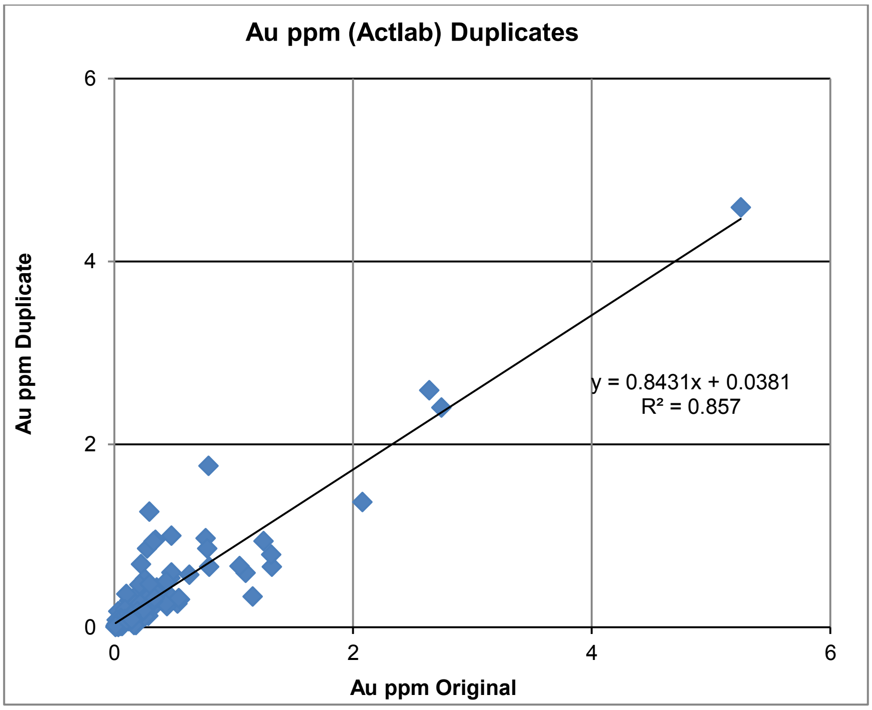


Figure 6-9: Actlab duplicates for originals with Au grades less than 50 ppm (8 pairs ommitted)

Further to the lab duplicates described previously Northern Lion has taken sample duplicates of the core and sent for analysis to Actlabs. Only a small number of these duplicates could be extracted from the database at hand for a number of batches of Phase 4 of the Northern Lion campaigns. Given the low precision performance at Actlabs it is expected that core duplicates would show a very low correlation coefficient as shown below 0.5 (Figure 6-10).

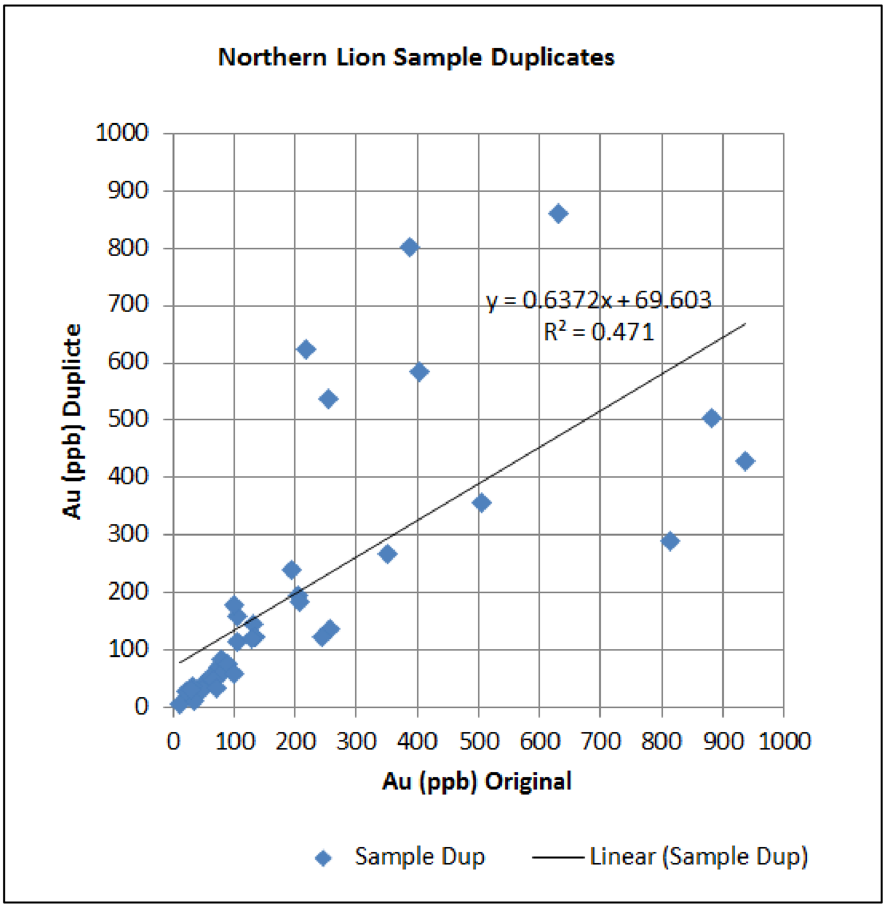


Figure 6-10: Northern Lion sample duplicates analysed at Actlabs.

Density Determinations

A total of 1,836 dry bulk density measurements were completed during NL's second drill phase. Initially density measurements were conducted for entire holes at intervals of 2, 3 or 5 m and subsequently based on lithology changes, mineralisation style changes and for intersects with higher Au-values.

The histogram of density values is shown in Figure 6-11. Removing the erroneous 11.778 value, the average is 3.03 cm^3/t. SRK also checked the average density per domain but noted little variation.

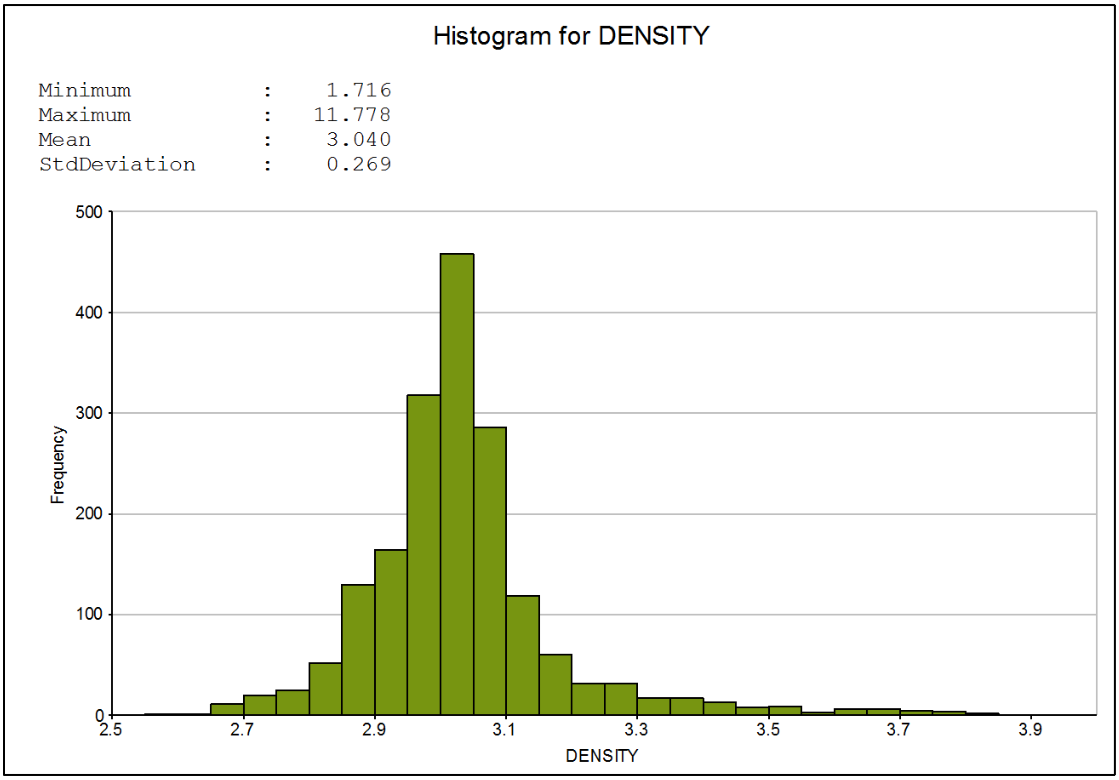


Figure 6-11: Histogram of density values

6.2.5 Historic Exploration Summary

The historic exploration conducted on the Haveri Project to date is summarised in Table 6-6. This includes all information recovered to date. The Vuokseniska and Outokumpu assays are a combination of historic and re-sampled assays by Glenmore Highlands, 70% and 10% of these respectively comprising re-assayed results.

Table 6-6: List of drilling campaigns and assays conducted to date

Company	No. Holes	Meterage	Holes with Au Assays	Holes with Cu Assays	No. Au Assays	No. Cu Assays	Meters Assayed (Au)	Meters Assayed (Cu)
Vuokseniska*	259	23,408	198	171	16,951	6,474	17,261	5,552
Outokumpu*	13	1,577	11	13	588	675	1,321	1,577
GMH**	161	9,826	161	161	7,654	8,141	7,292	7,747
NL*	76	20,887	76	76	13,904	13,851	19,226	19,258
Total	**509**	**55,698**	**446**	**421**	**39,097**	**29,141**	**45,100**	**34,134**

*Diamond drilling only.

**Diamond (80% of meterage), RC (10%) and percussion (10%) drilling.

The locations of drillholes split by exploration programme are shown in Figure 6-12. The average drill spacing is varied. In general the drill centres are spaced less than 25 m apart in the Haveri mine and Peltosaari areas but this reduces to less than 10 m in the central mine area.

No information has been recorded regarding the core recovery of any drilling campaigns.

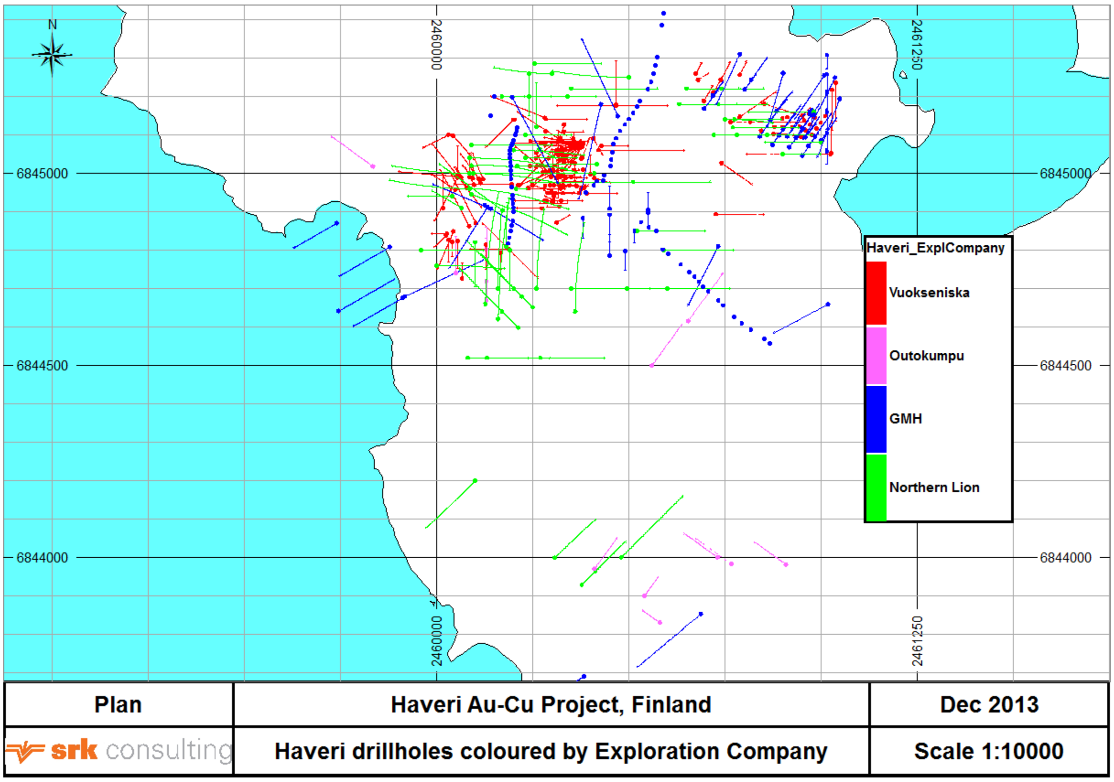


Figure 6-12: Location of historic drillhole collars

Typical cross-sections through the mineralisation at Haveri and Peltosaari are shown in Figure 6-13 and Figure 6-14, respectively; drillholes are coloured by Au (ppm).

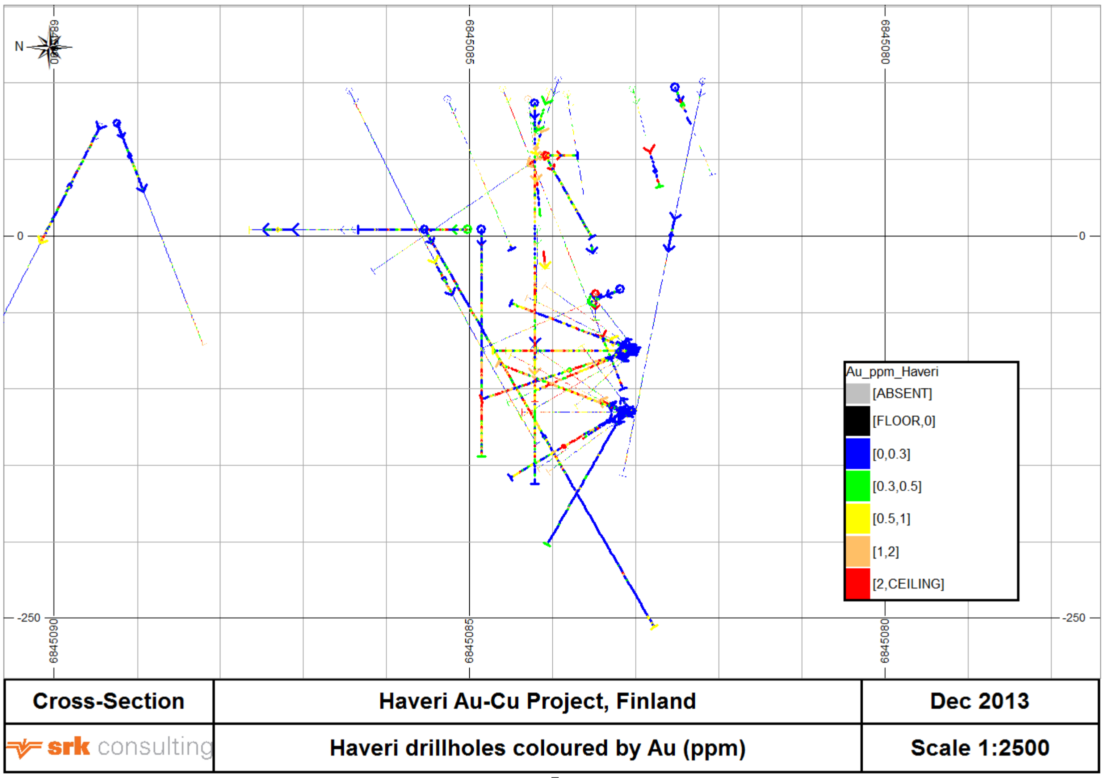


Figure 6-13: Cross-section through Haveri mineralisation

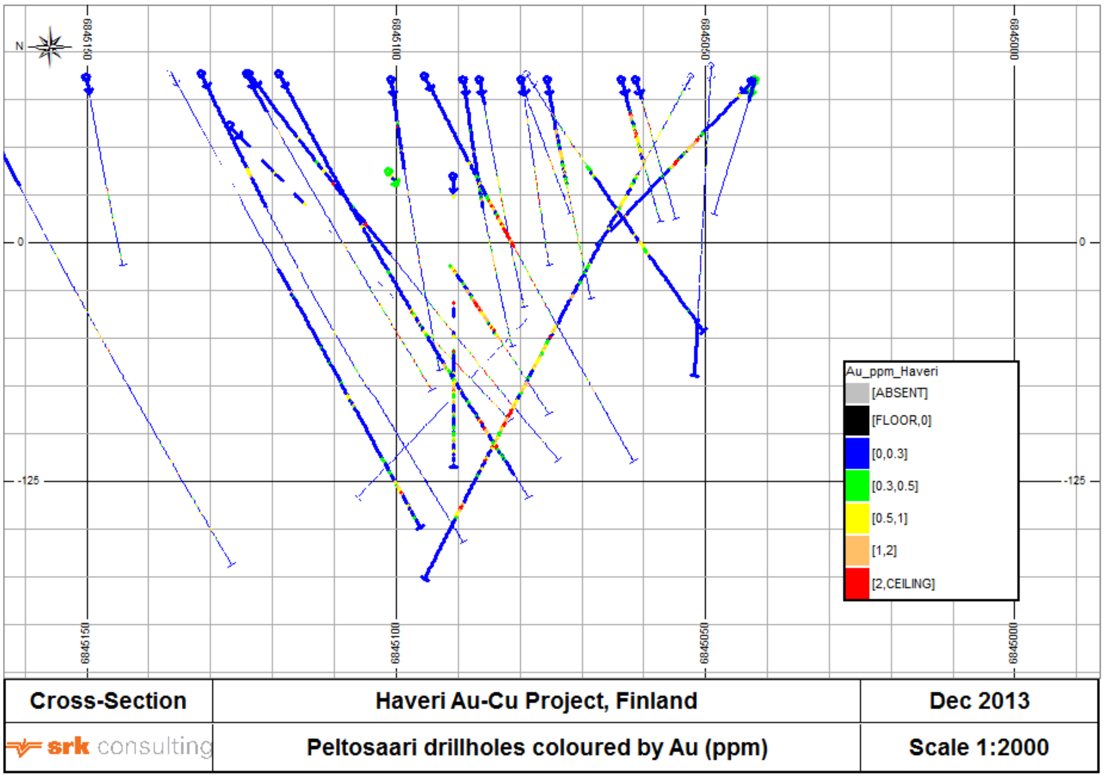


Figure 6-14: Cross-section through Peltosaari mineralisation

6.3 SRK Comments on Historic Exploration

SRK considers that the last operating company to explore the Haveri Project, Northern Lion, developed logging and sample preparation procedures that facilitated the appropriate handling of drill core from the rig through to sample selection, logging and data collection and dispatch of cut samples to the preparation laboratory. SRK considers the core logging facilities to be housed in a suitable building which is clean, modern and appeared to be well-managed. SRK did not witness any drilling taking place and cannot comment on chain of custody or security procedures in place at the time. Assaying by NL appears to have been carried out using appropriate techniques and by appropriately certified independent laboratories. Notwithstanding this, SRK considers that NL's quality control data is below industry standard, which lowers confidence in this data as a whole and is reflected in SRK's Mineral Resource classification as discussed below.

For operators prior to NL, only limited detail has been provided.

6.4 Historic Resource Estimates

A number of historic resource estimates have been completed by numerous operators. The most recent MRE was completed by Maptek in 2008 on behalf of Lappland Goldminers. SRK reviewed this MRE and the accompanying technical report (Reed, 2008) prior to preparing the MRE presented later in this report. The Mineral Resource statement from this report is shown in Table 6-7.

Table 6-7: Maptek Mineral Resource Statement (2008), presented above a cut-off grade of 0.5 g/t Au

Category	Tonnes (Mt)	Au g/t	Au KOz
Measured	17.1	0.92	504
Indicated	5.0	0.79	127
Meas+Ind	**22.1**	**0.89**	**632**
Inferred	-	-	-

With regard to the Maptek MRE, SRK notes the following:

- Limited verification of exploration data was carried-out. Data verification included visiting site, and comparing geological plans and sections to digital exploration data;
- Data from all four exploration campaigns was utilised. Samples extracted by holes drilled by RC and RAB methods, along with trenching, were omitted from the grade and tonnage estimation;
- Mineralisation domain wireframes were outlined using 0.2 g/t and 0.5 g/t Au cut-off grades for low- and high-grade domains, respectively. Mineralisation was orientated 020º dipping steeply to the west-southwest;
- A block model was produced with only Au (g/t) grades estimated. The estimation parameters used are stated below:
  - Search ellipse = 40 m (along strike), 30 m (down-dip), 20 m (across-strike) radii.
  - Minimum number of samples per block = 2.
  - Maximum number of samples per block = 32.
  - Maximum number of samples per octant = 2.
- Resource classification was conducted based on the sample distance between the drillhole data and the estimated blocks. CIM definition standards were reported to have been used. No Inferred Mineral Resources were outlined:
  - Measured = <30 m.
  - Indicated = >30<60 m.
- Resource reporting was based on a 0.5 g/t and 1 g/t Au cut-off grade.

The MRE appears to have utilised reasonable estimation parameters to derive the grade and tonnage estimates. However, SRK noted significant errors with the assay data used for interpolation. In addition, SRK was provided with additional exploration information, which was not utilised by Maptek.

Due to the limitations of the Maptek estimate, SRK does not consider the estimate to be suitable for inclusion in the PEA. SRK has undertaken a Mineral Resource estimate for the Haveri Project, based on the additional data supplied, and in recognition of the errors identified in the historical assay database. SRK considers that the Mineral Resource statement from Maptek has therefore been superseded by the SRK Mineral Resource statement presented herein.

6.5 Historic Mining

Historic mining occurred between 1842 and 1877, and again between 1942 and 1962. Mining by Vuokeniska between 1942 and 1962 used a combination of shallow open pit mining in a sulphide-rich body, and underground mining to a depth of 96 m below surface and reportedly produced a total of 1.5 Mt of material with an average grade of 3.5 g/t Au and 0.5% Cu.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

The Haveri Project is located within a sequence of meta-volcanic and meta-sedimentary units that constitute part of the 100 km long Tampere Schist Belt ("TSB"). The TSB is within the Svecofennian Domain of Proterozoic age (approximately 1.9 Ga) and forms part of a much larger sequence of Mid-Proterozoic supracrustal rocks which occur in the southern part of Finland.

All the supracrustal rocks have been deformed and regionally metamorphosed to greenschist or lower amphibolite facies during the Svecokarelian Orogeny (Lehtinen et al, 1998). Rock types in the TSB include mafic, intermediate and felsic volcanics, turbiditic and conglomeratic sediments, as well as black shales. The Tampere Schist Belt hosts a number of gold showings and prospects, of which, only three, Haveri, Orivesi (Kutemajärvi) and Ylöjärvi, are previous or current producers. Figure 7-1 shows the regional geology of the TSB, with the location of the Haveri project area and the currently operating Orivesi gold mine illustrated.

Haveri lies in a folded sequence of volcanics and sediments that has a pronounced structural fabric orientated 020º (north-northeast – south-southwest). Early thrusts are interpreted to be re-oriented into this direction. This orientation is the inferred transfer fault positions (Figure 7-1) while the east-west trending volcanic-sediment contact is taken as the location of major extensional faulting that create the primary sedimentary basin.

The mineralising late porphyries intrude into basement to the Haveri Schist Belt along the underlying 020º crustal break and form plutons at the basement cover sequence boundary.

The transfer fault direction has a characteristic spacing of approximately 20 km, which can be used to determine locations for other mineralised deposits along the TSB. This spacing concurs with the location of the Orivesi mine, along with other targets shown on Figure 7-1.

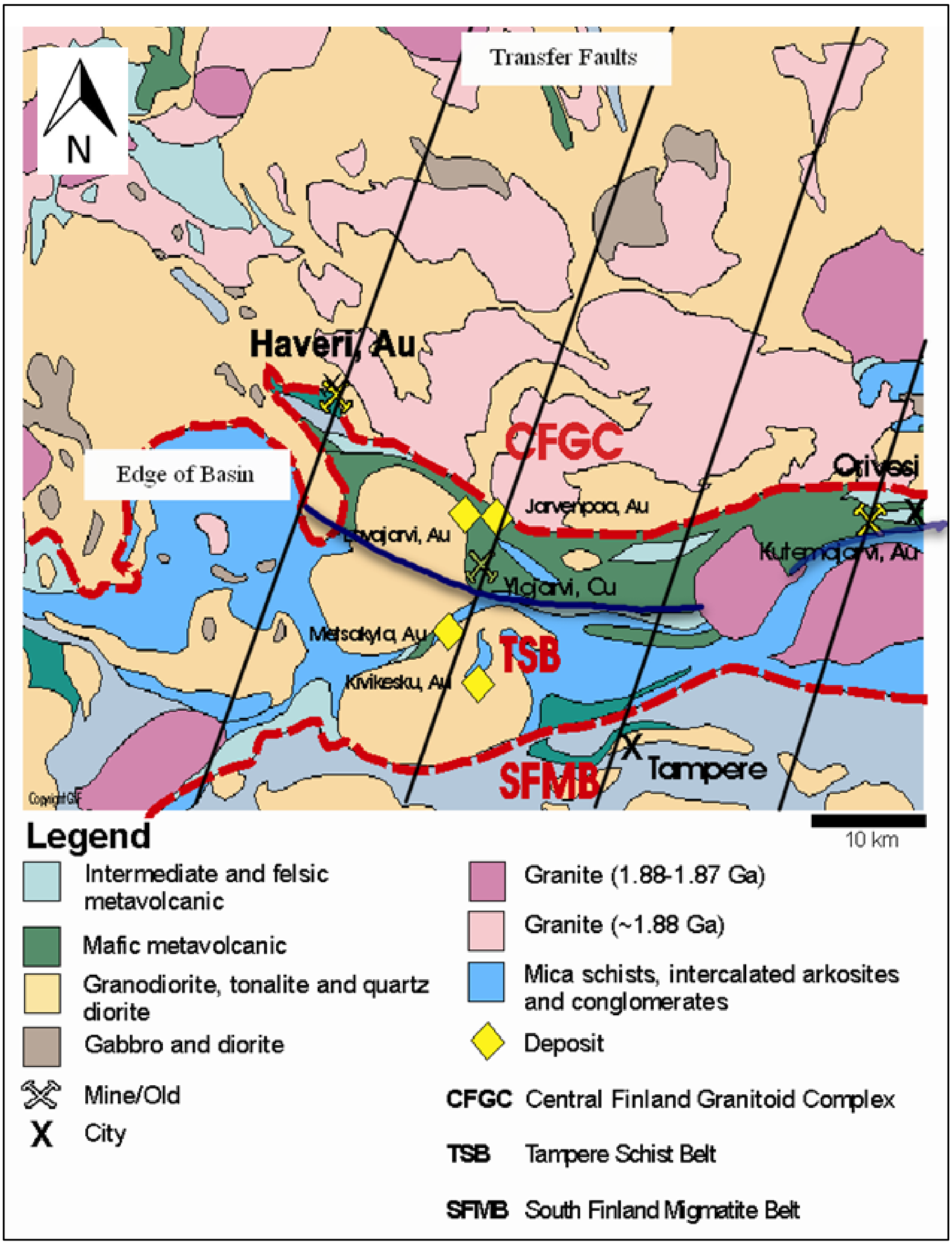


Figure 7-1: Regional geology map (Source: GTK, 2008)

7.2 Local Geology

The local stratigraphy upwards from the lowermost unit comprises mafic meta-lavas, lava breccia, tuffs and tuffites, and meta-sediments (turbidites metamorphosed to mica schists). Amphibolite facies metamorphism and intense deformation have modified these units considerably. A map of the local geology is shown in Figure 7-2, with a smaller-scale map shown in Figure 7-3.

Conflicting evidence has been produced regarding the geological structure of the Haveri area. The description from the structural geology report by Jigsaw (Standing, 2007), below, conflicts with geophysical interpretations which indicate that the Haveri geology is folded into an antiformal structure.

From Standing (2007): '*The structural evolution of Haveri is complex with up to three-fold deformations and numerous small-displacement fault deformations recorded. The basalt-sediment stratigraphy is folded about F_1 isoclinal folds without any significant axial planar foliation being formed, and the deposit is hosted in the hinge of an F_1 synform (east-west axial trace). Granitoids internal to the belt are inferred to have started developing during D_1 or shortly thereafter. The next deformation episode (D_2) resulted in refolding of F_1 isoclines by NNE-trending tight folds (F_2), development of brittle-ductile shear zones within felsic intrusions, and strong vertical stretching lineations proximal to the larger internal batholiths. A second granite intrusive episode is inferred to have occurred during D_2. At Haveri, D_2 shear zones formed parallel with the axial plane of the F_2 folds and mixed sulphide mineralisation concentrated along these in addition to bedding-parallel shears in sedimentary rocks and pillow rinds in basaltic rocks.'*

Drill core shows an S_2 cleavage defined by sericite cleavage replacing the earlier biotite selvedges. The shortening direction during D_2 is inferred to be west-northwest - east-southeast. D_2 was followed by north-south to north-northeast – south-southwest shortening which induced F_3 crenulations on the S_2 cleavage. A multitude of small-displacement brittle faults overprint the folded fabrics.

Ore-related sulphides are strongly controlled by S_2-parallel shear zones (generally trending 020º) as well as primary bedding and volcanic features and accordingly higher-grade shoot controls will be oriented parallel with F_2 fold noses as well as intersections between bedding-parallel shears and the S_2 shear zones (plunging shoots at 60º towards 280º).'

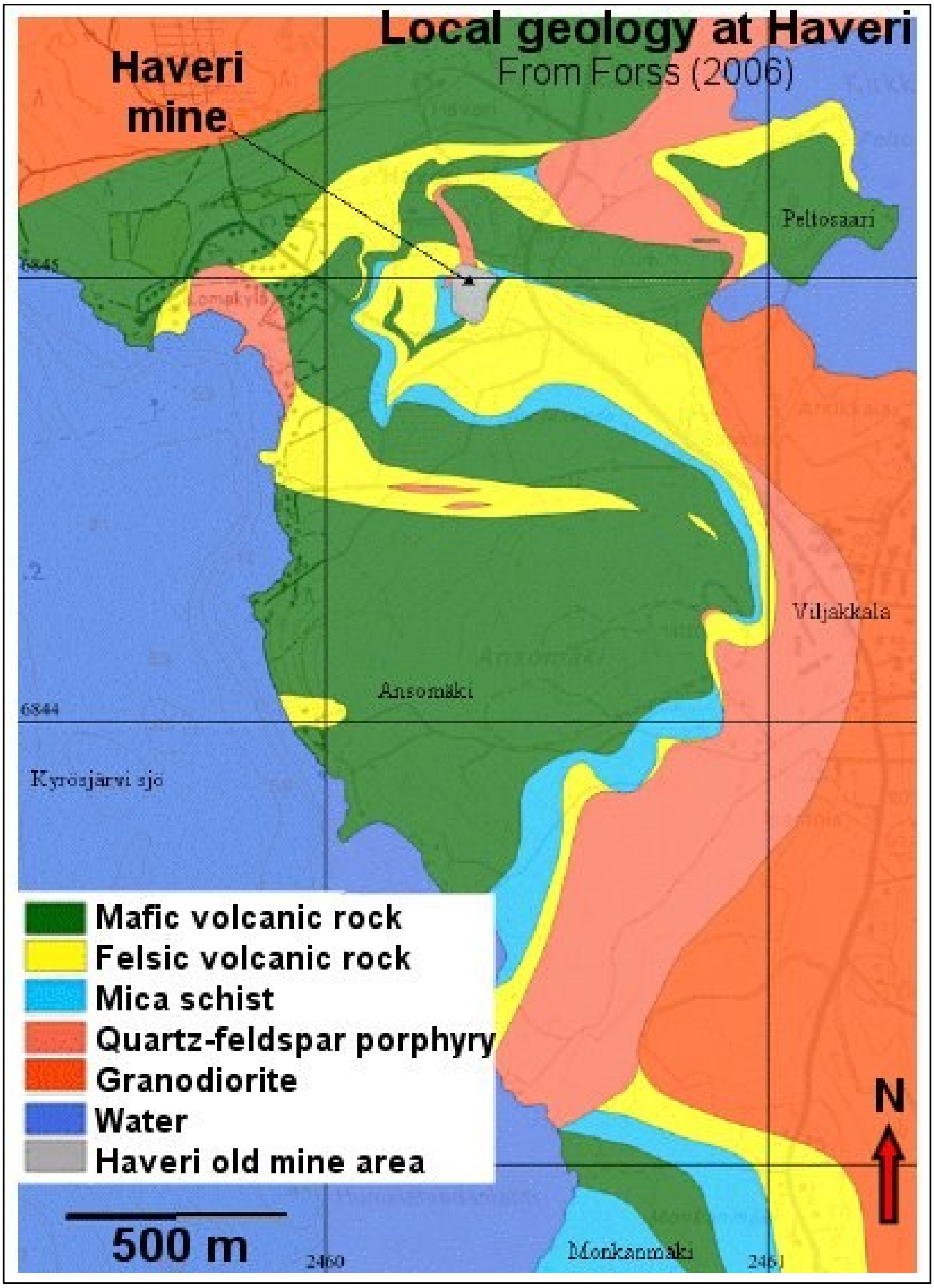


Figure 7-2: Haveri local geology (Source: Forss, 2006)

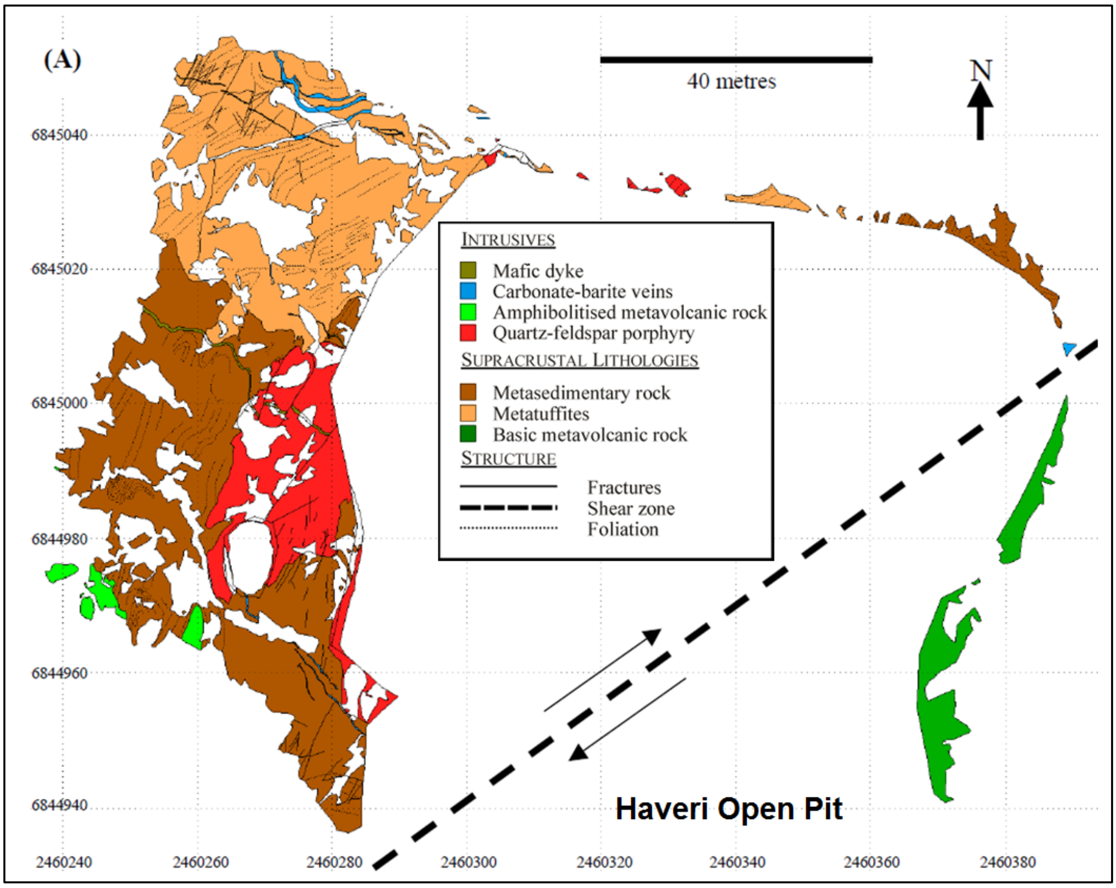


Figure 7-3: Haveri open pit geology (Source: Strauss, 2003)

7.3 Mineralisation

The mineralisation at Haveri is located within zones of iron-magnesium-calcium (Fe-Mg-Ca) alteration that is dominated by amphiboles, with minor pyroxene, chlorite, sulphides (pyrrhotite, chalcopyrite, pyrite), and garnet.

In general, the Au mineralisation is most concentrated along the transition between sediments and mafic volcanics. The highest Au values consistently occur in association with quartz veins/silicified patches and sulphide breccias within zones of strong pyroxene-carbonate-pyrrhotite alteration at the lava-sediment interface. The sulphide breccias are highly deformed and sheared. Another important setting for Au mineralisation is along the boundaries of the porphyries and, less importantly, gold is found associated with massive pyrite in the carbonate veins and as rare coarse-grained native Au disseminated in amphibolite, possibly reflecting fluid-wall rock interaction.

The main mineralisation type at Haveri comprises sulphide veins from a few millimetres to tens of centimetres wide to semi-massive zones 10 m thick and 50 m long. The mineralisation chiefly comprises pyrrhotite-chalcopyrite and magnetite patches, and pyrrhotite-chalcopyrite veins and vein networks. The dominant gangue is dark-green hornblende. The massive to semi-massive type grades into a disseminated type with no obvious change in the mineral assemblage, except for the decrease in the relative volume of gold-bearing minerals and amphibole (Eilu, 2012).

Both the massive and disseminated mineralisation types contain significant gold. Gold occurs at Haveri in the following three major settings (Strauss 2003):

- As free native gold closely associated with pyrrhotite and chalcopyrite,
- As native gold in quartz veins and their immediate wall-rock, and
- locally as very high-grade native gold in the amphibole gangue.

There is no information on whether the invisible gold occurs in sulphide lattice or as sub-microscopic inclusions. There appears to be no linear correlation between Au and Cu concentrations. These observations are in line with the earlier work on the deposit by Stigzelius (1944) and Mäkelä (1980).

A potential mineralising model for Haveri is shown in Figure 7-4 (taken from Jigsaw's 3D model report (Jigsaw, 2008). At depth, a required heat source is required to generate a source of oxidised hydrothermal fluids into the overlying deformed stratigraphy. The fluids are focused along D_2 shears which develop sub-parallel to the axial planar S_2 fabric. The preferential location (denoted by red square inset) is suggested to occur proximally to a magnetic-high, within a low-magnetic susceptibility zone. Mineralogical studies suggest that preferentially Biotite and Muscovite form within these zones. Note the structural position of Haveri is postulated to be within the synform.

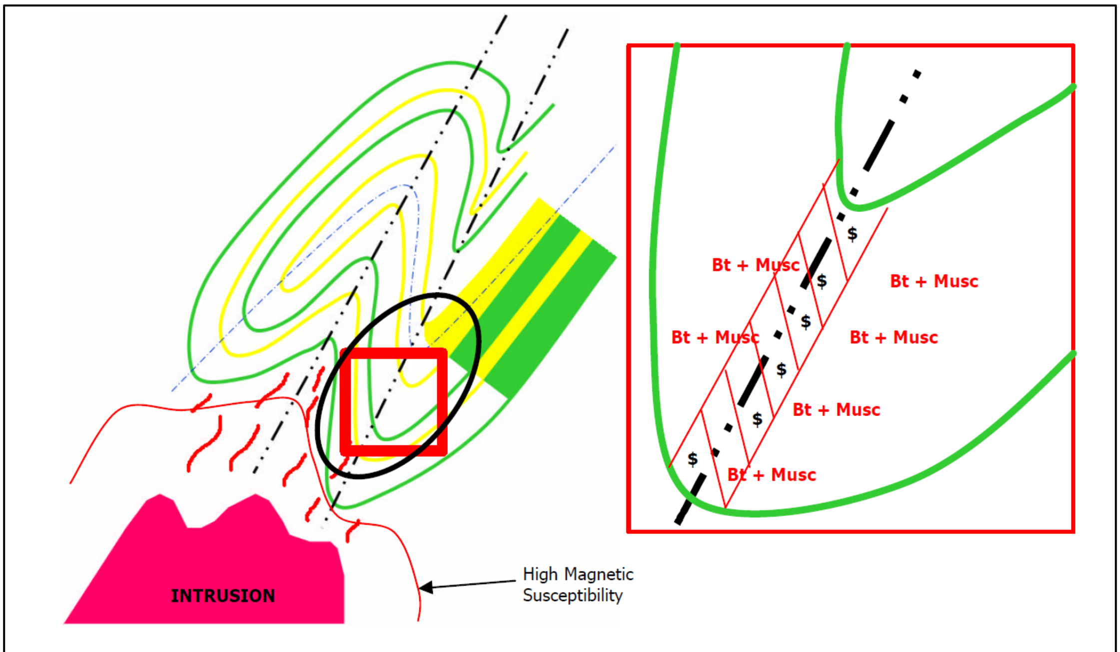


Figure 7-4: Potential Target Model for Mineralisation at Haveri (Source: Jigsaw, 2008)

8 DEPOSIT TYPE

Historically, the Haveri deposit was thought to represent a re-mobilised volcanogenic massive-sulphide ("VMS") deposit. Other studies considered the deposit to exhibit many of the characteristics of lode-Au deposits, certain Au-skarn deposits and Fe-oxide-copper-gold ("IOCG") deposits. Currently there is no one confirmed deposit type, however, the latest study by Eilu (2012) favoured the original theory of re-mobilized VMS theory. The possible conjectured types are described below:

Strauss (2003) concluded that based on 'tectonic setting, age, lithologies, alteration suite, proximity to certain granitoids and conditions of formation, Haveri is considered to be a high temperature, Ca-rich member of the recently recognized IOCG group of deposits.'

According to Hall (2007), 'a preferred geological model of Cu-Au mineralisation at Haveri consists of a porphyry copper-gold system operating during D_2 compression and overprinted by high temperature (amphibolite grade) metamorphism. The faults have breached the underlying magma chamber and caused the hydrothermal fluid to drain into these faults and from these faults into other structures rather than produce the characteristic stockwork quartz vein network in the carapace of the intrusion.'

Eilu (2012) concluded that *'lithological, structural, and primary and alteration geochemical and mineralogical evidence mostly support mineralisation in a VMS-like setting, and that the deposit was significantly affected by regional deformation and metamorphism. There appears to be no support for IOCG mineralisation, and only a little support for any significant orogenic gold overprint on a Cu-only VMS mineralisation. A number unanswered questions still remain for Haveri, such as the extent of primary geochemical haloes and their timing, chemical changes during alteration, the relationship between gold and biotite alteration, the exact siting of gold in its various mineralogical settings, and fluid, metal and heat sources during mineralisation.'*

9 EXPLORATION

Palmex has not conducted any exploration on the property to date.

10 DRILLING

Palmex has not conducted any exploration drilling on the property to date.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Palmex has not conducted any exploration sampling on the property to date.

12 DATA VERIFICATION

12.1 Introduction

In order to independently verify the Company's drill database, SRK conducted the following:

- QAQC analysis of historic drilling and assay data, as described in Section 6.2 above;
- Comparisons of hard copy data to the digital database (including collar locations, down-hole surveys and assays);
- A review of historic collar re-surveys undertaken by T&J Holmback Ab Oy (Holmback) in 2008;
- An inspection of several drill collars at the Haveri site to confirm location of these;
- Drill core inspection of 9 Haveri holes with good spatial representation across the deposit, cross-checking geology, mineralisation, sample interval and sample numbers against the Company's drill database; and
- Collection of 52 coarse reject samples for check assaying. These samples were selected by SRK on the basis of their spatial origin and grade representivity.

12.2 Database Checks

The input database used in the previous MRE by Maptek in 2008 contained errors. As a result, SRK audited the databases provided to create a new consolidated and validated database compiling all four exploration company data sets.

SRK compared 10% of the NL database assay values to the original laboratory assay sheets for Actlabs. SRK found a number of discrepancies amounting to approximately 10% of assay values not matching the original values when compared by sample identification number. This equates to the number of QAQC duplicate or repeat samples submitted by NL and Actlabs combined. SRK considers that the origin of the error is due to a calculation being used by the database compiler. Where a duplicate or repeat sample was encountered, the value for the assay was calculated as the average of the assay values. However, using an average of two assayed grades is not considered 'best practice', as the majority of samples are based on a single result and to do so would reduce the inherent variation in the gold content. It is SRK's opinion that the original sample assay should be retained in the database and the duplicate sample utilised for QAQC analysis only.

12.3 Historic assays versus GMH Re-assays

It is stated in the Maptek report (Reed, 2008) that 5,000 historic assay pulps were re-assayed by GMH in an attempt to verify the historic data. SRK found approximately 4,500 samples in the GMH database relating to Vuokseniska holes and also samples relating to drillholes without collars, which may be channel samples. The historic database appears to only contain original Vuokseniska assays where no re-assays were taken. It is therefore not currently possible to compare the historic and re-assayed values.

It is recommended that the original historic assays are located and compared to the GMH re-assayed values in order to help with verification of the historic data.

12.4 Maptek Check Assays

Maptek selected 135 check samples from NL drilling for re-assaying. The core selected comprised mainly half core, and occasionally quarter core where additional samples had been collected. Core lengths check-sampled were as close to the original lengths as possible in order to ensure a direct comparison.

Maptek reportedly supervised sample collection, sawing and placing of these samples into individual sample bags with an identifying tag. The bags were delivered to the assay preparation facility at ALS Chemex, Piteå (Sweden). Samples were prepared in Piteå, and shipped to the assaying laboratory in Vancouver. Maptek stated that ALS assayed the samples using the ME-ICP41 multi-element analysis. However, SRK contacted ALS and confirmed that this method does not report Au grades, and so SRK is unsure which assaying method was used.

The results of the check assay analysis by Maptek are shown in Figure 12-1. Despite a large dispersion in grades, the results demonstrated that highly elevated Au values exist at a similar order of magnitude to the original NL assay values. SRK concur with these findings.

