

NEWS RELEASE

9 April 2009

DUE DILIGENCE REPORT – KYLYLAHTI PROJECT

In 2008, as part of the financing process for the Kylylahti Project, Vulcan Resources Limited's ("Vulcan") (ASX: VCN, FSE: VUA, WKN: A0HHEF, Norwegian OTC: VCNR) financial advisors commissioned Snowden Mining Industry Consultants to prepare a technical review for potential lenders.

The financing of Kylylahti did not proceed due to deterioration of metal and credit markets and this report is attached to this release and is available to investors for review.

Kylylahti's key commodity exposures, copper and cobalt, have performed well this quarter. LME Copper opened the quarter at multi-year lows of \$1.25/lb and has risen to \$1.90/lb at the time of writing. Similarly, 99.8% cobalt as quoted on MinorMetals.com has increased from \$10/lb to \$18.00/lb. Whilst demand for major industrial metals is still depressed it is encouraging to see the recovery of prices from recent lows.

Vulcan continues to hold the Kylylahti project in abeyance whilst investigating opportunity for project synergies within Finland.

- ENDS -

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About Vulcan

Vulcan Resources Limited has base and precious metals development and exploration projects in Finland. Vulcan is currently focussed on acquiring near term development projects.

The Company's principal asset is the 800,000 tpa 100% owned Kylylahti copper-cobalt project located in eastern Finland which has a Resource of 7.85 million tonnes grading 1.17% copper, 0.24% cobalt, 0.22% nickel, 0.49% zinc and 0.70 g/t gold (for Resource Classification, see ASX release dated 26/06/07).

A Definitive Feasibility Study has been completed on a 10 year underground mine and concentrator producing copper-gold and zinc-cobalt-nickel concentrate for sale. All environmental and mining permits for the project are in place.

The Kuhmo Nickel Project is 95% owned by Vulcan and has a Resource containing 38,000 tonnes of nickel metal and over 80,000 ounces of platinum and palladium (for Resource Classification, see ASX release dated 22/08/06).

Vulcan is listed on the Australian Stock Exchange (VCN), the Frankfurt Stock Exchange (VUA) and the Norwegian OTC (VCNR).

Competent Person Statement

The information in this report that relates to Exploration Results, Mineral Resources or Ore Reserves is based on information compiled and reviewed by Dr Alistair Cowden BSc (Hons), PhD, MAusIMM, MAIG who is a full time employee of the Company and has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2004 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. Dr Alistair Cowden consents to the inclusion in the report of the matters based on their information in the form and context in which it appears.

Cautionary Statement

No stock exchange, securities commission or other regulatory authority accepts responsibility for the adequacy or accuracy of this release or has approved or disapproved the information contained herein.

Statements regarding Vulcan's plans with respect to its mineral properties are forward-looking statements. There can be no assurance that Vulcan's plans for development of its mineral properties will proceed as currently expected. There can also be no assurance that Vulcan will be able to confirm the presence of additional mineral deposits, that any mineralisation will prove to be economic or that a mine will successfully be developed on any of Vulcan's mineral properties. Circumstances or management's estimates or opinions could change. The reader is cautioned not to place undue reliance on forward-looking statements.

Vulcan Resources Limited: Review of the Kylylahti Feasibility Study
Project No. 7343

October 2008

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This report has been prepared by Snowden Mining Industry Consultants ("Snowden") for Azure Capital Pty Ltd.

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1 Summary

1.1 Introduction

This report provides a critical review of Vulcan Resources Limited's ('Vulcan's') Kylylahti polymetallic copper-cobalt-nickel-zinc-gold project in southeastern Finland. Kylylahti Copper Oy ('KCO'), which is a Finland incorporated, wholly owned subsidiary of Vulcan, is the operator of the mine.

A concise description of each of the key technical areas of the project is presented, along with brief commentaries which present Snowden's review of each area. These sections are identified with the title 'Snowden comment...'' in order to differentiate project summary from expert opinion.