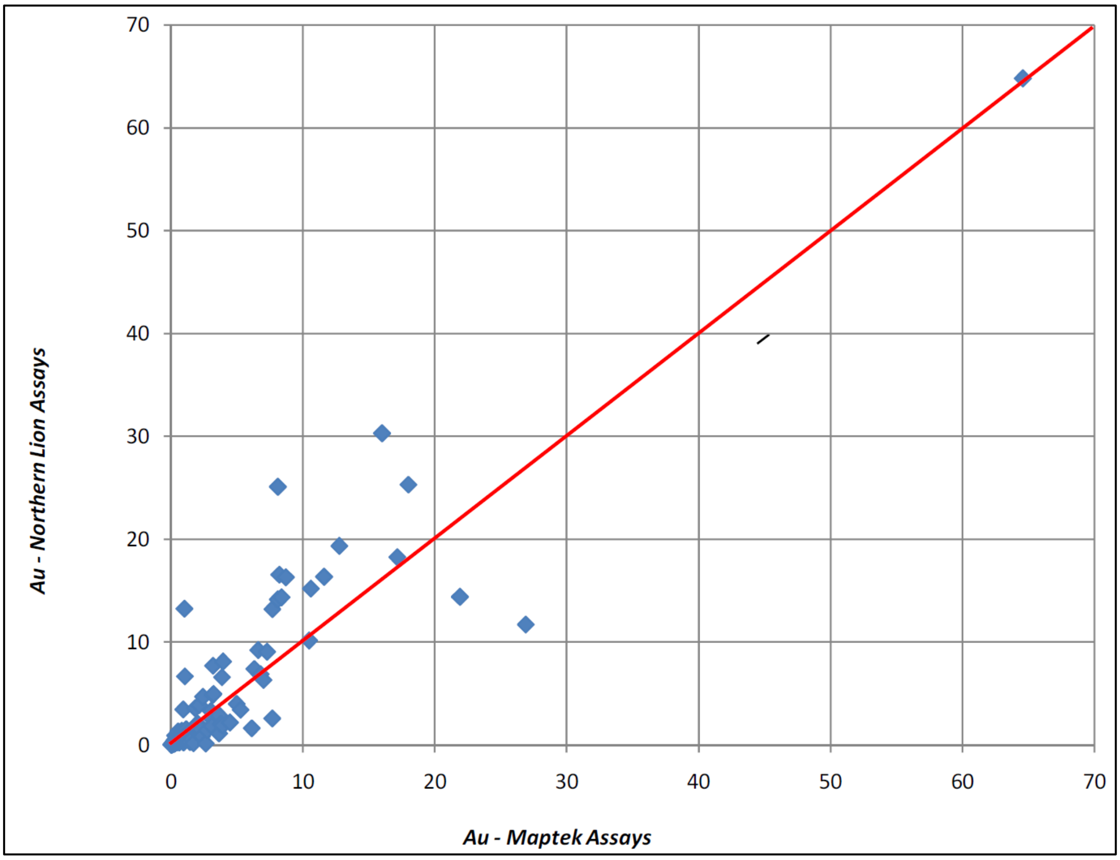


Figure 12-1: Maptek check Au assays vs original assays (Source: Reed, 2008)

12.5 SRK Site Visit and Northern Lion Drill Core Inspection

SRK visited the site and site facilities in Havari on February 5 and 6, 2014. A number of drillholes was selected prior to the visit based on relevance for the resource model and giving a good spatial distribution of holes across the deposit. The selected holes are listed in Table 12-1. These were delivered by Palmex's consultants from their logging facilities in Viljakkala. This facility is part of a small industrial complex of buildings on the edge of Viljakkala town. The building is secure, well-lit and heated, with purpose built roller tables and separate room for core. SRK inspected roughly 1km of drill core and found that all sample intersections were marked properly with sample number tags stapled to the boxes.

Table 12-1: Holes selected for core inspection during site visit.

BHID	Meterage
HN10	179.00
ME02	258.70
MD01	114.90
SW01	299.60
SW03	307.90
SZ05	262.45
P04	301.15
P06	319.50
P09	231.90

12.5.1 Collar Locations

During the field visit in February 2014 SRK inspected a small number of collar locations and took hand held GPS readings for several of these collars around the historic open pit.

12.5.2 Drill Core Storage

During the site visit SRK inspected the core storage site. The majority of the Northern Lion drill core is stored in a secure, heated farm shed about 2 km away from the deposit. The core shed is easily accessible and used for storage of farming equipment and as a garage. The core boxes were stacked in an orderly fashion and secured with metal straps on wooden pallets. All core boxes seemed to be in a reasonably good condition. SRK notes however, that the core boxes were stacked dangerously high, which could pose a risk to drill core and health and safety (Figure 12-2).

Figure 12-2: Northern Lion drill core storage (Source; SRK, 2014).

12.5.3 Coarse Reject and Pulp Storage

Coarse rejects and pulps are stored in a shed at the same farm as the drill core. The building is not heated and contains a large amount of farm equipment and firewood. SRK notes that whilst the bags containing the pulp / reject material appear to be well labelled by the laboratory, these are not kept in good order. Further, a number of bags had split causing loss off sample and possible contamination. The pallets on which samples were kept were placed were poorly stacked and these were at risk of collapse. SRK recommends that the Company catalogues the remaining samples and re-packs these in good order, ideally in a better, secure location.

12.6 SRK Check Assaying

SRK selected 52 coarse reject samples from 14 drillholes for assay at Labtium which were then re-assayed by method code 705P for Au and 511P for multi-element analysis ICP_OES.

Figure 12-3 and Figure 12-4 show scatterplots of the results from the check assaying of Actlab samples at Labtium. Qualitatively it can be said that the Actlab original assays appear to over report Au grades when compared to the Labtium results. In the low and medium grade range up to 5 ppm Au, significant relative over reporting can be noted. SRK notes that from this limited number of observations alone it might be premature to conclude systematic over-reporting. However, in connection with the poor QAQC results from the original sampling, an indication for a possible bias in the database could be inferred. This reduced confidence in the data is reflected in SRK's Mineral Resource classification presented below.

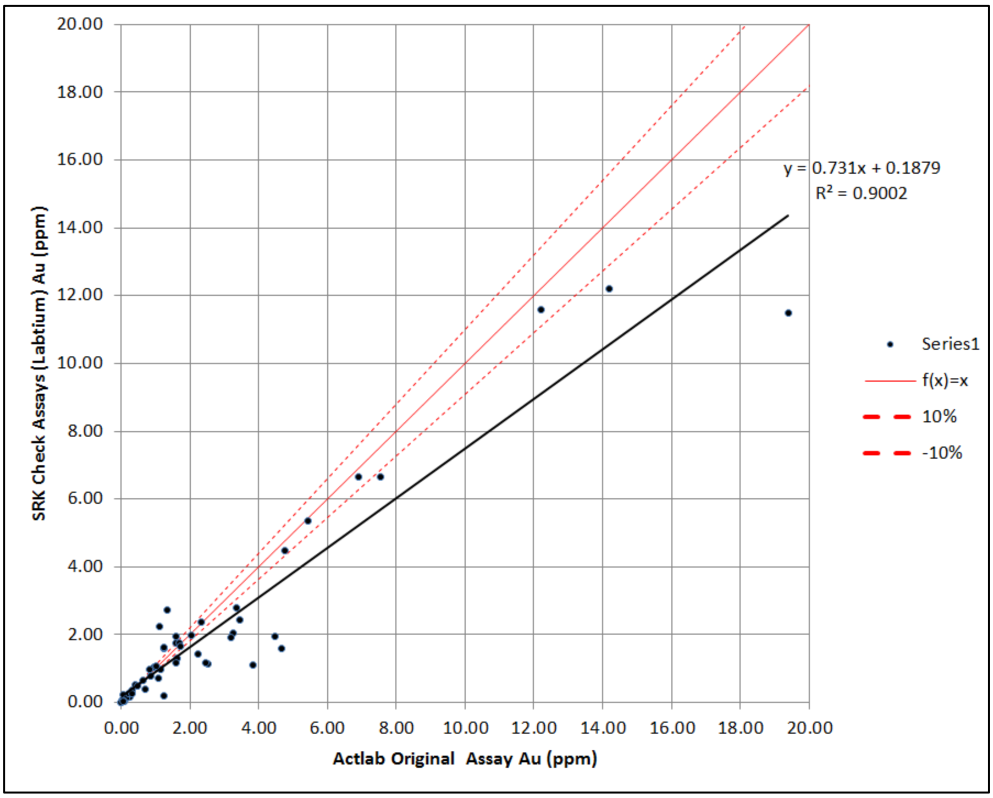


Figure 12-3: Scatter plot of SRK check assays for Actlab coarse rejects (complete results)

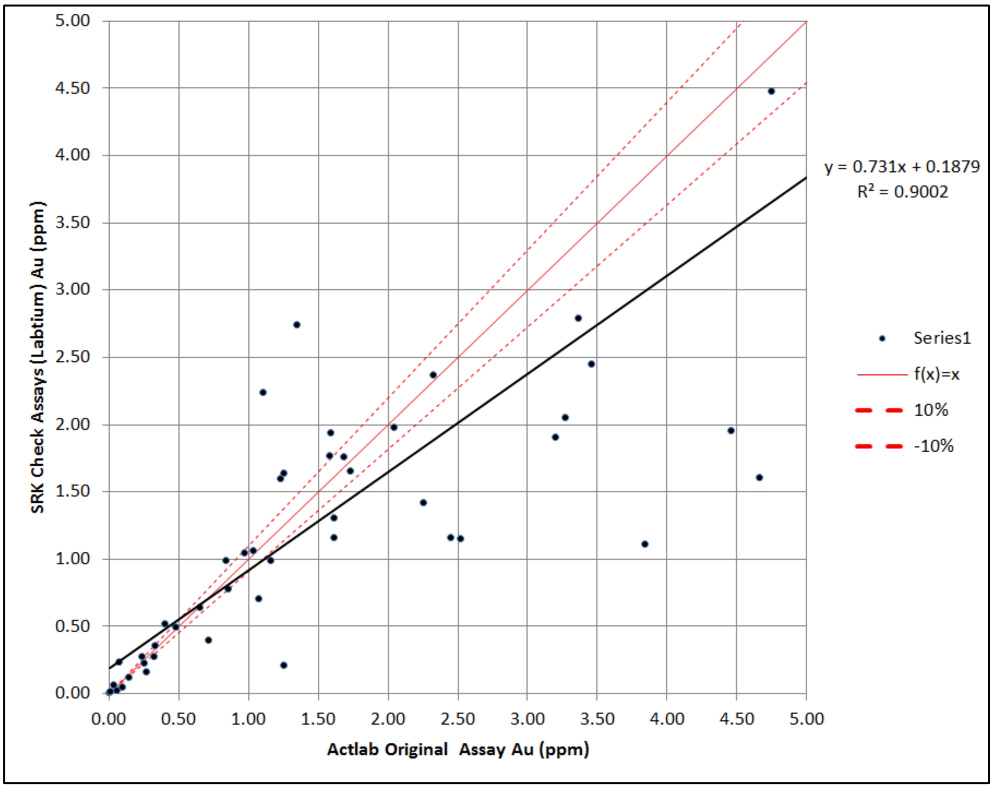


Figure 12-4: Scatter plot of SRK check assays for Actlab coarse rejects (low and medium grade results only)

12.6.1 Dataset Comparison

Due to the varied ages of the exploration campaigns, and the differing sampling and assaying methods used, the various datasets were compared to ensure they are compatible for use in SRK' independent MRE. Datasets were compared using quantile-quantile ("Q-Q") plots, which provide a breakdown of average grades of the datasets within stated quantile bins. Compatible datasets should show a strong correlation, as close as possible to a 1:1 ratio.

The results of the comparison are shown in Figure 12-5. As can be seen, the highest correlation exists between the Vuokseniska historic and re-assayed samples, with a good correlation, particularly at low grades. The other comparisons show markedly different results, with the Vuokseniska assays consistently higher grade than the NL and GMH assays. This can be explained by the location of the samples. The Vuokseniska holes are mainly clustered around the old mine area, with a high density in the area of known mineralisation. The other campaigns are more evenly distributed throughout the different deposit areas. This is shown by the higher correlation between NL and GMH holes, which are both widely spread, when compared to Vuokseniska holes.

The relatively strong correlation shown between the Vuokseniska historic and re-assayed results (by GMH) provides a degree of verification for the Vuokseniska assays. However, there is still a lack of supporting verification for the GMH data. Although large differences have been highlighted between the various datasets, due to the spatial differences between the datasets it is not possible to confirm whether the datasets are incompatible.

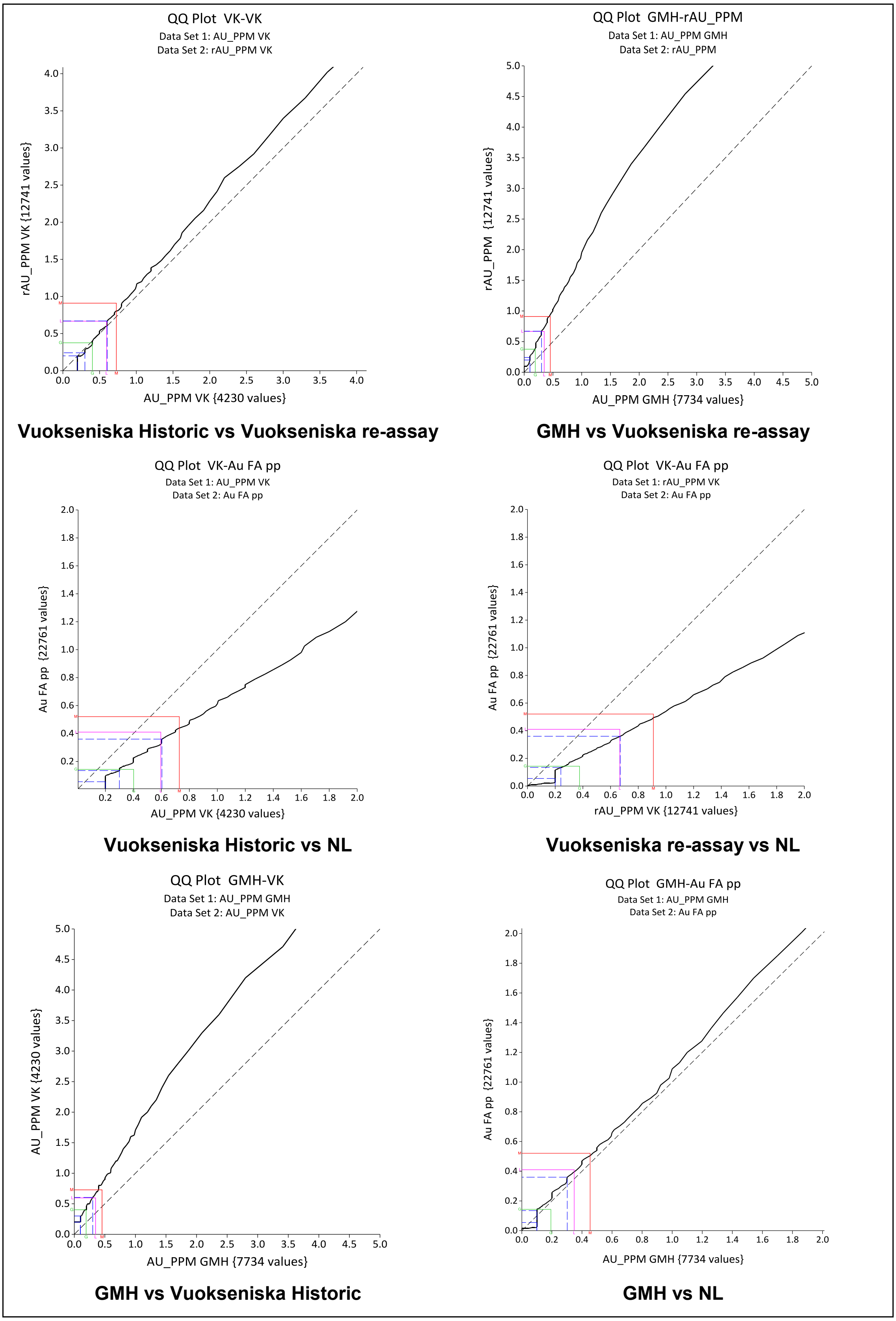


Figure 12-5: Q-Q plots comparing Au (ppm) assays from different datasets

12.7 Collar coordinates and Down-hole Survey Checks

12.7.1 Holmback Re-surveys

T&J Holmback Ab Oy ("Holmback") re-surveyed 31 of 43 GMH and 60 of 76 NL collar coordinates in 2008. The results showed a high degree of confidence can be attributed to the GMH and NL collar locations, with an average difference of <1 m in X, Y and Z values. SRK consider these differences immaterial on a deposit-scale and no further action needs to be undertaken to verify these locations further at this stage.

Holmback also checked down-hole surveying for 7 GMH and 20 NL holes. This resulted in an average deviation of 1.4º in azimuth, and 0.1º in dip. SRK consider these differences immaterial on a deposit-scale and similarly no further action needs to be undertaken to confirm the drillhole orientations at this stage.

12.7.2 SRK Collar Coordinate and Down-hole Survey Checks

SRK found three NL holes which did not contain down-hole survey data. In the previous MRE, these holes were presumed to be vertical, whereas the planned survey data (in the collar file) shows they are each inclined. The planned dip and azimuths were used by SRK for the purposes of the MRE.

Following the completion of the MRE, one of the missing drillhole surveys could be located for drillhole P03. The surveyed orientation for this hole is within reasonable bounds of the planned orientation and the difference between the planned orientation and the survey of less than 5º appears immaterial on a deposit scale.

12.8 SRK Comments

12.8.1 Northern Lion Data

The number of collars located in the field, drill cores reviewed and check samples selected for assay by SRK represents a small proportion of the overall number of drill collars and analysis carried out on the Project as a whole by Northern Lion. Notwithstanding this, the verification checks carried out on the Northern Lion data suggest this is of sufficient quality to be used as a basis for the Mineral Resource estimate as presented herein. It should be noted however, that there were no blanks inserted into the sample stream and that the overall results from NL's QAQC programme are poor, for the batches sent to Actlab.

12.8.2 Glenmore Highlands Data

No QAQC programs are documented for Glenmore Highland data. Therefore the validity of data from this period cannot be verified by SRK. The data has been used for purposes of domaining and geological modelling, but has been omitted in the estimation of grades in the resource block model. SRK understands that a significant amount of drill core from the Glenmore Highland exploration campaigns may be stored at the national core archive in Loppi, Finland and that a future programme of re-logging and verification sampling may improve confidence in this data.

12.8.3 Vuokseniska & Outukumpu Data

The historic Vuokseniska and Outokumpu data has not been verified by any form of QAQC procedure and the drill core was not available for verification by SRK.

12.9 Data Utilised for the SRK MRE

All available diamond drilling data from historic exploration campaigns as discussed above were used as the basis for SRK's independent MRE. Whilst the quality of the data is considered to be uncertain in some cases and poor in others in view of the QAQC results of the Northern Lion data (as discussed in Section 6.2.4), check assays carried out by SRK as well as the complete lack of QAQC for previous exploration campaigns, SRK has accounted for this through appropriate Mineral Resource classification, which is discussed in further detail below.

RC and percussion drillholes were not considered to be adequate for use in the MRE.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

No metallurgical testwork has been undertaken specifically in support of the production of this PEA. The following sections provide a summary of historic recoveries and testwork and present some conclusions regarding likely future metal recoveries.

13.2 Historical Operation

The historic mining and processing operations at Haveri are briefly described by Karvinen and Fraser (2003). Following extraction of magnetite ore from the area in the second half of the 19th Century, a copper and gold mining and processing operation was conducted by Vuokseniska Oy between 1942 and 1962. Over this period, the operation processed 1.5 Mt of ore with an average head grade of 3.52 g/t Au and 0.52% Cu. The annual production rate reached a maximum of 100 kt, and the “recovered grade” was reported as 2.80 g/t Au and 0.40% Cu, giving recoveries of 80% for Au and 77% for Cu. Karvinen and Fraser (2003) note that gold bars were produced on site, however they do not specifically describe in what form the Cu was recovered; a flotation concentrate is assumed.

13.3 Testwork, 2003

A metallurgical testwork program was conducted by the Geological Survey of Finland (Geologian Tutkimuskeskus, or “GTK”) in 2008 in support of a Technical Report conducted by Maptek Ltd on behalf of the then Project Owner AB Lappland Goldminers Oy (Reed, 2008).

Testwork was conducted on three composite samples. Two of the samples were of drill core from the Haveri deposit, one a composite of intervals from the drill holes SW01, SW02 and SW06, and the other a composite of intervals from the drill holes MD02, MD03 and ME02. The third sample was made up of surface rock blasted from the Peltosaari deposit.

Selected head assays of the samples are shown in Table 13-1.

Table 13-1: GTK Sample Head Assays

Element	Unit	SW	MD-ME	Peltosaari
Au	g/t	1.73	1.36	2.26
Cu	%	0.153	0.057	0.137
Ag	g/t	3.3	1.9	2.9
S	%	1.68	0.75	9.65

Rougher flotation tests concluded that the optimum grind sizes for flotation were 80% -60 μm for the SW and Peltosaari samples, and 80% -40 μm for the MD-ME sample.

Batch rougher-cleaner flotation tests produced results as summarised in Table 13-2.

Table 13-2: GTK Sample Rougher-Cleaner Flotation Test Results

Item	Unit	SW	MD-ME	Peltosaari
Cleaner Con Wt	%	0.4	0.1	0.7
Cleaner Con Cu Grade	%	26.7	28.9	14.0
Cleaner Con Cu Rec	%	67.8	59.2	66.6
Cleaner Con Au Grade	g/t	114	262	120
Cleaner Con Au Rec	%	21.7	22.6	32.6
Cleaner Con Ag Grade	g/t	246	134	54.5
Cleaner Con Ag Rec	%	34.7	16.1	10.2
Cleaner Con Fe Rec	%	2.3	1.0	1.4
Cleaner Con S Rec	%	8.0	5.8	2.2
Rougher Con Au Rec	%	77.8	68.9	68.5
Rougher Con S Rec	%	28.3	22.1	61.7

Cyanide leach tests were conducted on samples of the rougher tailings from the initial (i.e. Rougher only) flotation tests. The results of these tests are summarised in Table 13-3.

Table 13-3: Rougher Tailings Cyanidation Results

Element	Unit	SW	MD-ME	Peltosaari
Au Head Grade	g/t	0.44	0.26	1.20
Au Recovery	%	67.9	65.5	55.0
NaCN Consumption	kg/t	1.63	1.06	2.96
Overall Au Recovery (i.e. to flotation feed)	%	92.5	92.8	77.8

13.4 Tailings Re-Processing Testwork, 1980s

Outokumpu conducted an auger sampling campaign of the historic tailings in 1980 and 1983, and subsequently conducted metallurgical testwork on 1201 samples taken from 165 auger holes (Ketola, 1986). The average grade of the tailings was estimated to be 1 g/t Au and 0.1% Cu.

The testwork focussed on gold recovery only. Flotation recovered approximately 50% of the Au into a concentrate grading approximately 10 g/t Au. Cyanidation produced recoveries ranging between 75% and 80%. Gravity concentration recovered approximately 35% of the Au into a concentrate grading approximately 15 g/t Au.

13.5 Conclusions

Based on the limited testwork reported, there appears to be good potential for the recovery of both gold and copper from the Haveri material.

Based on the 2008 GTK work, the Cu appears in general to respond well to flotation, with high grade Cu concentrates produced from the Haveri sample, despite the relatively low head grades. While the flotation results for the Peltosaari sample were not as good, this sample had a very high S grade, which the GTK report notes was due to the presence of a significant quantity of pyrrhotite. SRK notes that the 95th percentile for S in the Haveri block model is only just above 2% S, and so the Peltosaari sample tested in the 2008 work program, with a S grade of 9%, is not representative of the orebody.

The gold recovery testwork conducted both by GTK and Outokumpu (on the historic tails) indicate that high Au recoveries are possible, by a combination of flotation (into the Cu concentrate) and cyanide leaching of the flotation tailings. Assuming that this combination of process units was used in the historic Haveri operation, the historic data supports the recoveries achieved in the testwork.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

SRK has produced a Mineral Resource estimate for the Haveri Project using the data from historic drilling campaigns. A database was compiled using data from 509 diamond drillholes, with collar, survey, geological and assay information, containing a total of 55,698 m of drilled metres (Table 14-1). In the process of completing the resource estimate, SRK validated and verified the database, interpretation and available data. The block dimensions selected for the block model was 20 m x 10 m x 10 m (X, Y, Z), which reflects the drilling pattern, spatial distribution and mine planning considerations. The grade estimate was produced using ordinary kriging ("OK"). Optimised pit shells were generated by SRK to restrict its estimate to material with potential to be exploited. Various economic parameters such as mining and processing and G&A costs, gold and copper recovery, and pit slope angle were used in as input parameters for the resource pit shells. All open pit resources are stated above a 0.45 g/t gold equivalent cut-off.

This section describes the work undertaken by SRK and summarises the key assumptions and parameters used to prepare the MRE.

Throughout the MRE, the following abbreviations are used:

- CU_PPM – copper grade, expressed as parts per million, also written as Cu (ppm).
- AU_PPM – gold grade, expressed as parts per million or grams per tonne, also written as Au (ppm) or Au (g/t).
- AG_PPM – silver grade, expressed as parts per million or grams per tonne, also written as Ag (ppm) or Ag (g/t).
- BI_PPM - bismuth grade, expressed as parts per million, also written as Bi (ppm).
- FE_PERC – iron grade, expressed as a percentage, also written as Fe%.
- S% - sulphur grade, expressed as a percentage, also written as S%.
- AS_PPM - arsenic grade, expressed as parts per million, also written as As (ppm).

14.2 Drillhole Database

The drillhole database used for the Haveri MRE comprises 509 diamond drillholes for a total meterage of 55,698 m, of which 45,100 m has been assayed for Au. This is summarised in Table 14-1 below.

Table 14-1: Available drillhole data

Number of drillholes	Total Drilled (m)	Assayed Metres (Au)	Assayed Metres (Cu)
509	55,698	45,100	34,134

14.3 Geological Modelling and Domaining

The geological modelling of the mineralisation zones was conducted in a combination of Leapfrog and Datamine Studio 3 software and comprised the following:

- importing the collar, survey, assay and geology data into both Leapfrog and Datamine to create a de-surveyed drillhole file;
- importing the topography data file;
- verification of previous lithology wireframes and the creation of mineralisation wireframes; and
- the creation of an empty block model coded by zone to distinguish the different geological domains identified.

14.3.1 Lithological Domain Modelling

SRK utilised lithological and structural geology wireframes created by Jigsaw in 2007 in order to code the model into lithological domains. The wireframes were imported and verified by SRK.

Three main lithological units were modelled by Jigsaw, namely basalt, metasediment and porphyry units. These units were modelled based on the geological map created by Forss (2006), as shown in Figure 7-2, along with lithogeochemical modelling by Jigsaw (2007) and geophysical anomaly surveys. The study by Jigsaw used a lithogeochemical spectral analyser (from ASD Inc.) to perform lithogeochemical characterisation of the lithologies in the Haveri area. The study indicated several useful pathfinders for identifying lithologies, such as biotite found only in the basalt, and a high Sc/Th ratio in mafic units compared to a low Sc/Th ratio in metasediments. The resulting lithological wireframes are of high quality and are geologically robust. Large fault zones interpreted from geophysical data were also created. These faults do not cross-cut the mineralisation, but provide possible fluid pathway information (for geological and hydrogeological purposes). The resulting lithological and structural wireframes created by Jigsaw are shown in Figure 14-1 and Figure 14-2.

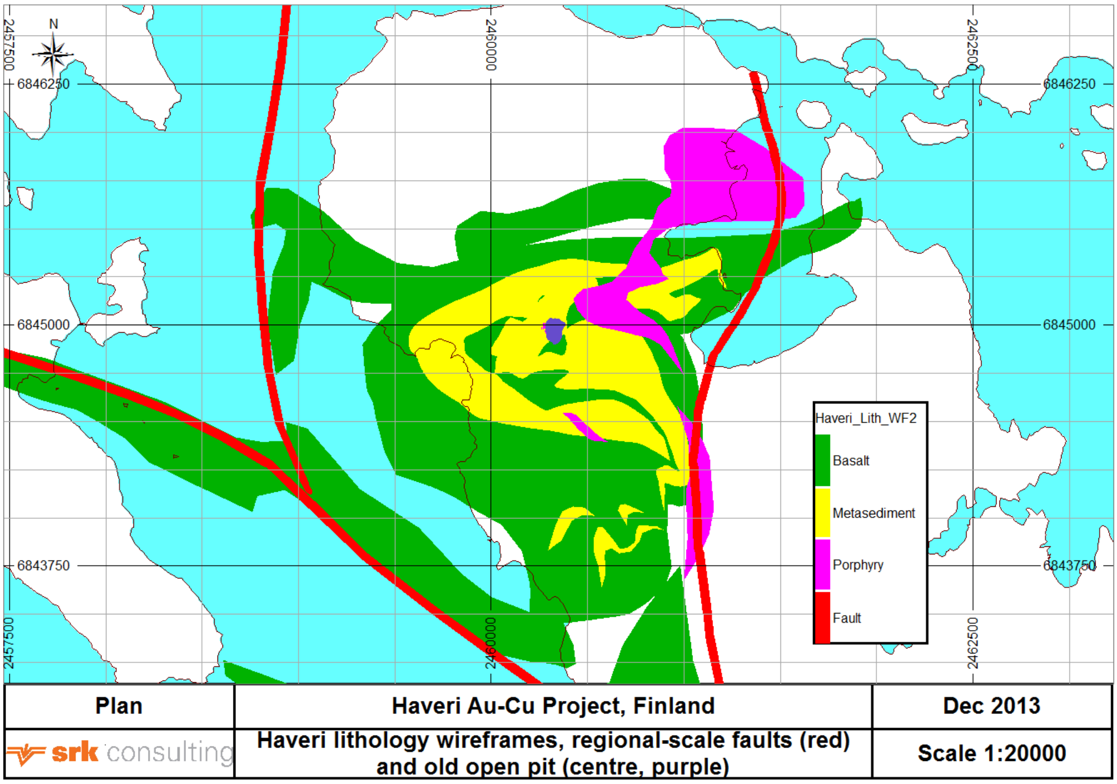


Figure 14-1: Haveri lithology wireframes and faults created by Jigsaw (Source: SRK, 2014)

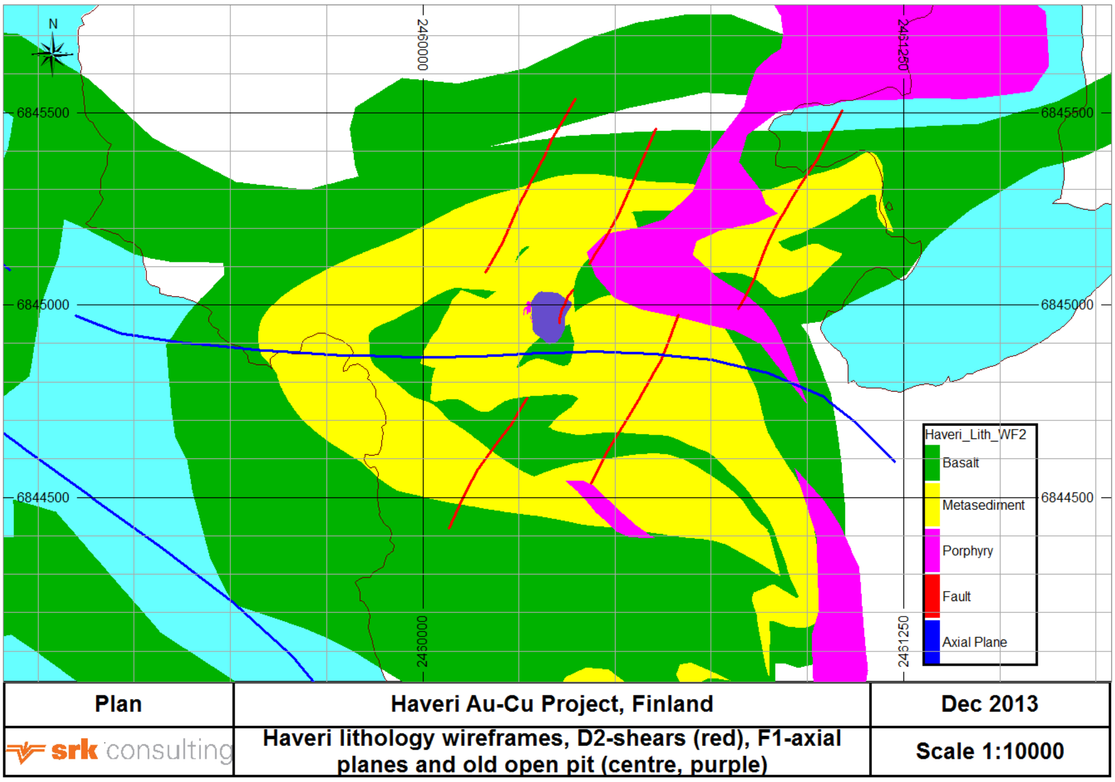


Figure 14-2: Haveri lithology wireframes, D2 shear zones and F1 axial planes created by Jigsaw (Source: SRK, 2014)

14.3.2 Overburden Surface Modelling

SRK used Leapfrog to create a surface for overburden material, including quaternary glacial till, other alluvial material and soil. This surface was based on overburden contact points using logging codes provided in drillholes. The overburden material varies in thickness from 0 to 27 m, with an average down-hole (not vertical) depth of 10 m.

14.3.3 Mineralisation Domain Modelling

The work by Jigsaw (2007) resulted in several conclusions for mineralisation targeting. The following observations were made to assist with exploration:

- The basalt-metasediment contact is a favourable mineralisation deposition zone.
- Mineralisation is well developed in low-magnetic susceptibility zones (within a general high-magnetic zone). The magnetite is interpreted as alteration overprinting of an already magnetic top of the basalt (tholeiitic basalt). This observation implies the basalt sequence is fractionated from magnesium-rich base to iron-rich top.
- Biotite is only present in the basalt. Biotite is interpreted as the product of alteration that is not destroyed by subsequent high temperature metamorphism. The biotite envelope is significant as a measure of the alteration pervasiveness
- D_2 shear-planes trending 020-045º (dipping steeply towards the west-northwest) are the main structural control on mineralisation. These structures are inferred as representing the main feeder zones from the underlying magmatic source. Grade bleeds out from these structures along cross-faults (and F_1 axial planes) along with lithological contacts (pillow margins).
- Intersection of D_2 shears and east-west trending F_1 axial planes provides the main target for higher-grade shoot-like structures (approximately 60º towards 280º).

On the basis of these findings, SRK used a dominant mineralisation direction of approximately 020º, dipping steeply (70º) to the west-northwest, in addition to attempting to identify high-grade plunging shoots. A mineralisation cut-off of 0.3 ppm Au was used in order to ensure the spurious 0.2 ppm Au grades did not have an adverse effect on the wireframing. Two high-grade (>1 ppm Au) plunging structures were created: one at Haveri and one at Peltosaari.

The resulting wireframes are shown in Figure 14-3 to Figure 14-5, with the high-grade wireframes shown inside the low-grade wireframes.

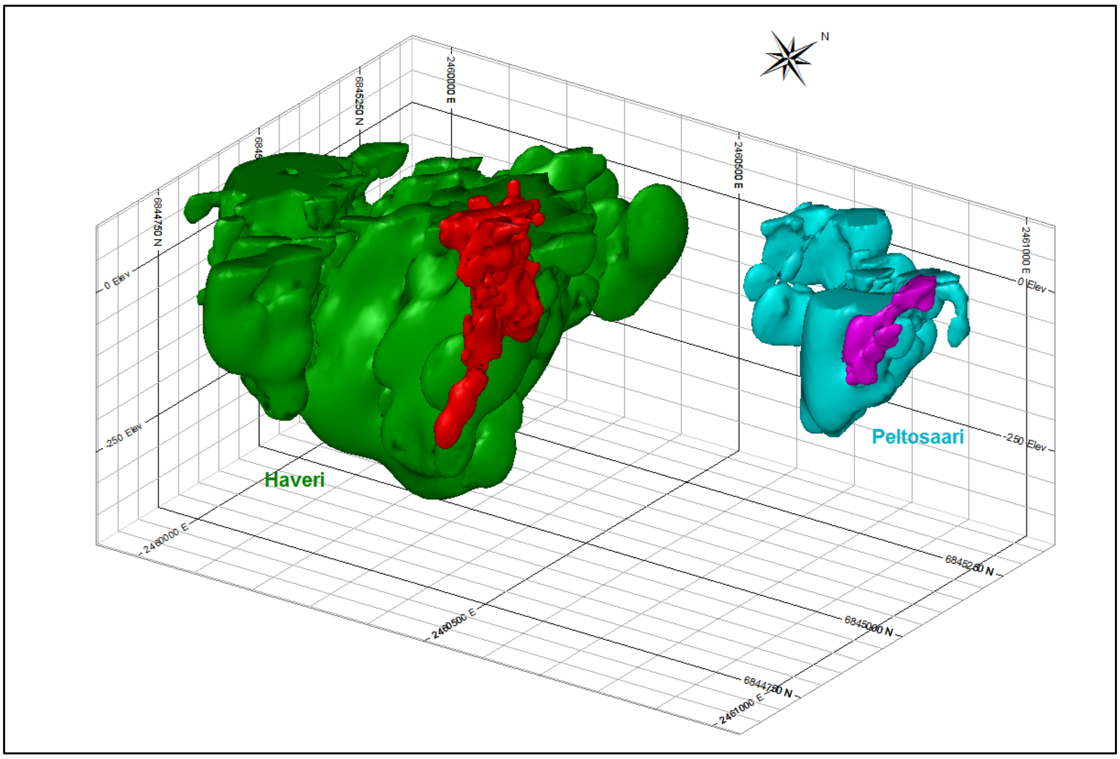
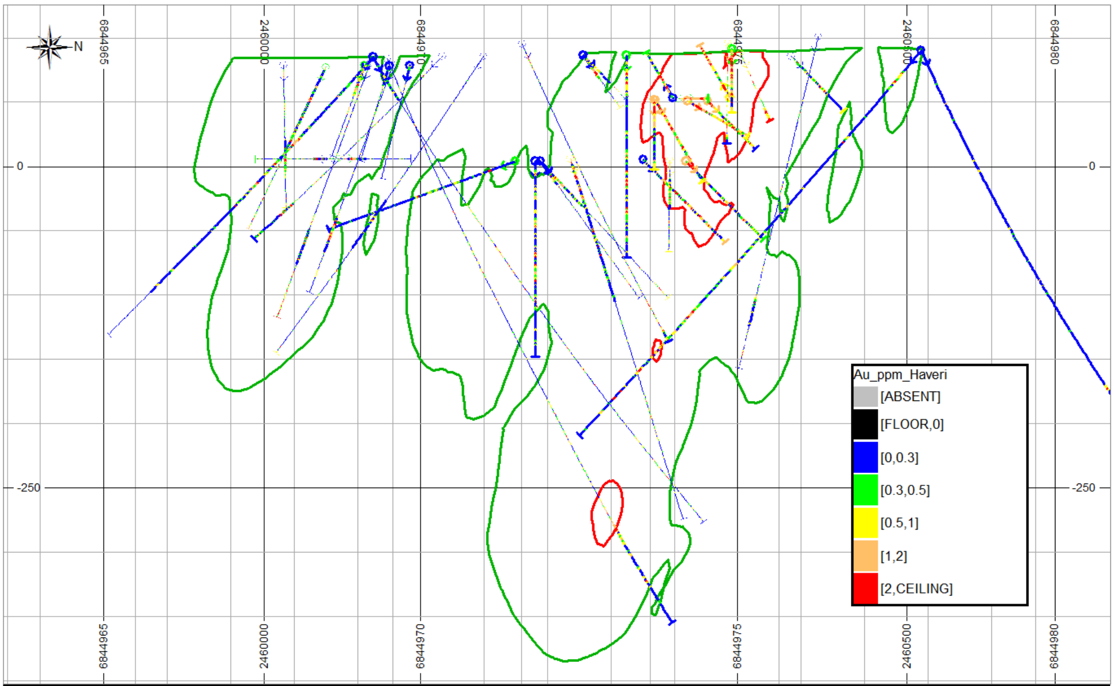
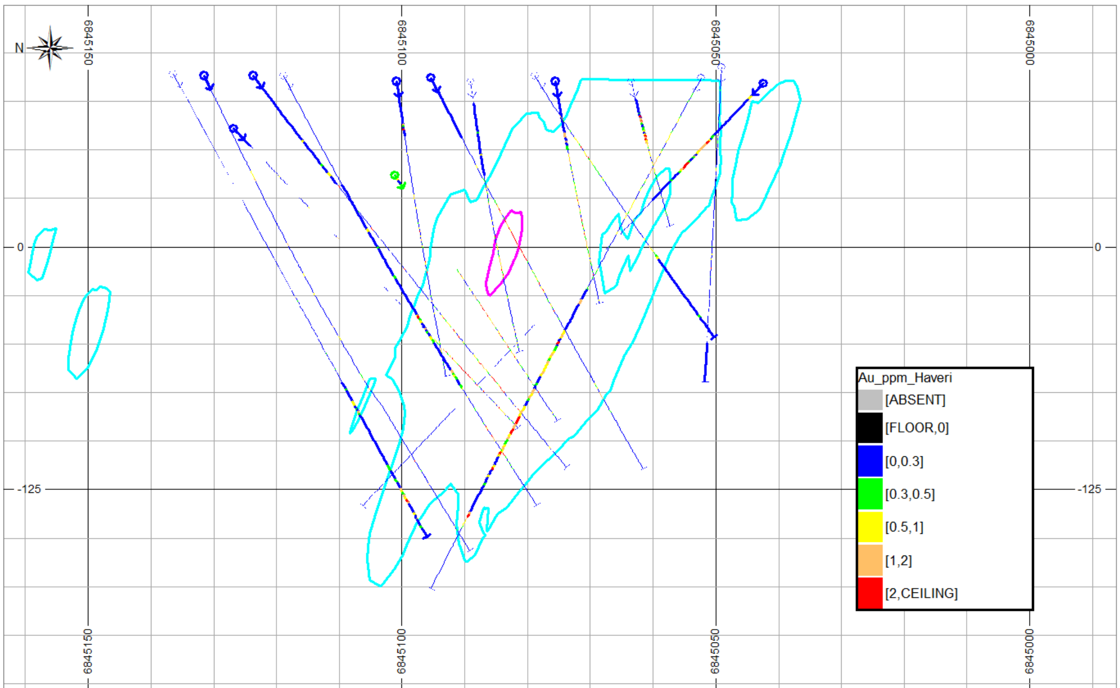


Figure 14-3: Mineralisation wireframes. Green and Blue = low-grade; Red and Pink = high-grade



Cross-Section	Haveri Au-Cu Project, Finland	Dec 2013
srk consulting	Mineralisation wireframes and drillholes coloured by Au (ppm)	Scale 1:3000

Figure 14-4: Cross-section through the central Haveri area, showing mineralisation wireframes (green = low-grade; red = high-grade) and drillholes coloured by Au (ppm) grades.



Cross-Section	Haveri Au-Cu Project, Finland	Dec 2013
srk consulting	Mineralisation wireframes (Peltosaari) and drillholes coloured by Au (ppm)	Scale 1:2000

Figure 14-5: Cross-section through the Peltossari area, showing mineralisation wireframes (blue = low-grade; pink = high-grade) and drillholes coloured by Au (ppm) grades.

14.3.4 Domain Coding

Two separate codes were created for the block model: a zone code relating to the mineralisation domains, and a zone code relating to the lithological domains. These are summarised in Table 14-2.

Table 14-2: Zone and lithzone codes created for Haveri Project

Description	Area	Code
High-grade mineralisation	Haveri	Zone 101
High-grade mineralisation	Peltosaari	Zone 201
Low-grade mineralisation	Haveri	Zone 102
Low-grade mineralisation	Peltosaari	Zone 202
Overburden	Both areas	Lithzone 10
Basalt	Both areas	Lithzone 100
Metasediment	Both areas	Lithzone 200
Porphyry	Both areas	Lithzone 300
Unspecified	Both areas	Lithzone 999

14.4 Statistical Analysis of Raw Assay Data

Table 14-3 shows the results of a classical statistical analysis of the raw assay data, within each of the modelled domains.

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 1 indicates the presence of erratic high values that may have a significant impact on the final grade estimate. As can be seen many of the CoVs are high and so compositing and grade capping were undertaken to lower these values.

Where grades were absent in the original sample files, absent values were replaced with detection limit values of 0.005% Cu, and 0.005 ppm Au.

Table 14-3: Length weighted statistics by domain (Zone)

Variable	Unit	Zone	No. Samples	Min	Max	Mean	Standard Deviation	CoV
Au	ppm	101	3358	0.005	760	2.43	17.04	7.00
		102	15638	0.005	176	0.76	3.31	4.37
		201	846	0.005	34.8	2.14	3.54	1.65
		202	3188	0.001	100.5	0.65	2.57	3.92
Cu	%	101	3358	0.0002	6	0.12	0.28	2.43
		102	15638	0.0002	14	0.07	0.20	2.95
		201	846	0.0007	0.8053	0.08	0.11	1.25
		202	3188	0.0001	2.0108	0.07	0.14	1.95
Ag	ppm	101	3358	0.001	238	1.86	5.83	3.14
		102	15638	0.001	52.628	0.94	1.90	2.02
		201	846	0.001	24	1.37	1.72	1.26
		202	3188	0.001	56.333	1.05	2.41	2.30
As	ppm	101	3358	0.05	17.774	1.57	2.19	1.39
		102	15638	0.0035	4190	7.65	81.39	10.63
		201	846	0.05	1170	9.68	43.46	4.49
		202	3188	0.05	990	7.26	21.42	2.95
S	%	101	3358	0.005	20.7	1.01	1.81	1.79
		102	15638	0.003	63.6	0.91	1.44	1.58
		201	846	0.005	23.4	1.81	3.85	2.12
		202	3188	0.005	19.4	0.68	1.66	2.42
Fe	%	101	3358	0.005	32.5	4.01	5.05	1.26
		102	15638	0.005	41.4	3.65	4.50	1.23
		201	846	0.005	48.8	9.65	10.80	1.12
		202	3188	0.005	51.5	6.13	7.14	1.16

14.5 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 support, and should therefore be of equal length.

Compositing was conducted down-hole, with the composite process being controlled by the wireframe surfaces relating to the geological domains. Due to the inherent short-scale variability of the Au grades, and limited width of the mineralised zones, a short composite length would be advisable. A large number of samples are between 1 – 1.5 m in length, therefore a 2 m composite length was chosen. Following a statistical investigation, all samples <1 m in length (which equated to 3% of the samples, averaging 0.8 ppm Au) were discarded from the final estimation drillhole file.

14.6 Grade Capping and Declustered Statistics

After compositing, the CoV of all domains was still considered high, and so grade capping was investigated. Logarithmic probability plots for Au were created per domain, and for all zones combined, in order to identify natural breaks in the grade populations. It was decided that Au should be capped at 20 ppm. Declustered statistics were run per domain following the grade capping, and are shown in Table 14-4. Although the CoV values are all elevated, they have been reduced significantly and the impact on the resultant grade interpolation has been lowered. The resulting logarithmic histograms for Au (ppm) and Cu% are shown in Figure 14-6 and Figure 14-7, respectively. As can be seen, the Au populations are near log-normal for each domain. The Cu populations show almost perfect log-normal populations, with the exception of the large number of detection limit values at 0.005% Cu. This heavily-skews the population and is an indication that there are fewer Cu assays than Au within the mineralisation wireframe.

Table 14-4: Composite statistics for Haveri by domain

Variable	Unit	Zone	No. Samples	Min	Max	Mean	Standard Deviation	CoV
Au	ppm	101	1754	0.005	20.00	1.91	2.42	1.27
		102	7141	0.005	20.00	0.68	1.38	2.03
		201	435	0.061	20.00	2.12	2.76	1.30
		202	1573	0.005	20.00	0.55	1.06	1.92
Cu	%	101	1754	0.001	4.75	0.12	0.26	2.28
		102	7141	0.001	7.40	0.07	0.16	2.44
		201	435	0.001	0.59	0.08	0.09	1.08
		202	1573	0.001	1.17	0.07	0.12	1.69
Ag	ppm	101	1754	0.001	50.00	2.02	5.20	2.57
		102	7141	0.001	27.44	0.94	1.53	1.63
		201	435	0.001	12.32	1.33	1.58	1.19
		202	1573	0.001	37.28	1.12	2.12	1.90
As	ppm	101	1754	0.05	13.00	1.23	1.91	1.55
		102	7141	0.034	400.00	9.62	87.36	9.08
		201	435	0.05	383.65	10.45	26.70	2.56
		202	1573	0.05	400	7.49	18.24	2.44
S	%	101	1754	0.005	16.08	1.14	1.73	1.52
		102	7141	0.00375	34.29	0.87	1.20	1.38
		201	435	0.005	22.03	2.05	4.12	2.01
		202	1573	0.005	19.40	0.65	1.50	2.29
Fe	%	101	1754	0.005	24.40	3.10	4.39	1.42
		102	7141	0.005	35.13	3.82	4.28	1.12
		201	435	0.005	40.50	9.42	9.53	1.01
		202	1573	0.005	37.72	6.01	6.98	1.16

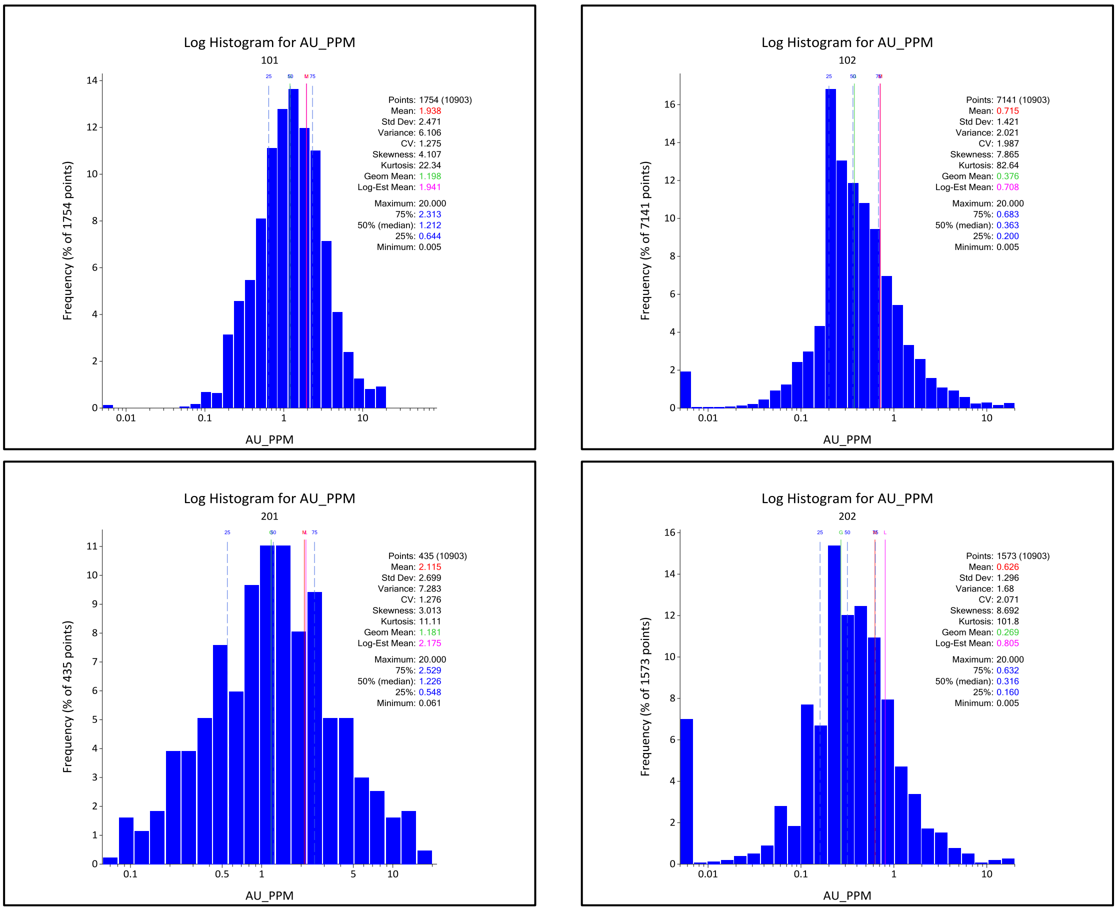


Figure 14-6: Au (ppm) Log-histograms of composited drillholes per domain.

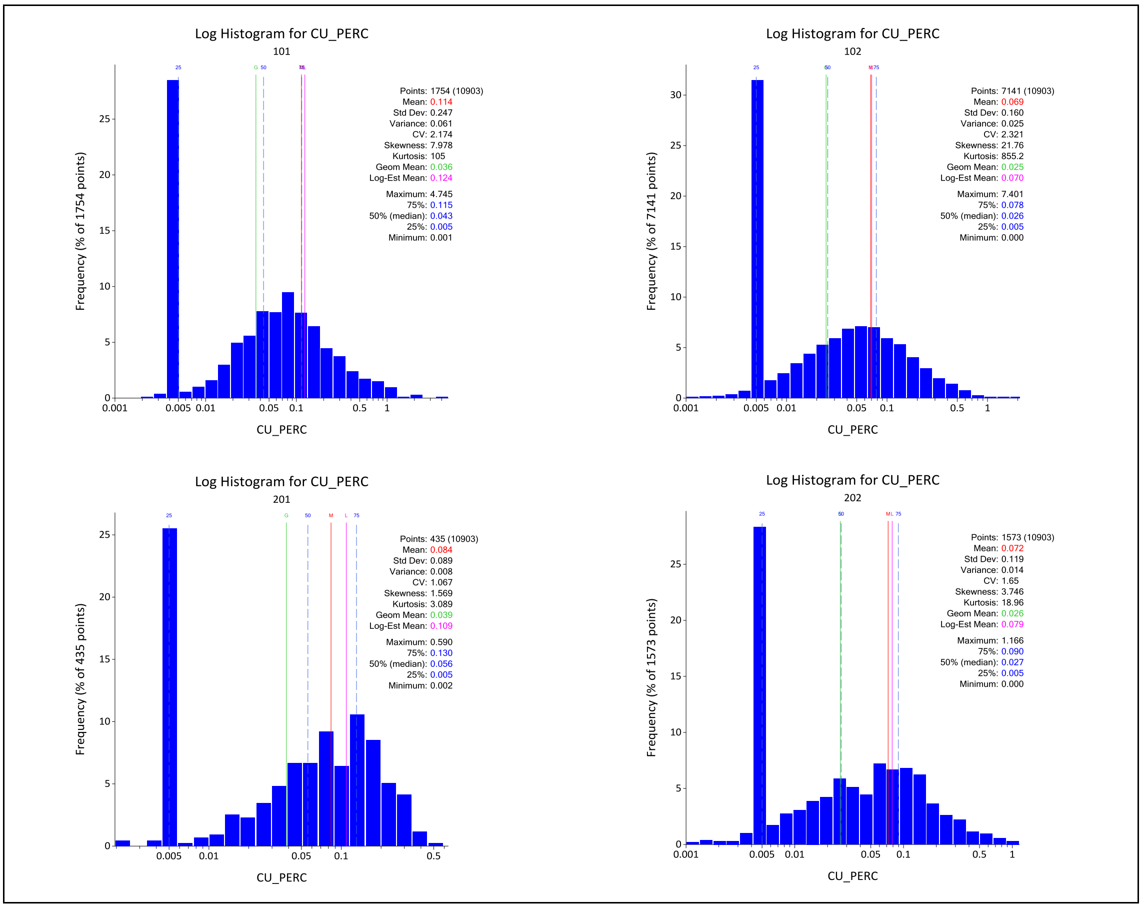


Figure 14-7: Cu% Log-histograms of composited drillholes per domain.

14.7 Density Analysis

A density value of 3.04 cm^3/t was assigned to every block within the estimation domains. SRK checked the average density per domain, which showed limited variation. The histogram of density values is shown in Figure 14-8. Without the erroneous 11.778 value, the average decreases to 3.03 cm^3/t, which is not considered a material difference.

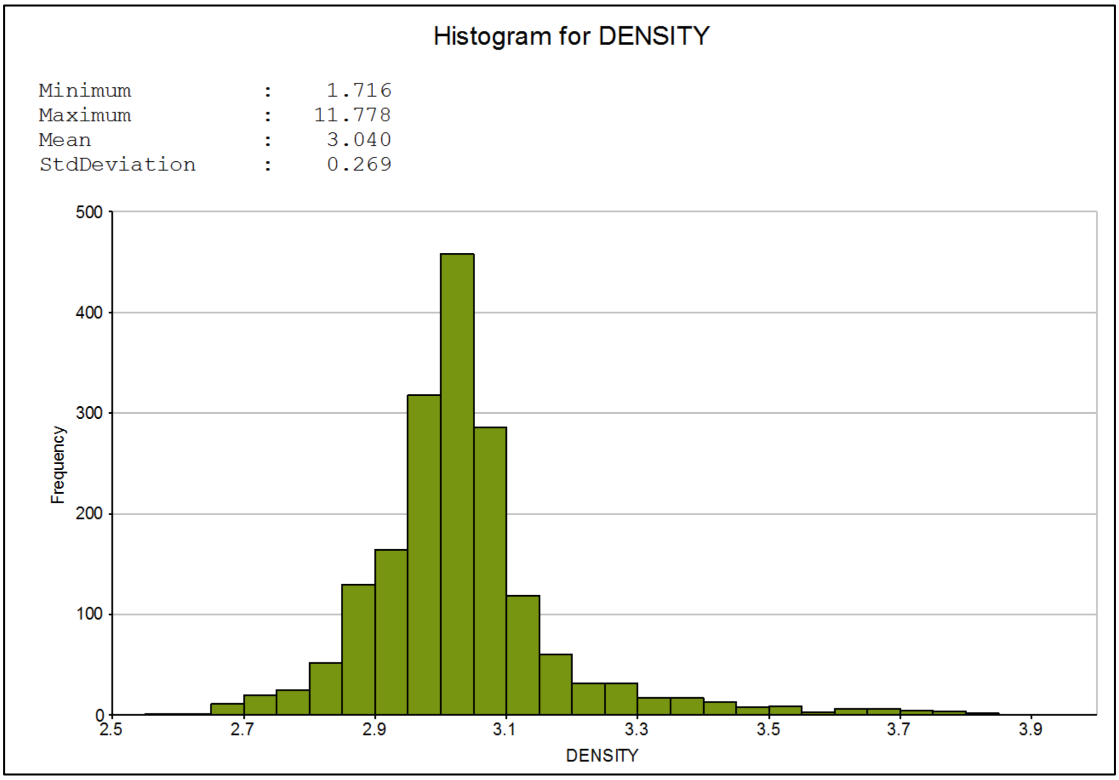


Figure 14-8: Histogram of density values

14.8 Variography

In order to ascertain the spatial correlation of the Au assays, a geostatistical analysis was undertaken. The optimal variogram directions were chosen using variogram maps, and the corresponding variograms modelled in the along-strike, down-dip and across-strike directions. Due to the logarithmic nature of the Au mineralisation, log-normal variograms were constructed and modelled.

A down-hole short-scale variogram was created per domain in order to ascertain the short-scale structures in the data, including the nugget effect. Subsequently, a larger-scale variogram was created per domain in order to ascertain larger-scale structures and the total range of the data. The variograms created are shown in Figure 14-9, with the parameters generated shown in Table 14-5 and Table 14-6, for zones 101 (Haveri high-grade) and 102 (Haveri low-grade), respectively. The Peltosaari zones do not contain adequate samples to define separate variograms, and so it was assumed that the spatial variation here would be similar to that at Haveri.

The variograms showed nugget effects of 45 – 60%, along with short-scale structures at between 5 - 15 m, and longer-scale structures at between 35 – 250 m. The majority of the variance is seen within 20 – 30 m for the zone 101 variograms, and within 100 m for the zone 102 variograms.

SRK has used a 2/3 total modelled range to derive potential search ellipsoid radii. The zone 102 variograms were less robust than zone 101, therefore the search radii for zone 102 is based on half the total variogram range. Numerous other factors need to be considered in deriving an optimum search ellipsoid, and this is discussed further in Section 14.10.

Due to the lower importance of the additional elements estimates (Cu, Ag, As, S, Fe), variograms were not attempted for these variable and the Au variograms were used to assist in developing interpolation parameters for these.

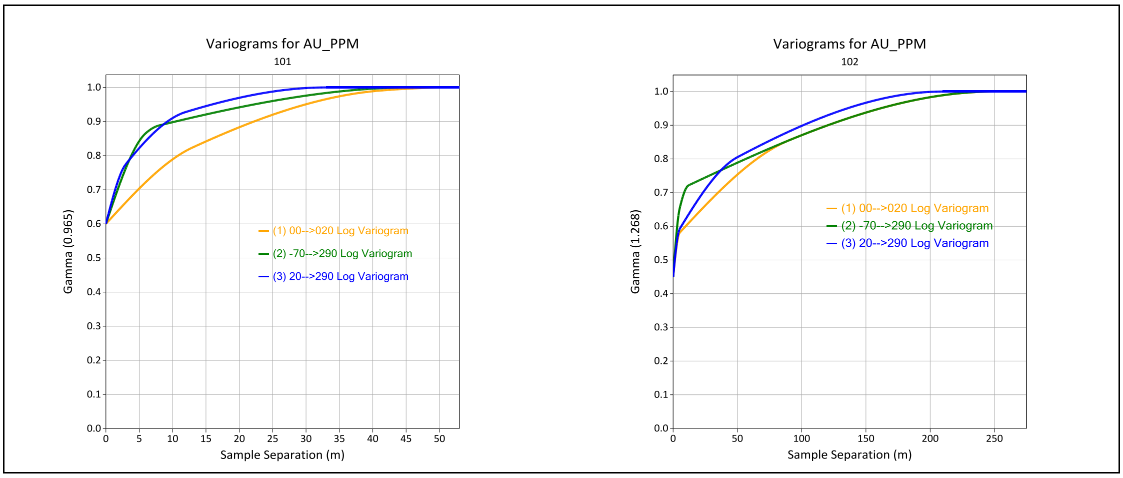


Figure 14-9: Modelled (logarithmic) Variograms by domain (Source: SRK 2014).

Table 14-5: Variogram parameters Zone 101 (Haveri high-grade)

	Along Strike	Down Dip	Down-hole
Nugget Variance (Co)	0.6		
Nugget Effect (%)	**60%**		
1st Range (A1)	15	5	5
1st Sill (C1)	0.1		
2nd Range (A2)	45	10	12
2nd Sill (C3)	0.15		
3rd Range (A2)	50	50	35
3rd Sill (C3)	0.15		
Total Sill (Co + C)		1 (normalised)	
Search Ellipsoid Radii	**35**	**35**	**25**

Table 14-6: Variogram parameters Zone 102 (Haveri low-grade)

	Along Strike	Down Dip	Down-hole
Nugget Variance (Co)	0.45		
Nugget Effect (%)	**45%**		
1st Range (A1)	5	5	5
1st Sill (C1)	0.1		
2nd Range (A2)	90	12	50
2nd Sill (C3)	0.15		
3rd Range (A2)	250	250	200
3rd Sill (C3)	0.15		
Total Sill (Co + C)		1 (normalised)	
Search Ellipsoid Radii*	**125**	**125**	**105**

*Note: zone 102 variograms less robust. Therefore search ellipse radii reduced from 2/3 to ½ total range.

14.9 Block Model

14.9.1 Block Model Framework

An empty block model was generated with block dimensions as shown in Table 14-7, and coded using the grade shell wireframes. These block dimensions approximate half the drillhole spacing at Haveri, however, the drill spacing is highly varied due to the dense drilling in the previously operating mine area. A block height of 10 m was chosen, being an estimated working bench height of the operating pit. Table 14-7 summarises the block model parameters.

Table 14-7: Block Model Framework

Coordinate	Origin	Block Size (m)	Number of Blocks
X	2459600	20	85
Y	6844400	10	120
Z	-400	10	55

14.9.2 Grade Interpolation

Grades for Au, Cu, Ag, As, S and Fe for all four mineralisation domains were interpolated into the block model using OK. Hard boundaries were used between the low- and high-grade domains in order to restrict the influence of high-grade areas.

14.10 Quantitative Kriging Neighbourhood Analysis ("QKNA") and Search Ellipsoid Optimisation

14.10.1 Search Ellipsoid Parameters

The strike of the mineralisation at Haveri is 020º, dipping steeply (70º) towards the west-northwest. Figure 14-10 shows the search ellipse generated for the Haveri deposit, with the dip and strike of the ellipsoid corresponding with the dip and strike of the mineralisation wireframes, and the search radii determined from the variography.

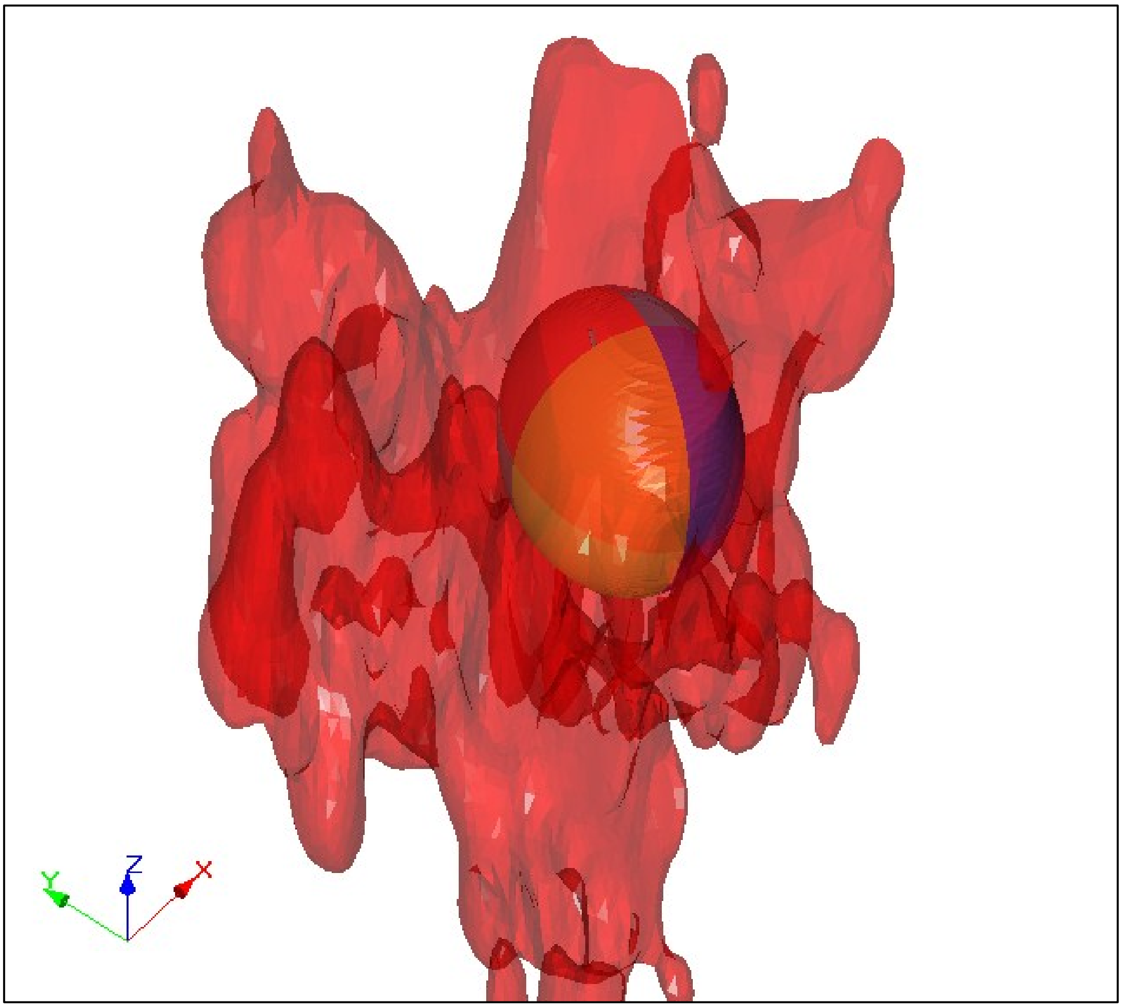


Figure 14-10: First pass search ellipse for Haveri high grade domain displayed with wireframe for Haveri high grade domain used for interpolation (Source: SRK, 2014).

Grades were interpolated in three separate runs. The first pass used the optimum parameters determined by the QKNA testing. The second run doubled the dimensions of the search ellipsoid, and the third run multiplied the original search ellipsoid by a factor of ten. The third run was designed to interpolate grades into any blocks not estimated in runs one and two. SRK notes that the confidence in the resulting grades is lower, as the search ellipsoid will have incorporated samples that are significantly outside the variogram range.

14.10.2 QKNA Introduction

To optimise the search parameters used in the interpolation, SRK has used a process of QKNA. QKNA, as presented by Vann et al (2003) and described below, is used to refine the search parameters in the interpolation process to help ensure 'conditional unbiasedness' in the resulting estimates. 'Conditional unbiasedness' is defined as 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:

- 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.

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. 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. 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%.

14.10.3 Haveri QKNA

The preliminary search ellipsoid radii were based on two thirds of the average Au variogram range for high-grade domains, and half the range for low-grade domains. SRK utilised a single search ellipsoid direction of 70/020º towards the west-northwest for all domains. In all QKNA runs, the negative kriging weights were noted, but did not exceed 0.5% of the total number of estimates, and so this was not considered in the QKNA analysis.

In total, 30 different QKNA runs were analysed for the Haveri zones separately. The parameters changed were:

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

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

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

The optimal parameters produced from the QKNA study, representing the final estimation parameters, are shown in Table 14-8.

Table 14-8: Chosen estimation parameters

Zone	Azimuth (°)	Dip (°)	Search Volume	Along Strike Radii	Down Dip Radii	Across Strike Radii	Minimum Samples	Maximum Samples	Max Samples per DH
			1	35	35	25	6	60	5
101/201	020	70	2	70	70	50	6	60	5
			3	350	350	250	6	60	5
			1	125	125	100	10	60	9
102/202	020	70	2	250	250	200	10	60	9
			3	1250	1250	1000	10	60	9

14.11 Block Model Validation

14.11.1 Introduction

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; and
- Validation plots.

14.11.2 Visual Validation

The block model was inspected by SRK on sections to compare the composite grades to the block model grades. The model and composite Au grades show moderate to good correlation, with example sections shown in Figure 14-11 and Figure 14-12.

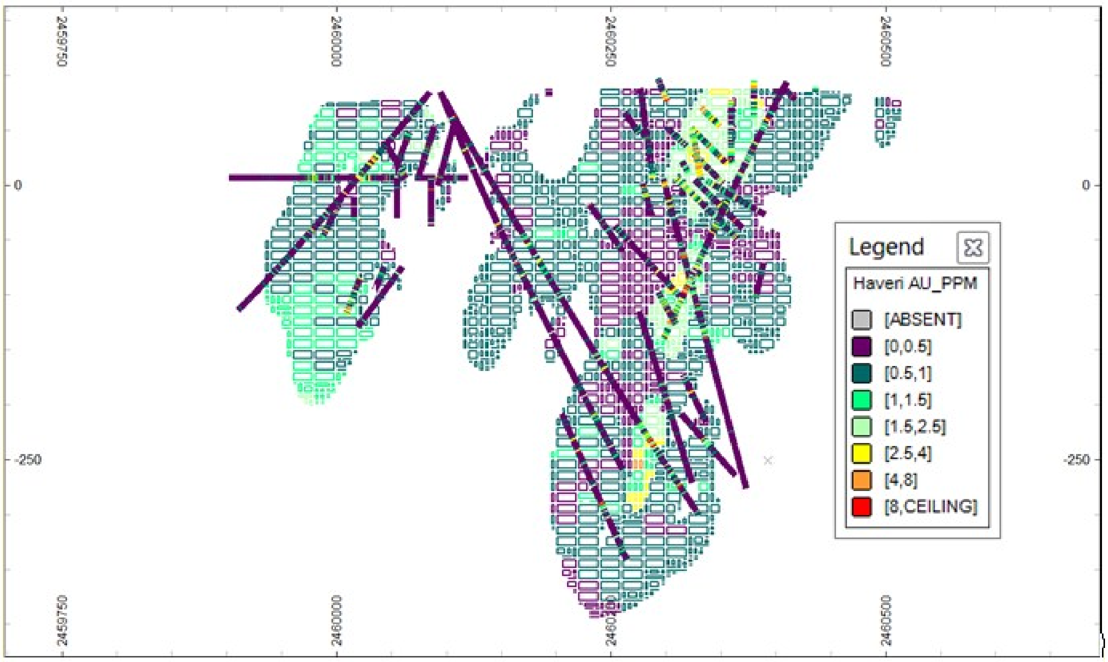
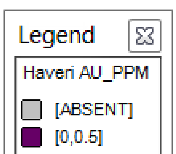


Figure 14-11: Visual validation plot on profile Y= 6845000 viewing North.



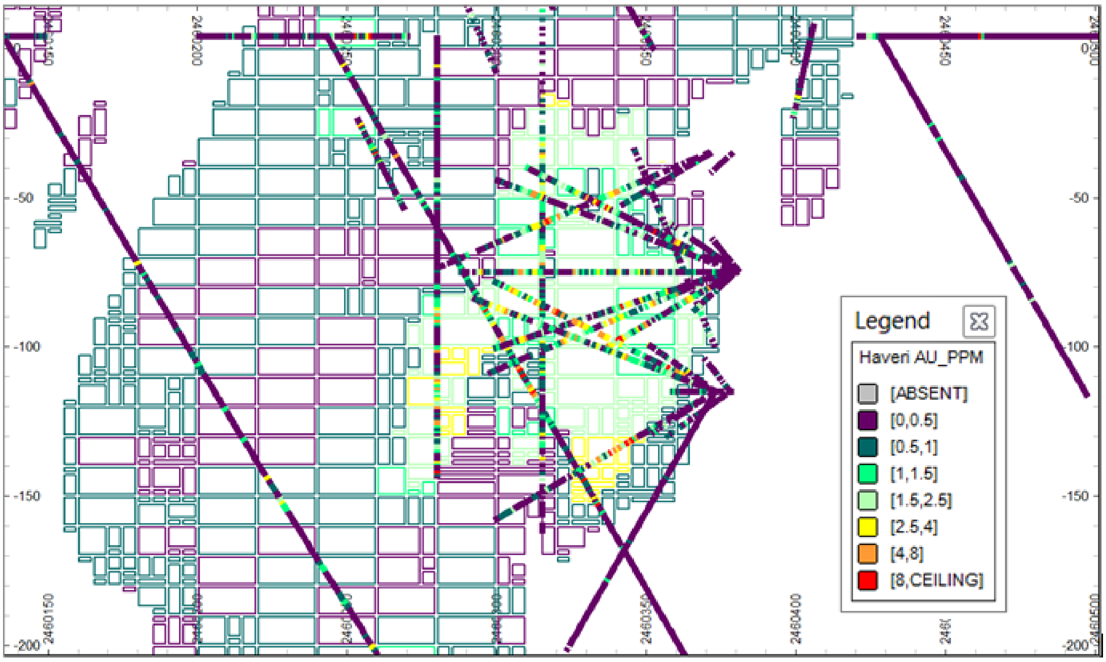


Figure 14-12: Visual validation plot on profile Y = 6845075 viewing North.

14.11.3 Statistical Validation

SRK has declustered the estimation composites in order to compare the estimation input data to the resulting block model. The results are plotted in Figure 14-13 and Figure 14-14. It can be seen that the data show little sensitivity to varying declustering cell sizes of approximately 10%. The blue line (Highest mean/Lowest mean) in the decluster plot depicts the average model grade for each of the zones analysed.

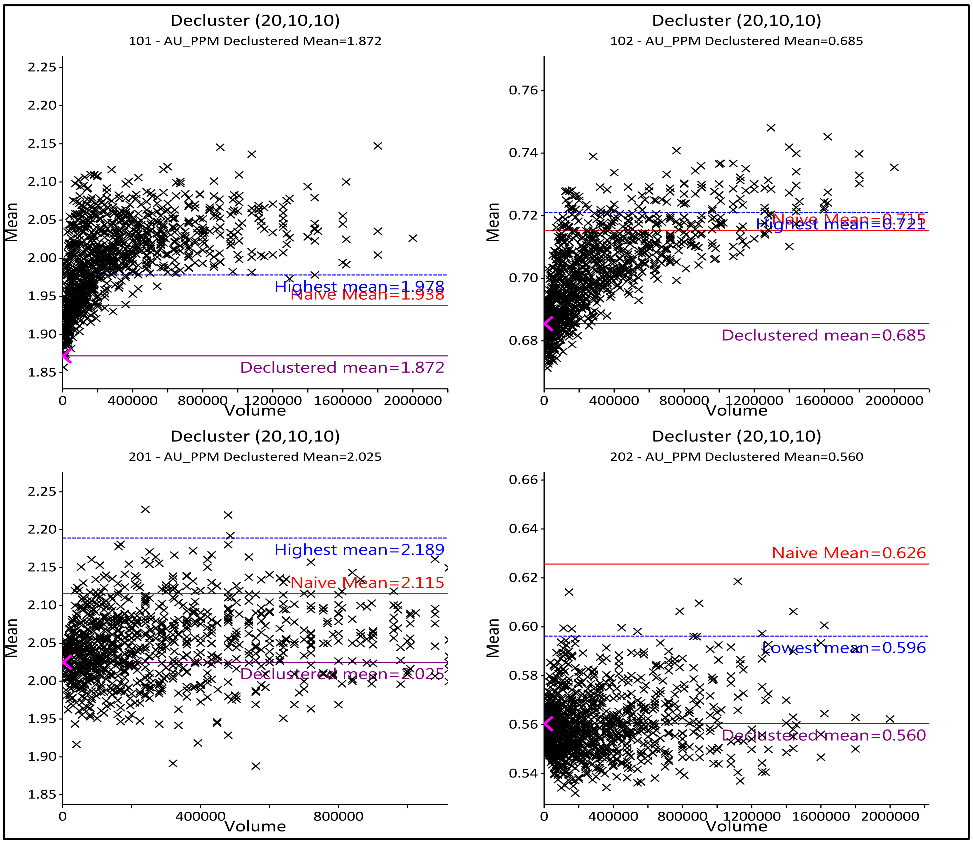


Figure 14-13: Declustered Au grades of estimation composites for all four mineralisation domains

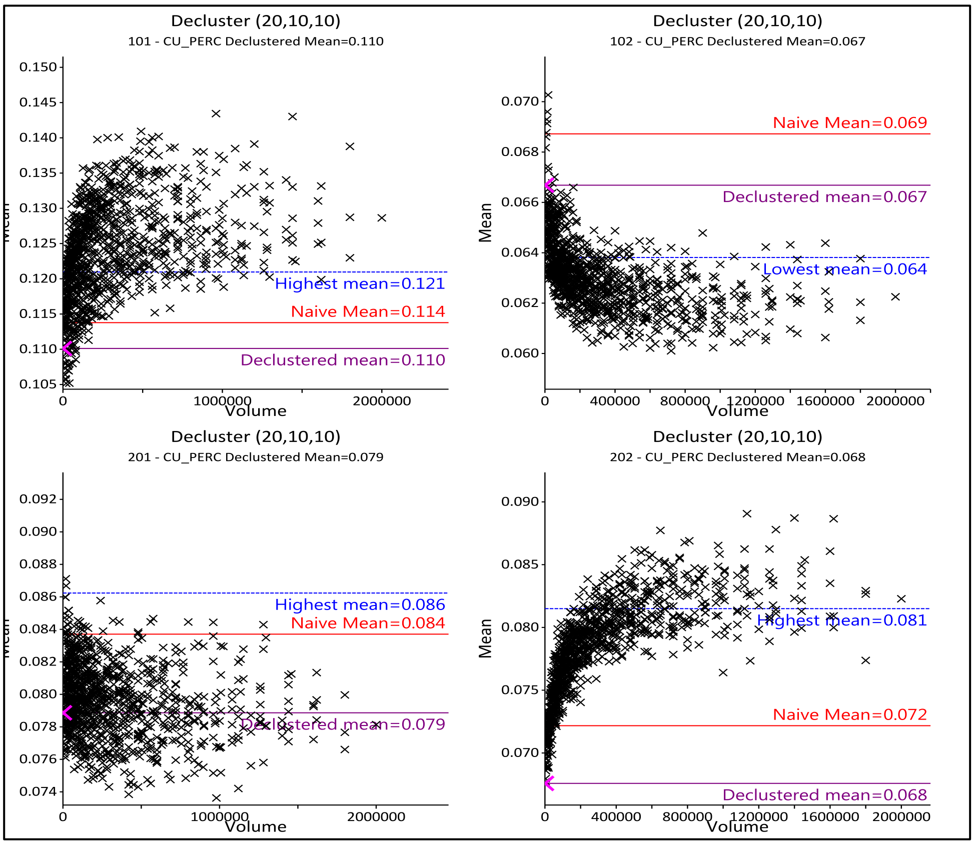


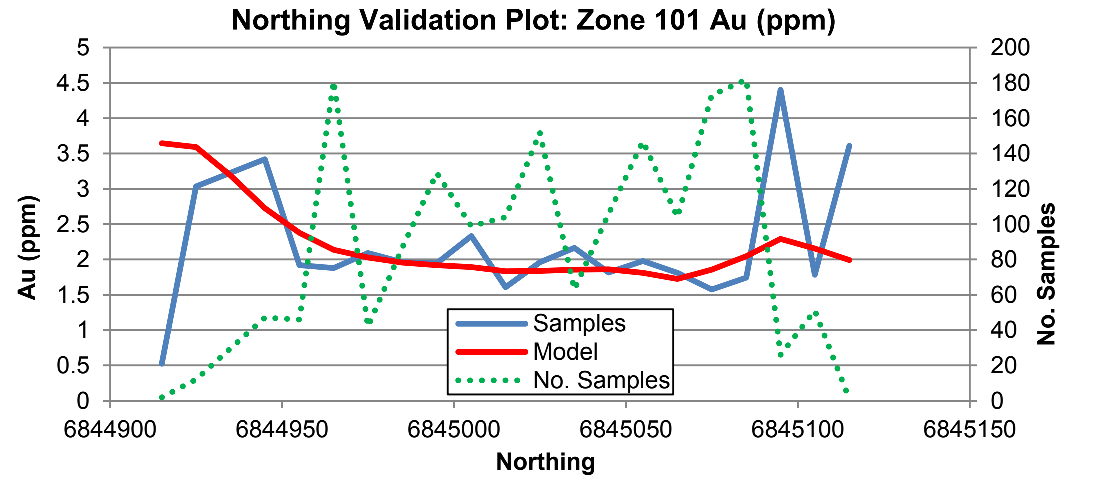
Figure 14-14: Declustered Cu grades of estimation composites for all four mineralisation domains (blue line depicts model grade)

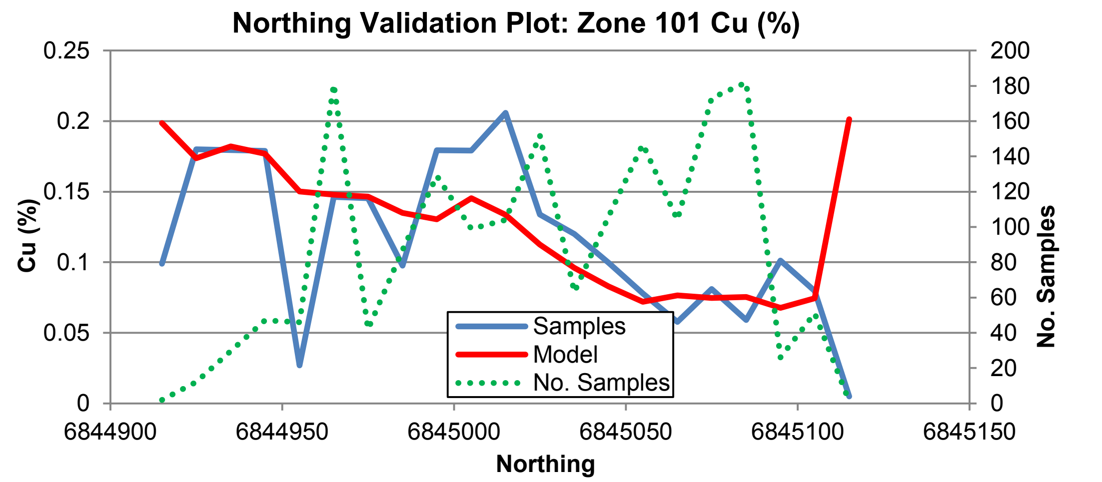
14.11.4 Validation Plots

As part of the validation process, the block model and input samples that fall within defined sectional or elevation criteria were compared and the results displayed graphically to check for visual discrepancies between grades.

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

Figure 14-15 shows the validation slices through the deposit. They show generally moderate to good correlation to the sample data, with a smoothing effect on the large outliers.


Northing Validation Plot: Zone 101 Au (ppm)
Au (ppm)
No. Samples
Northing
Samples
Model
No. Samples
0
0.5
1
1.5
2
2.5
3
3.5
4
4.5
5
0
20
40
60
80
100
120
140
160
180
200
6844900
6844950
6845000
6845050
6845100
6845150


Northing Validation Plot: Zone 101 Cu (%)
Cu (%)
No. Samples
Northing
Samples
Model
No. Samples
0
0.05
0.1
0.15
0.2
0.25
0
20
40
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100
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160
180
200
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6845000
6845050
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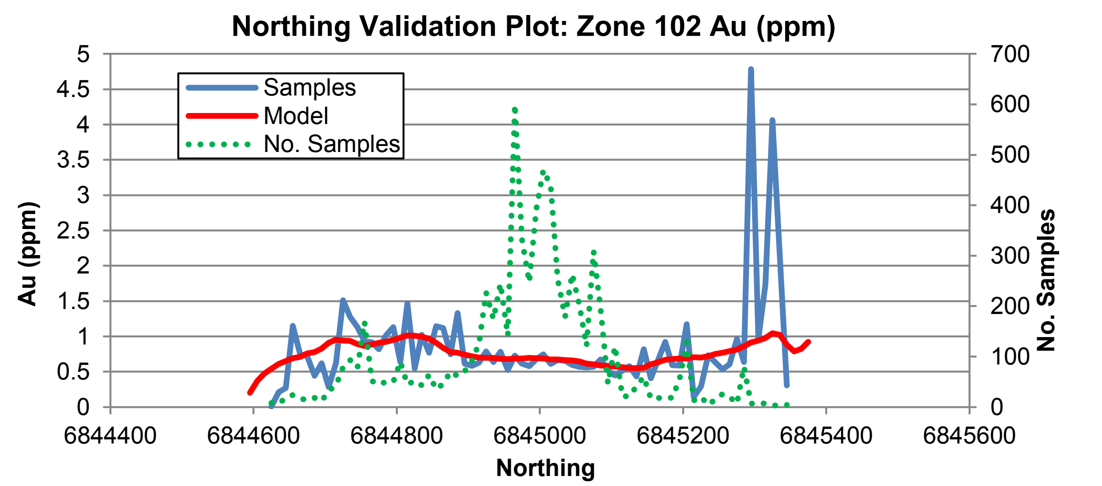
6845150


Northing Validation Plot: Zone 102 Au (ppm)
Au (ppm)
No. Samples
Northing
Samples
Model
No. Samples
0
0.5
1
1.5
2
2.5
3
3.5
4
4.5
5
0
100
200
300
400
500
600
700
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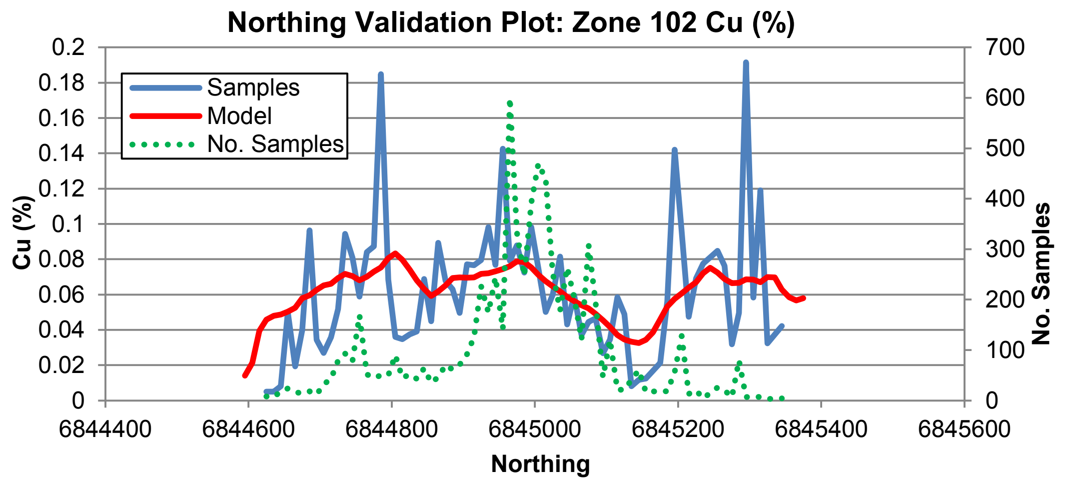
6845200
6845400
6845600


Northing Validation Plot: Zone 102 Cu (%)
Samples
Model
No. Samples
Cu (%)
No. Samples
Northing
0.2
0.18
0.16
0.14
0.12
0.1
0.08
0.06
0.04
0.02
0
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600
500
400
300
200
100
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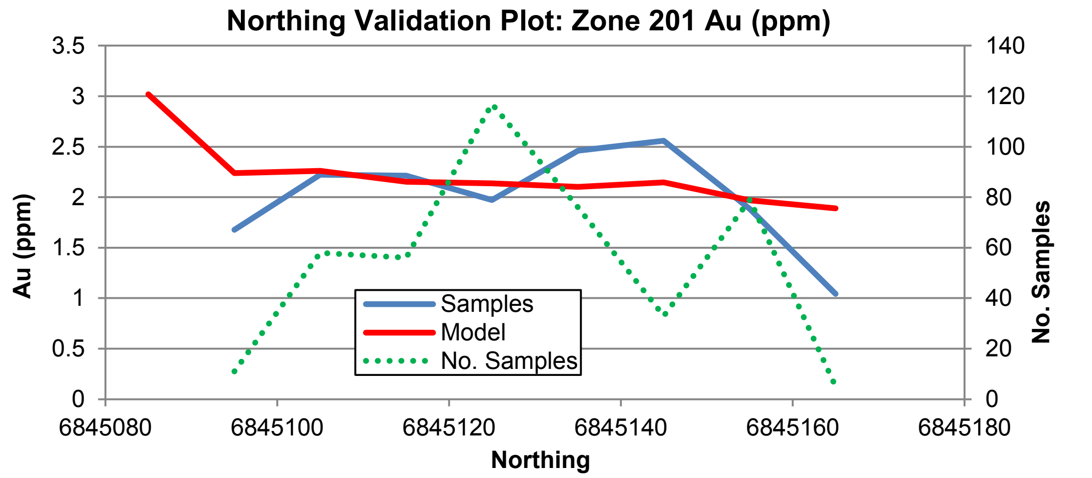
6845600


Northing Validation Plot: Zone 201 Au (ppm)
Samples
Model
No. Samples
Au (ppm)
No. Samples
Northing
3.5
3
2.5
2
1.5
1
0.5
0
140
120
100
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60
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20
0
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6845120
6845140
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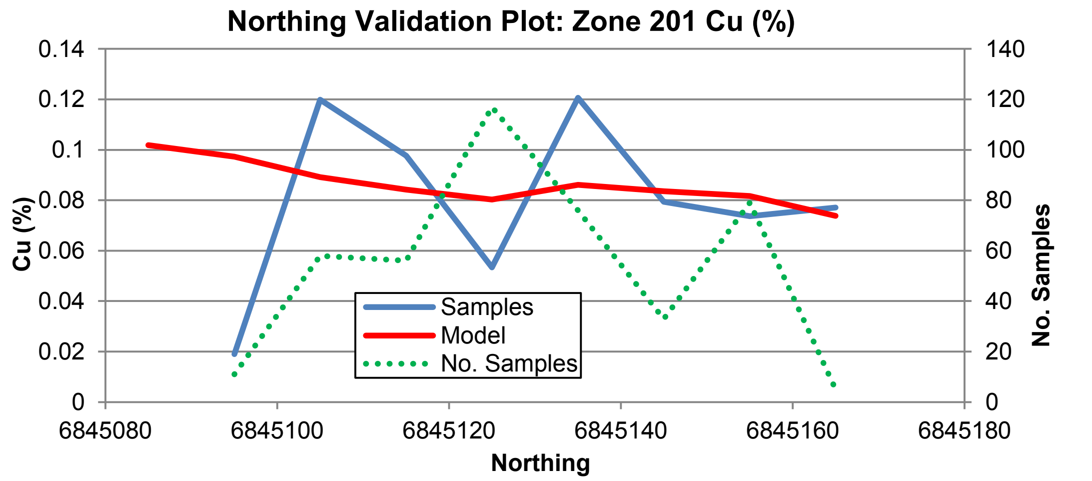
Northing Validation Plot: Zone 201 Cu (%)
Samples
Model
No. Samples
Cu (%)
No. Samples
Northing
0.14
0.12
0.1
0.08
0.06
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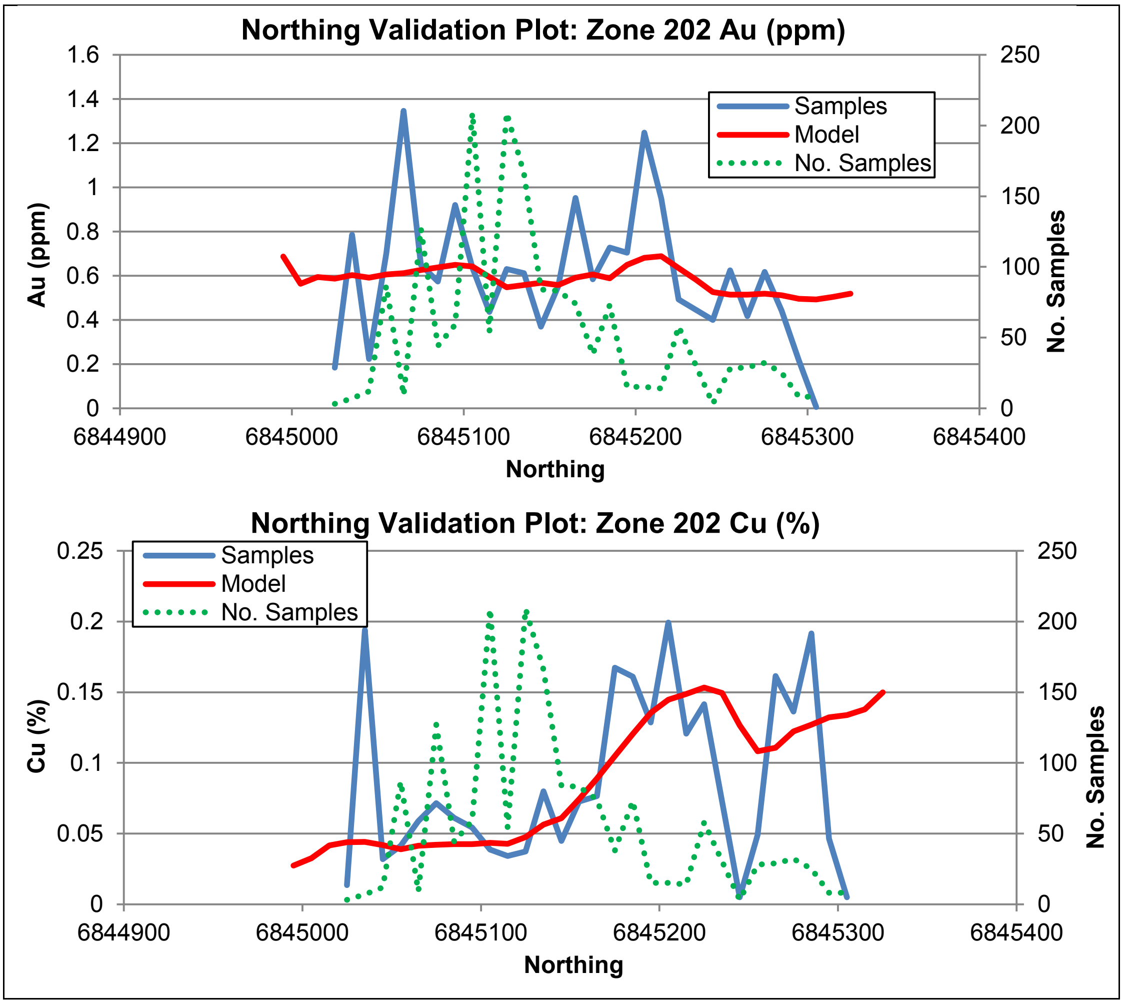

Figure 14-15: Validation Plots for Au and Cu grade in Haveri and Peltosari deposits

SRK is confident that the block model grades are a reasonable reflection of the composite sample grades.