The main groups of documents used as reference by Snowden in this work were firstly those relating to the Definitive Feasibility Study ('DFS') at Kylylahti, carried out under the management of SNC-Lavalin but including work by a wide range of parties (including Vulcan), and secondly those documents relating to Vulcan's subsequent optimisation and review of the DFS. The details behind their sources are contained in the References section at the end of this report.

1.2 Terms of reference

Snowden was appointed by Azure Capital Pty Ltd ('Azure') as the Independent Technical Expert ('ITE') with respect to Azure's role as financial advisors to Vulcan for the development of the Kylylahti project. Vulcan, together with Azure, is considering a number of financing options for the project, and the ITE's report is intended to provide a background to these financing options by giving a concise summary of the technical aspects of the project, along with a summary of the main risks inherent in the project. Thus this report represents a technical due diligence which may be used to assist financing or as a submission to the Credit Committees of potential lending institutions. Snowden has appointed a number of associates to provide expert opinion in certain areas, but takes overall responsibility for the conclusions.

While the review covers most technical aspects of the Kylylahti project, a number of areas are specifically excluded, viz:

- Snowden was not required to validate the legal status of tenements and permits held by Vulcan.
- A full review of any financial model was not required; however, a critical assessment of the key technical inputs to these models was required.
- Snowden was not required to opine on metal price and exchange rate assumptions used by Vulcan – these parameters will be validated by Azure.

Snowden undertook this work along with a number of associates. The various parts of the report and the responsibility thereof are detailed in Table 1.1.

Table 1.1 ITE report – areas of responsibility

Area of report	Responsibility
Overall compilation, review and editing, TSF review, project management and liaison	Ian Glacken, Snowden Francois Grobler, Snowden
Mineral Resource review	Stefan Zanon, Snowden
Ore Reserve, mine planning, scheduling, mining capital and operating costs	Peter Myers, Snowden
Geotechnical review	Richard Fry, Snowden
Paste fill plant review	Karl van Olden, Snowden
Metallurgy, process engineering, infrastructure, operational aspects of the concentrator	Jeff West and Damian Connelly, METS
Environmental and permitting review	Michael Wakefield, URS
Quantitative analysis of cost overrun	Francois Grobler, Snowden

1.3 Summary of risks

A detailed review of project risks, using the semi-quantitative approach detailed in Australian Standard 4360 along with detailed commentary, is presented in Section 14. A detailed risk assessment, considering a variety of project parameters, the probability, the impact and the overall risk rating is presented as Table 14.5. A summary of those risks, by major project area, is presented below in Table 1.2.

Table 1.2 Risk summary by area

Project area	Overall risk	Comments
Mineral Resources and data quality	Low	Data collection is to high standards. The geological model is well understood but has areas of uncertainty which recent drilling indicates represent orebody extensions. Estimation is carried out to an acceptable level of precision using industry standard techniques and respected consultants.
Ore Reserves, mine planning, mine schedule and cost estimation	Medium	The Ore Reserve is based upon detailed stope design using industry standard mining methods. The schedule has some flexibility and future pre-production work should address any scheduling or critical path issues. The costs have been estimated from first principles and are subject to normal industry cost pressures.
Geotechnical	Low	Good database, although there may be some locational biases in the information collected. Ground conditions are good. However, further assessment of stope designs and extraction sequences are needed from the viewpoints of stability and ground support requirements.
Metallurgy and processing	Low	Process route is tried and tested both globally and in the local area. Personnel risk is medium but lower than most other areas of the world. Set-up and fine tuning of the flotation circuit requires availability of key skilled personnel which should pose little problems given the local processing resources available.