14.12 Mineral Resource Classification

14.12.1 CIM Definitions

The definitions given in the following section are taken from the 2010 Canadian Institute of Mining Standing Committee on Reserve Definitions' Definition Standards on Mineral Resources and Reserves (CIM definition standards).

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

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

Inferred Mineral Resource

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

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

Indicated Mineral Resource

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

Mineralisation may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralisation. The Qualified Person must recognise 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 Haveri Classification

Introduction

To classify the Haveri block model, the following key indicators were assessed:

- Geological complexity;
- Quality of data used in the estimation;
- QAQC, density analysis;
- Results of the geostatistical (variography) analysis;
- QKNA results; and
- Quality of the estimated block model.

Geological Complexity

Due to the extensive, close spaced drilling, SRK considers that the geological continuity between sections is well understood and that the current geological interpretation is well supported. Detailed reports on the structural geological setting also provide a robust understanding of the mineralisation.

SRK considers that the associated risk relating to geological complexity is comparatively low.

Quality of the Data used in the Estimation

The historic drilling programmes by Vuokseniska Oy, Outokumpu Finn Mines Oy and Glenmore Highland Inc. are not supported by QAQC. The more recent sampling campaigns by Northern Lion Gold Corp are well-supported by QAQC samples, however the results of these are considered poor. Check sampling of Northern Lion assays by the author of the previous MRE provided some verification for the recent drilling. SRK carried out a campaign of re-assays on the Northern Lion coarse reject material and found that the re-assays reported within the same order of magnitude as the originals, however, overall the original assays appear to have a bias and over report the actual grade.

Overall SRK is not confident that the results of the QAQC analysis have validated the accuracy of modern drill sampling being used to generate the MRE, and sees a downgrade from previously applied Mineral Resource categorisation as appropriate. SRK has even lower confidence in the historic data, but due to the numerous samples associated with this historic data these were included in the MRE.

A dataset of density has also been generated by the Northern Lion and an average dry bulk density has been calculated. SRK has used that average density of 3.03 g/cm^3 for the estimate which SRK considers to be reasonably well constrained.

Statistical and Geostatistical Analysis

Geostatistical analysis of the composited assay data resulted in robust variogram models being produced for the deposit. This enabled the nugget and short-scale variation in grade to be determined with a high level of confidence. The 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

SRK has validated the models using both visual and statistical methods. SRK is confident that the block models reflect the input data on both local and global scales.

Classification

Given the above, the Haveri block model has been classified entirely as Inferred Mineral Resource. This decision is mainly based on the quality of the underlying database and the limited and poor quality of the QAQC procedures that have been carried out.

14.13 Pit Optimisation for Mineral Resource Estimation

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 pit optimisation exercise.

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

Table 14-9 shows the assumptions applied in the pit optimisation.

Table 14-9: Resource pit optimisation parameters

Geotechnical Parameters		
Overall Slope Angles FW/HW	47	°
Metal Selling Prices		
Copper Price	7850	USD/t
Gold Price	1510	USD/oz
Mining Cost Factors		
Total Open Pit Mining Cost (Base RL)	3.5	USD/t
Base RL for optimisation	80	m
Incremental Mining Cost below BRL	0.05	USD/10m depth
Processing Cost Factors (includes G&A)		
Crushing, Grinding and Flotation	14	USD/ t ore
Cyanidation	1	USD/t ore
Other Cost Factors		
Distance to Process Plant	0.7	km
Transport Cost	0.28	USD/t*km
Royalties	0.15	%
Mining Parameters		
Mining Recovery	97	%
Mining Dilution	5	%
Processing Parameters		
Recovery Cu	57	%
Recovery Au	75.6	%
Concentrate Grade Cu	25	%

The high metal prices used relative to most consensus forecasts of such at the present time is to ensure that the optimised shells encapsulate material with potential to be extracted not just material that could be economically extracted now. The resulting resource shell is displayed in Figure 14-16 together with the estimated block model. All resources are classified as Inferred and only material within the resource shell is reported in the Mineral Resource statement below.

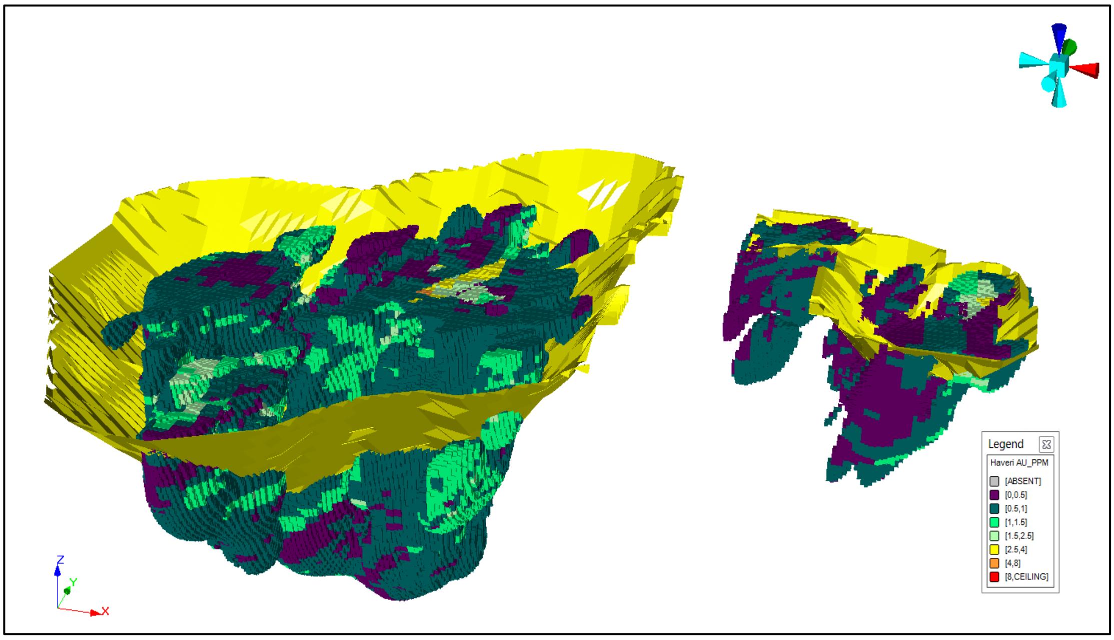

Figure 14-16: Resource shell (view direction towards north; Haveri and Peltosaari resources)

14.14 Gold Equivalent Calculation

Each block is assigned a gold equivalent (AuEq) based on the interpolated Au and Cu grades in each block as well as using long term metal prices and assumed recoveries, as described above. The following calculation was used to assign AuEq values to each block:

$$AuEq\ [g/t] = 0.994456*Au\ [g/t] + 1.288622*Cu\ [\%]$$

14.15 Mineral Resource Statement

The Mineral Resource statement generated by SRK has been restricted to that material falling within the resource pit shell and above a cut-off grade of 0.45 g/t Au Equivalent, representing the calculated marginal cut-off grade for the deposit. A USD 7850 / t copper price, and USD 1510 / Oz Au price, were used for the optimisation, which includes a 30% premium above the consensus long-term price determined from over 30 market forecasts, so as to include material with the potential to be extracted in the future not just that material that justifies extraction now. SRK consider that the material included within the pit shell and above the cut-off grade demonstrates reasonable prospects for eventual economic extraction, as required by CIM definition standards.

Table 14-10 shows the resulting Mineral Resource statement for Haveri. The statement has been classified by Lucy Roberts (MAusIMM(CP)) in accordance with the CIM definition standards. The effective date for this statement is 30 July 2014.

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 14-10: Mineral Resource statement (reported above a marginal cut-off grade of 0.45 g/t Au Equivalent and within the Whittle shell)

Category Inferred	Tonnes (Mt)	Au (g/t)	Cu (%)	AuEq (g/t)	Au (MOz)
Peltosari	4	0.84	0.07	0.93	0.12
Haveri	54	0.84	0.07	0.93	1.45
Total	58	0.84	0.07	0.93	1.56

In total, the Haveri Project has been estimated to contain an Inferred Mineral Resource of 58 Mt with an average grade of 0.84 g/t Au and 0.07 % Cu above marginal cut-off.

SRK is not aware of any environmental, permitting, legal or other external factors which may materially affect this resource estimate.

14.16 Grade Tonnage Curves

An overall grade-tonnage curve for AuEq [g/t] is shown in Figure 14-17. Figure 14-18 and Figure 14-19 show Mineral Resource grade tonnage curves for Peltosaari and Haveri respectively.

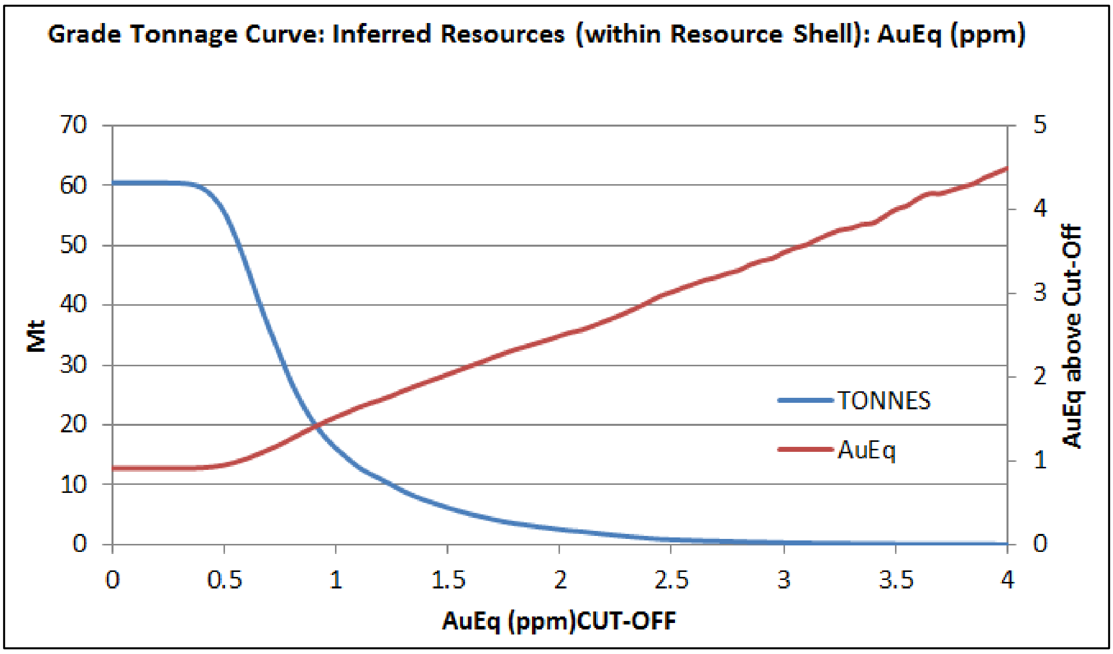

Figure 14-17: Grade Tonnage Curve for AuEq (g/t) – Inferred Resources above Resource pit shell (Source: SRK, 2014).

Table 14-11: Overall Cut-off grade-tonnage results (Inferred)

Cut-Off (AuEq)	Mt	AuEq
0	60.4	0.91
0.25	60.4	0.91
0.45	58.1	0.93
0.50	55.5	0.95
0.75	31.6	1.19
1.00	16.1	1.52
1.25	10.0	1.78
1.50	6.2	2.03
2.00	2.56	2.49
3.00	0.33	3.48
4.00	0.04	4.49

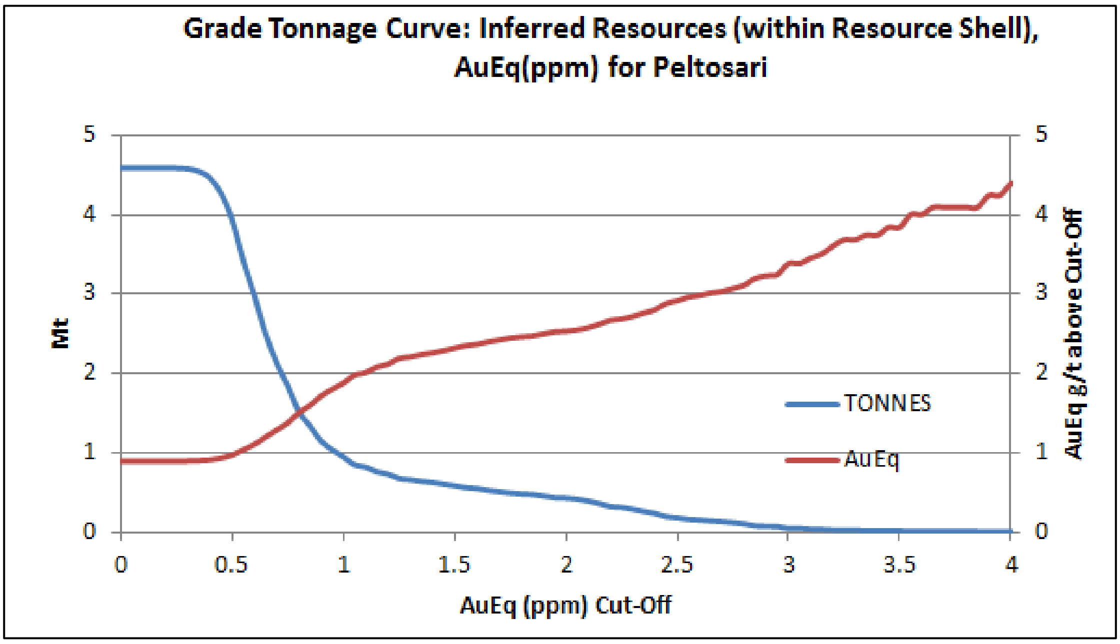

Figure 14-18: Peltosari Grade Tonnage Curve for AuEq (g/t) – Inferred Resources above Resource pit shell (Source: SRK, 2014).

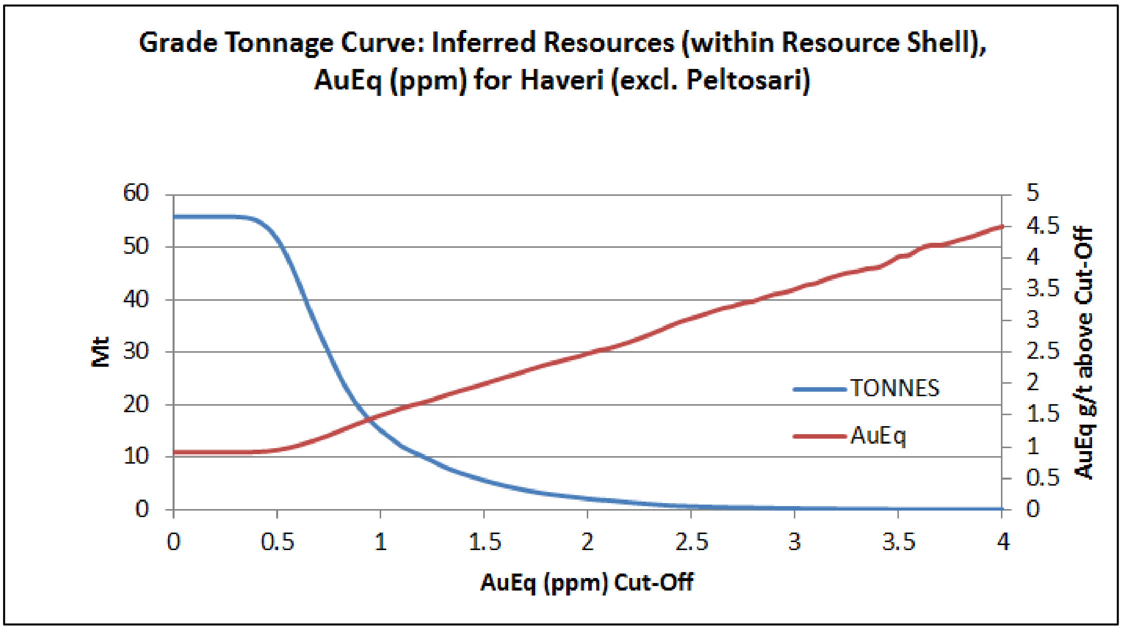

Figure 14-19: Haveri (excl. Peltosari) Grade Tonnage Curve for AuEq (g/t) – Inferred Resources above Resource pit shell (Source: SRK, 2014).

14.17 Comparison to 2008 Maptek MRE

The previous Mineral Resource estimate was prepared by Geoff Reed on behalf of Maptek for previous owners Lappland Goldminers, who is a Qualified Person (QP) as defined by CIM definition standards; the estimate has an effective date of 21 August 2008 and is reproduced in Table 14-12.

Table 14-12: Resource Statement by Maptek 2008 above a cut-off grade of 0.5 g/t Au

Category	Tonnes (Mt)	Au g/t	Au MOz
Measured	17.1	0.92	0.504
Indicated	5.0	0.79	0.127
Meas+Ind	**22.1**	**0.89**	**0.632**
Inferred	-	-	-

The key changes between the 2008 Maptek and 2014 SRK Mineral Resource statements are:

- Change in geological interpretation.
  - SRK considers the geology of the deposit as a relatively continuous zone of stringers and disseminated gold.
  - The previous interpretation constrained the mineralisation to relatively high grade vein structures. These structures are considered to have questionable laterally and down-dip continuity and SRK considers it unreasonable to maintain this interpretation, based on the data at hand.
- Change in classification
  - Based on the data at hand SRK considers it appropriate to classify the entire resource as Inferred. This is mainly due to the poor results from and the lack of QAQC for all the existing data.

- Change in reporting approach
  - SRK has limited the resource to only that material which satisfies the CIM definition for an economically extractable resource by defining a pit shell.

14.18 Exploration Potential

SRK notes that there is potential for improving the resource classification through a programme of re-assaying and a QAQC programme commensurate with industry best practice. This should improve confidence in the data supporting the estimate and if combined with a programme of re-logging and structural interpretation, may allow higher grade zones to be better defined.

Drilling and exploration to date has defined the sulphide bearing mineralised zones reasonably well. There is considered to be exploration potential at depth, chasing plunging lodes, although the perceived lack of continuity of high-grade structures may limit the possibilities of using selective (underground) mining methods.

In addition to this, historic drilling has identified potential areas in Tombstone, Haveri North and Casino Bay. These are all located within hundreds of meters of the historic mine. Karvinen (2003) notes that most of the high-grade intersections encountered in the Glenmore Highlands drilling are in areas of low sulfides and low magnetic response (e.g. hole R8). These areas, however, are the least explored and in many instances have not been tested by drilling.

Occasional high grade intersections suggest strong hydrothermal mineralisation events. These fluids would follow structural patterns of the deposit. It is important to note that very little structural work has been carried out on the deposit to date and SRK sees potential for further discovery in better interpretation of high grade mineralisation in better understanding of the underlying structural geological constraints.

Historic tailings, previously sampled by Outokumpu, should be resampled to a level sufficient to support a Mineral Resource estimate on these. This material could then in principal be incorporated into a life of mine plan and cashflow model, which could generate revenue during early years of operation.

15 MINERAL RESERVE ESTIMATES

SRK has not derived an estimate of Mineral Reserves at this stage as this technical study is limited to a PEA. Further work is required to evaluate the key parameters to a Pre-feasibility/Feasibility level of study and also to improve confidence in the data supporting the Mineral Resources in order to upgrade the category of these.

The principal tasks to be addressed before as part of a Pre-feasibility study are outlined in the recommendations section below.

16 MINING METHODS

16.1 Overview

This section presents the hydrological, geotechnical and mining inputs used to evaluate the Haveri deposit as part of this PEA, and the resulting mining schedules.

SRK has evaluated the potential to mine the deposit using both open pit and underground mining methods, selected an open pit only option for its Base Case analysis and reviewed the available geotechnical and hydrogeological information to determine suitable slope angles and hydraulic radius. Commercial pit optimisation software was then applied to the geological block model to determine the potential optimal pit boundary for economic analysis. SRK has also produced a preliminary production schedule and estimated the mining costs for input to the economic assessment presented later in this report.

Through discussions with the Client and in order to better understand some of the possible production scenarios, SRK undertook a high level analysis of several options as part of the mining study. In summary, these options included:

- Option 1: Peltosari open pit only, trucking and sale of ROM material to an existing third party owned process plant;
- Option 2: Haveri and Peltosari open pits, trucking and sale of ROM material to an existing third party owned process plant;
- Option 3: Haveri and Peltosari open pits with processing at a dedicated on-site process facility;
- Option 4A: Underground mining only with on-site processing; and
- Option 4B: Underground mining only, with trucking and sale of ROM material to an existing third party owned process plant.

Using benchmark analysis and the Infomine proprietary database, SRK undertook a high-level relative comparison of cashflow (CF) and net present value (NPV) for the each option. The results and summarised in Table 16-1 and Figure 16-1 below.

Table 16-1: High-level relative comparison of possible production scenarios

		OPEN PIT			UG	
		Option 1	Option 2	Option3	Option4A	Option4B
ROM GRADE	g/t	1.78	1.76	1.01	1.61	2.07
Run of mine tonnes	Mt	0.27	1.88	19.54	7.60	2.60
Waste	Mt	0.71	5.60	21.26	-	-
OPEX						
Mining costs	USD/t	7.00	8.00	3.50	20.00	30.00
Transport costs	USD/t	13.00	13.00	0.20	0.20	13.00
Processing costs	USD/t	-	-	15.00	15.00	-
G&A	USD/t	0.40	0.40	0.40	0.20	0.10
CAPEX						
Processing Plant		-	-	50.00	40.00	-
Mining equipment and development		-	-	15.00	30.00	20.00
CAPEX TOTAL	MUSD	-	-	65.00	70.00	20.00
Production rate	Mtpa	0.07	0.31	1.78	0.85	0.37
Mine life	Years	4.00	6.00	11.00	9.00	7.00
CF	MUSD	- 5.64	- 35.17	86.97	65.19	-3.38
NPV	MUSD	- 5.25	-24.56	- 8.08	-21.88	-19.86

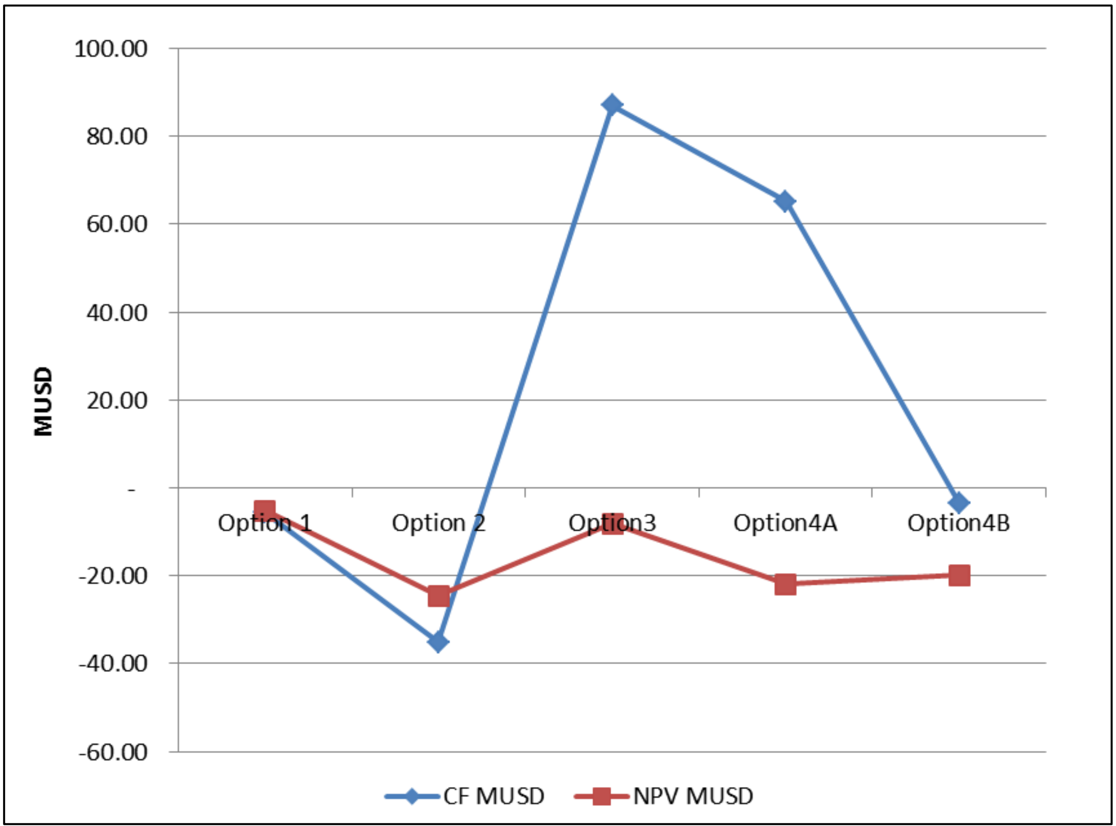

Figure 16-1: High-level relative comparison of possible production scenarios

In the context of the high-level nature of the above comparison, SRK notes that Option 3 appears to present the preferable alternative in terms of forecast cashflow and NPV. SRK also notes that:

1. Production through underground methods will likely be high risk, given the spotty nature of higher grade mineralisation as described in sections above; and
2. The owners of the existing third party processing facility may be reluctant to accept material which would likely be at a lower grade than is currently being processed. The plant would also likely require modifications and investment to the current process route.

Notwithstanding this, discussions with the owners of the third party facility should go-ahead and, if a mutually beneficial arrangement could be reached in principal, that this production scenario could continue to be assessed and refined during further studies. For the purposes of this report, Option 3 is considered as the base case and the other options are not discussed in further detail.

The base case envisages a conventional approach to open pit mining using an excavator-truck configuration. A run of mine production rate of 1.8 Mtpa is assumed. SRK has considered owner-operator for all mining operations.

16.2 Geotechnical Analysis

16.2.1 Introduction

The evaluation is based core photos, wireframes of the relic open pit and underground workings and from engineering judgments based on past experience of similar deposits. The analyses presented here, which are mostly empirical, are appropriate for a PEA level of study. Further data collection and more rigorous analyses will be required to support detailed mine design.

16.2.2 Geotechnical Characteristics

Geotechnical characteristics have been based largely on core photographs of exploration holes. As the data is limited, only one geotechnical zone is considered during this study. SRK considers that there is currently insufficient data to separately characterise the hangingwall waste, footwall waste rock and orebody, although further data capture will allow for this.

Core photos from resource drillholes were reviewed to obtain a global Rock Mass Rating (“RMR”) and Q value, which formed the basis for rock mass characterisation and determination of appropriate inputs into the empirical analyses.

The value of the intact rock strength is estimated from engineering judgement upon core photos and from knowledge gathered from working with similar geology.

The stress regime is thought to have no major adverse effect on the design of the pit walls and underground openings.

Currently, no information exists on the depth of groundwater below surface. For the purposes of this study, SRK has assumed that groundwater will be relatively near surface (due to surrounding water bodies) but will be able to drain naturally from the well jointed rock mass.

A summary of the rock mass characterisation results is presented in Table 16-2. It can be seen from the table that the rock mass comprises a good (to fair) quality rock mass.

Table 16-2: Summary Geotechnical Characteristics

Zone	Av RMR	RMR Range	RMR Class	Av Q'	Q' Range	Q Class
Global	65	50 - 90	Good	5.0	2.1 - 9.0	Fair

Note: The Q classification Q’ (Q prime) has been used here. Q’ is Q value with the inputs for ground water (Gw) and stress reduction factor (SRF) omitted. The Q value may increase or decrease when Gw and SRF are included.

16.2.3 Geotechnical Design Criteria

Open Pit - Slope Angle Estimation

The Laubscher Mining Rock Mass Ratings (MRMR) was used to calculate overall slope angles by referring to the Haines-Tebrugge chart. The RMR values calculated from the core photos are modified to account for the potential effects of mining and exposure to produce an MRMR value. MRMR adjustments were applied as follows:

Weathering – 1.0
Stress – 1.0
Orientation – 0.9
Blasting – 0.9

The Haines-Terbrugge empirical slope design chart, which relates adjusted MRMR to slope angle and slope height for specified factors of safety, was used to estimate to estimate slope angles. A maximum vertical slope height 150 m and a nominal factor of safety of 1.5 were used.

The overall slope angle for the open pit at Haveri derived and used for this PEA is 47°.

Underground – Stope Dimensions

For the purpose of the underground assessment, the Stability Graph Method was used to estimate stable stope spans. Stability graph input parameters for an average (assumed) stope dip of 70° using an average hangingwall Q’ value of 5.0 are presented in Table 16-3. The N’ values for each vein orientation relate to hydraulic radii for stable stope hangingwall dimensions (Table 16-4).

Table 16-3: Stability Graph Method Input Parameters

	80° Dipping Vein	Comments
Q'	5.0	Fair rock mass
UCS (MPa)	125	Estimated
Sigma 1	5.6MPa	300 m mining depth assumed. Rock mass density - 2.8 t/m^3
Stress:strength ratio	22.32	
Factor A	1.0	
Angle between stope face and daylighting joint	10°	No structural data available so conservative value used
Factor B	0.2	
Potential Failure Mode	Gravity	Assumed. No structural data
Dip of Stope Face	70°	Assumed from orebody wireframe (more conservative than 90°)
Factor C	2	
N' = Q' x A x B x C	5.9	

Table 16-4: Stope Hydraulic Radii and Stope Stability Condition

Hydraulic Radii for Various Stope Geometries											
OB Thickness	Stope Length (m)										
Stope Span (m)	10	20	30	40	50	60	70	80	90	100	110
10	2.50	3.33	3.75	4.00	4.17	4.29	4.38	4.44	4.50	4.55	4.58
20	3.33	5.00	6.00	6.67	7.14	7.50	7.78	8.00	8.18	8.33	8.46
30	3.75	6.00	7.50	8.57	9.38	10.00	10.50	10.91	11.25	11.54	11.79
40	4.00	6.67	8.57	10.00	11.11	12.00	12.73	13.33	13.85	14.29	14.67
50	4.17	7.14	9.38	11.11	12.50	13.64	14.58	15.38	16.07	16.67	17.19
60	4.29	7.50	10.00	12.00	13.64	15.00	16.15	17.14	18.00	18.75	19.41
70	4.38	7.78	10.50	12.73	14.58	16.15	17.50	18.67	19.69	20.59	21.39
80	4.44	8.00	10.91	13.33	15.38	17.14	18.67	20.00	21.18	22.22	23.16

N	5.9	
Key		Hydraulic Radii (m)
	Stable below	4.90
	Unsupported Transitional	7.5
	Stable with Support	10.2
	Supported Transitional	12.2
	Unstable above	12.2

This analysis indicates ranges of stope dimensions that satisfy various stability criteria. An appropriate hydraulic radius for initial design purposes will lie between the Stable HR value and the Stable with Support HR value. The underground spans at Haveri should have a hydraulic radius of less than 10.20.

16.2.4 Conclusions and Further Investigation

SRK has reviewed the methodology used to estimate open pit slope angles and underground spans and considers this to be to internationally accepted standards and appropriate for a PEA.

The overall slope angle for the open pit at Haveri to be used for the PEA is 47°.

The underground spans at Haveri should have a hydraulic radius of less than 10.20.

Open Pit

Given the preliminary nature of the study, the lack of geotechnical data and the empirical method by which the slope angles have been estimated, SRK considers that the slope angles could be increased when greater confidence in the geotechnical parameters are achieved in later studies. More detailed geotechnical investigations 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.

Underground potential

As a general rule, underground mining method selection is determined by the geological and geotechnical characteristics of a deposit and the productivity required from the mine. The process is, however, often carried out early in the development cycle of a mine and often the fundamental data required to make an informed assessment, such as extensive geotechnical drilling, has not been undertaken. This can make the process in early stages of mine design very subjective and based largely on benchmarking against similar deposits. The range of underground spans presented by SRK for the PEA is considered to be conservative given the lack of geotechnical data available. Opportunities to optimise the spans could therefore be achieved through further geotechnical investigations.

Further Investigations

A geotechnical and hydrogeological drilling programme should be implemented to explore the rock mass, structural and hydrogeological properties of the hangingwall and footwall rocks. Geotechnical information of the orebody can be gathered from resource holes. Geotechnical boreholes should be located near the perimeter of the preliminary open pit whittle shell and drilled at various dip and azimuth angles to reduce the structural bias generated by the resource boreholes. A geotechnical logging, sampling and rock mechanical laboratory testing programme should be undertaken. Boreholes should be orientated and logged for structural parameters.

Given the small amount of information on the rock mass strength in the footwall and hangingwall rocks, SRK recommends the following:

- Selected future boreholes should be logged geotechnically, orientated and piezometers installed to measure groundwater levels.
- A limited number of specific geotechnical boreholes behind or near the pit crest wall into the footwall and hangingwall waste rocks should be drilled as part of future studies.
- Selected exploration holes should be extended at least 50 m into the footwall/hangingwall.
- Outcrops located near the final pit walls should be mapped geotechnically (if available/accessible).
- All core should be strength tested using a portable point load tester which allows testing samples directly at the core shed in conjunction with a limited geotechnical laboratory testing programme.

16.3 Hydrogeology

SRK reviewed the potential impact of water inflows and the results are presented elsewhere in this report.

16.4 Seismicity

SRK concludes that seismic risks are low though additional analysis of seismic data from the regional digital seismic stations is required to determine the design criteria for buildings and pit slopes.

16.5 Open Pit Optimisation

SRK used the Whittle 4X pit optimisation software to determine the economic pit limits for the Inferred Resources in the base case option. The key parameters for optimisation are summarised in Table 16-4. The metal prices and smelter charges were estimated using recognised sources including the Mining Cost Service (Infomine). The mining cost used in the optimisation is estimated using a base mining cost and an incremental cost for depth. The mining losses and dilution factors were considered suitable for the nature of the geological contacts, dip and shape of the mineralised zone, mineralised thickness, maximum thickness of interburden and minimum thickness of mineralisation, and mining equipment selected. These suggested an average waste dilution factor of 5.0% and ore losses of 3.0%.

The currency used for the purposes of the mining study is United States Dollars (USD). The base case metal prices used in the study are based on consensus market forecasts and are therefore lower than those used to report the MRE presented earlier in this report.

Table 16-4: Pit optimisation criteria

Metal Prices, Q4 2013	**Units**	**Input/Calculation**
Copper Price – Mean CMF long term price	c/lb	274
	$/t	6 050
Copper Price - 30% premium	c/lb	356
	$/t	7 850
Gold Price - Mean CMF long term price	$/g	37
	$/oz	1 160
Gold Price - 30% premium	$/g	49
	$/oz	1 510
Whittle Analysis		
Mining Ore Losses	%	3%
Mining Dilution	%	5%
Dilution Grade Gold	g/t Au	0.00
Dilution Grade Copper	% Cu	0.00
Processing - Gold Concentrate to Cyanidation		
Recovery Au	%	60%
Conc. Grade Au	g/t Au	19.0
Mass recovery Au	%	3%
Processing – Copper/Gold Concentrate		
Recovery Au	%	20%
Recovery Cu	%	60%
Conc Grade Au	g/t Au	97.0
Conc Grade Cu	%Cu	25%
Smelter Recovery Au	%	95%
Smelter Recovery Cu	%	95%
Mass recovery	%	0.20%
Operating Cost Breakdown		
Reference Mining Elevation	m	80
Ref Mining Cost Waste	$/t	3.5
Ref Mining Cost Ore	$/tRoM	3.5
Incremental Mining Cost	$/t/10m	0.05
Processing Cost Flotation	$/tore	14.0
Process Cost Cyanidation	$/tconc	1.0
G&A Cost	$/tRoM	Include in Processing
Transport Ore to the Process Plant	$/tkm	0.28
Transport Distance	km	1.5
Transportation Cost	$/tRoM	0.42
TC/RC - Copper		
T/C Copper Conc.	$/tconc	90.0
R/C Copper	$/lb Cu	0.09
	$/t Cu	199

TC/RC - Gold		
Refining Deduction Au	%	0.5%
Refining Charge Au	$/oz	6.0
Taxes & Royalties		
Royalty Cu	%	0.15
Royalty Au	%	0.15

The nested pit shells produced by Whittle are graphically presented below in Figure 16-1 with the highlighted option indicating the final selected pit shell.

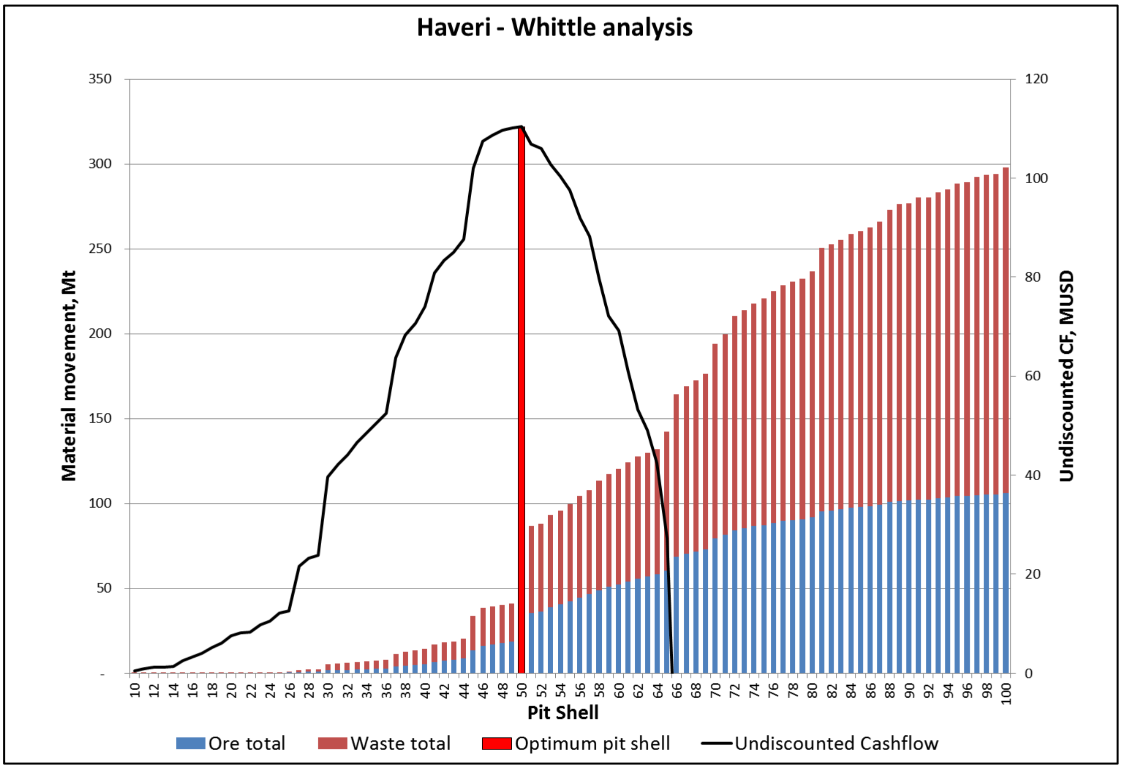

Figure 16-1: Pit optimisation results (Source: SRK, 2014)

SRK notes that the maximum undiscounted cash flow is achieved by shell 50 with the gold price 1160 USD/oz and copper price 5550 USD/t and SRK selected this shell for the further analysis. The selected pit shell is projected onto an aerial photograph in Figure 16-2 below.

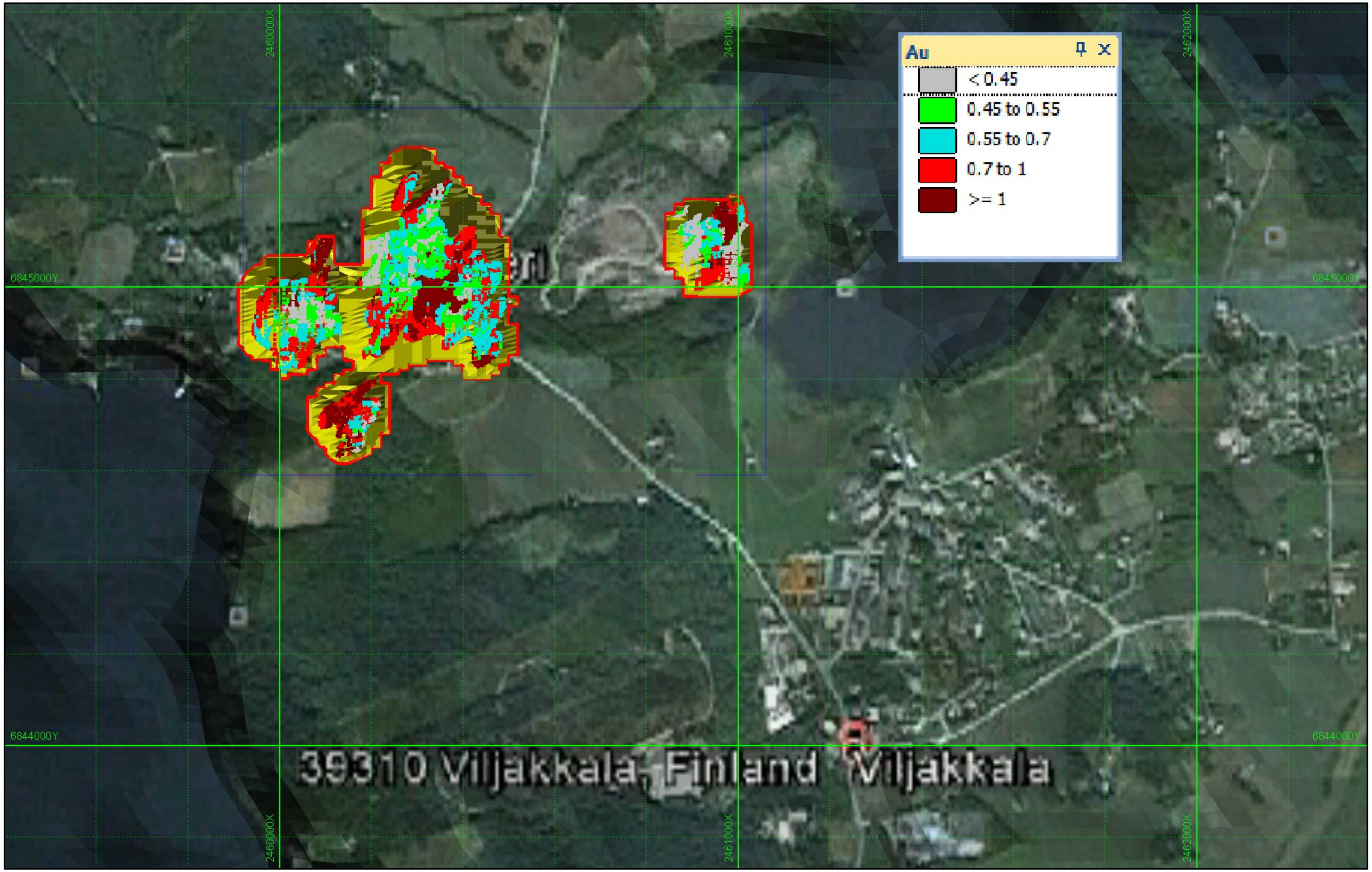

Figure 16-2: Selected pit shell with block model showing grade distrbution. Aerial photo as background (Source: SRK 2014)

16.6 Life of Mine Plan

The mine plan is based on a production rate of 1.8 Mtpa which generates an overall mine life of 11 years. SRK considered a mining sequence based on average strip ratio in the final optimum pit shell – no pushbacks were selected.