Project area	Overall risk	Comments
Environmental/permitting	Low	Permit to operate granted. Tailings storage facility (TSF) only classified as landfill. TSF able to be used as water supply buffer. Community concerns are minor and have been addressed by Vulcan. Monitoring has started.
Operational	Low - medium	Costs may be higher than budget, although contingency in capital expenditure is within industry range. Finely-balanced nature of mining schedule may affect downstream areas. Good availability of skilled workforce in local area, plus government training grants. Should be easy to retain Finnish staff that can carry out the vast majority of tasks. Vulcan expects to have a 100% Finnish workforce.
Infrastructure	Low - Medium	Climatic issues with respect to potable water and power supply have been anticipated and catered for by Vulcan in the project design. TSF expansion beyond design capacity is possible and should cause no geotechnical problems. Possible mechanical delays and supply problems.
Engineering Procurement Construction and Management (EPCM)	Medium	Delays and cost escalation always a potential issue. Long planned construction period provides flexibility in case of delay in long lead time items. Combination of Australian design skills and Finnish procurement and construction management expertise. Mine development will be tendered amongst Finnish and Scandinavian contractors.
Capital and operating costs	Low - medium	Capital costs seem reasonable. Contingency for concentrator is within the acceptable range. Paste plant and TSF costs are acceptable although TSF expansion needs to be considered. Mining operational costs very well estimated from first principles. Main components of concentrator operating costs (power and labour) are likely to be stable.

1.4 Completeness of the study

One of Snowden's terms of appointment was to opine on the 'bankability' of the DFS and Vulcan's subsequent optimisation. While Snowden does not favour the term 'bankable', an alternative and more rigorous definition of a feasibility study is one which:

- describes key aspects of a project
- justifies continued investment in that project
- provides technical, commercial and financial recommendations regarding the project
- identifies areas of risk.

Using this definition, Snowden believes that Vulcan has covered all of the technical, commercial and financial areas of the project (within the areas of Snowden's brief) and that the DFS and subsequent optimisation presented and reviewed by Snowden justifies continued investment in the Kylylahti project. Snowden, as ITE, has

identified areas of risk, and Vulcan has, in its comments relating to a draft of this report, highlighted how it intends to mitigate significant risks.

1.5 Study disclaimers

Snowden has based its findings upon information made known to it as at 31 July 2008. Snowden has endeavoured, by making reasonable enquiry of Vulcan, to ensure that all material information in the possession of Vulcan has been fully disclosed to Snowden. However, Snowden has not carried out any type of audit of the records of Vulcan to verify that all material documentation has been provided.

A draft version of this report was provided to Azure and to Vulcan for comment in respect of omission and factual accuracy, and this draft reflects Vulcan's comments. Azure has agreed to indemnify Snowden from any liability arising from its reliance upon inaccurate or incomplete information provided.

Snowden has prepared this report on the understanding that all Vulcan's granted tenements are currently in good standing and that there is no cause to doubt the eventual granting of any tenement applications. Snowden has not attempted to establish the legal status of the tenements within each project area with respect to potential environmental and access restrictions. Snowden also has not independently verified ownership and the current standing of Vulcan's agreements and is not qualified to make legal representations in this regard.

Snowden is an independent subsidiary of Downer EDI Limited which provides specialist mining industry consultancy services in the fields of geology, exploration, resource estimation, mining engineering, geotechnical engineering, risk assessment, mining information technology and corporate services. The company, with its principal office at 87 Colin Street, West Perth, Western Australia, also operates from offices in Brisbane, Johannesburg, Cape Town, Vancouver, Belo Horizonte and London and has acted as Independent Technical Experts on a variety of projects covering a range of mineral commodities in many countries.

Neither Snowden nor those involved in the preparation of this report have any material interest in Vulcan or in the mineral property considered in this report. Snowden has been remunerated for this report by way of a professional fee determined according to a standard schedule of rates which is not contingent on the outcome of this report.

Snowden and its associates confirm that all reasonable care has been taken to ensure that all information contained within this ITE report is, to the best of its knowledge, in accordance with the facts and contains no omissions likely to affect its import.