SRK split the mineralised material into three categories using gold equivalent grade ("Au EQ"). The formula to calculate the equivalent gold grade:

(Au EQ) (g/t) = 0.994456 x Au (g/t) + 1.288622 x Cu (%)

The three categories are based on cut-off grade calculations as follows:

- High-grade: Au EQ > 0.70 g/t.
- Low-grade: 0.55 g/t < Au EQ < 0.7 g/t.
- Mineralised waste: Au EQ < 0.55 g/t.

Mineralised waste is stockpiled for possible processing, should an increase in future gold price warrant this. This material is not included in the production schedule.

SRK notes that a pushback based, bench-by-bench schedule should be provided as part of the next stage of mining study.

16.7 Operating Strategy

The operating strategy is based on the mine schedule to provide:

- a preliminary estimate of mining equipment requirements;
- a preliminary estimate of mining personnel; and
- a basis of the mine cost estimate.

Equipment requirements have been determined using the following methods:

- 261 workings days per year and 16 working hours per day;
- truck and excavator requirements were calculated based on productivities and cycle times;
- 3 m^3 capacity excavators and 24 t articulated trucks have been assumed for rock mass movement
- drilling requirements has been based on 5 m benches with 115 mm blasthole drills for the run of mine and 10 m benches with 152 mm blasthole drills for the waste;
- ancillary equipment has been based on material movement and primary fleet requirements;

The mine equipment requirements and the mobile and auxiliary equipment requirements are shown on an annual basis in Table 16-5.

Table 16-5: Equipment requirements (per year)

Mobile Mining Equipment	
Ore Percussion drill rig	1
Waste rotary rig	1
Hydraulic Shovel 3m^3	3
24t truck	10
Cat D8 type Bulldozer	4
GRADER	1
Wheeled Loader 6m^3	1
W/Bowser	1
Auxiliary Equipment	

Tractor & trailer	1
Explosives Truck	1
Light Tower & gen set	4
Hydraulic rock breaker	2
Diesel pump 150mm	3
Pick up twin cab	2
Pick up single cab	4
Fuel & Lube Truck	1
Low bed and tractor	1
Service truck with Hi-ab	1
180 psi compressor	2
Rough terrain hi-ab truck	1
3t tyre handler	1
Crew bus	1
Fuel Bowser	1
Road wagon	1

Personnel requirements have been based on:

- material movements; and
- equipment requirements;

An estimate of the mine staff and maximum personnel required for the life of mine is shown below in Table 16-6.

Table 16-6: Personnel required

Mine staff	
Mine Manager	1
Maint Supt	1
Shift Foreman	2
Mine Trainer	1
Workshop Supervisor	2
Senior Planning Engineer	1
Planning Engineer	1
Senior geologist	1
Shift geologist	2
Senior Surveyor	1
Survey Asst	2
Welders	2
Fuel & Lube	2
Tyre	2
Maint Planner	1
Service Crew	2
Blasting Gang	2
Mine personnel	
Drillers	4
Shovel Operators	6
Truck Drivers -24t	20

Dozer Operators	8
Grader Operators	2
Wheel Loader Operators	2
Water Truck Operators	2
Fitters	15

16.8 Capital and Operating Costs

16.8.1 Equipment

SRK estimated the mining capital costs and fleet requirements, which are summarised in Table 16-7 below, using the following assumptions:

- The truck cycle times for run of mine, waste and overburden are based on the average location of the benches in the pit for each cut-back.
- Based on typical productivities for a 24 t articulated truck with matching excavator and drills, and an average operating time of 2,731 hours per year.

Table 16-7: Average operating time – Haveri

	Generalised Shift Times			
	Calendar Days	(days)		365
	Days per week	(days)		5
	Available Days	(days)		261
	Holidays			
	Weather			10
	Scheduled days	(days)		251
	Shifts/day	shifts		2
	Annual Work Shifts	shifts		502
	Hours/day	hrs/day		16
	Scheduled Hrs			4 016
	Shift Breakdown			
	Overall Shift Pattern	(hrs)		8
	Shift Change	Min/Shift		30
	Lunch/Coffee Break	Min/Shift		30
	Fuelling	Min/Shift		15
	Blasting	Min/Day		30
	Maximum Work Hours per Day	(hrs)	13.0	
		hrs/shift		6.5
	Mechanical Efficiency			**85%**
	Mining Utilisation			**79%**
incl	Shift stoppages			81%
incl	Effective work - mins/hr	58		97%
	Work hours per shift	hrs/shift		4.3
	Total Hrs/yr			8 760
	Available hrs/yr			**4 016**
	Mechanical Efficiency	85%		**3 414**
	Utilisation	80%		**2 731**

Initial capital expenditure is defined as the investment in the first two years to achieve full production. No replacement has been planned due to the life of the operation and the fact that material movement declines in the later years. Mining equipment capital costs are presented

in Table 16-8.
salary.

Table 16-8: Mining equipment capital cost – Haveri

Main equipment	Unit cost (USD)	Costs (USD)
Ore Percussion drill rig	653 072	653 072
Waste rotary rig	437 621	437 621
Hydraulic Shovel 3m3	604 095	1 812 285
24t truck	339 128	3 391 280
Cat D8 type Bulldozer	624 431	2 497 724
CAT 12M GRADER	277 328	277 328
Wheeled Loader 6m3	612 902	612 902
CAT W/Bowser	307 455	307 455
Sub Total		**9 989 667**
Auxiliary Equipment		
Tractor & trailer	61 800	61 800
Explosives Truck	606 258	606 258
Light Tower & gen set	22 706	90 824
Hydraulic rock breaker	98 159	196 318
Diesel pump	14 626	43 878
Pick up twin cab	61 800	123 600
Pick up single cab	41 200	164 800
Fuel & Lube Truck	83 173	83 173
Low bed and tractor	144 200	144 200
Service truck with hi-ab	164 285	164 285
Compressor	25 750	51 500
Rough terrain hi-ab truck	82 400	82 400
3t tyre handler	67 980	67 980
Crew bus	92 700	92 700
Fuel Bowser	83 173	83 173
Road wagon	92 700	92 700
Sub Total		**2 149 589**
Total		**12 139 256**

Summary Mining Capital Costs		
Mine Facilities & Haulage Dispatch System	**(USD)**	6 124 049
Haul Roads	**(USD)**	742 933
Mobile Mining Equipment	**(USD)**	9 989 667
Auxiliary Equipment	**(USD)**	2 149 589
Total	**(USD)**	**19 006 238**

16.9 Labour

SRK used benchmarked annual salaries to estimate mining labour costs. Personnel requirements from have been used to determine the associated operating costs.

Statutory social costs required in Finland have been included in the employee salaries as well as shift allowances of USD 5788 per month and vacation salary which is 5% of the annual salary.

Table 16-9: Labour costs – Haveri

Mine Staff	(EUR/year)	USD/year
Mine Manager	148 000	197 333
Maint Supt	99 000	132 000
Shift Foreman	44 000	58 667
Mine Trainer	49 000	65 333
Workshop Supervisor	67 000	89 333
Senior Planning Engineer	91 000	121 333
Planning Engineer	66 000	88 000
Senior geologist	91 000	121 333
Shift geologist	66 000	88 000
Senior Surveyor	91 000	121 333
Survey Asst	38 000	50 667
Welders	46 000	61 333
Fuel & Lube	46 000	61 333
Tyre	46 000	61 333
Maint Planner	49 000	65 333
Service Crew	49 000	65 333
Blasting Gang	43 000	57 333

Operators & Fitters	(EUR/year)	USD/year
Drillers	53 000	70 667
Shovel Operators	53 000	70 667
Truck Drivers -25t	53 000	70 667
Dozer Operators	53 000	70 667
Grader Operators	53 000	70 667
Wheel Loader Operators	53 000	70 667
Water Truck Operators	53 000	70 667
Fitters - Shifts	46 000	61 333
Fitter Assistants	41 000	54 667

17 RECOVERY METHODS

17.1 Process Plant

The conceptual flowsheet design developed by SRK for Haveri for the purpose of this PEA is based on the following two process aims:

- The production of a marketable copper concentrate; and
- The production of the bulk of the tailings essentially devoid of sulphide and arsenic.

The proposed flowsheet will therefore produce three process streams, as follows:

- A “clean” copper concentrate;
- A bulk sulphide flotation concentrate, containing the remaining sulphides in the run of mine. This concentrate would be cyanide leached for gold recovery, and the tailings, following cyanide detoxification, stored in a lined and capped Tailings Storage Facility (“TSF”), to prevent the generation of acid and/or toxic metal containing effluent; and
- A “clean”, i.e. essentially free of sulphides and heavy metals, tailings stream for disposal in the primary TSF.

This flowsheet is identical in concept to that recently proposed by SRK for the Kopsa project in central Finland, for material of a similar Au but slightly higher Cu grade. It is also similar in concept to the flowsheet used at another operating mine in the region. In this case, the only valuable metal is gold, and so both the sulphide concentrate and the flotation (and gravity) tailings are cyanide leached – in separate circuits. Both tailings are detoxified following cyanidation, after which the sulphide tailings are stored in a lined facility, and the main flotation tailings stream is stored in a paddock facility.

17.2 Process Design Criteria

Based on the metallurgical investigation data presented in Section 13, SRK has developed the following design criteria for the processing of the Haveri material, in support of the mine schedule and Mineral Resource estimate developed for the PEA.

Table 17-1: Process Design Criteria

Item		Unit	Value
RoM Production		tpa	1,800,000
Flotation Feed Grade	Cu	%	0.09
	Au	g/t	1.00
	S	%	1.24
Copper Concentrate		tpa	3,900
	Cu Rec	%	60.0
	Au Rec	%	20.0
	Cu	%	25.0
	Au	g/t	92.6
Sulphide Concentrate		tpa	45,000
	Au Rec	%	60.0
	Au	g/t	24.0
Cyanidation Recovery	Au	%	95.0
Recovery to Doré	Au	%	57.0
Overall Recovery	Cu	%	60.0
	Au	%	77.0

The design criteria is based on a number of assumptions, detailed as follows:

- The copper flotation criteria are based on the 2003 GTK results. While the GTK tests were open circuit, they did achieve concentrate grades and open circuit recoveries above the values assumed in the design criteria, even for the MD-ME sample, which had a very low head grade of 0.06% Cu.

- The 2003 GTK testwork did not aim to produce a bulk sulphide concentrate, and the S recoveries to the rougher flotation stage (see Table 13-2) were quite low, especially for the Haveri (SW and MD-ME) samples. The bulk concentrate flotation criteria assumes a mass recovery of 2.5% of the feed; at the listed head S grade this is equivalent to recovering essentially all of the sulphide to a concentrate assaying approximately 92% pyrite. SRK notes that the technical feasibility of producing a “clean” flotation tailings has not yet been demonstrated.
- The assumed cyanidation stage recovery is significantly higher than the cyanidation recoveries reported both for the GTK work (see Table 13-3) and the Outokumpu work conducted on the historic tailings (see Section 13.4). However, the assumed bulk sulphide concentrate Au grade is also significantly higher than the head grades of the samples reported in the historic testwork; the tailings grade for a 95% recovery from a 24 g/t Au sample is higher than the head grades of the historic samples.

17.3 Process Plant Capital Cost Estimate

SRK has estimated a capital cost for the proposed conceptual Haveri process plant based on information from a subscription database.

The estimated capital cost for the process plant – flotation and cyanidation of the bulk sulphide concentrate, including tailings delivery (i.e. pump station) – is USD 50 million. This figure, which is an estimate suitable for a conceptual / scoping level of study only, can be considered to be inclusive of indirect costs such as EPCM and Contingency.

17.4 Process Plant Operating Cost Estimate

SRK has estimated operating costs for the proposed conceptual Haveri process plant based on information form a subscription database, as well as recent operational data from a Finnish flotation plant processing sulphide ore.

The estimated operating cost for the process plant, suitable for a conceptual / scoping level of study only, is as follows:

- Flotation (including crushing and grinding): USD 14 /t RoM ore; and
- Bulk Sulphide concentrate cyanidation: USD 1 /t RoM ore.

17.5 Recommendations

No metallurgical testwork was undertaken in support of the production of this PEA and the metallurgical parameters developed are based on a relatively limited amount of historical data. In addition, virtually no specific engineering was conducted with regard to the process plant design, and the process plant capital and operating costs subsequently generated are very high level estimates.

Further development of the Haveri project will therefore require the execution of metallurgical testwork and plant engineering programs commensurate with the level of study being undertaken. Given the flowsheet proposed in this PEA, a future metallurgical testwork program should focus on the production of a saleable copper concentrate, with the recovery of the remaining sulphide minerals into a bulk sulphide concentrate, with the corresponding production of a benign bulk tailings stream. The gold would then be recovered from the bulk sulphide concentrate by cyanide leaching.

Samples selected for the metallurgical testwork program should cover the expected range of potential variability within the Haveri deposit. The variability parameters should include grade

– Cu, Au and As, as well as location within the orebody, such as lateral extent and depth.

As well as flotation and cyanidation response, other relevant properties that should be tested include hardness, i.e. crushing and grinding work indices, as well as settling characteristics and environmental risk parameters, e.g. ARD and heavy metal mobility.

On the basis of more specific process parameters developed from this testwork, a more detailed plant engineering study can be undertaken, again commensurate with the precision of the overall study.

18 PROJECT INFRASTRUCTURE

18.1 Site Access, Power and Water

Given the location of the project, and the fact that it hosted a historic mining and processing operation, albeit of a small scale, SRK has assumed that site access and the provision of electrical power and water to the project site will be relatively straightforward.

The project's electrical power requirements are likely to be of the order of 6-8 MW. The project's make-up (i.e. net) water requirements are likely to be of the order of 1 Mm^3/annum (equivalent to 2,800 m^3/day or 30 l/s).

19 MARKET STUDIES AND CONTRACTS

No market studies have been conducted and no contracts have been signed to date.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Hydrogeology

20.1.1 Previous Work and Data Sources

No previous detailed work has been carried out in relation to water management at Haveri. Data sources that have been used as a basis for the assumptions in this PEA include:

- A report pertaining to environmental studies, permitting and social or community impact (SRK, 2014a);
- Various reports relating to the geology of the Haveri deposit (Karvinen and Fraser, 2003 and Strauss, 2003);
- A technical report and current resources estimate (Reed, 2008);
- Core photos from a selection of exploration hole drilling;
- Anecdotal information provided by the Oxy Dive Club;
- Regional climatological data derived from the FAO Lo-Clim database; and,
- The Hertta 5.2 environmental database, supplied by OIVA (http://wwwp2.ymparisto.fi).

20.1.2 Site Characterisation

Location and Site Description

Haveri is a historical gold-copper mine in Ylöjärvi municipality located in Southern Finland about 175km north of Helsinki and 35km west of Tampere. The nearest village is Viljakkala, located 1km east of the mine. The setting is rural, with farms, forests and lakes.

Figure 20-1: Haveri mine location (Google Earth, accessed February 26, 2014) with nearby surface water bodies and preliminary catchment delineation (blue lines) calculated using Aster topographic data (Source: SRK 2014)

As described by various authors (Reed, 2008, Karvinen and Fraser,2003, and Stauss, 2003) the Haveri deposit has been the site of periodic mining for over 250 years. In the most active part of its history (1942 to 1960), the main commodities were copper and gold which were extracted from open pit and shallow underground workings. Since 1962 the property has been inactive, except for limited exploration in the immediate area.

The existing, water-filled open pit is approximately 80 m deep and 100 m long. Relic underground workings also exist which are approximately 200m deep and up to 500m wide.

Topography

As described by Reed (2008), the local topography consists of rolling hills, with a strong glacial influence that has imparted a NW-SE direction that is apparent in the shape of the lakes and hills. The area lies at approximately 90-100mRSL on the gently undulating, shallow dipping slopes of Viljakkala, some 10m above the level of the surrounding lakes (approximately 83mRSL).

The elevation of the pit rim is highest on the northern and eastern walls (95-100mRSL) and lowest on the western and southern walls (90-95mRSL).

Hydrology

Haveri is located on a peninsular in Lake Kyrösjärvi (to the west) and Lake Viljakkalanselkä (to the east). The town of Kyrösjärvi has one of the main physical water monitoring stations in Finland and the hydrological status is publicly available online.

The current water management plan ("Kokemäenjoen-Saaristomeren-Selkämeren vesienhoitosuunnitelma" translated to "The Kokemäenjoki Archipelago and Bothnian Sea Management Plan") is valid to the end of 2015 and the preparation of the new water management plan to cover the period 2016 to 2021 is on-going.

A preliminary sub-catchment delineation has been undertaken (see Figure 20-1) for the immediate area surrounding Haveri. It can be seen that the pit is located on a watershed divide between two sub-catchments.

Geology

A description of geology is given in previous sections of this report. Further assessment at PFS level will be required to assess the implications of faulting for groundwater management, as well as an overall evaluation of structural controls on groundwater flow.

Climatological Data

No locally sourced data for the mine site has yet been obtained. In its absence, regional climate data has been obtained from FAO Lo-Clim estimation software. Figure 20-2 illustrates the monthly variation in precipitation, temperature and potential evapotranspiration. Average annual precipitation is about 566mm, with July and August having most precipitation with 70 and 75mm respectively. Annual potential evapotranspiration is estimated to be 545.5mm, indicating a minimum precipitation excess of 20.5mm. Temperatures in the region generally fall below freezing from November to April. Around half of the annual runoff is estimated to occur in the three months during the spring melt.

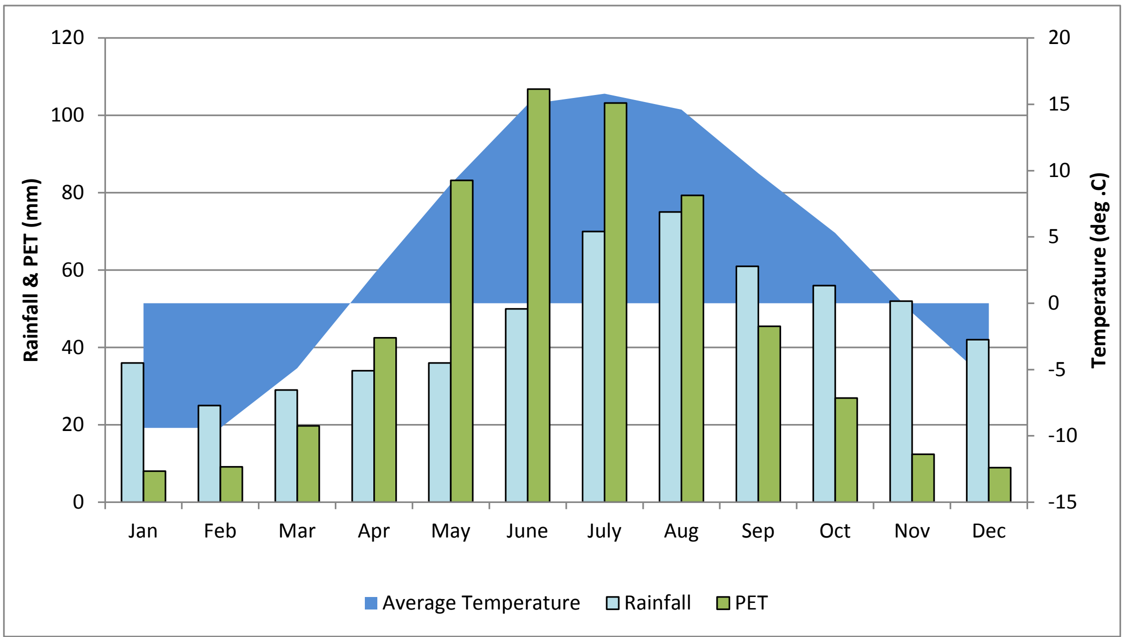

Figure 20-2: Precipitation, temperature and evaporation estimates for Haveri (FAO Lo-Clim)

Recharge

No site specific data exists for groundwater recharge. Conceptually, however, recharge will only occur when temperatures are above freezing which, according to Figure 20-2, occurs on average from around April to November. Many workers (Bengtsson, 1982) have shown that snow-melt is a significant source of groundwater recharge as the rate of melting is slow enough to encourage infiltration, whilst others suggest that the frozen ground under the snow impedes groundwater recharge. It is likely that both mechanisms may exist under differing conditions. More detailed hydrological analysis as part of any future study will be required to allow an estimate of groundwater recharge rates to be made.

Water Balance

In order to obtain estimates of snowmelt, snow storage, actual evapotranspiration, runoff and recharge, a simple monthly water balance was developed for the area around Haveri , based on regional monthly climatic data (as described in Section 0) and was based on a model developed for the NOPEX[1] region by Xu et. al, (1996).

Snowfall and subsequent melting is described as a function of precipitation and temperature according to the following equations:

$$Snowfall = precipitation \ x\ (1\text{-}\{exp[-(temperature - a_1)/(a_1 - a_2)]\}^2)$$

$$Snowmelt = snowpack \ x\ (1\text{-}\{exp[-(temperature - a_2)/(a_1 - a_2)]\}^2)$$

Equation 1: Snowfall and snowmelt as a function of precipitation and temperature

where a_1 and a_2 are constants.

Figure 20-3 illustrates calculated monthly snowfall and snowmelt according to the above equations. It can be seen that snowmelt is predicted to start in April with residual snowmelt in June.

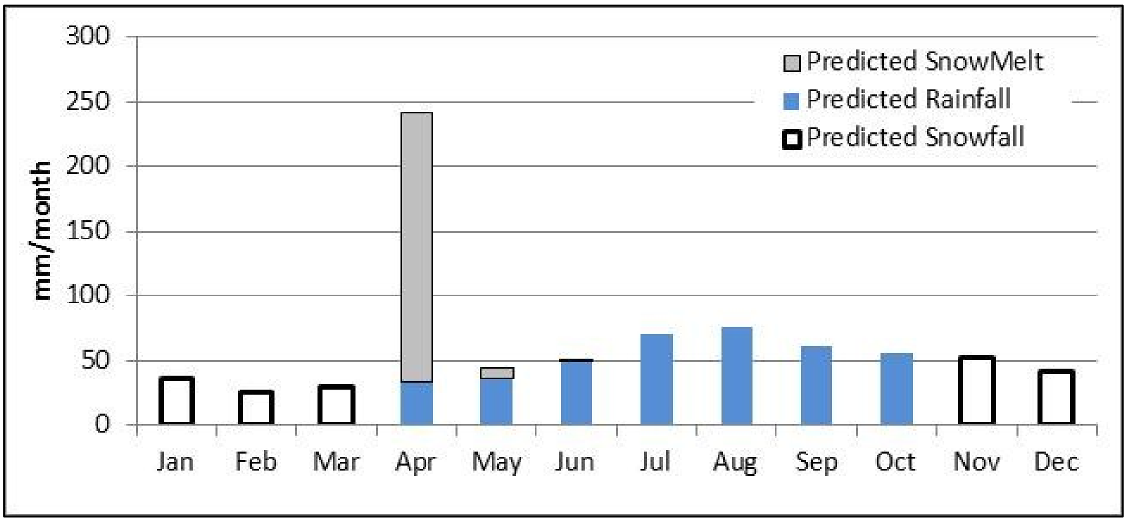

Figure 20-3: Predicted Rainfall, snowmelt and snowfall water balance for the Haveri region

20.1.3 Hydrogeology

Introduction

No studies have been undertaken at Haveri relating to hydrogeological characterization or water management. The characterization of the hydrogeology is based on evaluations made by SRK from core photos, wireframes of the relic open pit and underground workings and judgements based on previous experience.

The analyses presented here, which are mostly empirical, are appropriate for a PEA. However, further data collection and analysis will be required to support any further study.

Hydrogeological Setting

The presence of groundwater identification has largely been based on core photographs of exploration holes and geological and structural interpretation provided by previous workers (Reed, 2008, Karvinen and Fraser,2003, and Stauss, 2003).

[1] *Northern Hemisphere Climate Processes Land Surface Experiment Project*

There are two main groundwater zones that are considered during this preliminary study. These are:

- The glacial overburden;
- The deeper, more competent bedrock.

Within these zones, there are also likely to be differences in groundwater flow regime corresponding to lithology and structure within the bedrock which may or may not be controlled by lithology.

Glacial till is typically quite heterogeneous in nature with boulder clay dominating but also with pockets/lenses of sand gravel that could offer enhanced permeability. The till is likely to play a relatively minor role in saturated groundwater flow at Haveri due to its limited thickness (generally ≤3m as cited by Strauss, 2003) relative to the other rock types present. That said, it may play a significant role in terms of control of groundwater recharge to the underlying strata.

Groundwater flow in the bedrock will be limited to fractures (faulting or jointing) and the weathered zone. Often, the upper portion of the bedrock is significantly weathered, however previous studies suggest that this is not the case at Haveri (Karvinen and Fraser, 2003). Where fractured rock is present, permeability may be high.

A high-level review of core photos suggest that discrete zones of intense fracturing are present within the fresh rock (see Figure 20-4 and Figure 20-5 for examples), which may result in a higher permeability. Equally, storage will be controlled by the fracture network, with fracture connectivity playing a key role in storage. Where fractures are poorly connected, groundwater inflows to an excavation would be short-lived as fracture storage is exhausted.

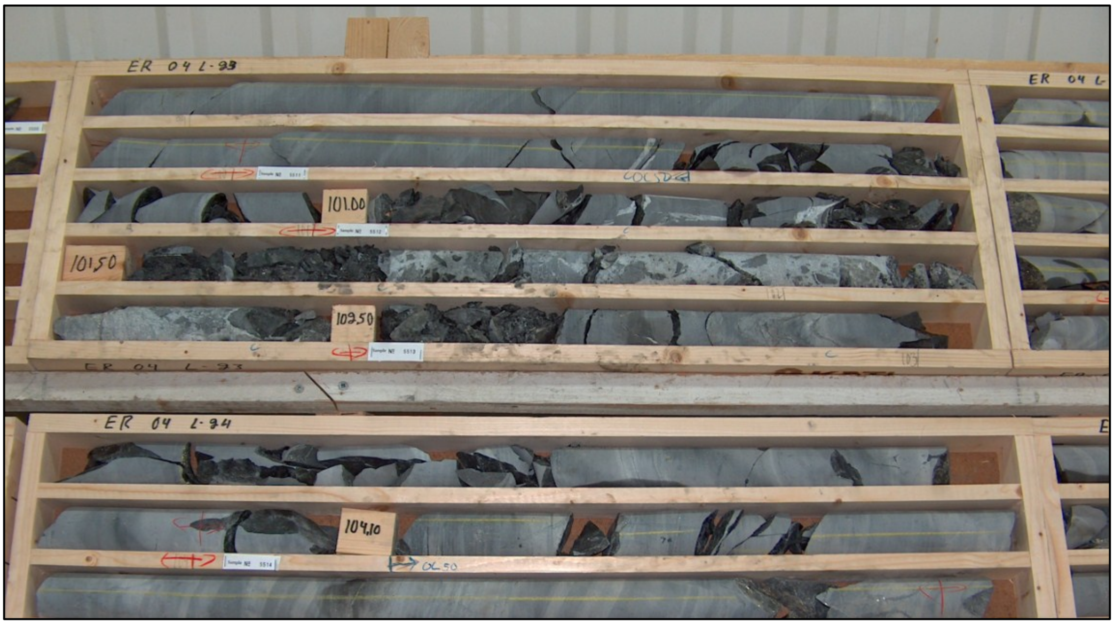

Figure 20-4: ER04 core photo illustrating a zone of intense fracturing

Figure 20-5: HN06 core photo illustrating a zone of intense fracturing

Groundwater Measurements

Available groundwater monitoring data for Haveri is limited.

The only publicly available groundwater measurements have been obtained from the Hertta 5.2 database which shows seven boreholes nearby to the mine (see Table 20-1 and Figure 20-6). Six groundwater elevations are available, all measured on August 3, 1976, which range between 83.15 to 85.19mRL.

Caution must be exercised when using data that might not reflect present day, or indeed representative, conditions. However, the groundwater elevations indicate a westerly groundwater flow direction suggesting that groundwater may be discharging into the lake. Furthermore the average elevation of Lake Kyrösjärvi is 83.2mRSL (Hertta server, 2014) corresponding closely to observed groundwater levels, reinforcing the evidence of a connected groundwater/surface water system.

Table 20-1: Historical water elevation measurements

Map Location / ID	No of Measurements	Date(s)	Groundwater Elevation (mASL)
1 / (HP3)	1	03/08/1976	83.69
2 / (HP4)	1	03/08/1976	84.85
3 / (HP1)	1	03/08/1976	83.17
4 / (HP7)	1	03/08/1976	85.19
5 / (HP6)	1	03/08/1976	83.16
6 / (HP5)	0	-	-
7 / (HP2)	1	03/08/1976	84.04

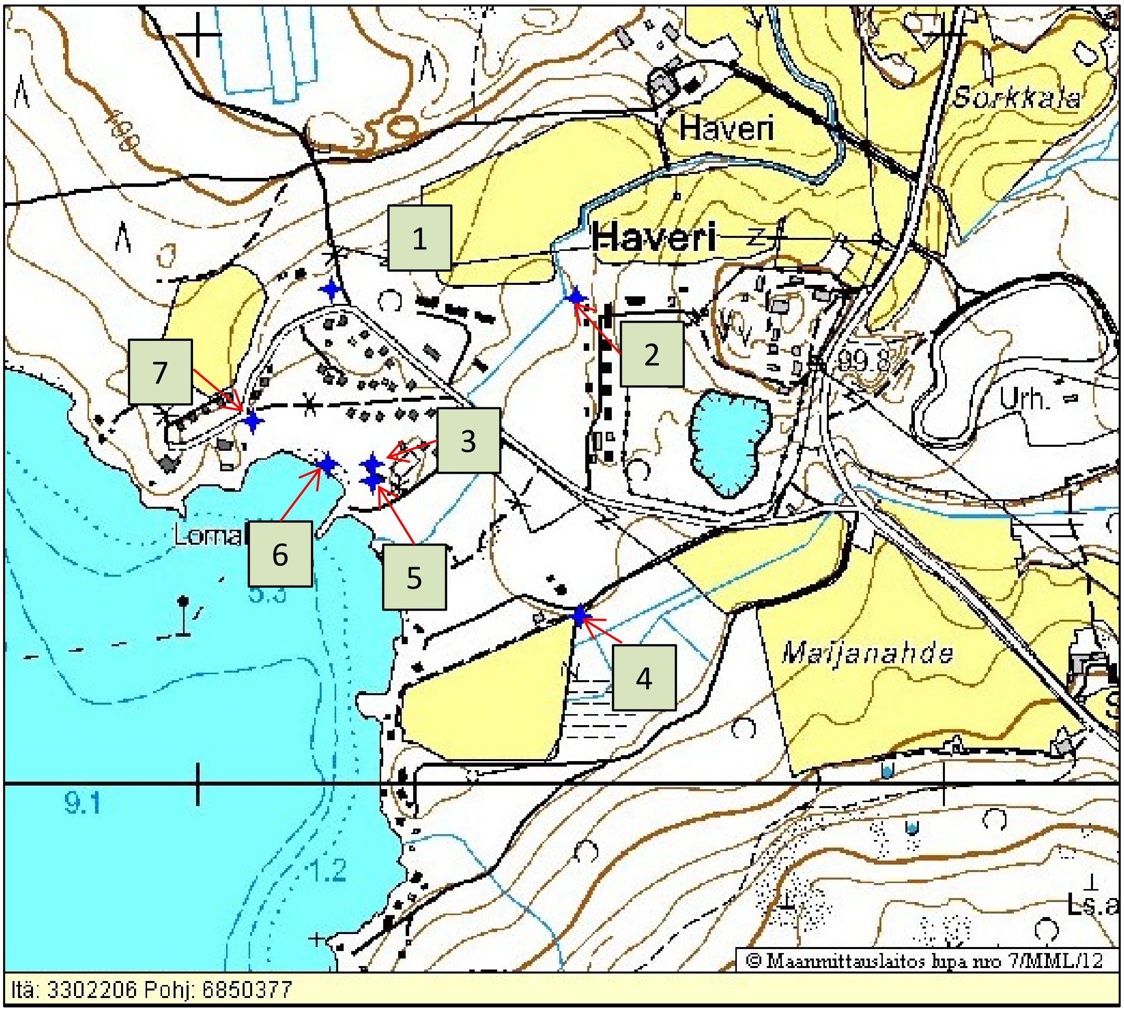

Figure 20-6: Location of historical groundwater levels (Source: Hertta 5.2 Database, 2014)

Preliminary Hydraulic Properties

No site specific measurements have been taken of the hydraulic properties of the hydrogeological units present at Haveri. The following is a discussion of some plausible ranges of hydraulic properties based on values for similar geological units published in the public domain.

The preliminary conceptualisation indicates that the majority of groundwater inflow may originate from zones of intense fracturing within the bedrock. Hydraulic conductivity of fractured bedrock may range between 1x10-4 and 1m/day (Heath, 1983). The conductivity of fractured rock is largely controlled by the infill and condition of fractures. For example, a fracture is likely to be water bearing if it is open of contains sand/gravel filled fractures. Alternatively, fractures can be less impermeable if they are filled with clay gouge or mineralisation.

For Haveri, it is conservatively assumed that all fractures, where present, will have no localised infill or mineralisation and will therefore be open to groundwater flow. This conservative assumption has been taken as there is no data to support a less conservative approach (such as detailed geotechnical logging including infill characteristics, fracture aperture, etc.). Based on this assumption, hydraulic conductivity of fractured bedrock at Haveri may be between 1x10-1 and 1m/day (towards the upper range of values estimated by

Heath). This range corresponds closely to previous fractured bedrock estimates from analogous SRK studies in Finland.

The host bedrock (un-fractured) may have a much lower hydraulic conductivity of between 2x10-9 and 2x10-5 m/day (Heath, 1983).

It has been assumed that 10% of the bedrock will be subject to intense fracturing, with the remaining 90% characterized as relatively impermeable. This is a "best-guess" hypothetical assumption that has been made following a high level review of core photographs. Further structural evaluation would be required during the next phase of study and the implications for groundwater management reviewed. Based on this assumption, a "bulk" hydraulic conductivity for the entire saturated thickness is estimated between 1x10-1 and 1x10-2 m/day (Table 20-2).

Table 20-2: Estimated hydraulic conductivity for the Haveri PEA

	Hydraulic Conductivity (m/d)		Estimated Proportion of Total Mine Depth (%)
	Maximum	**Minimum**	
Fractured Bedrock	1	$1x10^{-1}$	10%
Un-fractured Bedrock	$2x10^{-5}$	$2x10^{-9}$	90%
Representative "Bulk" K	**$1x10^{-1}$**	**$1x10^{-2}$**	**-**

Recent Pit Lake Formation

Water from the mine workings was pumped out by Glenmore in 1997 during is exploration program on the property (Karvinen and Fraser, 2003). It is assumed that the entire mine workings were dewatered to facilitate underground sampling and exploration that is reported to have been completed. Historical volumes of dewatering were not available at the time of this assessment.

It is reasonable to assume that pumping of the mine ceased in 2000 at the same time the Glenmore Highlands exploration programme ended. Photos from the year 2000 (Figure 20-7 – top left) show a pit that is still dry although it is unclear to what level groundwater might be at in the underground mine workings.

Anecdotal information from the Oxy Dive Club suggests that in May 2001 the pit lake water level was at approximately 35m below ground level.

A later photo from 2004 (Figure 20-7 – top right) indicates an estimated pit lake level at approximately 14-20m below ground level.

The Oxy Dive Club provides further anecdotal information suggesting that in 2008 the lake water level had recovered to the "normal level", which is interpreted to be approximately 1 to 5 meters below ground level, dependant on relative location to the pit rim. This is the approximate level of the present day lake (Figure 20-7 - bottom), equivalent to an estimated elevation of 89mRSL.

Figure 20-7: Air photo of the pit in 2000 (top left)[2], northern and western walls of open pit in 2004 (top right)[3], and northern and western walls in 2013 (bottom)[4]. (Source: Palmex, 2014)

The total water volume based on existing void space is summarised in Table 20-3 is approximately 508,919m^3.

Table 20-3: Water Volume of Existing Workings

Wire Frame	VOLUME (m^3)
Old_UG_drifts	89,436
Old_Drifts_intopit	15,685
Old_Pit_Ut	435,069
Toal Water volume, m3	508,819

Neighbouring Claims & Operations

A search of the public domain relating to water management issues of neighbouring claims has been conducted. Two nearby mines are of most interest, namely Ylojarvi copper mine (15km away) and Orivesi gold mine (40km away).

Ovivesi (currently operational) water management involves collection of water in underground sumps which is then pumped to the surface. Sources indicate ("Orivesi goldmine pollutes", 2013) that in 2013 water pumped from Ovivesi mine into Ala-Jalkajärvi and Peräjärvi lakes resulted in elevated levels of sulfate, nitrogen and metals.

The Ylojarvi mine (active from 1943-1966) is reported to be the principle origin of arsenic

[2] *Source: "Haveri – Gold database" (2008)*
[3] *Source: "Haveri – Gold database" (2008)*
[4] *Source: "Haveri Pit" (2013)*

contamination in the surface and groundwater of the area (Parviainen, 2008). The mining activities left behind two tailings areas, two open pits and underground galleries. Water from Lake Parosjärvi was used in processing the ore in a closed circulation system, and in the first open pit as a clarification pool. As the mining works proceeded under the lake, the latter was dewatered. After closing the mine, the lake filled up again with water and as a result part of the tailings, open pits and underground galleries filled with tailings were flooding. This resulted in sulphide containing material subject to leaching into surface and groundwater.

No information was found regarding groundwater/surface water inflow rates or historical dewatering volumes.

20.1.4 Water Resources

Groundwater Resources

In Finland, groundwater resources are classified by the Finnish Environment Institute (SYKE in Finnish) into three categories: I, II and III:

Class I: An important groundwater area for water intake. Groundwater is or will be utilized within 20-30 years or should be reserved for emergency water supply. Water is used by at least 10no, households or industry;

Class II: A groundwater area that is suitable for water intake. Groundwater is not currently utilized but the potential exists for supply; and

Class III: A groundwater area that may be utilized for water intake but where available data is insufficient to classify them further. Class III groundwater areas may subsequently be investigated by SYKE and either moved to category I or II or alternatively removed from the groundwater classification system.

Classified groundwater areas are presented in Figure 20-8.

Part of the potential mining area is within a small (0.52km^2) Class I groundwater area called Haveri. According to Hertta 5.2. database (data accessed 13th January 2014) Haveri groundwater resource is located under clay, in a narrow gravel and sand deposit. This area is currently classified as an important resource for groundwater production and has got a valid groundwater protection plan in place. This groundwater area is not currently in use and potential water supply for drinking water production is calculated to be 180 m^3/day. According to a representative of the Municipal Water Company (Ylöjärvi Vesi Liikelaitos) iron and manganese concentrations in the groundwater are problematic for water supply, which is why this resource is not currently in use. It is currently unclear whether the elevated iron and manganese concentrations are naturally occurring or due to anthropogenic sources.

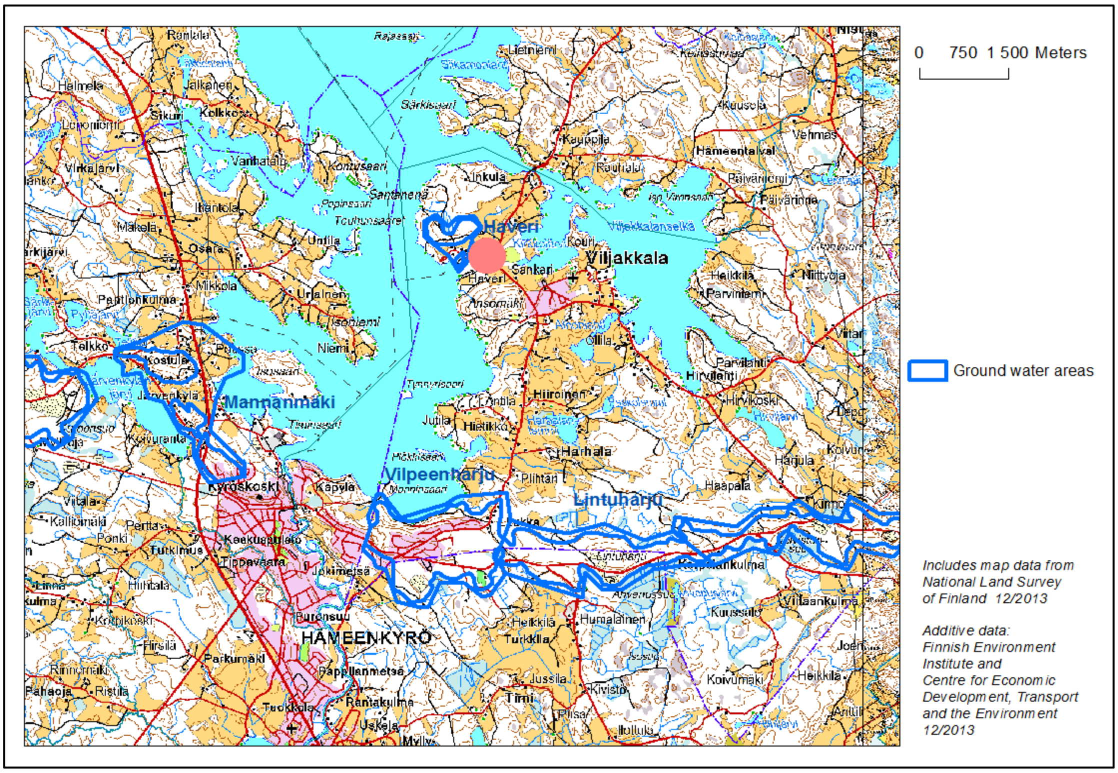

Figure 20-8: Regional groundwater areas (Hertta 5.2 database, 2014)

If the Haveri groundwater area remains as an official groundwater area in the future, it should be recognised that mining in Haveri may present a significant risk for the groundwater resource.

A second, larger classified groundwater area called "Vilpeenharju" is around 4-5km to the south of the project site. This aquifer is currently acting as the primary municipal water supply for surrounding villages.

Surface Water Resources

Potential surface water resources within the area consist of lakes that surround the project to the north, east and west.

20.1.5 Mine Water Management

Prediction of Inflows

Groundwater inflows to the mine development have been estimated using an analytical groundwater flow equation. The equation, as cited in Krusseman and De Ridder (1979) and Singh et al. (1985), is based on well hydraulics and can be used for estimation of steady state inflow rate to a mine. This equation was derived from the Theim-Dupuit equation and can be applied for an unconfined aquifer:

$$= \frac{\pi K(H^2 - h^2)}{In(\frac{R}{r_p})}$$

In the above equation:

Q = groundwater inflow (m3/day);

K = permeability of the unconfined aquifer - $2x10^{-1}$ m/d (max) and $2x10^{-2}$ m/d (min);

H = The initial water table elevation – 199m above base of workings;

h = The target water table elevation – 0m above base of workings;

R = radius of influence – 3300m (max) and 1000m (min);

r_p = equivalent radius of the mine workings – 28m.

Although the Theim-Dupuit equation assumes steady-state conditions, the model is quasi steady-state as the radius of influence (R) has been estimated using the Theis (1935) approximation for transient drawdown in an unconfined aquifer for a specific time (t) based on an assumed storage coefficient typical of fractured rock (1.4%). Time (t) is usually taken as the life of mine. However given that this has not yet been determined for the project, a nominal 10 year mine life has been assumed.

A single layer model has been developed represented by a "bulk" hydraulic conductivity (K) of between 2x10-1 and 2x10-2 m/day.

The initial groundwater table elevation (H) has been assumed to be 1m below ground level, which represents the current level of the pit lake. Flows to the mine are calculated for a target water level (h) equivalent to the base of the mine which is at 200m below ground level. This provides an estimate of the total groundwater inflows to the mine assuming there is sufficient sump or bore pumping to keep the mine dry.

The radius of the mine workings (rp) has been calculated with consideration to the total water volume of existing workings (total of 508,919m^3). Using the maximum depth of mining (200m) an equivalent radius has been calculated such that an equivalent "well" has a volume equal to the existing workings.

Surface water inflows have been calculated using the preliminary water balance. Surface water inflows include contribution from direct rainfall and snowmelt over the area of the pit. The area of the pit has been estimated as 40,000m^2 (200 x 200m). This is a reasonable assumption for the present day conditions given that the pit is located on a watershed dividing two sub-catchments (based on preliminary catchment delineation), therefore much of the surface water will likely drain away from the pit.

The average monthly surface water inflows are summarised in Figure 20-9, which shows a daily average surface water inflow of approximately 65m^3/day. It can clearly be seen that there is a large increase in April as the temperature rises above freezing causing an increase in the contribution from snowmelt. April has the largest inflow, almost 320m^3/day. Snow melt contribution also exists in May (albeit to a lesser degree than April). Contribution from direct rainfall is predominant from April to October. The water balance indicates that there will be zero surface water inflow from November to March, primarily due to a below freezing temperature.

It is noted that an assessment of extreme rainfall magnitude events has not been undertaken at this level of study. It is recommended that this is evaluated during the next level of study for return periods 1:5, 1:10, 1:20, 1:50 and 1:100. Once this has been undertaken the water management requirements should be reviewed.

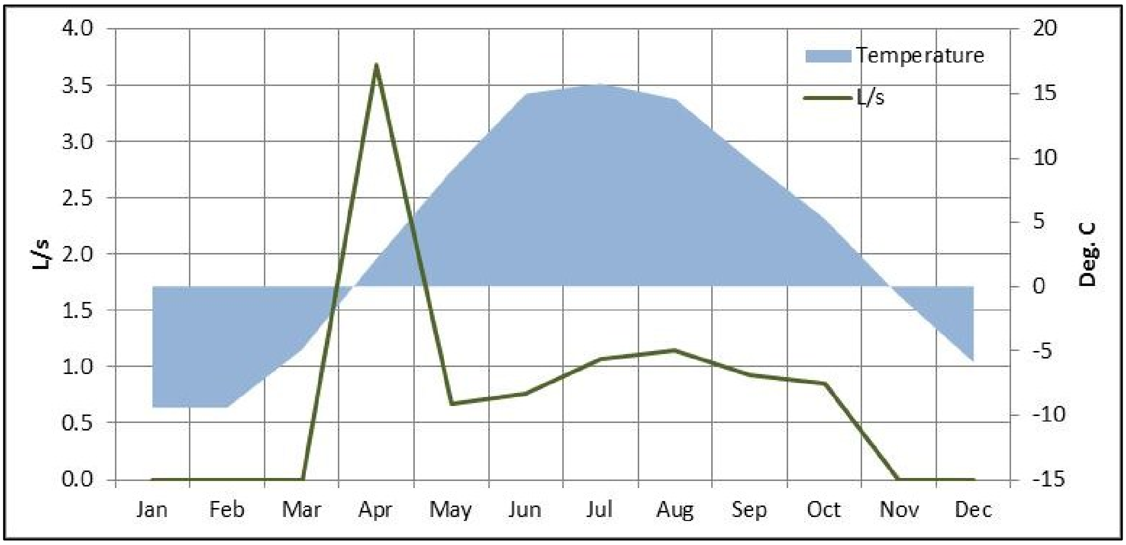

Figure 20-9: Surface water inflows (direct precipitation and snowmelt)

A summary of the estimated groundwater and surface water inflow volume into the Haveri mine is presented in Table 20-4. The large range of modelled inflows (5 to 32L/s) highlights current uncertainties in hydraulic properties at the Haveri project. The results also indicate that the majority of inflow is predicted to be from groundwater.