2 Project overview

2.1 Introduction

Vulcan Resources Limited ('Vulcan') owns the Kylylahti polymetallic copper-cobalt-nickel-zinc-gold project in southeastern Finland. It is located about 400 km northeast of the national capital Helsinki, 24 km northeast of the historical mining centre of Outokumpu, and about 2 km west of the town of Polvijärvi. Regional infrastructure is good, with grid hydroelectric power available, sealed roads, and an extensive rail network with a railhead at Vuonos, 15 km to the west of Polvijärvi. Regular scheduled air services connect the regional centres of Joensuu, 40 km to the southeast, and Kuopio, 100 km to the west, with Helsinki.

Kylylahti was discovered by the Finnish mining company Outokumpu Oy ('Outokumpu') in 1984, and was considered for development by Outokumpu in 1986 and 1996. The project was sold to Australian company Dragon Mining NL in 2003, which on-sold it to Vulcan in December 2004. Vulcan completed a Pre-Feasibility Study ('PFS') for the development of the project in October 2005, which included reviews of previous metallurgical studies, a new resource estimate, and mining and environmental studies.

The project is secured by four mining leases covering 181 hectares ('ha'), and four mineral claims covering 234 ha over a total of about 4.1 km². The underlying land is privately owned, mostly by the Polvijärvi municipality.

2.2 Definitive Feasibility Study

The Definitive Feasibility Study ('DFS') for Kylylahti was prepared by SNC-Lavalin Australia Pty Ltd ('SNC'), in conjunction with a large number of subcontractors. The draft DFS was issued in November 2007 and initially considered the PFS base case of a 500,000 tpa operation producing copper-gold concentrate for sale to smelters and a bulk zinc-cobalt-nickel concentrate ('bulk concentrate') for further processing. The bulk concentrate was to be transported to a third party roaster for roasting. A pressure acid leach vessel and a hydrometallurgical plant were to be constructed for the production of high value, high payability intermediate products. The DFS was carried out on the PFS base case but, as costs were made available and it became apparent that a new roaster would be required, alternative project options were pursued. The DFS subsequently focussed on optimisation and value engineering of the base case, de-bottlenecking the flowsheet and considering alternative approaches to processing bulk concentrate.

Vulcan, together with a group of subcontractors, carried out reviews and optimisation of the DFS between November 2007 and March 2008. The optimisation and reviews have resulted in the current case which does not include a concentrate processing facility, but which considers the sale of bulk concentrate for processing by others. This report is based upon information produced by Vulcan and its subcontractors following the optimisation and review stage; however, the technical evaluation, simplification and cost reduction of the various aspects of the process continues to be investigated by Vulcan..

2.3 Mineral Resource estimate

The current Mineral Resource estimate for the Kylylahti deposit is that generated in May 2007. This was generated by Vulcan staff in conjunction with noted industry consultants The Quantitative Group ('QG'), a staff member of which assumed the role of Competent Person, as required by the Australasian Code for the reporting of Exploration Results, Mineral Resources and Ore Reserves (the JORC Code, 2004).

The resource is tabulated in Table 2.1 . In addition to generating the resource estimate QG audited and reported on the quality of the geoscientific data supporting the estimate and the integrity of the database, which includes historic Outokumpu data.

Table 2.1 Current (July 2008) Kylylahti Mineral Resource estimate

Resource Classification	Tonnes	Cu (%)	Co (%)	Ni (%)	Zn (%)	Au (g/t)
Measured	686,000	1.17	0.24	0.22	0.36	0.50
Indicated	6,973,000	1.17	0.24	0.22	0.50	0.72
Inferred	195,000	1.34	0.28	0.23	0.62	0.96
Total	7,854,000	1.17	0.24	0.22	0.49	0.70

The Kylylahti deposit comprises two lenses; the upper Wallaby zone and the lower Wombat zone. These have been well defined by drilling apart from the lower limit of the Wombat zone (wherein orebody extension potential exists) and the zone of overlap of the Wallaby and Wombat zones. In Snowden's opinion the resource categorisation applied fairly reflects the quality of the data, the precision of the estimation technique, and the relative knowledge of the respective parts of the orebodies.

It appears that recent drilling by Vulcan will result in an expansion of the Mineral Resource; however, this has not been considered in this study.