The estimates are considered sufficient in order to inform a PEA. SRK recommends these results be used conservatively for the scenario being investigated:

- Maximum inflows: for scenarios relating to design and costing of dewatering systems and associated discharge; and,
- Minimum inflows: for scenarios investigating potential water shortfall and make-up water requirements.

Table 20-4: Predicted final inflows for the Haveri mine from analytical modelling

	Maximum Inflows (L/s)	Minimum Inflows (L/s)
Groundwater	31	4
Surface Water (direct precipitation and snowmelt)	1	1
Total	**32**	**5**

A combination of anecdotal and photographic evidence relating to the pit lake formation following dewatering by Glenmore Highlands is described in elsewhere in this report. The evidence provides estimates of the Haveri pit lake elevation since pumping ceased in the year 2000. This information has been used to construct a bespoke pit lake recovery model to estimate the required inflow in order to allow the pit lake to recover to normal levels from the year 2000 to 2006/2008 (6 to 8 years). The results of the model are presented in Figure 20-10.

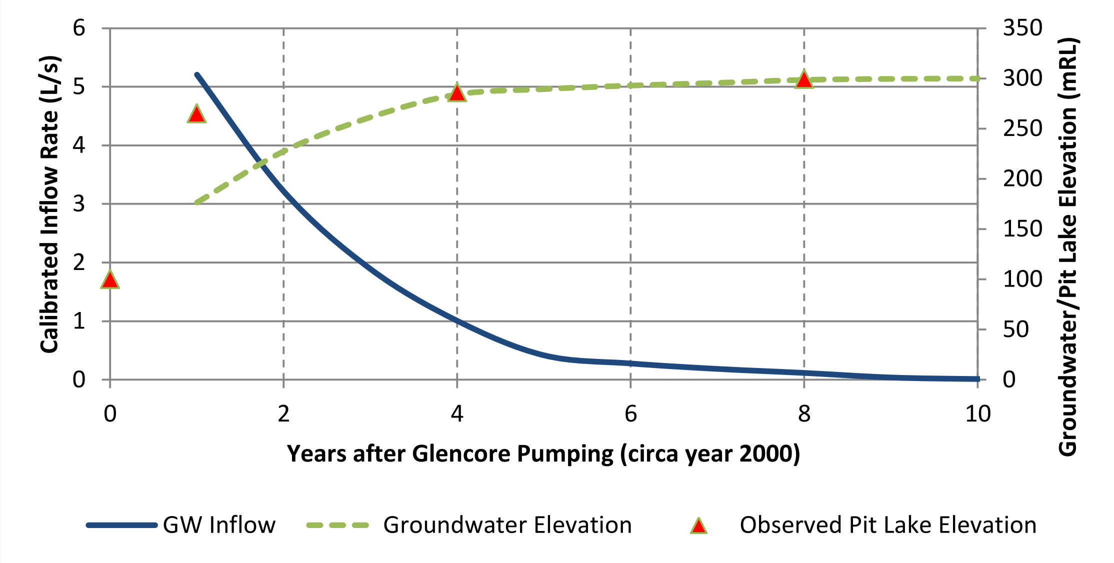

Figure 20-10: Preliminary pit lake formation model for a groundwater inflow of 450m^3/day

In the absence of detailed pumping information, it has been assumed that Glenmore Highland Inc. dewatered the entire mine of the total water volume (estimated in Table 20-3). The total volume (approximately 508,000m^3) has been represented in the model in a simplified form to a depth of 200m (the bottom depth of mining).

Surface water inflow contribution has been included as a constant of approximately 1L/s, calculated as direct input onto the area of the pit.

Groundwater inflow has been calibrated such that the modelled groundwater recovery/pit lake elevation broadly resembles the distribution of observed levels. The calibration results in a groundwater inflow of approximately 5L/s when the mine is fully dewatered (when hydraulic gradients are at their maximum). Groundwater inflow into the mine is dependent on hydraulic heads adjacent to and below the pit, the lake stage, and the aquifer properties of the rock. The groundwater inflow rate is initially high and decreases with time as heads in the aquifer approach the lake stage.

The results suggest that groundwater inflows will likely be towards the minimum range of inflows derived from analytical modelling (Table 20-4).

This evaluation has been based on very limited and uncertain data sources and therefore the results must be treated with caution. However, it may be postulated that actual groundwater inflows are likely to be in the lower range of presented values in Table 20-4.

Impacts from Dewatering

A cone of depression will surround Haveri in response to dewatering. The Theis (1935) analytical approximation has been used to calculate an estimated distance that the cone of depression may extend from the edge of the workings in order to assess the impacts on local receptors. The results are presented in Figure 20-11 after 10 years on mine dewatering (assumed life of mine). It can be seen that the cone of depression may extend between 1 and 3.5km away from the mine.

A high-level review has been undertaken to identify water courses and local borehole supplies that may be affected. Lakes in proximity to the project include Lake Kyrösjärvi (to the west) and Lake Viljakkalanselkä (to the east). The boundaries of both these lakes are within the calculated cone of depression and may therefore become impacted by mine dewatering. In reality, the cone of depression will not extend any further than the lakes as they will act as a recharge barrier.

This interpretation assumes that there is sufficient hydrogeological connection between the lakes and the mine itself. This assumption is supported by the similar water elevations of historical groundwater (see Section 0) and surrounding lakes (see Section 0).

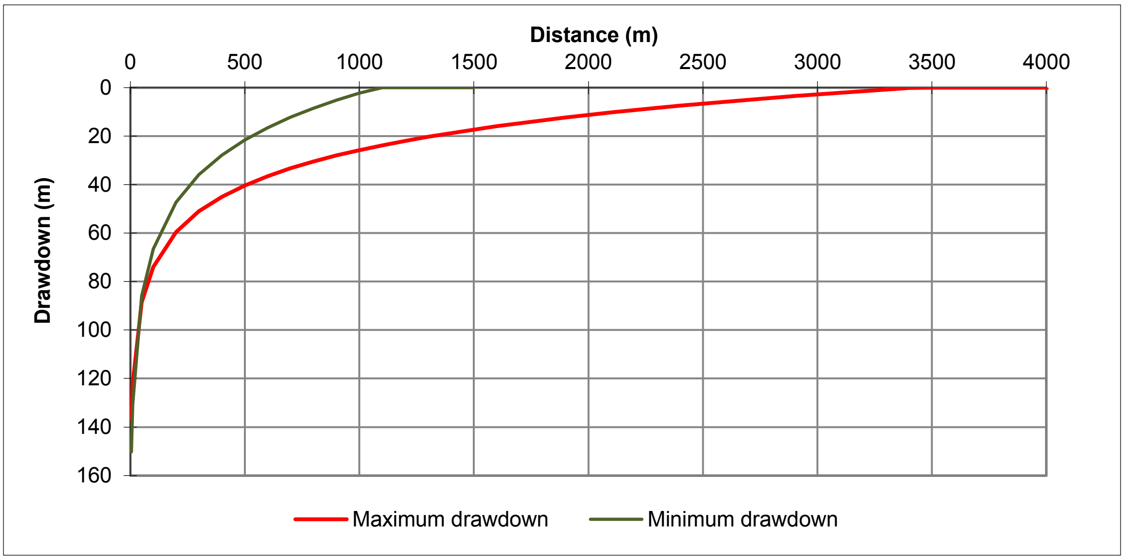

Figure 20-11: Extent of drawdown from the edge of the mine workings after 10 years of mining (Theis, 1935)

The Haveri Class I groundwater area (which the project borders) will likely be impacted by dewatering of the Haveri mine. Existing information suggests that this groundwater area is not currently in use (Section 0), however a groundwater protection plan is currently in place by the local authority. The impacts of mine dewatering and the implications this has for the groundwater protection plan will need to be reviewed during any future study.

Groundwater Management

Predicted groundwater inflows can likely be managed to ensure a “dry” pit floor though simple sump pumping. The slope angle estimations by SRK (SRK, 2014b) have been derived assuming that slopes are naturally draining and therefore at this early stage there doesn’t appear to be a requirement to depressurise. This will need to be evaluated in greater detail during future investigations.

Surface Water Management

The project area is subject to moderate intensity rainstorms which represent a potential risk to mining operations if not adequately managed. An evaluation of flood lines and corresponding flow rates and constraints will be required. These studies will allow design of flood mitigation measures needed to protect the pit and associated infrastructure as well as to mitigate any potential environmental impacts.

The first step in developing a surface water management plan is to understand flow conditions in local rivers and streams. This data, combined with the mine design and local topography, will form the basis for the definition and design of drainage or river/stream diversion requirements.

Preliminary Water Management & Cost Estimate

The groundwater and surface water inflows predicted at Haveri through analytical modelling, if properly managed, are unlikely to cause significant operational problems. However, it is important that dewatering infrastructure, including sumps, pumps and pipelines are adequately sized such that they can easily handle the predicted groundwater and surface water inflows with a suitable capacity in reserve to accommodate uncertainty.

A very preliminary estimate of water management associated capital costs (CAPEX) has been calculated in order to inform the PEA (Table 20-5). The preliminary design and costs presented here should be treated with caution and subject to complete review at the next level of study. Unit costs have been obtained from an analogous SRK study in Finland. A modular system has been costed for with several pumps used in unison to add greater flexibility to the dewatering system. Sumps should be located at several levels and water collected and pumped to ground surface by means of a series of in-line booster pumps. This preliminary dewatering design assumes there will be two sumps in total. The size of sumps has not been evaluated as part of this study.

Table 20-5: Preliminary CAPEX costs for dewatering at Haveri

Unit	Unit Cost (USD)	Quantity	Total Cost (USD)
6" Pipe with couplings USD/100	$6,130	150*	$919,563
Sump pump	$12,000	4	$48,000
Booster Pump + VFD	$68,700	4	$274,800
Flotation Unit	$2,846	2	$5,692
Tanks	$12,891	2	$25,781
	SUM (+15% Contingency)		**$1,464,912**

* 150 x 100m = 1500m

The proposed pumps incorporate VFD (variable frequency drives) units and consequently a single sump pump is generally able to handle both the low inflows during the winter months and the high inflows predicted during the spring melt. The number of pumps covers the maximum anticipated water inflow (Table 20-4) as well as additional spare pumps of equal amount. Flotation units for each sump pump have been included and water storage tanks for each booster station.

Discharge lines of 6” diameter have been considered to extend from the sumps to the pit edge. For costing purposes, a relatively large amount (1500m) has been included to cover variations in the mine design and locations of sumps. The diameter of the pipeline has not been considered in detail and will require further review at the next phase of study with regard to dewatering volumes and frictional head losses.

20.1.6 GAP ANALYSIS AND FURTHER WORK

Groundwater

Existing data is limited to public domain information, general geological reports and data and anecdotal information. No detailed hydrogeological investigations have been completed for Haveri to date. The next phase of investigation will therefore be designed to gather site specific data on the hydrogeological regime present and use this data to develop a robust conceptual model. This conceptual model will then be used to better constrain pit inflows, dewatering requirements, potential dewatering strategies and potential impacts on other water users.

In particular the following work should be undertaken in order to reach a PFS level of confidence in groundwater management requirements and costs:

- Drilling and hydrogeological testing programme;
- Development and instigation of a groundwater monitoring programme;
- Review of structural data including geotechnical logging and development of a structural model;
- Review and update conceptual hydrogeological model;
- Prediction (including numerical modelling) of pit inflows;
- Design and cost dewatering strategy (including numerical modelling);
- Water impact assessment including prediction (including numerical modelling) assessment of drawdown and potential contamination impacts;
- Assess water demand and potential water supply options; and
- Review closure issues.

Surface Water

There is very limited existing data relating to surface water flows around the Haveri area. This will need to be addressed as part of the PFS. The work required to bring the study up to the level of PFS include the following:

- Baseline data analysis and hydrological assessment;
- Catchment mapping and boundary definition;
- Assessment of water supply options with respect to available water resources;
- Long-term rainfall analysis and assessment of return periods;
- Determination of floodlines with respect to identified catchments and rainfall data;
- Development of a monthly water balance for the mine; and
- Review of surface water quality sampling and assessment of data.

It should be noted that the above list is based on information available at this time and may be adjusted according to the development path of the proposed mine.

20.2 Geochemistry

This evaluation presented here is based on the following sources:

- Environmental data service by Finland´s Environmental Administration "Hertta", http://wwwp2.ymparisto.fi/
- Spatial datasets in Finnish national geoportal, http://www.paikkatietoikkuna.fi/web/en/sdi-in-finland
- Annika Parviainen, 2012: Evolution of sulphide oxidation and attenuation mechanisms controlling acid mine drainage in decommissioned low-sulfide tailings. Aalto University publication series Doctoral Dissertations, 107/2012

- Exploration data (combined, all campaigns)
- Geological maps and material at http://www.geo.fi/en/index.html
- Marja Liisa Räisänen, Anna Tornivaara, Teija Haavisto, Kaisa Niskala ja Matti Silvola, 2013: Suljettujen ja hylättyjen kaivosten kaivannaisjätealueiden kartoitus (Mapping closed mines and mining waste areas). YMra24/2013 (Environmental Ministry's report 24/2013)

At the time of this project geochemical information is available only in a very limited form, namely anecdotal, and key waste characterisations have not yet been completed. Some recommendations considering the next steps are presented in the end of this geochemical assessment.

Acid Rock Drainage and Metal Leaching (ARDML) is probably the greatest long term environmental liability facing mining operations within sulphide deposits. ARDML arises when the in situ stable sulphide minerals are exposed to air and water through their excavation or ground disturbance. The resulting leachate produced from the natural weathering of the exposed rocks can range from highly acidic to neutral effluents and is highly dependent on both the acid generating sulphide minerals and acid consuming carbonate minerals present in the deposit. In addition, these leachates can also mobilise metals/metalloids from surrounding minerals. Even at neutral pH effluents can mobilise sufficient quantities of environmentally sensitive elements such as arsenic, as to cause an issue.

As a characterisation of new tailings (pilot concentration waste) is not yet completed, the existing tailings in Haveri have been used as an anecdotal reference for considering future environmental impacts. The total quantity of historical tailings at Haveri is 1.4 Mt, according to Environmental Ministry´s report 24 /2013. According to the same report there are also some 5,000t of waste rocks stored above the surface. SRK assumes that this waste rock is from a later pilot extraction, and not from the original mining period.

Tailings were deposited into a bay of Kirkkojärvi lake, where the dam was constructed of the overburden and rock material covering the ore. The tailings dam is 18.4 ha and 2 to 9 m deep. The bottom of the dam includes natural silts and clays. In addition, SRK also understands that parts of the latter tailings were deposited outside of the dam and as a result are in direct contact with the lake water. The tailings have not been formally covered, only revegetation attempts have been undertaken to minimise dust emissions.

Parvianinen (2012) has studied Haveri mine (and Ylöjärvi mine) tailings and reports that the tailings from the earlier mining in Haveri Cu and As concentrations are significant and exceed also the Finnish limit values for contaminated soils. In places Co and V concentrations exceeded the upper guideline values and Ni and Zn exceeded the lower guideline values. The average S concentration in the tailings was 2.9 %, whilst the highest measured S concentration in an individual sample was 6.5 %. Surficial (dry) parts of the tailings are oxidised. However, secondary Fe(III) minerals have been providing attenuation for potential impact by retaining As, Cu and Zn to some extent. Fe(III) (oxy)hydroxides have an important role in Haveri tailings as mobilised metals are both adsorbed on to these secondary mineral surfaces and are co-precipitated within the secondary Fe(III)-minerals. Yet, elevated metal concentrations (Co, Cu, Fe, Ni, and Zn) are still evident in the low pH waters near the tailings. In water samples from 2011 (Hertta service) sulphate concentrations are also elevated in the drainage channels around the tailings.

Preliminary depth sampling of pit water was carried out by SRK in February 2014 down to a depth of approximately 50 m. Only total concentrations were analysed. Some of the total metal concentrations are elevated, and the same metals are present as with the water coming from the tailings area (Co, Cu, Fe, Ni, Zn). These concentrations in connection with low pH (lowest measurement was 3.0) give indication of potentially sulphide oxidation related water quality within the pit. In the water quality data from 'Hertta', ditch number 3 represents partially the micro catchment area, where the old pit is located and partially the impact of tailings. Elevated sulphate and metal concentrations are present in this ditch, though not in as significant as in the water representing the tailings area alone.

Parviainen (2012) studies indicated a less oxidised levels within the lower tailings compared to the surficial part of the facility.

Although no ARDML geochemical assessment has been completed to date for the project the reported geology and mineralogy suggests that:

- The sulphide minerals are dominated by pyrrhotite and chalcopyrite, with trace pyrite, sphalerite and molybdenite.
- For this preliminary assessment, potential waste rock was identified within the drill total assay data and based on the current pit shell model the potential average sulphur concentration in the waste is of the order 0.85 %, with an associated copper concentration of 0.04 %.
- The host rocks seem to have relatively small neutralisation potential but there is some reported calcite (calcite-scapolite) mineralization within the mineralised zone.
- Looking at the concentrations in the exploration data in general, the most elevated elements that could occur as potential pollutants are As, Co, Cu and Ni.

20.2.1 Waste legislation

Whilst the geochemical characterisation will aid in defining suitable mitigation controls, within Finnish legislation there are also guidelines against which waste materials should be assessed.

EU Directive 2006/21/EC – Management of Waste from Extractive Industries. This Directive uses arbitrary classification to determine if waste is inert or non-inert based on its sulphur content. The Directive states that waste:

- Materials with a sulphide sulphur content <0.1% can be classified as inert, so long as other criteria for potential contaminate release are met, i.e. there is no potential for environmental impacts from metal leaching.
- Materials with sulphide sulphur contents between 0.1% and 1% may be classified as inert so long as the ratio of acid buffering/consuming to acid generating potential is greater than 3 and other criteria for contaminate release can be met.
- Materials with sulphide sulphur content greater than 1% must be classified as non-inert.
- So called PIMA-threshold values (Table 20-6) are developed for contaminated soil, but are even recommended as a reference for classifying waste from extractive industries to inert, by Ministry of Environment in the report "Kaivannaisjätteen luokittelu pysyväksi" extractive waste classification to inert)", Suomen Ympäristö volume 21/2011, where also the accepted analysis methods are addressed. Classification as inert can take place also directly according to lithology, if material includes only certain rock types, which are listed in the report named above.

The Tampere region is an area with elevated background values for certain elements. Due to this special geochemical environment PIMA –limits are difficult to apply in this specific region for As, Pb and Zn. This means, that information from background concentration register "Tapir" (administrated by the Geological Survey of Finland) should be taken to consideration in all evaluations.

Table 20-6: PIMA Threshold Values used in evaluation of soil contamination and treatment requirement evaluation according to Government decree (VNa) 214/2007 and the highest recommended regional background values for Tampere area in "Tapir".

Parameter	PIMA Value (mg/kg)	Regional (mg/kg)
As	5	**19**
Cd	1	0,55
Co	20	29
Cr	100	82
Cu	100	55
Hg	0.5	0,11
Pb	60	**77**
Ni	50	37
Sb	2	1
V	100	95
Zn	200	**208**

20.2.2 Conclusions

As a preliminary assessment of the potential wastes, based on the limited available data, the waste rock material may be directly classified as non-inert. This is based on the total sulphur concentration in the waste rock which is expected to be of the order of 0.85 %, as assessed against the EU mine waste directive. Further geochemical assessment could potentially redefine this but for the purpose of this PEA the waste rock is regarded as potentially acid generating.

The majority of the mining wastes at Haveri could potentially exceed the As, Co, Cu and Ni values noted above, due to the elevated occurrence of these elements in the rocks of Haveri area; this is considering both the PIMA threshold and regional background values. The existing tailings dam has an historical ARDML issue in the form of metal leaching to the surrounding environment and it is highly probable that the new tailings will also be acid generating due to the potential sulphide minerals present. Several metals (Co, Cu, Fe, Ni, Zn) are still currently leaching from the historical tailings.

Due to practicalities the depositing of new tailings on the top of the old tailings, is unlikely. However from an environmental perspective depositing new wet tailings on the top of old tailings could decrease the oxidation within the old tailings. However, the change in the environment (turn to more acidic or reductive conditions) might also cause remobilisation of metals adsorbed on secondary iron mineral surfaces or even the mobilisation of the secondary minerals themselves. If the water level in the old tailings lowered due to pit dewatering, to enable future mining then the oxidation of the old tailings is likely to increase. This would have a negative water quality impact and result in an increase of metal leaching and sulphate concentrations.

For the purposes of this PEA, it has been assumed that treatment and containment facilities for tailings will be required. Treatment will be needed also for the water coming from the waste rock and pits. In addition the mine dewatering could potentially result in the need for likely requirement to treat both the old tailings facility and any drainage waters. Alternatively, if reprocessing of tailings is an economical alternative, it might also have positive environmental impact in connection to the project.

20.2.3 Summary of risks related to mining waste

ARDML

Characterisation of the mining waste has not yet been completed, but both the mineralogy and total sulphur content indicates that the waste is potentially acid generating. The exploitation of a sulphide deposit is likely to be result in a similar classification for the tailings as well. The majority of the mining wastes will potentially have elevated concentrations of environmentally sensitive elements; namely As, Co, Cu and Ni. As such the mine wastes will have a metal leaching potential, with a risk of receptor impact. This potential for impact both from the operational and post-closure phases means that water treatment will probably be required. In addition, these issues will also need to be addressed within baseline and ongoing monitoring programmes.

Risks related to dust with high metal concentrations

As the mining wastes will have elevated metal concentrations in addition to the leaching potential the issue of dust control must also be addressed. This will require addressing within the ESIA, permitting and any environmental management plans. As both the mine and the waste storage facilities are going to be relative near to housing and water courses, dust management practises will be essential for the permitting.

Risk arising from draining the old tailings area for mining

The Peltosaari pit is located inside the old tailings area and operating this pit will cause a significant change in the water table within these tailings. The likely Haveri pit is further from the tailings, but the dewatering of this pit could also cause water table to lower within the old tailings. Changes within the water table will initiate oxidation of the fresh sulphide portions of the tailings and lead to significant water quality issues in form of metal leaching. This needs to be addressed both within the water management planning and monitoring. Impacts can be expected in all stages from mine preparation to post closure. Possible direct seepage from the oxidising tailings towards the lake need to be investigated as another potential risk.

20.2.4 Gap analysis and further work

At this stage, waste materials are being treated as potentially acid generating, but this preliminary classification is based only on sulphur content from the exploration database. No geochemical characterisations of the mining wastes have been undertaken. Currently, due to the lack of characterisation of potential new mining waste, it is not possible to estimate the scale of potential environmental impacts from proposed mining and processing activities.

Geochemical data availability and information gaps are presented in Table 20-7.

Table 20-7: Gap analysis

Study Area	Existing Data	Data and Information Gaps
Haveri/ surroundings: Current and historical water quality	Drainage water quality from around the tailings. Lake sediment study considering the receptor lake Kirkkojärvi. Some quantity of (public) historical and recent water quality data from receptor lake areas.	Ground water quality data for tailings area and surroundings
Haveri: Tailings	Study of existing tailings and their oxidation status.	Static and kinetic testing (tailings from trial concentration), leaching tests

Study Area	Existing Data	Data and Information Gaps
Haveri: Waste rock	Exploration data (total assays)	Static and kinetic testing of waste rock, leaching tests
Haveri: Mine water	Preliminary pit water sampling	Pit water chemistry in detail and complete stratigraphy Recovery of historical mine water data

20.2.5 Recommendations

A geochemical assessment programme is recommended. The programme would comprise both static and kinetic characterisation. This would result both absolute classification of materials, and a definition of the rates at which the predicted classification occurs. Justifiable mitigation controls can then be attained through assessment of these test results.

One option for the safe disposal of high sulphur wastes is as a cemented backfill deep within underground workings. If underground mining is considered, then the use of long term leaching tests (and possibly leaching tests for material from batch cementing tests) would be recommended for demonstrating the case-specific suitability of this disposal option. Above the ground, a cover with a hydraulic barrier is needed, if the waste is classified as potentially acid generating. Water cover (within the abandoned pits) could also be investigated as an alternative. The close proximity of water bodies must be also taken to consideration in mining waste management.

20.2.6 Water Treatment

As any contact waters from the mining operations are likely to have elevated concentrations of environmentally sensitive elements then water treatment would be a necessity for the environmentally safe operation of the mine. Mine dewater flows are predicted to range between 18 and 115m^3/hr (see hydrogeology section). No predictions have been undertaken for flows after closure or flows from the TSF.

Based on the potential detrimental elements identified within the geochemical assessment a simple pH correction and metal precipitation process may be all that is required. Depending on the iron levels present in the effluents generated some addition of iron salts may be of benefit, as these like the secondary minerals in the tailings will aid in the attenuation of the environmentally sensitive elements. The precipitation process will also be enhance by a recycling of the previously precipitated elements such as in the high density sludge (HDS) processes and could be further improved with microfiltration post treatment. The microfiltration captures the colloidal suspensions that do not settle out in the clarifier.

Assuming a total flow of 120m^3/hr a typical HDS plant, utilising lime as the neutralising reagent would cost of the order of US$1.6M (with no civils), an additional 10% of this cost would be required if micro filtration was required. Such treatment plants are in common use throughout the mining industry. The operational cost is very much dependent on the amount of lime required. Lime addition is required for both pH correction and metal precipitation. Other operational costs include flocculents and power. Typical HDS plants have an operational expenditure of US$1 to 2 per m^3 treated. The plants are fully automated and only require 1 person on the day to oversee the plant.

However, if sulphate was also found to be an issue then the additional use of a reverse osmosis (RO) unit may be required in addition. This would not be required to treat the entire waste stream as the treated purified water could be used as a dilutant for the untreated water. Assuming of the order of 50% treatment requirement then an RO plant would be of the order of US$0.75M. The brine reject from the RO could be recirculated through the HDS plant for disposal. Typical RO expenditure costs are a further US$1 to 2 per m^3 treated. This is dependent on the salinity of the water entering the treatment plant. In addition RO membranes will require replacement every 5 to 10 years. This will be approximately 50% of the initial capital costs.

20.3 Tailings Storage Facility

20.3.1 Background

SRK has completed an options assessment to identify suitable Tailings Storage facility (TSF) locations and methods for the approximately 19.8 Mt of tailings material forecast to be produced at the plant over the LoM.

Specifically, SRK has:

1. Identified appropriate tailings storage options for the Haveri Project which take into account the proposed Plant development options and location setting;
2. Completed high level volumetric calculations for each storage method in order to outline suitable footprint areas for TSF development;
3. Undertaken a multi—criteria options assessment, which considers the environmental and social impacts of each development option;
4. Prepared capital and operating cost estimates for each development option, in order to define a preferred concept for consideration the economic model (accuracy ±50%.);

20.3.2 Design Criteria

The TSF options assessment was based upon the following design criteria, which have been defined primarily from the SRK mining schedule and previous project experience at similar operations:

	Design Criteria	Unit	Value	Source / Comment
1	**Production Rates**			
1.1	Total Tailings Produced	Mt	19.8	SRK Mining Schedule
1.2	Life of Mine	Years	11	SRK Mine Schedule
2	**Tailings Properties**			
3.1	Bulk dry density of settled tailings	t/m^3	1.4	SRK Assumption
3.3	Beach angle	%	0	
3.4	Percent Solids (w/w)	%	50%	
3	**TSF Main Dam Properties at pipeline discharge**			
4.1	Freeboard requirement	m	2m	Assumed (experience)
4.2	Minimum crest width	m	10	Vehicle Access
4.3	Upstream slope	-	3:1 (H:V)	Slope Stability
4.4	Downstream slope	-	3:1 (H:V)	
4	**Sub-grade key trench of TSF Main dam**			
4.1	Width at bottom of key trench	m	3	Assumed (experience)
4.2	Depth	m	3	Assumed
4.3	Side Slope inclination	-	1:1 (H:V)	Assumed (experience)

Tailings dry bulk density and beach slope angles are the two key parameters that will have a direct impact on the storage requirements and consequently the footprint and height of the TSF. The estimated bulk dry density of the in-place tailings and the beach slope will provide the parameters required to estimate the storage volume for the TSF.

SRK assumes that the tailings will pumped to the TSF as a slurry and has selected 50% solids (v/v) as an indicative value for conventionally thickened tailings material. For the purposes of this high level assessment, a beach slope of 0% has been assumed for modelling purposes.

20.3.3 Site Selection

The site selection process used the available data with the aim of selecting economically viable sites that offer sufficient required storage capacity with minimal impact upon the local population and environment.

Using public domain satellite imagery and the Global Mapper software tool a suitable site was selected potential TSF sites with sufficient storage capacity in the area surrounding the mine. The software allows for 3D imaging which aids in the identification of valleys and natural land depressions where existing topographic contours would provide natural containment.

The following factors were considered during the site selection process:

- As the topography around the mineralised body is gently undulating, there is limited opportunity to develop valley impoundment structures; therefore surface containment methods were considered (paddock storage for slurry tailings). Wherever possible, the local topography was utilised to minimise volumes of embankment fill.
- The potential locations were limited to a radius of 5 km from the mine site as transporting soil, rock fill and tailings over greater distances would likely make the location less economically viable.
- The mine site is located on an isolated peninsula, which is surrounded by water bodies on three sides. Due to the large volume of tailings to be stored over the project LoM, there is only one suitable site located on the peninsula itself. All other options involve a 50m river crossing to reach the respective sites.
- In order to ensure that all options for TSF development we examined, sites on the surrounding land across the water body were examined (which would require installation of a pipe crossing section across the 50m wide stream).
- Due to the presence of local communities in proximity to the mine site, the potential for wind blown dust contamination is high, therefore due consideration was given to localities for tailings storage that would minimise this risk.

Based upon the assessment requirements, four potential TSF outlines were mapped within the selected 5km range, utilising the natural land contours to maximise the available storage capacity while minimising embankment fill requirements. The relative locations of location options are shown in Figure 20-12 below:

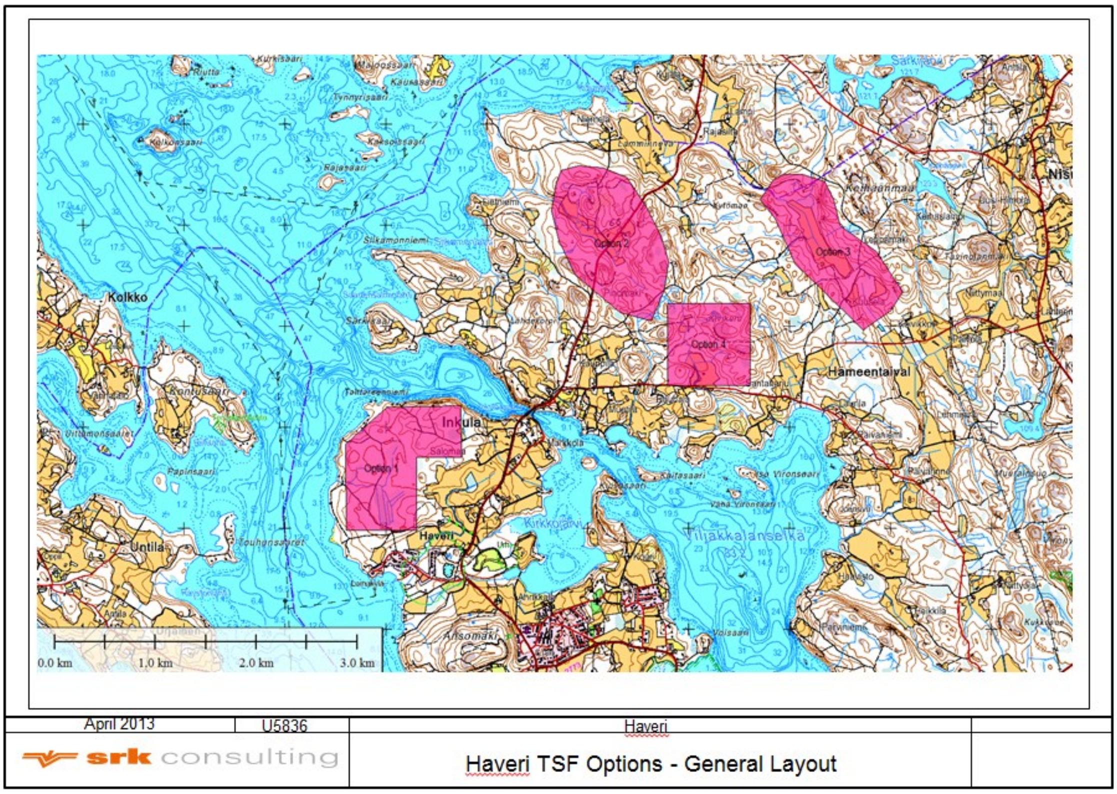

Figure 20-12: Mine Site Plan

An indicative schematic cross section showing the key features of the likely perimeter dam design configuration is included in Figure 20-13 below:

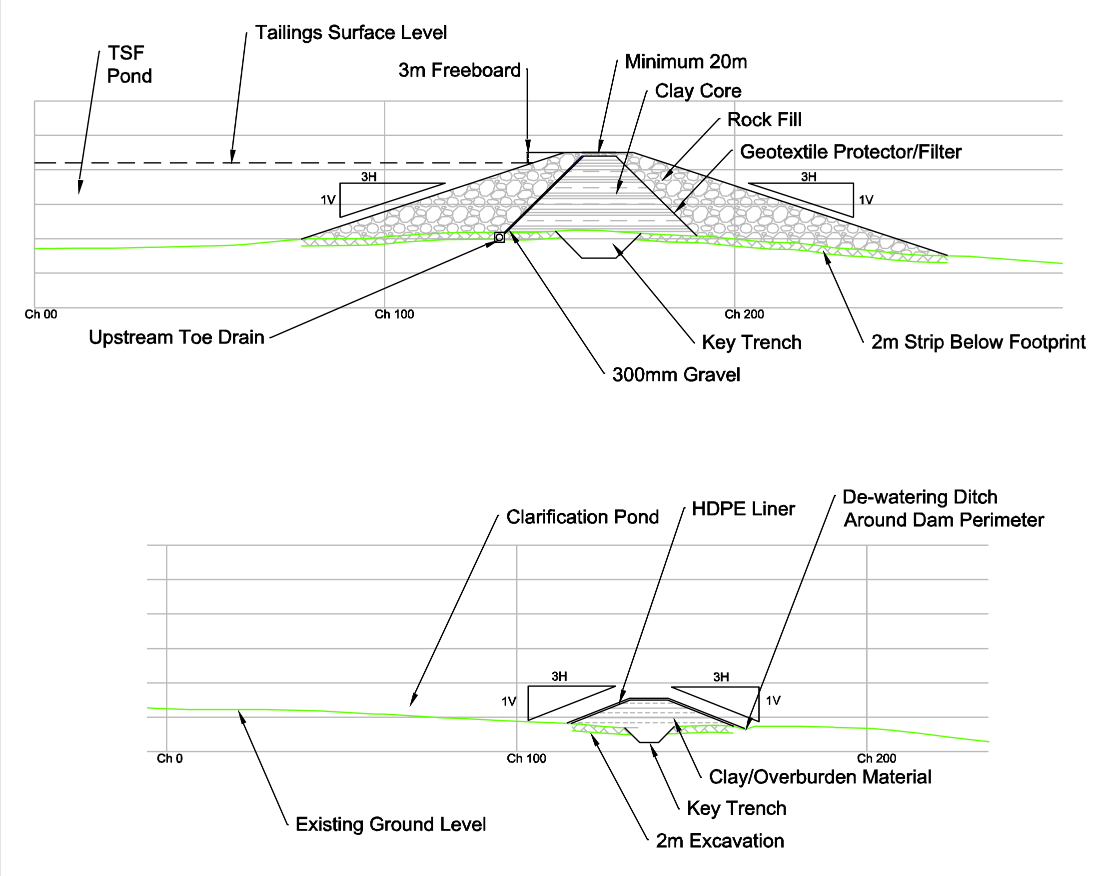

Figure 20-13: Schematic Embankment Cross Sections

20.3.4 Proposed TSF development options

The following four locations were assessed during this scoping study exercise:

Option1

- Paddock impoundment structure, located approximately 1.0km to the NE of the processing plant, on relatively flat land;
- Paddock style impoundment simplifies water management considerations, volumes of upstream runoff will be zero, due to installation of perimeter drainage channels;
- Based upon volumetric calculations carried out using Global Mapper Software, the maximum height of the facility will be 34m over a maximum footprint area of 1 km^2. The total perimeter length of the impoundment will be 5.0km;
- Starter embankments shall be constructed of locally sourced rock fill material.
- Embankment lifts shall be carried out using the downstream raise method and will consist of waste rock fill sourced from the open pit mining operation. The results of geochemical test work during subsequent phases of design should be considered when sourcing waste rock material, as there is potential for mine waste to be acid generating.
- Tailings shall be transported to the site via a 200mm diameter plain steel standard pipeline. Tailings will be distributed via a series of spigots, rotated around the perimeter embankment wall. Two 25kw pumps shall be required to transport tailings to the TSF site.
- The embankment impoundment shall be lined with HDPE, to ensure that seepage from the base is minimised during the operations phase.

- Water reclaim shall be undertaken via a floating barge decant, through which collected fluids from the impoundment will be pumped back to the plant for use as make up water. SRK assumes that a sedimentation pond will be constructed at the plant site, to ensure that suspended solid can settle out prior to re-use of return water.

Option 2

- Paddock style impoundment located approximately 3.5km north of the plant site.
- Involves a river crossing (approximately 50m length), SRK has assumed that a pipeline crossing will be required, which will be located adjacent to the existing road bridge structure.
- Based upon volumetric calculations carried out using Global Mapper Software, the maximum height of the facility will be 33.6m over a maximum footprint area of 1.1 km^2. The total perimeter length of the impoundment will be 3.7km;
- Starter embankments shall be constructed of locally sourced rock fill material.
- Embankment lifts shall be carried out using the downstream raise method and will consist of waste rock fill sourced from the open pit mining operations.
- Tailings shall be transported to the site via a 200mm diameter plain steel standard pipeline. Tailings will be distributed via a series of spigots, rotated around the perimeter embankment wall. Two 50kw pumps shall be required to transport tailings to the TSF site.
- Water reclaim shall be undertaken via a floating barge decant, through which collected fluids from the impoundment will be pumped back to the sedimentation pond at the plant for use as make up water.

Option 3

- Paddock style impoundment located approximately 5.1km NE of the plant site.
- Involves a river crossing (approximately 50m length). As per Option 2, SRK has assumed that a pipeline crossing will be required.
- The maximum height of the facility will be 36.2m over a maximum footprint area of 0.94 km^2. The total perimeter length of the impoundment will be 4.3km;
- Starter embankments will be constructed of locally won borrow material (potentially sourced from a local quarry);
- Embankment lifts shall be carried out using the downstream raise method, with HDPE liner being installed on the upstream face of each embankment raise.
- Water reclaim shall be undertaken by a floating barge decant structure, with return water being pumped back to the sedimentation pond at the plant site via a steel pipeline.

Option 4

- Paddock style impoundment located approximately 3.7km north of the plant site.
- Involves a river crossing (approximately 50m length) to site located north of the peninsula.
- The maximum height of the facility will be 45.3m over a maximum footprint area of 1.1 km^2. The total perimeter length of the impoundment will be 3.2km;
- Starter embankments shall be constructed of locally sourced rock fill material.
- Embankment lifts shall be carried out using the downstream raise method and will consist of waste rock fill sourced from the open pit mining operations.
- Tailings shall be transported to the site via a 200mm plain steel standard pipeline.

Tailings will be distributed via a series of spigots, rotated around the perimeter embankment wall. Two 50kw pumps shall be required to transport tailings to the TSF site.

- Water reclaim shall be undertaken via a floating barge decant, through which collected fluids from the impoundment will be pumped back to the plant for use as make up water.

20.3.5 Multi-Criteria Analysis

To take into account the environmental and social impacts of each proposed TSF development option, multi criteria analysis was undertaken, in which all sites were ranked based upon specific criterion.

A weighting value was assigned for the relative importance of each factor as part of the site selection process. The weighting values range between 1 (little overall significance) and 5 (high overall significance). For each factor, the proposed construction location is assigned a 'negative impact ranking' that ranges from 1 (lowest relative negative impact) to 5 (highest relative negative impact). For each selected construction locality option, the results of the assessment are presented as:

- the ranking total – sum of individual rankings from all factors considered; and
- The weighted total – sum of rankings multiplied by weightings from all factors considered.

The lower the weighed total, the more preferable the option location is for the environmental and social factors considered. A summary of results from the comparison exercise are included in Table 20-8 below.

In summary, the weighted totals show that:

- Option 3 has the lowest weighted total of all the options, primarily due to the relatively isolated nature of the site and the minimal impacts related to dust, noise and visual disturbance. SRK notes the higher environmental impact associated with increased energy use to transport tailings to this site.
- Option 2 has the highest weighted total, due to the required diversion of a main road and the disturbance of a historical site within the footprint area.
- Option 1 incurs a low weighted total, however the impacts of dusting on the local community will be significant.

The results of the comparison exercise are discussed further in the Conclusions section below.

Table 20-8: Results of Multi Criteria Analysis

Locality Name				Option 1	Option 2	Option 3	Option 4
ITEM	**CRITERION**	**WEIGHTING**					
1	Geology	4	Ranking:	2	2	2	2
2	Ground Stability	4	Ranking:	2	2	2	2
3	Current Land Use	4	Ranking:	3	4	2	3
4	Sensitive Environments / Heritage Resources	4	Ranking:	2	4	2	3
5	Surface / Ground Water	5	Ranking:	2	3	2	3
6	Visual impact	3	Ranking:	3	4	3	4
7	Nuisance Dust	4	Ranking:	4	3	2	3
8	Nuisance Noise	4	Ranking:	3	4	2	4
9	Consequence of pipe failure between processing plant and TSF	5	Ranking:	2	4	4	4
10	Disruption of existing / current transport routes (existing public roads, rail, etc)	3	Ranking:	2	4	2	3
11	Public safety and environmental issues relating to the TSF side slope failure zone of influence	5	Ranking:	3	4	2	4
12	Energy usage and carbon footprint (power of pumping system, haul route length, etc)	5	Ranking:	2	3	4	3
			Unweighted Total	**30**	**41**	**29**	**38**
			Weighted Total	**124**	**170**	**123**	**159**

RANKING TABLE:	
Highest relative negative impact	5
High relative negative impact	4
Moderate relative negative impact	3
Low relative negative impact	2
Lowest relative negative impact	1

20.3.6 Cost Estimation

SRK has prepared capital and operating cost estimates for development of the tailings facilities, the unit costs are based upon project experience in the region.

In order to assess the cost implications of each TSF development option outlined above, a conceptual level (±50% accuracy) CAPEX and OPEX estimate was prepared for each. The following items were included:

Capital Expenditure

- Clearing and grubbing of embankment footprint area, including removal of all shrubs and grassland;
- Embankment Earthworks – Bulk excavation of 2.0m of material around the entire starter embankment perimeter area, upon which the embankment can be keyed into the natural ground surface;
- TSF starter dam construction – The volumes of imported fill for perimeter starter embankment construction have been calculated based upon sufficient storage for a period of 2 years. A 10m wide crest has been assumed with 1V:3H slopes on either side.

- Perimeter Ditch Construction – Perimeter ditches shall have be 2.0m wide at the base, 1m wide and have a side slope of 1V:1.5H;
- Floating barge decant system – In order to safely remove supernatant fluids from the TSF, a floating barge decant system has been included in the costing exercise ;
- Tailings delivery system construction (pipeline for slurry transport) – SRK has assumed that a single 200mm plain steel standard wall pipeline will be used to transport tailings from the plant to the TSF.
- Tailings Pumps – Based upon the annual throughput centrifugal pumps have been included as part of the slurry distribution network. These have been sized based upon predicted slurry characteristics and distance/head calculations from the plant to the TSF site.
- Water return pipelines and pumps from the TSF to the plant site clarification pond.

OPEX:

- Installation of additional embankment raises over the LOM. These will be constructed of mine waste material generated from the open pit operation (pending the results of geochemical characterisation of this material).
- Installation of additional HDPE sections on the upstream face of each embankment raise to ensure that seepage is minimised through the embankments.
- Maintenance, labour and power costs for the pipelines and pumps per annum over the LOM;
- Haul road maintenance over the LOM.
- Perimeter ditch extensions around the TSF perimeter during downstream raises.

20.3.7 Labour Requirements

SRK has estimated the following labour requirements for the TSF operation, based upon experience of similar scale projects. These are summarised in Table 20-9 below:

Table 20-9: Haveri TSF Labour Requirements

Category	No. Personnel	Notes
Tailings Superintendent	1	Permanent (In Country)
Mechanical Engineer	1	Permanent (In Country)
Surveying Field Crew	4	Part Time (As Required)
Dozer Operator	2	12hr Shifts (1 Dozer)
Front End Loader Operator	2	12hr Shifts (1 Loader)
Excavator Operator	2	12hr Shifts (1 Excavator)
Water Cart Operator	2	12hr Shifts (1 Bowser)
Supervisor	2	12hr Shifts (For construction only)
Leading Hands	2	12hr Shifts (For construction only)
Field Assistants	4	12hr Shifts

20.3.8 Capital and Operating Expenditure Summary

The CAPEX and OPEX outcomes of the trade-off study are summarised in Table 20-10 below:

Table 20-10: CAPEX and OPEX summary for TSF development options

Case	CAPEX (USD)			OPEX (USD)	
	Total Earthworks	Total Pipelines and Pumps	Total	Total	USD/tonne tails
Option 1	16,736,085	818,520	**17,554,605**	**20,541,322**	**1.0**
Option 2	8,341,462	2,039,360	**10,380,822**	**10,731,622**	**0.6**
Option 3	6,601,894	3,268,440	**9,870,334**	**10,062,257**	**0.5**
Option 4	11,022,015	2,488,440	**13,510,455**	**29,656,385**	**1.5**

20.3.9 Conclusions

Introduction

SRK has prepared a conceptual level study, which details four tailings storage development options for the Haveri project. Based upon an assessment of conceptual level capital and operating cost estimates (±50% accuracy) for each, Option 3 incurs the lowest CAPEX/OPEX costs, due to the significantly lower volumes of embankment fill and associate earthworks required for development.