2.4 Mine plan and Ore Reserve estimate

As part of Vulcan's optimisation of the DFS one of the major changes was to increase production rates from 550,000 tonnes per annum ('tpa') up to 800,000 tpa from an underground mine, which will be accessed by a decline from surface. This work was carried out by consultants SRK, which in conjunction with the production rate increase revised the mine design, the stoping and development schedule and the mining cost estimates. In general operating costs were reduced from the DFS by optimising stope sizes, increasing capital development, reducing DFS errors and by improving the paste fill schedule. The current life of mine plan is for ore production over 11 years. The associated Ore Reserve (again reported in accordance with the JORC Code, 2004) is provided in Table 2.2.

Table 2.2 Current (July 2008) Kylylahti Ore Reserve

Reserve classification	Tonnes	Cu (%)	Co (%)	Ni (%)	Zn (%)	Au (g/t)
Proved	603,714	1.11	0.23	0.20	0.36	0.50
Probable	6,342,095	1.17	0.24	0.20	0.50	0.72
Total	6,945,809	1.17	0.24	0.20	0.49	0.70

The mining method chosen for the Kylylahti orebody is a combination of longitudinal open stoping and transverse open stoping (to be used for the wider ore zones in the Wombat zone). Both methods are well established and appropriate for the morphology and the spatial distribution of ore zones. The stopes will be sequentially filled using paste fill technology (delivered from a custom-built paste fill plant on the surface and using predominantly concentrator tailings) which allows for a high extraction ratio. Stopes were designed according to the guidelines imposed by the mining methods and the fill sequence, leading to a mining inventory. This

was then converted in to an Ore Reserve through the addition of modifying factors, predominantly dilution and ore loss (which was incorporated into a single stage adjustment). An economic filter was also applied to the stope inventory to generate the Ore Reserve. It is Snowden's view that the mining inventory to Ore Reserve conversion process is sub-optimal and could be tightened up to increase the overall value of the mining schedule.

Ventilation has been designed as a push/pull system, with fresh air forced into the mine via surface-mounted downcast fans and exhaust air drawn out through exhaust raises. Intake air will be heated to improve the underground working environment. Snowden believes that the ventilation plan has been constructed to a high degree of detail.

The mine schedule is designed to achieve the 800 ktpa production rate. This is based upon reasonable development and stoping assumptions and presents an achievable project outcome. The proposed equipment fleet also appears reasonable.

Vulcan has carried out a full geotechnical investigation programme based upon a variety of test holes. All such holes were fully structurally logged and subject to the normal battery of rock strength and failure tests. Snowden believes that the geotechnical investigation programme has been carried out in accordance with industry standard approaches, with one possible issue that the actual geotechnical information is not evenly spread throughout the orebodies, raising the potential issue of bias.

2.5 Processing details

The DFS was based upon the concept of producing two concentrates, namely a copper-gold concentrate (containing between 27 and 28% copper and between 10 and 14 g/t gold) and a bulk concentrate, containing cobalt, nickel, zinc and copper. Approximately 90% of the concentrate produced over the life of the mine will be the bulk concentrate. The DFS further went on to investigate roasting of the bulk concentrate and the sale of various products. This rather complex scheme has been greatly simplified in the Vulcan optimisation to the generation of the two concentrates as before and then the sale of each individual concentrate product via off-take agreements. This approach has greatly reduced the risk profile of the entire project, which now comprises industry standard, tried and tested crushing, grinding and flotation activities.

Mineralogy, crushing and flotation testwork has been carried out by well respected organisations such as Outotec in the local mine region and by AMMTEC in Perth. As a result of the DFS review a single stage, fully autogenous grinding mill with associated pebble crusher has been adopted. Of note is the fact that this scheme was adopted in the old Outokumpu mines in the area.

The flotation circuit is fairly standard and consists of a copper removal stage followed by cleaning of the tail to generate a bulk concentrate. The copper-gold concentrate is optimised for metal (i.e. copper and gold) recovery, whereas the bulk concentrate constitution has been optimised for sulphur recovery, which is a key driver for bulk concentrate purchasers.