Despite the large CAPEX associated with development of Option 1, this location is situated within the mining licence area and would minimise overall environmental disturbance. There is potential to optimise the CAPEX and OPEX expenditures associated with starter embankment installation and subsequent embankment raises by dividing this structure into a series of smaller cells. This option should not be discounted until earthworks optimisation measures are considered in more detail at PFS level.

Based upon multi-criteria analysis, Option 3 also incurs the lowest risks associated with environmental and social impacts. SRK notes however, that this assessment is based upon preliminary review of public domain information and that a site specific assessment of these risks should be completed at the next stage of design. Option 1 incurs similar level of risk, this option should be considered at PFS level, as there is potential to significantly reduce CAPEX by developing Option 1 in a series of smaller cells.

A key design consideration for the TSF is dust management; SRK recommends that dust management measures are considered during the next stage of design. Due to the remote nature of the Option 3 site, the impacts of dusting on local communities will be minimised.

Despite the relatively low CAPEX and OPEX associated with Option 2, the requirement to divert a main road and disturbance of historical sites in the area has lead SRK to discount this Option.

Option 4 incurs significant OPEX penalties associated with embankment raises and due to proximity to the main road, has been deemed less suitable for development.

Risks and Opportunities

- Tailings Thickening – SRK has assumed that tailings can be produced with a 50% solids content (w/w). Should lower solids content be realised, the volumetric calculations for the slurry impoundment and associated earthworks quantities will need to be revised accordingly.
- SRK is not aware of current surface water management practises in each area, which could have an impact upon TSF development. Local land use patterns must be determined prior to further assessment of Options 1 and 3.

- This study assumes that the tailings material has the potential to be acid generating or exhibit metals mobility. A liner system has therefore been included in the initial costing exercise, to ensure that the risk of groundwater pollution through seepage is minimised.
- SRK recommends that a full geochemical assessment is carried out during the PFS stage, in order to characterise the mine waste material generated from the open pit operation. Should this material prove to be net acid generating, then alternative borrow sources may be required for embankment fill, which will significantly increase OPEX expenditure for all options.

Recommendations

At PFS level, the following items should be considered for further investigation:

- SRK recommends that Options 1 and 3 are investigated in more detail during the PFS phase. Option 3 incurs the lowest CAPEX, however this development option may have unforseen risks due to surface water regimes be disrupted in the area. In addition, the costs associated with pipeline installation and maintenance could be increased, due to unforseen ground conditions etc. Whilst the CAPEX associated with Option 1 is high, optimisation measures such as sequential cellular development should be investigated. Land acquisition and development at this site will be significantly more straightforward, due to its position on the current mining licence. In addition, Option 1 does not require a pipeline crossing over the river.
- TSF volumetric study – Once an accurate mass balance for each processing option has been established, the sizing and configuration of each TSF arrangement should be re-visited, in order to optimise the height versus area of each impoundment.
- The current land-use, land purchase and permitting status of all sites should be verified, to ensure that there are no fatal flaws associated with development of the TSF on any of the aforementioned locations. Current surface water management practices should be examined in particular, to ensure that current water use practices are not influenced adversely by development at sites 1 or 3.
- Environmental Impacts – In order to minimise dusting over the LOM, sequential cellular development should be considered for Option 1, to allow progressive rehabilitation through the LoM. Such measures would include capping of sections of the TSF with a soil cover system.
- Site Investigation – A preliminary site investigation should be carried out at the proposed TSF sites, to accurately characterise the foundations for design. The field investigation should consist of trial pitting and boreholes, so that samples can be extracted for geotechnical test work.
- Stability/Seepage Analyses – Limit equilibrium stability analyses should be carried out for all slurry impoundment structures, to ensure that the facilities are stable under both static and seismic loading conditions.
- The costs associated with a pipeline crossing over the river to the north of the mine site should be examined in more detail during the next stage of design. This could have a negative impact upon the CAPEX estimates for the distal options outlined above.

20.4 Environmental and social Impact Management

20.4.1 Environmental and Social setting

Haveri and Viljakkala villages are the nearest communities to the Haveri and Peltosaari pits. Both villages are located in Ylöjärvi Town in the Pirkanmaa Region, which is part of Länsi-Suomen lääni (Province of Western Finland).

Haveri and Viljakkala villages and planned Haveri and Peltosaari pits are located on a peninsular in Lake Kyrösjärvi between the Viljakkalanselkä and Haverinselkä sections of the lake, east and west of the cape respectively. Lake Kyrösjärvi is the surface water receptor for Haveri mine. The lake is located in the lowest part of the Ikaalinen sub-catchment, which belongs to the greater Kokemäenjoki River catchment area. It belongs to the 'Kokemäenjoki-Saaristomeri-Selkämeri' watercourse management planning area (watercourse management plans set catchment specific objectives, which authorities must take into consideration during permitting).

The latest publicised water quality data for Haverinselkä is from the late 1990´s. Haverinselkä water had a pH 6.8, electrical conductivity (EC) 4.8 mS/m, total nitrogen concentration 700 µg/l and oxygen concentration 8,5 mg/l (15.8 C°) at the surface and pH 6.3, EC 5.0 mS/m, total nitrogen (N) concentration 760 µg/l and oxygen concentration 5.4 (10.4 C°) at a depth of 21 m. Metals were not analysed.

Lake sediment studies carried out by Parviainen (2012)5 show evidence of historical mining related impacts at Kirkkojärvi and Viljakkalanselkä bays. From the mid-sixties to mid-seventies (soon after mine closure), Cu concentrations in bay sediments were elevated but have decreased gradually. Currently Cu concentrations remain higher than background levels. Ag, As, Cu, Ni, and Zn also had short term peak concentrations after mine closure, but Cu, Fe and S have remained elevated over the long term. The cause has been attributed by Parviainen (2012) to oxidation of the tailings after the top layers of the material dried when the mine closed. Use of the tailings storage facility ended before current waste legislation, which means the original mining company (if it still existed) would be unlikely to be economically liable for any required closure measures. SRK's understanding is that Ylöjärvi Town is the current land owner and is coordinating management associated with the old mining areas. Palmex will need to clarify the legal status of any historical liabilities with Ylöjärvi Town during the next phase of the Project development.

Part of the proposed mine area occurs within a classified groundwater resource area. Further detail is given in Section 20.1. Due to elevated metals in the water, an alternative water supply is currently used by Viljakkala and a long term solution is being investigated by the municipality and Water Company.

The nearest creeks in a natural state are north of the Haveri groundwater area. These creeks form a micro catchment, where the main creek (located approximately 0.5 km north of a proposed pit) flows westwards to Kyrösjärvi. There are impacted creeks and ditches in contact with the historical tailings dam, although the upper part of one of the creeks is in a relatively natural status. This creek is north of the proposed mine. No springs are thought to occur in the immediate surroundings of the mine or historical tailings dam but this must be confirmed with specialist investigations.

[5] *Annika Parviainen, 2012: Evolution of Sulphide Oxidation and Attenuation Mechanisms Controlling Acid Mine Drainage in Decommissioned Low-Sulfide Tailings. Aalto University Publication Series Doctoral Dissertations,107/2012.*

Haveri is in the Boreal Forests/Taiga biome characterized by coniferous forests consisting mostly of pines, spruces and larches[6]. The nearest protected Natura 2000 area is Alhonlahti (Figure 20-14). The area, located approximately 1.5 km southeast of Haveri, is a shallow bay in the Lake Kyrösjärvi catchment, which has conservation importance as a wetland for bird life. It is surrounded by housing and agricultural land. A road transects the wetland. Approximately 2.5 km south of Haveri are 2 additional bird water protected areas.

According to the draft Ylöjärvi Town Nature Protection Plan (2013), Ansonmäki outcrops, approximately 0.5 km south of Haveri, has conservation importance as a geological site, as does a few herb-rich forest patches on southern and eastern slopes. These areas are recorded in the currently valid Pirkanmaa Regional Plan as areas of conservation importance. The area also serves as an outdoor recreational area for Viljakkala (a village located ca 0.5 km from the mine).

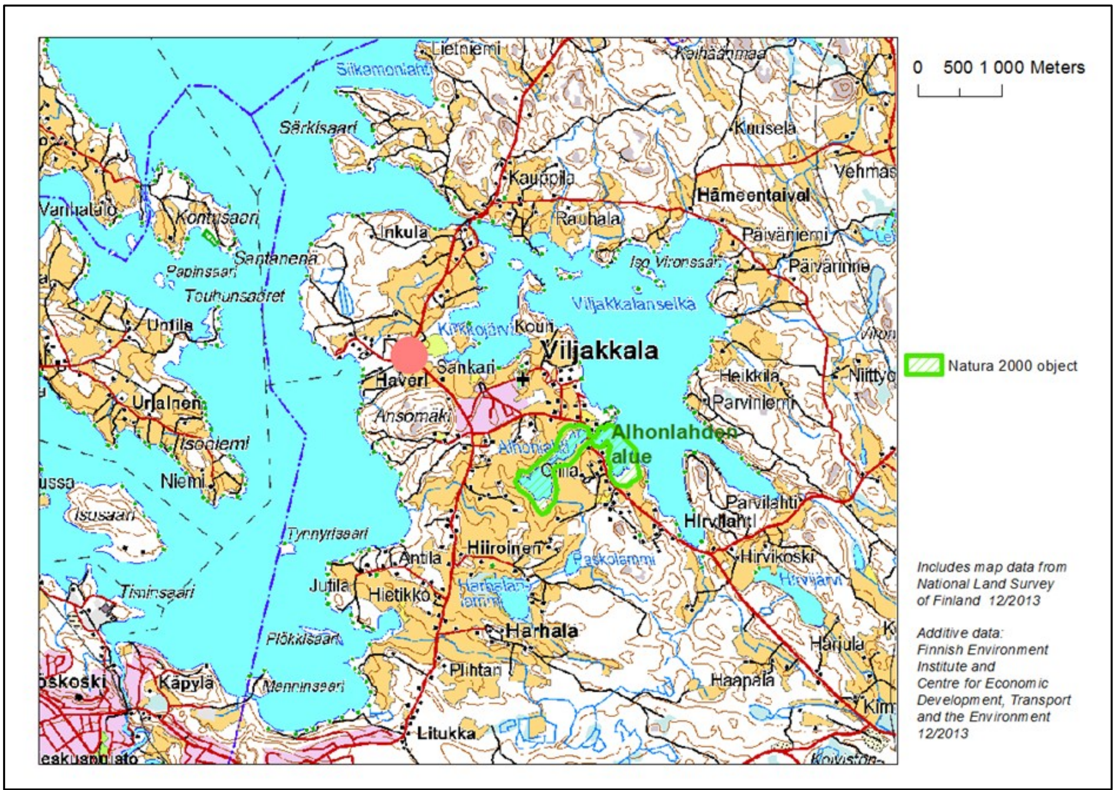

Figure 20-14: Natura 2000 areas in the vicinity of Haveri

The project's field of influence will extend to agricultural and forestry land and built-up areas.

In Finland, both existing land uses and future land use reservations are dictated by Regional and Local Plans administered by authorities. Haveri and Viljakkala villages and the proposed mine are included in a town plan administrated by Ylöjärvi Town. However, the proposed mine's field of influence may extend outside the area covered by the Ylöjärvi Town Plan. Pirkanmaa Regional Council's Regional Plan applies to the whole area. Figure 20-15 shows land uses in the Regional Plan.

[6] *Wildlife Finder, after World Wildlife Fund - http://worldwildlife.org/science/wildfinder/.*

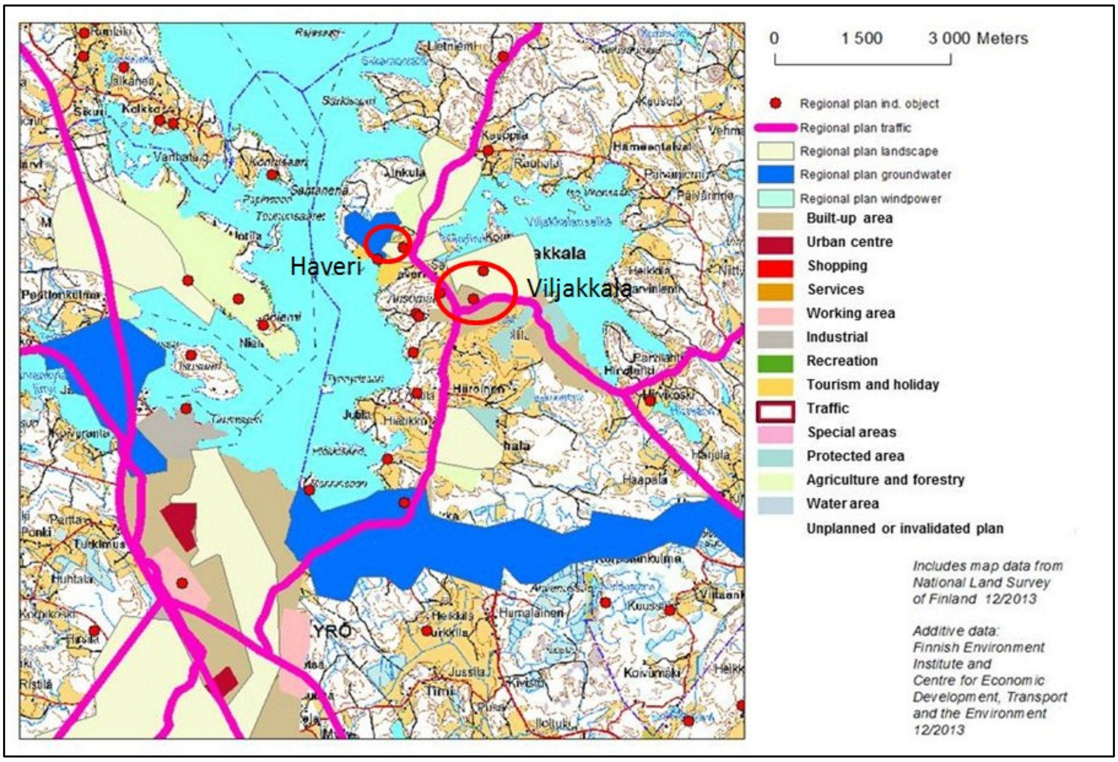

Figure 20-15: Land uses in the vicinity of the Project as designated in the Regional Plan

There are currently no land use reservations for mining in the Ylöjärvi Town Plan or regional plan. Land potentially impacted by the proposed mine is classified as natural or cultural landscape, tourism, roads and groundwater resource in the Pirkanmaa Regional Plan. South of the Project area there are reservations for nature protection/conservation importance mentioned earlier. The agricultural area north of Haveri is assigned in the Pirkanmaa Regional Plan as an ‘agricultural area with special environmental values’. East of Haveri, where the proposed Peltosaari pit is located, the historical tailings dam is partially within the footprint of the pit. Just west of Haveri village, there are tourist facilities (Kulta-Haverin Vapaa-aika and Kulta Casino). Significant parts of the tourist facilities are inside Haveri pit borders. Some homesteads are within the footprint of the pit.

The population of Viljakkala village south-southeast of the Project was 569 in 2011 (unofficial statistics). The permanent population of Haveri village includes a few households. The official population of Viljakkala postcode area is 1038 people, but this covers the whole peninsular and even some areas (Hämeentaival) on the other side of the lake, north of Haveri. The shortest distance between areas historically disturbed by mining and Viljakkala households is 500 m.

No detailed statistics are available about the livelihoods on Viljakkala and Haveri communities. In Ylöjärvi Municipality the main livelihoods are industry, health and social services, trade and construction. In Viljakkala agriculture is likely to have a larger role than in Ylöjärvi Town. Tourism is also a source of income in Haveri. No reindeer husbandry takes place in this part of Finland.

The most important cultural monument in the area is the historical Haveri Mine with its buildings and museum. The historical housing area near the mine (within the proposed Haveri pit footprint pit) is recognised as a high value object in the recent inventory of built cultural environments administered by the Museum Authority. This inventory is significance because it forms part of the basis for regional planning. According to a local resident, there is private graveyard site located south west of the Haveri pit in real estate 980-441-1-11, which is partly inside the planned Haveri pit borders.

20.4.2 Environmental and social approvals

It is understood the authorisation process to obtain environmental and mining permits has not been initiated. Some consultations with stakeholders have taken place. The Project will have to follow the Finnish legislative framework (Appendix A) to obtain an environmental permit for the mining development.

The estimated duration of the environmental authorisation process, excluding an appeals process, is as follows:

- Best case – 19 months;
- Probable (average) case – 27 months; and
- Worst case – 36 months.

A detailed description of the steps required to obtain the environmental permit is given in Appendix A, together with typical linkages to the project development study phases.

20.4.3 Management system

Palmex does not yet have an environmental, community or sustainability policy and at this early stage of the Project, there is no formal Environmental and Social Management System (ESMS) in place. Initial management by the company has focussed on stakeholder engagement, as described below.

20.4.4 Stakeholder engagement

Palmex Mining Oy has carried out consultation with local landowners and other interested local parties by means of a single public information event in support of the exploration permit and PEA. A separate meeting was arranged with Ylöjärvi municipality.

During the public meeting, people expressed interested in, for example: land-owner payments related to exploration work, water impacts and mining waste. They also wanted to know if there is uranium in the mineralisation as the presence of uranium in mining waste has been discussed in Finnish media during the last few years and therefore become a common subject of suspicion. Locals/summer cottage owners in the Osara area (on the other side of the lake looking from Haveri), have expressed that they oppose exploration activities in Osara. (Possible mineralisation in the Osara area is not included in this study.) In the Haveri area at least one cottages owner expressed worry in the public meeting and wanted information of potential mine´s impact on cottage owners.

20.4.5 Environmental and social issues

Based on the review undertaken by SRK, the principal substantive environmental and social issues relating to the asset are listed below.

Class 1 Groundwater Resource

Mining may impact on the Haveri Class 1 groundwater resource (in glacial overburden), which is designated as a resource for potable use. Groundwater from this area is currently not utilised, but a nearby pumping station has status as a backup water plant. The protected groundwater formation area for this pumping station is partly inside the planned Haveri pit borders. Part of the groundwater protection area is also in the least utilised part of the cape and could be topographically suitable for future tailings storage.

Although future use of the resource is likely to have a low priority to Ylöjärvi Town, mining the

Haveri pit (and possibly Peltosaari pit) would require administrators to take a final/permanent decision to relinquish this groundwater resource for potable use and to register the change of protection status, adding the Project's administrative requirements, with subsequent consequences on the cost of relevant supporting studies and time needed to obtain the necessary permissions to proceed with operations.

Current Land Use Amendment Status

For environmental permitting it is necessary that planned operations are not in conflict with existing land-use plans. Mining is not included in the currently valid town or regional plans.

The regional plan has a strong legal status. It is not possible to issue an environmental permit for a development (for example a mine), which may compromise the aims of land use plans. Because of this, the process to explore possibilities to adjust the regional plan must be explored early in the Project development. Changing the Regional Plan must be initiated by the Pirkanmaa Regional Council. Mining project´s compatibility to the national land use objectives is also evaluated in this land use planning review process. Updating the regional plan is a slow process with extensive stakeholder engagement, which in a typical case can take years. It can however be timed, in co-operation with the relevant authorities, to take place simultaneously with project´s EIA and permitting processes. This way work done for the EIA, including stakeholder engagement, will also support the regional planning process.

For major building activities, including mining, a valid town/zoning plan is needed and these plans will have to be initiated or changed accordingly, further adding to the administrative requirements. 'Suunnittelutarveratkaisu' (a planning decision) by the Ylöjärvi Town will be required to start the process. This means Ylöjärvi Town must make a decision to initiate a local planning process for the relevant area. Communication between mining company and municipality leads to preparation and presentation of the issue to the relevant council, which can take the decision to start the process. Town and zoning plans must not disagree with the regional plan, which means the regional plan update process must be taking place at the same time.

Protected Alhonlahti Natura 2000 Area

Although the Alhonlahti Natura 2000 area is only 1.8 km from the Project, SRK considers impacts on the are unlikely based on a preliminary review of topography and probable surface and groundwater flow paths. Alhonlahti bay is upstream of any potential mine water discharge points considering a preliminary assessment of flow direction in Lake Kyrösjärvi. However, there is insufficient information available at this stage to categorically preclude mining impacts on the Alhonlahti. Considering the Natura 2000 area is a bay in the receptor lake, there is a possibility that extensive investigations will be required for environmental permitting. A formal Natura 2000 assessment will be probably required. The need for formal Natura 2000 assessment must be formally confirmed from Pirkanmaa Centre for Economic Development, Transport and the Environment (ELY).

Disturbance of Viljakkala and Haveri Communities

The mine will likely result in dust, noise and vibration disturbance to Viljakkala and Haveri area households. Blasting will be a key factor (together with loading, transport and crushing) in dust emissions and noise. There are some natural geographical barriers between Viljakkala village and Haveri pit area, but no significant geographical barriers to prevent dust emissions from Peltosaari pit area towards Viljakkala village. Noise, dust and vibration impacts on the affected communities will need to be assessed in the EIA and if necessary mitigation put in place to minimise impacts. Alternatively the communities and/or affected individual households will need to be relocated. Osara and Hämeentaival areas on the other sides of the lake may also be partly within the impact area.

Size of the Safety Buffer Zone

In recent years authorities have required larger safety buffer zones than in the past (to ensure community safety from blasting, noise, chemicals on site etc). A number of factors affect the size of the safety buffer zone, as discussed below.

The nearer the open pit (or underground mine portal/shaft) to houses, the greater restrictions imposed. Because of general explosive safety regulations designated in Explosives Decree 1993/473 (not specific for mining), multiple charges are not allowed and individual charges are limited to approximately 1 kg within 200 m distance of houses. Using large charges near the surface requires large safety zone. The geotechnical characteristics of the area will also have an impact on the size of the safety buffer zone. These restrictions could impact on mine planning and result in loss of access to some of the mineralised material, or require additional relocation to be undertaken to enable access, which has a cost and social implication.

Safety buffer zones are addressed in updates to local and regional plans. If the Project is classified by authorities as ‘Seveso-plant’, then the safety zone determination in regional/local plans is likely to result in a large buffer zone. The classification status depends on the types and quantities of chemicals stored in the operation area, defined in Directive 96/82/EC. There are always some Seveso-classified chemicals used in mining operations, but the stored quantities on site are the determining factor for classification. Based on preliminary information, Haveri is not a large scale Project and thus may fall outside of Seveso-requirements.

Based on the pit shell model for this PEA, the shortest distance from the (Peltosaari) pit borders to the Viljakkala village is approximately 400 m. The specific number of households impacted will have to be investigated based on the actual extent of the buffer zone requirement and on site specific impact evaluation. However, the majority of Viljakkala centre households and a few houses on the northern side of Haveri (including a holiday village) are directly affected or within 1000 m of the pit borders. Some of these households may have to be relocated to allow for sufficient buffer zone.

The total scale of relocation and compensation issues cannot be defined without at least preliminary impact models for noise, dust and vibrations, and evaluations related to blasting safety. Negotiations related to relocation need to start early in the process as compensation needs to be addressed as a significant issue to avoid delays.

Acid rock drainage and metal leaching

There is a risk waste materials (both tailings and waste rock) will be classified as potentially acid producing which may need to be controlled by both operational mitigation measures and closure measures. Also future pit lakes might contribute to acid rock drainage and metal leaching. The risk of acid rock drainage and metal leaching is described in more detail in Section 20.2.

If mitigation measures are inadequate, acidic water with elevated metals could reach the surrounding environment, including both ground water and the receptor lake. Mitigation measures may relate to both the waste rock dump and the tailings storage facility design. For example, a liner or other base with similar characteristics may be required. A water treatment facility may need to be considered. In the closure stage, possible mitigation measures include both cover and passive water treatment.

Closeness of water

The proximity of the mine operations to natural water bodies is a risk for the permitting process. Generally a buffer zone is required between the open pit and the edge of a water body. This requirement is based on water discharge, dust transport and safety perspectives. For example boat traffic restrictions in water bodies may be required considering for example the risk of falling rocks from blasting. The extent of the required buffer zone will need to be based on site specific evaluation.

Aesthetic impacts

The TSF and waste rock dumps are likely to be visible to affected communities including Osara across the lake. The Peltosaari pit is likely to be visible from the Viljakkala village and Haveri pit is likely to be visible from the Osara area.

Relocation of historical sites

The Haveri historical mine headgear and related housing area will have to be relocated because they are in the footprint of the proposed pit. The building remains of the historical Haveri mine currently has protection status and may therefore already be a permitting risk. The historical housing area will likely be recognised in the next update of Pirkanmaa Regional Plan. Investigations and authority communications should be initiated as early as possible to avoid delays. The potential relocation of a grave site also needs to be investigated and, if necessary, the appropriate permits obtained to move this. To avoid delays, investigations and negotiations should be started simultaneously with the EIA process.

Lack of space

Few alternatives are available for locating infrastructure on the peninsular due to a lack of land space, so there will be competition for land use between urbanisation, groundwater resource and mining. Alternatively tailings must be located outside of the peninsular.

Management of the historical liabilities

The Peltosaari pit is located in the footprint of the historical tailings storage facility. Dewatering this pit may result in significant changes to the water table within the remaining parts of the tailings on the site (parts of the historical tailings storage area west from planned Peltosaari pit). Depending partly on the final use of the historical tailings area in mining operations, the historical tailings may or may not cause additive costs in managing the acid rock drainage and metal leaching from larger area during the preparation, operation and closure of the mine. Risks related to the historical tailings are addressed in more detail in section 20.2.

The Project could be (partly) liable for the historical tailings storage facility and associated environmental and social issues if the Project impacts on the tailings. Operating or changing the facility together with possible future land ownership may cause changes to the current liabilities. A legal assessment of these and possible other historical liabilities in future mining scenario is needed to address this issue.

Transportation to existing process facility

Whilst off-site processing is not being considered as the base case, SRK notes that the shortest transport route to the nearest Concentrator in Sastamala passes through the centre of Kyröskoski (a town 10 km from the mine). Truck movements may disturb people who live in the area or use its services (in terms of noise, visual and increased traffic congestion). There is also one bridge with low weight limits along this route which may need strengehtening, thus interaction with the relevant authority (Traffic Administration in ELY centre) should be commenced early to determine who will be responsible for any necessary upgrades. The alternative route without town centre or bridge-related complications is approximately 4 km longer.

20.4.6 Closure requirements and costs

Legal requirement and economic deposit related to the closure are described in Appendix A. The main requirements of European Directive (EU) 2006/21/EC are:

- The first closure criteria group includes requirements for general safety. Physical stability of any remaining structures must be acceptable from the perspective of long term climate, individual floods and cumulative impacts, frost and smelting, erosion and degradation processes of disposed materials.
- The second closure criteria group concerns cover, chemical and biological stability. Planned post closure land-use must be suitable for surrounding land use requirements and landscape.
- The third closure criteria group addresses social and economic requirements, which are widely defined in the local legislation but should result in minimal negative impacts on socio-economic environment.

Site specific closure objectives will need to be defined in a conceptual closure plan as the mine development process proceeds.

From a cost perspective the most critical parts are usually requirements relating to the chemical and/or mineralogical quality of the mining waste and associated cover (or backfilling or other relocating) costs. Because of this the preliminary cost evaluations in Table 20-11 only covers mine residue deposit closure. Aspects such as demolition of the plant, closure of the mine workings, social closure costs, post closure monitoring and maintenance and closure management and engineering have not been costed for at this time. A conceptual closure plan addressing all aspects of closure will need to be done during the PFS to enable adequate provision in the financial model. Full closure costs are likely to be in the low 10s of millions of Euros, possibly in the order of Euro 20million, depending on the outcome of the project engineering and ESIA.

The preliminary mine residue deposit closure costs consider the waste as potentially acid generating, however no mining waste characterisation has taken place so these numbers may be conservative. If mining waste is confirmed as acid generating, then the waste will require cover with a sealing layer and protection/growing media layer. Depositing waste rock in a location where there is permanent water cover (backfilling) could be investigated as an alternative.

At cessation of mining operations the active water treatment plant may need to be replaced with suitable passive treatment schemes. Such schemes will minimise on-going operational costs as they require minimal external input once operational. If the water table is lowered in the old tailings area for mineral extraction, without reprocessing the old tailings, oxidation of old tailings may add to the water treatment requirements. Primary mitigation measures may become necessary to limit the transport of metals released in sulfide oxidation (caused in old tailing by the Project). At this scoping stage closure costs relating to the old tailings, other than water treatment, have not yet been addressed.

Table 20-11: Preliminary mine residue closure cost estimate, euros

Item	Type	Total Cost (Euros)	Percentage Cost
Cover for TMF	Implementation	5 850 000	64%
Cover for waste rock	Implementation	2 370 000	26%
Demolition concentrator, other structures and management of contaminated soil	Implementation	440 000	5%
Water treatment	Implementation	150 000	2%
Additional investigations, authority work	Work	60 000	1%
Monitoring, maintenance	Work	200 000	2%
Total Cost	**-**	**9 100 000**	**100 %**

The mine residue closure cost calculation has been based on following assumptions:

- The production of 19,8 million tonnes of tailings and 17,5 million tonnes waste rock
- That the mining waste is potentially acid generating.
- The cover requirement is ca 0.5 m sealing layer (clay moraine), ca 0.5 m protection layer (moraine), composted sewage sludge or fertilisation and vegetation.
- Only minor surface profile adjustments are required for both waste rock dump and tailings facility.
- Active water treatment will be needed during closure and passive after closure.
- The potential for a small amount of surface area soil contamination (metal contamination, 22,500 m^3).
- A follow up period of 30 years for monitoring, maintanence of stuctures and roads.
- That confirmatory investigations (soil contamination mapping etc) are needed in the end of operations.
- That possible additive inspections and permitting procedures wil be needed in the end of operations and the Project is responsible for the costs of authority work.

20.4.7 Gap analysis and further work

The proximity of the communities in Haveri and Viljakkala is a significant issue for the Project and investigations related, for example, to noise, dust, vibrations, blasting safety will be essential. An early EIA with its assessment of Project alternatives should be initiated as soon as possible. The extent of the impact area should be evaluated in early stages of the Project development to assess the likely extent of any buffer zone and confirm relocation requirements. Likely methods required for site specific evaluations are described in Table 20-12.

Table 20-12: Gap Analysis

Study Area	Existing Data	Data and Information Gaps
Baseline	Drainage water quality from around the historical tailings. Lake sediment study considering the receptor lake Kirkkojärvi. Small quantity of historical (public) ground water level and quality data from Haveri ground water area. Some quantity of (public) historical and recent water quality data from receptor lake areas.	Complementary baseline assessment (water), which will give background data for water bodies potentially affected by future discharges or mine seepage. Baseline assessments for flora and fauna in mining area and identification of possible sensitive receptors (e.g. protected areas or areas of conservation importance).
Noise, air quality, vibrations, visual impacts	None	Baseline for noise, air quality and vibrations. Preliminary models of the impacts on the area's population are required as soon as possible to define resource sterilisation or relocation needs.
Blasting safety	None	Preliminary analysis of site specific additive requirements related to blasting safety and explosives storing (due to closeness of the community).
Burial place Impact	Real estate code for site, where private burial place exists	Exact coordinates and alternatives for management.
Historical tailings: Responsibility		Liability investigation of current facilities (to understand pre-project liabilities) and consideration in the EIA of the likely impacts of the new mining project on the historical tailings.

20.4.8 Recommendations

With respect to environmental and social aspects, the critical recommendation is the need to start the EIA process. In particular the following aspects need to be considered:

- The Project will require resettlement of some households. This issue should be addressed as early as possible. Noise, vibration and air quality modelling will facilitate assessment of impacts on the communities and safety buffer zone requirements. These studies should start as soon as practically possible.
- It will be necessary to study baseline conditions and carry out dust and noise modelling for selected scenarios as soon as possible. Surface water and groundwater monitoring programmes should be developed to provide background data for EIA.
- Early communication with Pirkanmaa Regional Council and Ylöjärvi Town is recommended considering the need of changes in land-use plans.
- The need for a Natura 2000 assessment should be queried with the relevant authority (Pirkanmaa Centre for Economic Development, Transport and the Environment) at a relatively early stage of the process to avoid potential Project delay.
- Because Viljakkala and Haveri are nearby the Project and due to space limitations, it is recommended that likely required chemical/explosive storage quantities are studied

during the PFS to exclude possible Seveso-classification.

- Historical buildings are a risk for permitting and therefore early communication with museum authorities is recommended.
- Early communications with Pirkanmaa ELY (Pirkanmaa Centre for Economic Development, Transport and the Environment) and Ylöjärvi Town should take place to evaluate the possible decommissioning of the Haveri ground water protection area and reserve water plant, as well as to address the Project´s potential surface water impacts.
- Detailed records of all communications with stakeholders should be kept.
- An early stage stakeholder consultation considering the private burial site is needed. Investigations should be carried out as a part of EIA process.
- Undertake a legal assessment to clarify responsibility for the historical liabilities
- Stakeholder engagements plans and grievance mechanisms need to be developed early in the Project to guide consultation going forward.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The following sections provide an overview of the operating and capital cost assumptions made as described in the preceding sections of this report.

21.2 Operating Costs

An overview of operating costs for the major costs centres is presented in Table 21-1 and illustrated in Figure 21-1 over the Project life of mine. No overall operating cost contingency has been assumed, however contingency (and G&A) is included in processing operating costs.

Table 21-1: Overview of operating costs by major cost centre

	USD/t moved	USD/t processed	Percentage of total
Mining	3.2	6.7	29%
Processing	7.3	15.0	64%
Tailings	0.2	0.5	2%
Environmental & Closure	0.6	1.3	5%
Total	11.3	23.5	29%

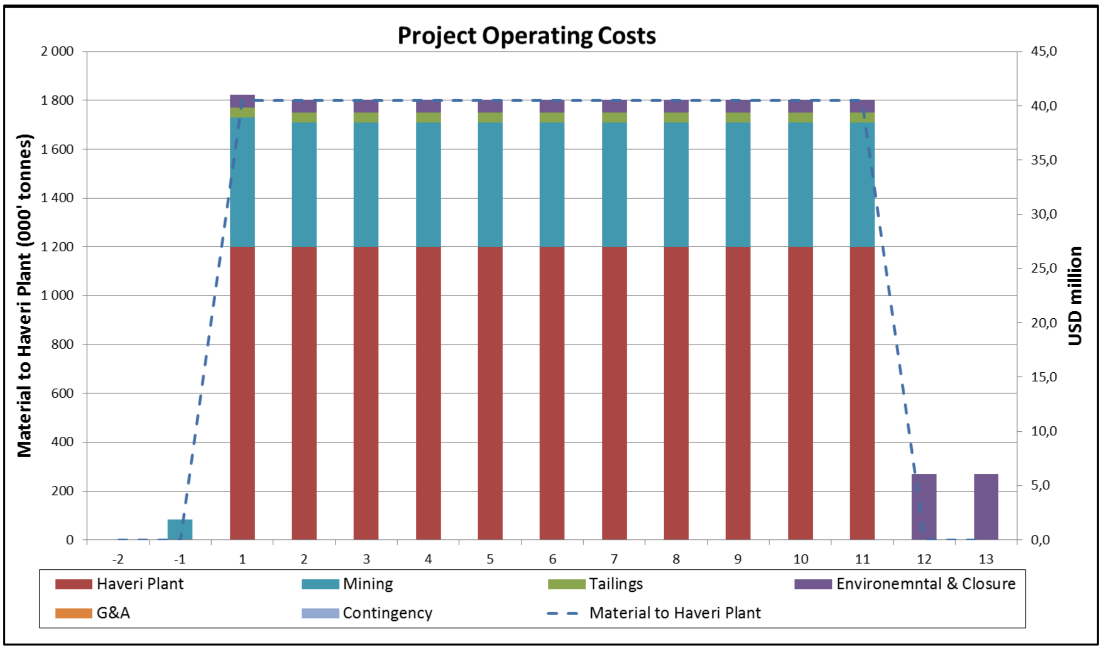

Figure 21-1: Summary of operating costs over the life of mine (Source:SRK, 2014)

21.2.1 Mining

The assumed operating costs are presented below. These estimated costs are based on the selected mining production schedule (1.8 Mtpa) and corresponding equipment usage. Increasing costs with pit depth are accounted for as is the cost of re-handling material from stockpiles into haul trucks.

Table 21-2: Mine operating costs

Mining Cost Centre	USD / tonne total material
Drilling	0.04
Blasting	0.24
Loading	0.25
Hauling_In pit	0.34
Stockpile Excavation	0.11
Haulage_Mine to plant	0.00
Mobile Mining Equipment	0.27
Auxiliary Equipment	0.11
Labour	1.63
Mine Facilities & Other (incl. grade control)	0.18
Total Mining	3.16

21.2.2 Processing

Table 21-3 presents the assumed operating costs for processing Haveri. These costs include a provision for general administration (G&A) and contingency.

Table 21-3: Process operating costs

Processing Cost Centre	USD / tonne
Crushing, Grinding & Flotation	14.0
Cyanidation	1.00
Total Processing (includes contingency and G&A)	15.0

21.2.3 Tailings

For the purposes of this report, tailings site Option 3 has been selected (see Section 20.3). As presented in Table 21-1 above, the assumed operating costs for disposal of tailings is USD 0.5 per tonne of material processed. There is currently no separate provision for the treatment of the high-sulphide tails and it is assumed that this material will be blended with tailings from the bulk sulphide concentrate for deposition in the tailings facility. Further work will be required to confirm that this is alternative is acceptable to the permitting authorities.

21.2.4 Environmental, Rehabilitation & Closure

As presented in Table 21-1 above, operating costs for environmental aspects and closure amount to USD 1.3 per tonne of material processed, or USD 25 million over the life of mine. The major costs items in this figure comprise USD 12 million for closure of the tailings and waste rock dump facilities and USD 13 million for water treatment.

21.2.5 Treatment Charges and Refining Costs

In addition to the costs presented in Table 21-1 above, the following treatment charges and refining costs (TCRC's) have been are assumed.

Table 21-4: Treatment Charges and Refining Costs

TCRC's	(Unit)	Cost
Cu Treatment Charge	**(USD/t)**	63
Cu Refining Charge	**(USD/lb)**	0.063
Au Refining Charge	**(USD/oz)**	5.0

21.3 Capital Costs

The capital costs estimated as part of this study have been derived SRK and are discussed in detail elsewhere in this report. The following section presents a summary of these costs, which total USD 92.2 million. SRK notes the following:

- Contingencies of 10% have been applied to all capital costs;
- Working capital has been assumed at 20% of operating costs incurred during the first year of production;
- No provision has been made for sustaining capital, which for the purposes of this study is accounted for in operating cost provisions.
- In general (with the exception of tailings construction), capital costs have been profiled with 80% of expenditure occurring in the first two years preceding production, and the remaining 20% occurring in the first year of production.

Figure 21-2 gives a breakdown of the envisaged capital expenditure over the life of mine and split between the major cost centres, including contingency and working capital.

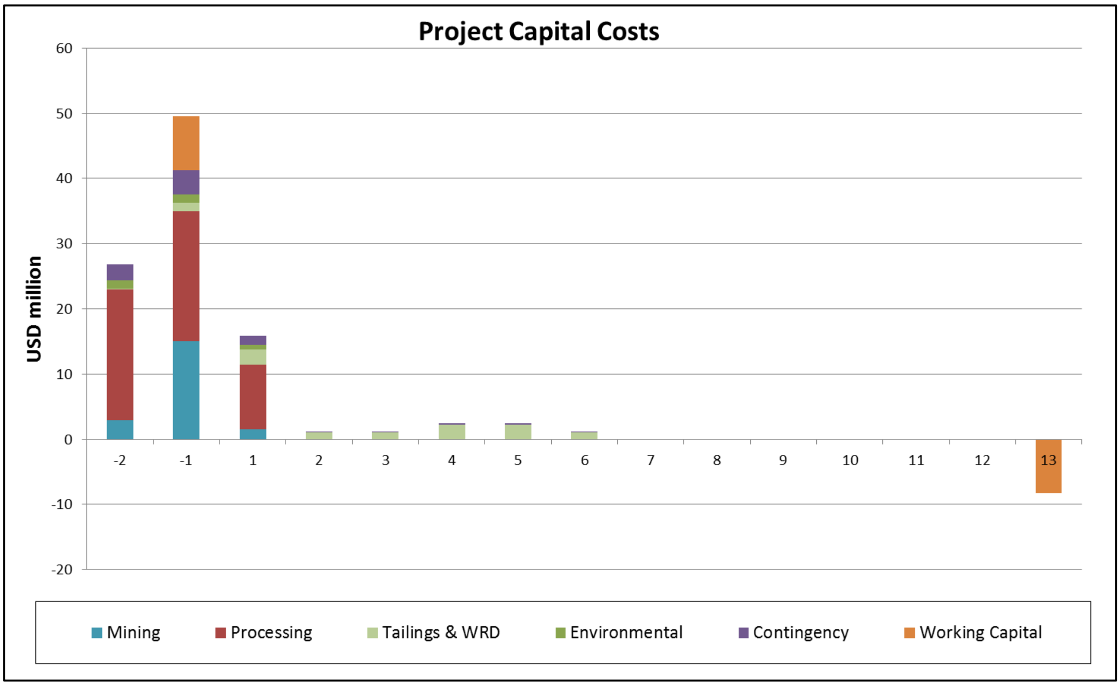

Figure 21-2:

Table 21-5 below presents capital cost assumptions, with a high-level breakdown under the major costs centres. Over 90% of capital is assumed to be required in the pre-production years and subsequently the first year of production.

Table 21-5: Capital cost assumptions

Description	Value (USD million)
Mining	
Mine Facilities & Haulage Dispatch System	7.0
Haul Roads	0.3
Mobile Mining Equipment	10.0
Auxiliary Equipment	2.1
Total Mining	**19.4**
Processing	
Process plant (incl. EPCM & contingency)	50.0
Total Processing	**50.0**
Tailings & WRD	
Tailings construction costs	9.9
WRD Construction (incl. ground prep & liner)	1.2
Total Tailings & WRD	**11.1**
Environmental	
Water Management Facilities	1.5
Water Treatment Plants	1.6
Land purchase	0.3
Total Environment	3.3
Contingency (10%)	8.4
Total	**92.2**

22 ECONOMIC ANALYSIS

22.1 Introduction

SRK has constructed a technical economic model (TEM) to derive a post-tax Net Present Value (NPV) for the Project. The TEM is based on the technical assumptions developed by SRK during the course of this PEA, as commented on in the previous sections of this report.

The economic analysis contained in this report includes Inferred Resources only and is preliminary in nature. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves, which would also require the support of a pre-feasibility level study. There is no certainty that the reserves development, production, and economic forecasts on which this Preliminary Assessment is based will be realised.

22.2 Valuation Process

22.2.1 General Assumptions

The model is based on production from a main open pit at Haveri and a satellite pit at Peltosaari, with on-site crushing, grinding and flotation for production of a marketable copper sulphide concentrate and smelted gold doré through conventional flotation, cyanide leaching and Carbon-in-Pulp (CIP) / Carbon-in-Leach (CIL).

For the purposes of this study, the material contained within the selected pit shells has been assumed to remain constant at 1.8Mtpa over the life of mine.

SRK has constructed a post-tax and pre-finance TEM which assumes:

- a US Dollar (USD) valuation currency, with any Euro (EUR) derived costs being converted at a EUR:USD exchange rate of 1:0.75;
- a base case discount rate of 8%;
- the TEM is in real 2014 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 cashflow starts in year -2;
- working capital based on 25% of the operating costs from the first year of production;
- depreciation on a 20% declining balance basis; and
- corporate tax rate of 24.5%.

The TEM considers the revenue and cost implications of both a marketable copper sulphide concentrate and smelted gold doré.

22.2.2 Commodity Price Assumptions

The following commodity price assumptions have been used:

Copper USD 6,500 / tonne

Gold USD 1,300 / troy ounce

22.3 Mine and Process Physical Assumptions

22.3.1 Mining

A summary of the combined mass movement of material is presented in Figure 22-1 and below. It is assumed that marginal material is processed along with run of mine material.

Table 22-1: Summary of movement of material from the open pit

Mining	Unit	Value
ROM	(tonnes '000)	14 180
Marginal Material	(tonnes '000)	5 620
Waste Rock	(tonnes '000)	19 160
Glacial Ovb	(tonnes '000)	1 740
Total Material Mined	(tonnes '000)	40
Strip ratio	(w:o)	1.1
Life of mine	(years)	11
Grade Cu	(%)	0.08%
Grade Au	(g/t)	0.90

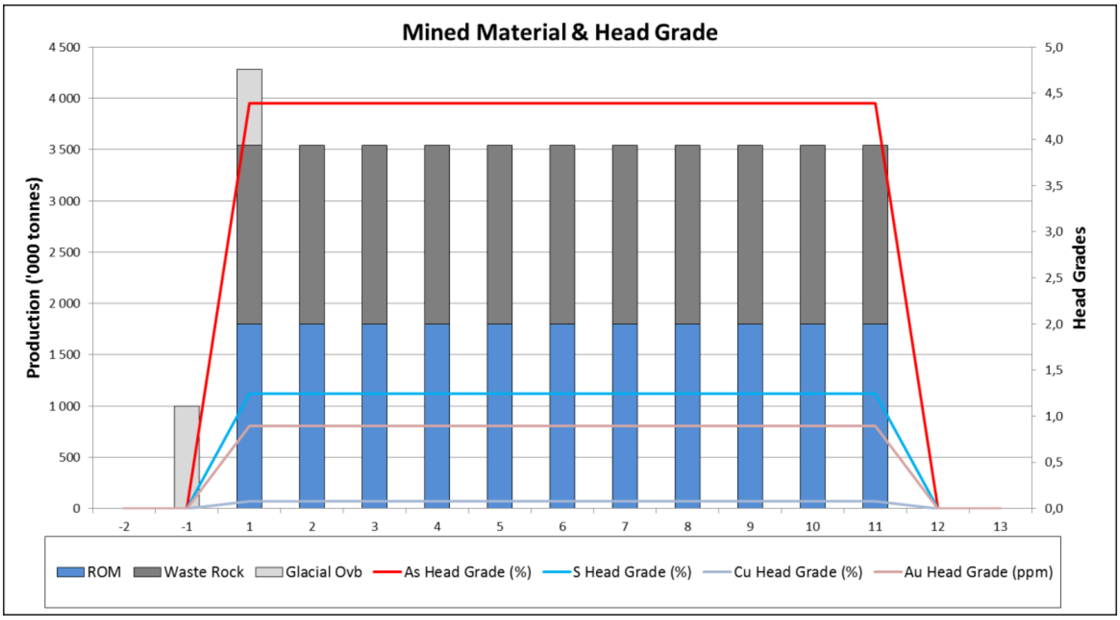

Figure 22-1: Summary of mass movement of material (Source:SRK, 2014)

22.3.2 Process, Smelting and Refining

Process recovery and concentrate grade assumptions are discussed and presented in Section 17 and Table 17-1 above. Table 17-1 is reproduced below (Table 22-2). Smelting and Refining assumptions are presented in Table 22-3.

Table 22-2: Process Design Criteria

Item		Unit	Value
RoM Production		tpa	1,800,000
Flotation Feed Grade	Cu	%	0.09
	Au	g/t	1.00
	S	%	1.24
Copper Concentrate		tpa	3,900
	Cu Rec	%	60.0
	Au Rec	%	20.0
	Cu	%	25.0
	Au	g/t	92.6
Sulphide Concentrate		tpa	45,000

Item		Unit	Value
	Au Rec	%	60.0
	Au	g/t	24.0
Cyanidation Recovery	Au	%	95.0
Recovery to Doré	Au	%	57.0
Overall Recovery	Cu	%	60.0
	Au	%	77.0

Table 22-3: Smelting and Refining assumptions

Item	Unit	Value
	Copper Concentrate Losses & Deductions	
Cu Payable	(%)	95.0
Cu unit deduction	(%)	1.0
Au unit deduction	(g/t)	1
	Leach Doré	
Au Payable	(%)	99.5

SRK notes that no penalties have been assumed for contained arsenic. For the purposes of this study, it is assumed that these costs are non-material and will be covered by the deduction, treatment and refining charges.

22.4 Revenue & Cash Flow Projections

Figure 22-2 below provides an overview of net revenue for Cu and Au over the life of mine.

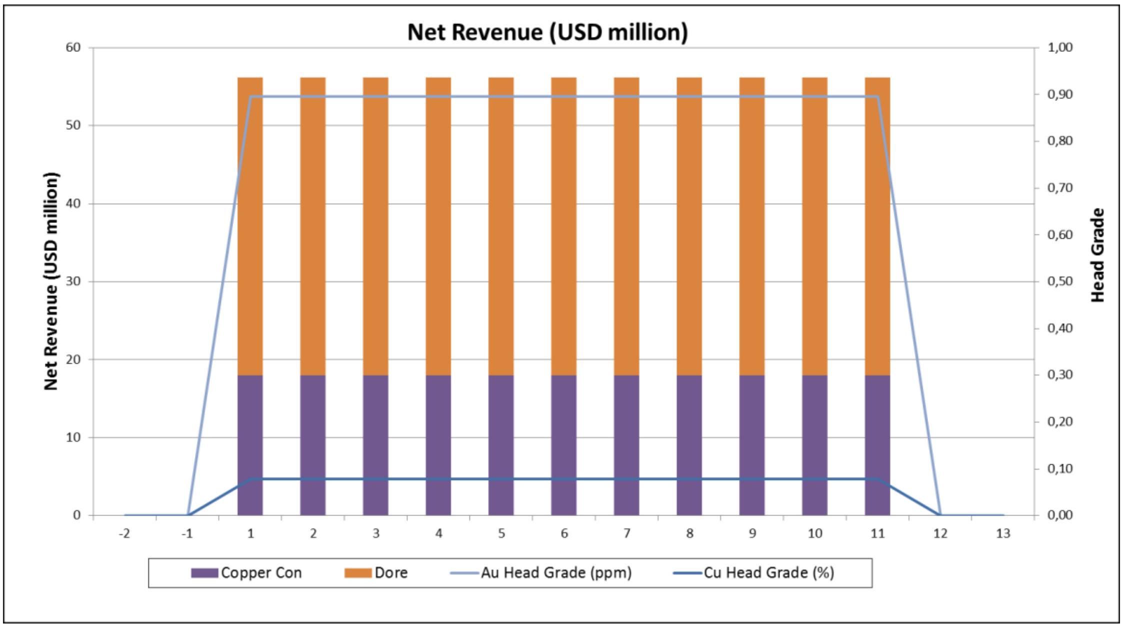

Figure 22-2: Contribution to net revenue of copper concentrate and Au doré (net of TCRC's, losses and deductions). (Source:SRK, 2014)

The annualised net post-tax, pre-finance cashflow is summarised in Figure 22-3, Table 22-4 and Table 22-5 below.

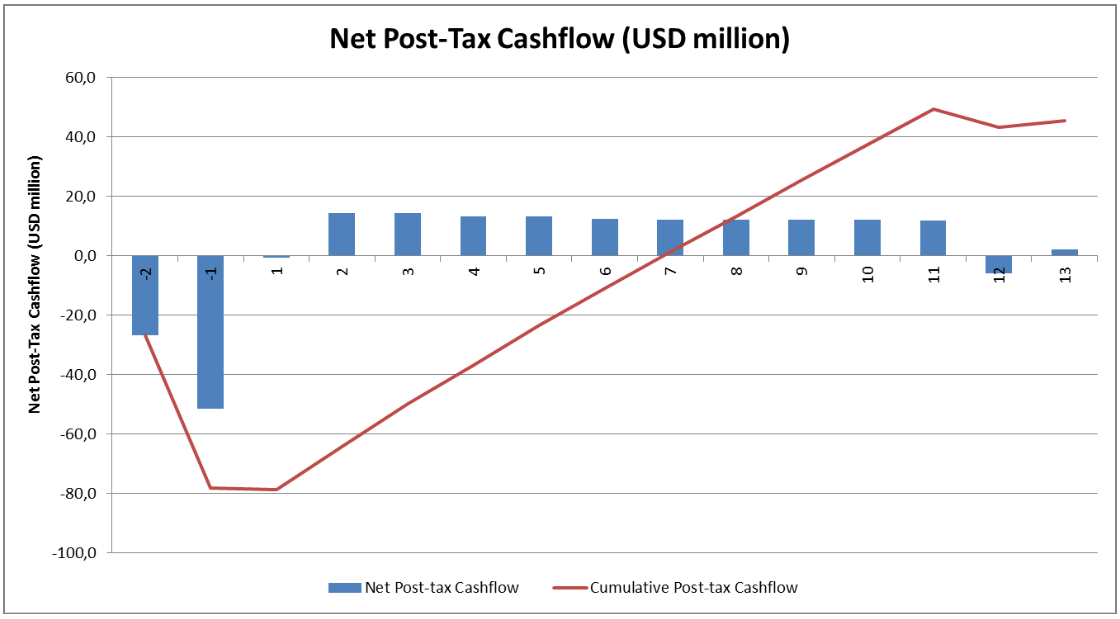

Figure 22-3: Annual and cumulative net post-tax cashflow. (Source:SRK, 2014)

Table 22-4: Summary undiscounted net-post tax cashflow

Description	Units	Total
Gross Revenue	(USDM)	622
Operating costs / t ROM	(USD/t)	23
Capital costs	(USDM)	92
Net pre-tax cashflow (undiscounted)	(USDM)	65
Net post-tax cashflow (undiscounted)	(USDM)	46
Payback period	**(years)**	6.5

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, for the base case, at a Cu price of USD 6 500/tonne and Au price of USD 1,300 / troy ounce, a 8% discount rate the project has a **post-tax, pre-finance NPV of USD -1.4 million (IRR 7.6%)** for production of both a copper concentrate and Au doré.

On the basis of current knowledge and given input assumptions discussed above, the Project would appear to be marginal to sub-economic. The Projects sensitivities are discussed below and recommendations are presented in Section 26 below.

Table 22-5: Summary Annual Cash Flow

SE471 Haveri PEA			Year														
Summary Annual Cashflow	**Units**	**Total**	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
CASHFLOW																	
Mining																	
ROM	**(000' tonnes)**	**19 800**	0	0	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	0	0
Waste Rock	**(000' tonnes)**	**19 160**	0	0	1 742	1 742	1 742	1 742	1 742	1 742	1 742	1 742	1 742	1 742	1 742	0	0
Glacial Ovb	**(000' tonnes)**	**1 739**	0	1 000	739	0	0	0	0	0	0	0	0	0	0	0	0
Total Material Mined	**(000' tonnes)**	**40 699**	0	1 000	4 280	3 542	3 542	3 542	3 542	3 542	3 542	3 542	3 542	3 542	3 542	0	0
Stripping Ratio (waste / ROM)	**(w:o)**	**1,06**	0,00	0,00	1,38	0,97	0,97	0,97	0,97	0,97	0,97	0,97	0,97	0,97	0,97	0,00	0,00
Processing																	
Material to Plant	**(000' tonnes)**	**19 800**	0	0	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	1 800	0	0
Au Head Grade (ppm)	**(ppm Au)**	**0,90**	0,00	0,00	0,90	0,90	0,90	0,90	0,90	0,90	0,90	0,90	0,90	0,90	0,90	0,00	0,00
Cu Head Grade (%)	**(% Cu)**	**0,08**	0,00	0,00	0,08	0,08	0,08	0,08	0,08	0,08	0,08	0,08	0,08	0,08	0,08	0,00	0,00
Copper Concentrate Product	**(tonnes)**	**37 372**	0	0	3 397	3 397	3 397	3 397	3 397	3 397	3 397	3 397	3 397	3 397	3 397	0	0
Dore - Au	**(oz)**	**323 245**	0	0	29 386	29 386	29 386	29 386	29 386	29 386	29 386	29 386	29 386	29 386	29 386	0	0
Revenue																	
Gross Revenue																	
Copper Con	**(M USD)**	202	0	0	18	18	18	18	18	18	18	18	18	18	18	0	0
Dore	**(M USD)**	420	0	0	38	38	38	38	38	38	38	38	38	38	38	0	0
Total	**(M USD)**	622	0	0	57	57	57	57	57	57	57	57	57	57	57	0	0
Net Revenue																	
Copper Con	**(M USD)**	198	0	0	18	18	18	18	18	18	18	18	18	18	18	0	0
Dore	**(M USD)**	420	0	0	38	38	38	38	38	38	38	38	38	38	38	0	0
Total	**(M USD)**	618	0	0	56	56	56	56	56	56	56	56	56	56	56	0	0
Operating Costs																	
Mining	**(M USD)**	**128,7**	0,0	1,9	11,9	11,5	11,5	11,5	11,5	11,5	11,5	11,5	11,5	11,5	11,5	0,0	0,0
Processing	**(M USD)**	**297,0**	27,0	27,0	27,0	27,0	27,0	27,0	27,0	27,0	27,0	27,0	27,0	0,0	0,0	0,0	0,0
Tailings	**(M USD)**	**9,6**	0,0	0,0	0,9	0,9	0,9	0,9	0,9	0,9	0,9	0,9	0,9	0,9	0,9	0,0	0,0
Environemntal & Closure	**(M USD)**	**25,4**	0,0	0,0	1,2	1,2	1,2	1,2	1,2	1,2	1,2	1,2	1,2	1,2	1,2	6,0	6,0
G&A	**(M USD)**	**0,0**	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Contingency	**(M USD)**	**0,0**	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Total Operating Costs	**(M USD)**	**460,8**	0,0	1,9	41,0	40,6	40,6	40,6	40,6	40,6	40,6	40,6	40,6	40,6	40,6	6,0	6,0
Unit Operating Costs	**(USD / oz AuEq)**	**963**	0	0	943	933	933	933	933	933	933	933	933	933	933	0	0
Capital Costs																	
Mining	**(M USD)**	19,4	2,9	15,0	1,5	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Processing	**(M USD)**	50,0	20,0	20,0	10,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Tailings & WRD	**(M USD)**	11,1	0,0	1,1	2,2	1,1	1,1	2,2	2,2	1,1	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Environmental	**(M USD)**	3,3	1,3	1,3	0,7	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Contingency	**(M USD)**	8,4	2,4	3,7	1,4	0,1	0,1	0,2	0,2	0,1	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Working Capital	**(M USD)**	0,0	0,0	8,2	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	-8,2
Total	**(M USD)**	92,2	26,7	49,4	15,8	1,2	1,2	2,4	2,4	1,2	0,0	0,0	0,0	0,0	0,0	0,0	-8,2
Cashflow																	
Net Pre-tax Cashflow	**(M USD)**	**65,1**	-26,7	-51,3	-0,6	14,4	14,4	13,2	13,2	14,4	15,6	15,6	15,6	15,6	15,6	-6,0	2,2
Cumulative Pre-tax Cashflow	**(M USD)**	**0,0**	-26,7	-78,0	-78,7	-64,3	-49,9	-36,7	-23,5	-9,1	6,5	22,1	37,7	53,4	69,0	62,9	65,1
Corporation tax	**(M USD)**	**-19,6**	0,0	0,0	0,0	0,0	0,0	0,0	0,0	-1,9	-3,4	-3,5	-3,5	-3,6	-3,6	0,0	0,0
Net Post-tax Cashflow	**(M USD)**	**45,5**	-26,7	-51,3	-0,6	14,4	14,4	13,2	13,2	12,5	12,2	12,1	12,1	12,0	12,0	-6,0	2,2

22.5 Project Sensitivities

For illustrative purposes the following analysis presents the sensitivity of the Project under various scenarios for variation in single and twin sensitivity parameters.

22.5.1 Single Parameter Sensitivities

Figure 22-4 shows the varying NPV for varying single parameter sensitivities at an 8% discount rate for revenue, operating costs, capital costs and EUR:USD exchange rate.

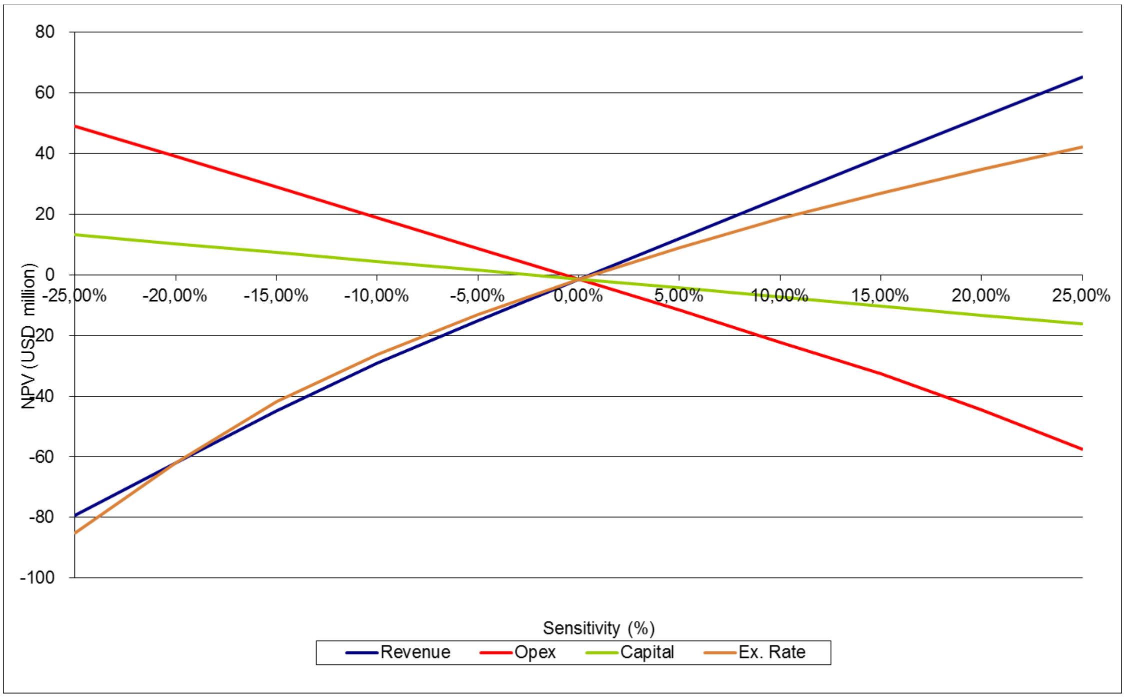

Figure 22-4: Single parameter sensitivity post-tax, pre-finance NPV at 8% discount rate. (Source:SRK, 2014)

SRK notes that Project is most sensitive to changes in commodity price and least sensitive to changes in capital cost. For illustrative purposes, a summary table of production physicals, costs, revenue and cashflow is presented in the below, for three different gold price scenarios; 1 100 USD/oz (low), 1 300 USD/oz (base case) and 1 500 USD/oz (high).

Table 22-6: Summary results for three gold price scenarios

	Unit	Low	Base Case	High
Gold price scenario	**USD / oz**	**1 100**	**1 300**	**1 500**
Cut-off grade ROM	**(g/t Au)**	0,7	0,7	0,7
Cut-off grade Marginal Material	**(g/t Au)**	0,55	0,55	0,55
ROM	**(000' tonnes)**	19 800	19 800	19 800
Waste Rock (incl. overburden)	**(000' tonnes)**	20 899	20 899	20 899
Material to Plant	**(000' tonnes)**	19 800	19 800	19 800
Au Head Grade (ppm)	**(g/t Au)**	0,90	0,90	0,90
Cu Head Grade (%)	**(% Cu)**	0,08	0,08	0,08
Dore Au produced	**(oz)**	323 245	323 245	323 245
Dore Au produced	**(kg)**	10 053	10 053	10 053
Copper Concentrate produced	**(tonnes)**	37 372	37 372	37 372
Overall Au Recovery	**(%)**	77%	77%	77%
Overall Cu recovery	**(%)**	60%	60%	60%
Total Gross Revenue (Dore & Copper Con)	**(USD million)**	579	579	579
Total deductions (TCRC's & losses)	**(USD million)**	4	4	4
Total Net Revenue (Dore & Copper Con)	**(USD million)**	575	575	575
Operating Costs				
Mining	**(USD/t)**	6,5	6,5	6,5
Processing (incl. G&A, transport)	**(USD/t)**	15,0	15,0	15,0
Tailings	**(USD/t)**	0,5	0,5	0,5
Environmental & Closure	**(USD/t)**	1,3	1,3	1,3
Total	**(USD/t)**	23,3	23,3	23,3
Unit operating cost / oz AuEq	**(USD/oz AuEq)**	956	956	956
Capital Costs	**(USD million)**	92	92	92
Net Pre-tax Cashflow	**(USD million)**	-22	65	152
Corporation tax (24,5%)	**(USD million)**	0	20	-41
Net Post-tax Cashflow	**(USD million)**	-22	45	111
NPV (post tax, 8% WACC)	**(USD million)**	-41	-1	36

22.5.2 Twin Parameter Sensitivities (Base Case)

Table 22-7 shows the sensitivity of the Project at an 8% discount rate to simultaneous changes in two parameters, specifically; revenue and operating costs, revenue and capital costs, operating costs and capital costs respectively.

Table 22-7: Twin Parameter Sensitivities for base case post-tax, pre-finance NPV at 8% discount rate

TWIN PARAMETER SENSITIVITY												
REAL												
REVENUE V OPEX SENSITIVITY		REVENUE										
DISCOUNT FACTORS	-1,4	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
	-25%	(18,3)	(4,5)	9,0	22,5	35,8	49,0	62,2	75,4	88,6	101,8	114,9
	-20%	(28,9)	(14,8)	(1,2)	12,3	25,7	39,0	52,3	65,5	78,6	91,8	105,0
	-15%	(40,5)	(25,4)	(11,5)	2,2	15,6	29,0	42,3	55,5	68,7	81,9	95,1
	-10%	(53,4)	(36,2)	(21,9)	(8,1)	5,5	18,9	32,2	45,5	58,7	71,9	85,1
	-5%	(66,4)	(49,1)	(32,6)	(18,5)	(4,7)	8,8	22,2	35,5	48,8	62,0	75,1
OPEX	0%	(79,3)	(62,0)	(44,7)	(29,1)	(15,1)	(1,4)	12,1	25,4	38,7	52,0	65,2
	5%	(92,3)	(74,9)	(57,6)	(40,3)	(25,6)	(11,7)	1,9	15,3	28,7	42,0	55,2
	10%	(105,2)	(87,9)	(70,6)	(53,3)	(36,3)	(22,1)	(8,3)	5,2	18,6	31,9	45,2
	15%	(118,1)	(100,8)	(83,5)	(66,2)	(48,9)	(32,8)	(18,7)	(5,0)	8,5	21,9	35,2
	20%	(131,1)	(113,8)	(96,5)	(79,2)	(61,8)	(44,5)	(29,3)	(15,3)	(1,7)	11,8	25,2
	25%	(144,0)	(126,7)	(109,4)	(92,1)	(74,8)	(57,5)	(40,2)	(25,8)	(11,9)	1,7	15,1
REVENUE V CAPEX SENSITIVITY		REVENUE										
DISCOUNT FACTORS	-1,4	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
	-25%	(62,0)	(44,7)	(27,9)	(13,9)	(0,2)	13,2	26,5	39,7	52,9	66,1	79,3
	-20%	(65,4)	(48,1)	(31,0)	(16,9)	(3,2)	10,3	23,6	36,9	50,1	63,3	76,5
	-15%	(68,9)	(51,6)	(34,3)	(19,9)	(6,1)	7,4	20,7	34,0	47,3	60,5	73,6
	-10%	(72,4)	(55,1)	(37,8)	(22,9)	(9,1)	4,5	17,9	31,2	44,4	57,6	70,8
	-5%	(75,8)	(58,5)	(41,2)	(26,0)	(12,1)	1,5	15,0	28,3	41,6	54,8	68,0
CAPEX	0%	(79,3)	(62,0)	(44,7)	(29,1)	(15,1)	(1,4)	12,1	25,4	38,7	52,0	65,2
	5%	(82,8)	(65,5)	(48,2)	(32,1)	(18,1)	(4,4)	9,2	22,6	35,9	49,1	62,4
	10%	(86,3)	(68,9)	(51,6)	(35,2)	(21,1)	(7,3)	6,2	19,7	33,0	46,3	59,5
	15%	(89,7)	(72,4)	(55,1)	(38,3)	(24,1)	(10,3)	3,3	16,7	30,1	43,4	56,7
	20%	(93,2)	(75,9)	(58,6)	(41,4)	(27,2)	(13,3)	0,4	13,8	27,3	40,5	53,8
	25%	(96,7)	(79,3)	(62,0)	(44,7)	(30,2)	(16,3)	(2,6)	10,9	24,3	37,7	51,0
OPEX V CAPEX SENSITIVITY		OPEX										
DISCOUNT FACTORS	-1,4	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
	-25%	63,2	53,2	43,2	33,3	23,2	13,2	3,1	(7,2)	(17,5)	(28,1)	(40,1)
	-20%	60,3	50,4	40,4	30,4	20,4	10,3	0,1	(10,1)	(20,5)	(31,2)	(43,6)
	-15%	57,5	47,6	37,6	27,5	17,5	7,4	(2,8)	(13,1)	(23,6)	(34,3)	(47,1)
	-10%	54,7	44,7	34,7	24,7	14,6	4,5	(5,8)	(16,1)	(26,6)	(37,6)	(50,5)
	-5%	51,9	41,9	31,8	21,8	11,7	1,5	(8,7)	(19,1)	(29,7)	(41,1)	(54,0)
CAPEX	0%	49,0	39,0	29,0	18,9	8,8	(1,4)	(11,7)	(22,1)	(32,8)	(44,5)	(57,5)
	5%	46,2	36,1	26,1	16,0	5,9	(4,4)	(14,7)	(25,2)	(35,9)	(48,0)	(60,9)
	10%	43,3	33,3	23,2	13,1	2,9	(7,3)	(17,7)	(28,2)	(39,0)	(51,5)	(64,4)
	15%	40,4	30,4	20,3	10,2	(0,0)	(10,3)	(20,7)	(31,3)	(42,1)	(54,9)	(67,9)
	20%	37,6	27,5	17,4	7,3	(3,0)	(13,3)	(23,7)	(34,3)	(45,5)	(58,4)	(71,3)
	25%	34,7	24,6	14,5	4,3	(5,9)	(16,3)	(26,7)	(37,4)	(48,9)	(61,9)	(74,8)

23 ADJACENT PROPERTIES

Not applicable.

24 OTHER RELEVANT DATA AND INFORMATION

Not applicable.

25 INTERPRETATION AND CONCLUSIONS

At the Company's request, SRK have proposed a phased budget to progress the Project to the next level of study, based on the work carried out and the recommendations presented in this report. This proposed budget focuses on work to improve confidence in the underlying geological data and assesses processing options as part of Phase 1. Subject to the results of this, Phase 2 could use these findings to update the high-level mine plan and assess this in the financial model to gauge overall impact on Project viability.

SRK stress that the results of work should be critically assessed by the Company before committing to further expenditure.

The budget presented in Table 25-1 below has been proposed by SRK and agreed to in principal by the Company (with the exception of Phase 1, which was proposed by the Company). SRK anticipate that Phase 1 could be carried out over two months, Phase 2 over a 6 to 12 month period, with Phase 3 following over a subsequent 3 to 6 month period.

Table 25-1: Haveri proposed work budget over 12 to 18 months

Phase	Item	Estimate (USD)
Phase 1	Due diligence of previous resource estimates and comparison with SRK estimate	50 000
Phase 2	Verification of historic assay data	37 500
	Re-logging and sampling of GMH core at Loppi	60 000
	Re-logging and structural interpretation of drill core with update to geological model	25 000
	Subject to results of the above, update of the Mineral Resource Estimate, including Ag	45 000
	Laboratory scale testwork including initial optical / XRT sorting tests	90 000
	Subject to results of testwork, discussions with existing third party process facilities regarding toll treatment	7 500
	Total Phase 1	**265 000**
Phase 3	Revise pit optimisation and high-level mine plan based on above findings	30 000
	Update financial model to assess likely impact on project viability	30 000
	Total Phase 2	**60 000**
	Total (Phase 1 & Phase 2)	**372 500**
	Contingency (10%)	37 250
	Grand Total	**409 750**

25.1 Risks and Opportunities

25.1.1 Introduction

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.

25.1.2 Risks

There are a number of risks inherent to the mining industry, including the stability of the markets, uncertainties related to Mineral Resource and Mineral Reserve estimation, equipment and production performance. The specific risks SRK has identified relating to Haveri are summarised below.

- The Project is currently marginal to sub-economic, which is largely due to the nature of the mineralisation and low average Au and Cu grades. It is uncertain at this stage whether a further work will improve the Project's overall viability;
- The limited land area on the Haveri peninsula restricts the options available for placement of mine related infrastructure; and
- Environmental and social issues present a key risk, given population densities and competing land-use in the immediate area.

25.1.3 Opportunities

SRK consider there to be specific opportunity to improve project economics as follows:

- Re-sampling and logging may improve confidence in the data supporting the current Mineral Resource, improve the Resource category and possibly facilitate higher grade zones to be better defined;
- Optical and/or X-ray transmission (XRT) sorting technologies could potentially be utilised to reduce the quantity of material to be processed, for a minimal loss of contained metal in the plant feed. Whilst this technology is still relatively unproven in the mining industry, SRK notes the positive results of initial testwork undertaken by Belvedere for the Kopsa Cu-Au Project, also in Finland and with Cu Au grades; and
- Tailings material from historic production at Haveri may be re-processed during initial years of an operation to provide early cashflow.

26 RECOMMENDATIONS

Based on the findings of this PEA, SRK makes the following recommendations:

- The locating of the original historic assays and the comparison of these with the GMH re-assayed values in order to help with verification of the historic data;
- Review, assay and re-logging of the GMH drill core at the GTK archive in Loppi;
- Complete re-assaying of the coarse reject material from the Northern Lion drilling along with an appropriate QAQC programme;
- Subsequent to the re-sampling, the systematic re-logging of drill sections with a view to developing a detailed structural interpretation and improving the understanding of the geological controls on mineralisation;
- The modelling of the Ag mineralisation and the inclusion of this in the Mineral Resource statement to enable this to be considered as a by-product in the cashflow model;
- The collection of representative metallurgical samples;
- New metallurgical laboratory scale test-work to assess how to produce a higher grade concentrate (in order to reduce freight costs) whilst maintaining a high processing recovery and suppressing contamination (arsenic in the copper concentrate; copper in the gold concentrate);
- Optical / XRT sorting test-work;
- Detailed analysis of the geotechnical domains to determine appropriate slope angles which comply with pit design standards;
- Detailed analysis of hydrogeological and hydrological factors and the impact on dewatering and the design of water management systems;
- Commencement and/or continuation of discussions with the owners of existing third party processing facilities to determine whether the sale or toll treatment of crushed Haveri ROM is possible and if so, what terms may be reasonable to assume for the purposes of comparison during further phases of study;
- Further work to improve the accuracy of cost estimates; and
- An assessment of whether the cost of developing the infrastructure can be shared with the regional authorities.

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Abbreviations

CAPEX	Capital expenditure
OPEX	Operating expenditure
TMF	Tailings management facility
CIM	Canadian Institute of Mining, Metallurgy and Petroleum. Produces definitions and guidelines for the reporting of Exploration Information, Mineral Resources and Mineral Reserves
PEA	Preliminary economic assessment (as defined by CIM). A study, other than a pre-feasibility or feasibility study, that includes an economic analysis of the potential viability of mineral resources

Units

masl	Metres above sea level
Mt	Million metric tonnes
Mm^3	Million cubic metres
Ktpa	Thousand tonnes per annum
Mtpa	Million tonnes per annum
€	Euro
M€	Million Euro
SEK	Swedish Kronor
MSEK	Million Swedish Kronor
USD	US Dollars ($)
M USD	Million US Dollars ($)
%	Percentage
g/t	Parts per million
g/t	Grams per tonne
Au	Gold
Ag	Silver
As	Arsenic
Cd	Cadmium
Co	Cobalt
Cr	Chromium
Cu	Copper
Fe	Iron
Hg	Mercury
Ni	Nickel
Pb	Lead
S	Sulphur
Sb	Antimony
V	Vanadium
Zn	Zinc

APPENDIX A

A FINNISH LEGISLATIVE FRAMEWORK

PRIMARY AUTHORISATIONS AND LEGISLATION FOR A NEW MINING DEVELOPMENT

In Finland, a project proponent must obtain the following primary authorisations for a new mining development:

- Environmental permit;
- Mining permit;
- Water permit;
- Land use amendment;
- Building permit;
- Certain derogation permit(s)*; and
- Natura 2000 authorisation*.

*Potential authorisations required depending on whether the development impacts any Natura 2000 areas and/or impacts on certain fauna, flora and habitats.

The principle legislation regulating developments includes:

- Mining Act of Finland (621/2011) and Finnish Government Decree on Mining Activities (391/2012);
- Environmental Protection Act (86/2000) and Decree (169/2000);
- Water Act (587/2011) and Decree (1560/2011);
- Act on Environmental Impact Assessment (EIA) Procedure (468/1994) and Decree (713/2006);
- Waste Act (646/2011) and Decree (179/2012);
- Cultural Heritage Act (295/1963);
- Reindeer Keeping Act (848/1990);
- Land Use and Building Act (132/1999); and
- Nature Conservation Act (86/2000).

1.1 Environmental authorisation

The Finnish environmental authorisation process takes places over two procedural phases. An EIA procedure is administered by one authority and then the permitting process starts with the approval[1] of the EIA, which is used to support applications to other authorities for the various permissions needed to construct and operate the development, including the environmental permit, water permit and building permits. In the case of the environmental and water permits, these are granted following a joint application submission in accordance with the Environmental Protection Act and Decree and Water Act by the Regional State Administrative Agency / Aluehallintovirasto (AVI).

[1] *EIA approval is not an approval of the Project. EIA approval is conducted to assure, that EIA report is complete enough, provides the information needed in permitting processes and gives good insight to the project alternatives. In EIA approval it is also controlled, that EIA procedure with all stakeholder consultations is adequately carried out.*

1.2 EIA procedure

The EIA procedure is administered by one of the 15 Centres for Economic Development, Transport and the Environment (ELY) and regulated by the Act on EIA and associated Decree. ELY is responsible for the environment and natural resources, and it monitors compliance with environmental permits. ELY also confirms the requirement for the EIA procedure [2]. In the case of the Haveri Project, the Länsi-ja Sisä-Suomen AVI (Western and Central Finland Regional State Administrative Agency) and the Pirkanmaa ELY are the principle authorities. The proponent initially compiles an EIA programme and thereafter an EIA report. Both documents are submitted to ELY who publicise the reports and issue statements on their adequacy taking into consideration comments from other government departments, landowners, other rights holders and the public. The content of the EIA programme and EIA report is described as follows.

EIA programme:

- Motivation for following EIA procedure;
- Conceptual project description, planning status and location, land use requirements and possible connection to other developments;
- Project alternatives including the 'zero' alternative (if no specific reason can be presented for excluding an alternative);
- Information on existing and planned environmental impact investigations, decisions, plans and permits, including proposed impact assessment methodology and assumptions;
- Preliminary baseline environmental description;
- Suggested impact area definition;
- Stakeholder consultation plan; and
- Timeframe for EIA procedure and planned public consultations.

EIA report:

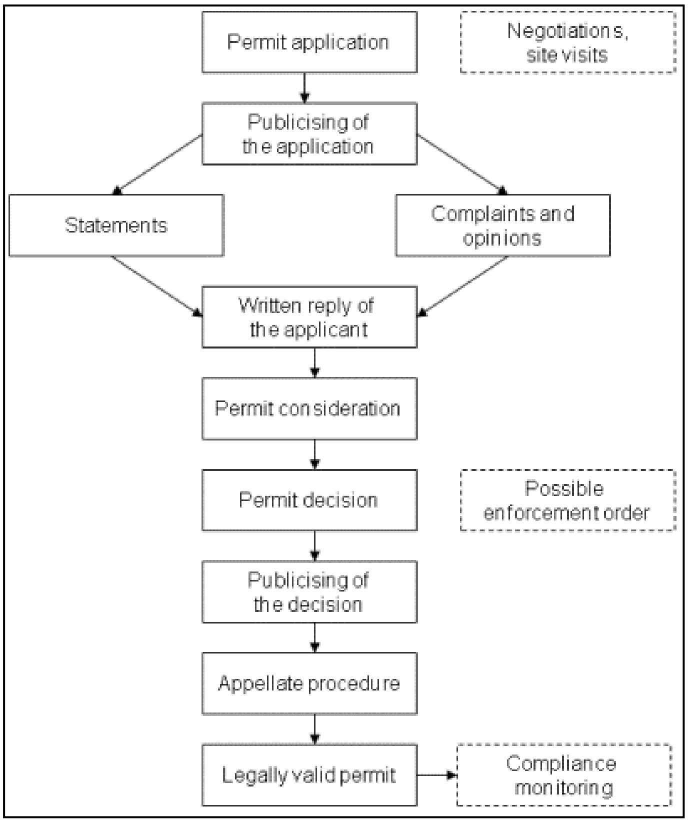
- Information from EIA programme;
- Project´s relation to land-use plans and relevant programmes for natural resource management;
- Technical description of planned activities and production, traffic, raw materials, waste and emissions, including building and decommissioning stages;
- Background material used in EIA;
- Baseline environment description and impacts of project (including alternatives), information gap analysis and assessment risks of environmental accidents;
- Feasibility and comparison of project alternatives;
- Suggestions for environmental impact management
- Suggestion for monitoring programme; and
- Declaration of EIA procedure, including stakeholder consultation and explanation how ELY statement on the EIA programme has been implemented into the report.

1.3 Permit application procedure

The proponent must also follow a permit application procedure for the development. The procedure is administered by the AVI and regulated by the Environmental Protection Act (86/2000) and Environmental Protection Decree (169/2000). The proponent must compile an

[2] *The two-step EIA procedure is required when 550 000 ton extraction limit is exceeded or open pit is larger than 25 ha. In practice, mining projects almost without exception follow the two-step EIA procedure, because site specific factors (current land-use, potential disturbance, potential pollution, shoreline, protected objects) may lead to requirement to carry out two-step EIA procedure. In this Project´s case there are several site specific factors, which would each probably lead to the requirement of the two-step EIA procedure. If the two-step procedure was not applied, environmental impacts would still need to be evaluated in detail in permit application stage.*

Environmental Permit report. Usually ELY's statement on the EIA report is appended to the Permit Application report together with the EIA report and submitted to AVI in support of an environmental permit application. App A Figure 1 shows the environmental permit application handling process following submission of the Environmental Permit report.



App A Figure 1: Environmental permit handling process following submission of the Environmental Permit report to AVI (Source: Ministry of Environment, 2013)

AVI reviews the Environmental Permit report and, if required, requests complimentary information from the proponent. The proponent revises the report to the satisfaction of AVI after which the authority publicises the permit application (and potentially the complementary information). Public complaints and opinions are solicited during the publication period (60 days). Public participation meeting(s) are convened by the proponent during the permit application procedure according to an agreement with the permit authority. AVI also simultaneously obtains formal expert opinion on the development.

After publication the proponent responds to public comments and opinions in a separate response document. The authorities carry out a site inspection around this time, if it has not taken place earlier during the procedure. Thereafter AVI considers the environmental permit application and proposes a record of decision (RoD) based on the submitted data. A positive RoD requires proof the development will not damage people's health or risk relevant pollution. If the damage and pollution cannot be prevented with permit conditions, then the RoD may be negative. If granted, then permit conditions regulate the mine design, emissions, construction, operation, monitoring and closure requirements.

The proponent may request an order from AVI to enforce the authority's RoD to grant the environmental permit. If granted and AVI's RoD of the permit is appealed, then the proponent may start preparatory works before the appellate procedure involving the court (which could take a number of years) is finalised, providing other authorisations (see below) have been obtained and appropriate guarantee placed as stipulated in the RoD. In some instances the enforcement order is not granted, in which case the appellate procedure must be finalised before the permit is legally valid and preparatory works permitted to begin.

1.4 Water permit

Water permits are required for activities impacting on water and/or water supply (including groundwater). The proponent applies for a water permit to AVI at the time of filing the Environmental Permit report. Relevant legislation includes the Water Act (587/2011) and associated Decree (1560/2011), which covers activities impacting on ground and surface water levels. Water pollution is regulated by the Environmental Protection Act (86/2000) and Decree (169/2000). AVI issues the water permit in conjunction with the environmental permit (integrated permit).

1.5 Natura 2000

If a project or plan, either individually or in combination with other projects and plans is likely to have significant adverse effect on the ecological value of a Natura 2000 site, the planner or implementer of the Project is required to conduct an appropriate assessment of its impact (including cumulative impacts). The assessment can also be undertaken as part of the Environmental Impact Assessment Procedure (468/1994), when agreed so with the authority.

The authority in charge of granting the environmental permit will see that the Natura assessment is carried out. ELY centre in charge of the Natura 2000 site in question must give a statement within six months at the latest. The environmental permit can be granted if a project's impacts do not compromise the purpose of the protection. Removing a site completely or partially from the Natura 2000 network requires EU Commission's acceptance.

1.6 Timing of environmental authorisation process

The estimated duration of the environmental authorisation process is given in App A Table 1. The timeframes for the various scenarios to complete the procedures and obtain an environmental permit are based on 2002-2012 processing times. Information source is the Ministry of Employment and Economy.

App A Table 1: Estimated duration of the environmental authorisation process

	Scenarios in months		
	Best	**Probable**	**Worst**
EIA procedure			
EIA programme	4	6	8
EIA report	5	9	20
Permit Application procedure			
Environmental permit report	Preparation mainly during the EIA procedure		
Environmental permit handling	9	12	16
Total duration (excluding appeal process)	19	27	36
Potential appeal handling	0	12-24	42
Total duration (including appeal process)	18	45	86

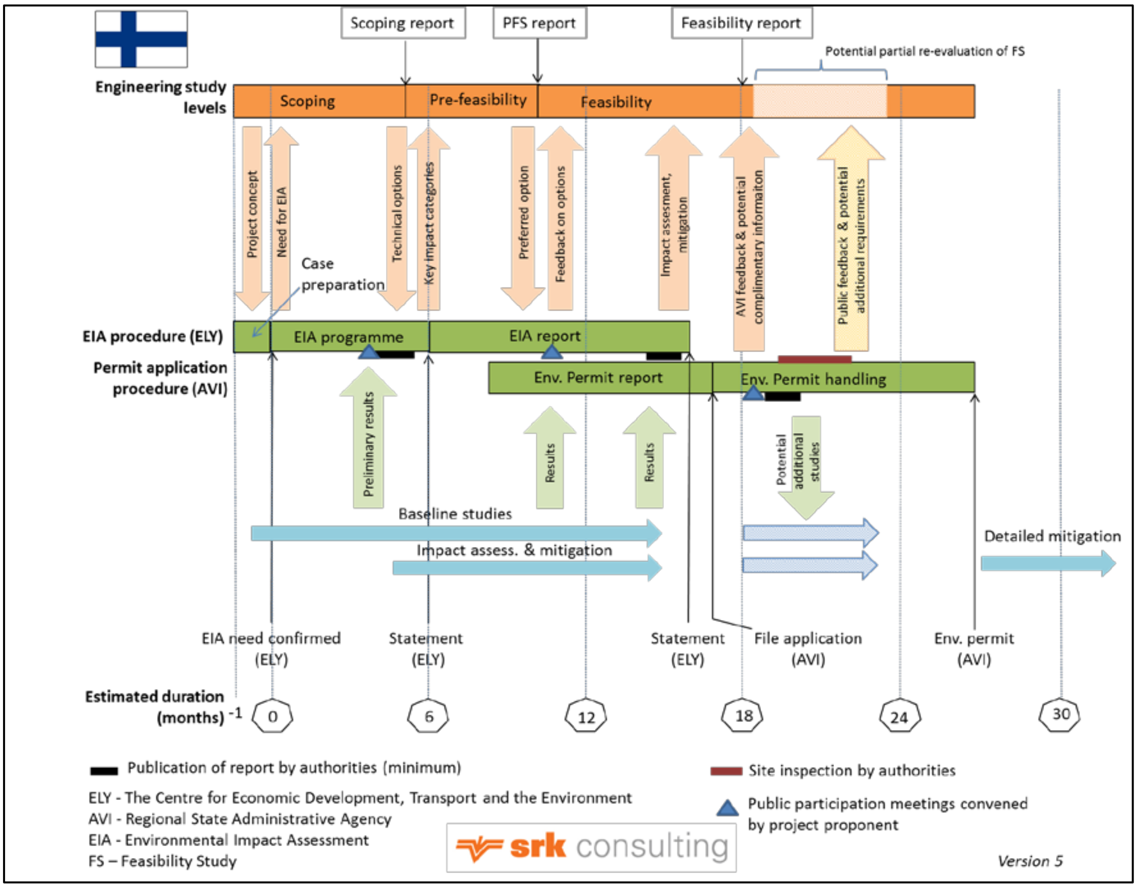
The probable scenario to obtain a permit is 27 months if the RoD is not appealed.

1.7 Links between environmental authorisation and engineering studies

An illustration of how the environmental authorisation process and engineering studies may be integrated is given in App A Figure 2. The linkages are based on the probable authorisation scenario in App A Table 1 for development studies for a small open pit mine. Please note these timeframes are rough estimates and linkages indicative. They will need to be determined specific to the project.

1.8 Legal requirements related to the closure

The European Directive (EU) 2006/21/EC – Management of Waste from Extractive Industries is the basis for Finnish requirements for mining waste management. The European Commission Reference Document on Best Available Techniques (2009) for the management of tailings and waste-rock in mining activities is a key reference point for management solutions. Funds for closure are required (in terms of the Environmental Protection Act 86/2000) and will be defined based on the mitigation measures in the conceptual closure plan. The Environmental Ministry has defined instructions to environmental authorities deciding on how to determine closure costs (9th March 2005 No. YM2/401/2003).



App A Figure 2: Environmental authorisation process links to engineering studies based on a probable scenario.

Other primary authorisations required are as follows.

1.9 Mining permit

The mining permit is primarily an access to a mineral resource. The proponent applies to The Safety and Chemicals Agency / Turvallisuus- ja Kemikaalivirasto (Tukes) for a mining permit for the development. The new Mining Act (621/2011) requires the EIA report and ELY's statement on the report to be appended to the permit application. Tukes publishes the permit application and collects statements from other concerned authorities and the public before making a RoD. Usually Tukes states in its RoD that environmental impacts will be regulated by the environmental permit conditions. Initially Tukes issues a draft RoD and thereafter instructs the local land survey authority to survey the concession. A final RoD on the concession is granted after the survey.

1.10 Land use plans

Regional Land Use Plans exist for all areas in Finland. In many cases there are also local land use plans (zoning and town plans) to guide construction and other land use changes in areas where land is used intensively. Land use and building planning is regulated by the Land Use and Building Act (132/1999). The project area´s Regional Plans are administrated by the Pirkanmaa Regional Council). Local Land Use Plan is administrated by Ylöjärvi Town. If a land use plan conflicts with the project, then the proponent suggests an amendment to the land use plan. The administrating authority initiates the process for re-planning and approval of the updated plan. The mining project cannot get environmental permit if it conflicts, for example, with a valid Regional Land Use Plan (and National Land Use Objectives).

1.11 Building permit

Approval of the building permit is required from the municipality prior to commencing construction. The application for a building permit is required to demonstrate rights of access to land for the project. The building permit also requires compliance with existing land use plans.

1.12 Land rights

The proponent's application for a building permit must demonstrate right of access to land in the mining permit. Land rights are generally settled in agreements between the proponent and landowner, however in some cases the proponent applies for a redemption right from the government. A redemption right requires the proponent demonstrate the project is needed from the community/society perspective. Forming a mining area via redemption right leads to the need for surveying of the mining areas by the land survey authority. In general, the mining permit area is registered to the real estate data base.

1.13 Derogation permit

The proponent may require derogation permits in terms of the Water Act (587/2011) if the development impacts on natural springs or creeks. Derogation permits may also be required in terms of the Nature Conservation Act (1096/1996, Chapters 48 and 49) and Forest Act (1093/1996, Chapters 10 and 11) if the development impacts on certain plant species, habitats and forests of conservation importance. Derogation permits are administered by ELY in terms of the Nature Protection Act (1096/1996) or Law of Cultural Heritage (295/1963). Deviations from the Water Act (587/2011) are handled by AVI. Nature protection and conservation requirements may also be required in terms of other legislation, e.g. impacts on land protected under the Mire Conservation Decree (852/1988) must be permitted by the Ministry of Environment.