The change from two mills to one has had a major impact on the project's capital cost but has slightly increased the risk associated with this area of the process; however, this arrangement has been shown to work successfully on this type of ore, most notably in the Outokumpu district. Snowden believes that the mill throughput has been conservatively designed and does have some flexibility for increasing the crushing rate. The flotation circuit has also been designed with some flexibility, although a slight risk is the possible lack of expert technical help to fine-tune this circuit in the early days of the project.

In general the process plant is not overly complex or sophisticated, and Snowden expects that Vulcan will be able to achieve design production and efficiency within a three month time frame from the start of wet commissioning.

2.6 Infrastructure, tailings disposal and paste plant

The infrastructure and transport networks in Finland are excellent. The project is in a rural area with good access to the main road network (the concentrator will be 400 m from a sealed highway) and within 2 km of the village of Polvijärvi, which is expected to supply a significant proportion of the permanent workforce.

The mine site layout has provision for a concentrator as previously described, administration offices, temporary stockpiles for acid generating and non-acid generating waste material, a box cut area which provides surface access to the underground mine, a mine ore pad adjacent to the concentrator, a paste plant for supply of paste fill to the underground mine, and a tailings storage facility ("TSF"). The project will be connected to the Finnish grid power supply system (which currently offers power at reasonable rates), and to the Polvijärvi potable water supply system and the local sewerage system.

It is intended that up to 70% of the concentrator tailings are returned underground as paste fill, along with the vast majority of the waste rock from underground development; thus the ultimate capacity of the TSF is approximately 1.5 Mt. The potential tailings material has been tested and found to be non-acid generating; notwithstanding this Vulcan has elected to line the TSF to prevent discharge to the local groundwater system. This is mainly due to a change in the solids content of the tailings material from the 65% allowed for in the DFS to 35% in the review and optimisation. This lower solids content allows Vulcan more flexibility in handling mine water sources, including the option of using the TSF as a temporary water storage area.

The TSF design and associated testwork has been carried out by prominent consultants Golder Associates. Snowden views the TSF design and construction as low risk, particularly as the decision to line it has now been taken. However, Vulcan may wish to plan for an expansion of the TSF over and above the design in consideration of an ultimate and currently unplanned increase in the size of the mine and a consequent increase in the amount of tailings.

The paste fill plant was designed initially by Golder PasteTec and the review and optimisation work was carried out by Turnberry RPA. Paste fill comprising 35% solids will be delivered to the plant from the concentrator and will be thickened at the plant before delivery underground via a reticulation system under the force of gravity. Turnberry's estimate of the utilisation of the paste fill plant (largely confirmed by Snowden) is 72%, which provides some flexibility if changed and fill and stoping schedules are to be adopted.

Snowden views the design, operating specifications and cost of the paste fill plant to be within normal expectations.

2.7 Permitting, environmental and social aspects

The Vulcan DFS for the Kylylahti project has addressed the requirements of the Environmental Impact Assessment Act (468/1994), the Environmental Protection Act (86/2000) and the Water Act (264/1961) for granting of an environmental permit for commencement of mining. The proponents have instigated broad community consultation and have attained a reasonable degree of support from both the community and the municipal authority. The requirement for employment opportunities and development in the district has no doubt reinforced the positive outlook on the development.

The waste rock has been characterised with respect to acid generating potential. The majority of the material has been characterised as non-acid generating, with a small amount classified as potentially acid generating.

Vulcan has submitted two samples of ultramafic rock from Kylylahti to the GTK (Finnish Geological Survey) for the purposes of testing for fibrous minerals. The tests revealed no suspect minerals. Visual examination of some diamond drill core indicated some potentially fibrous minerals, but these were neither confirmed nor eliminated by further testing.

The mine and plant developments are in close proximity to sensitive surface water ecosystems. However, the Kylylahti mining district is a well established industrial zone and there is expectation and acceptance of environmental impacts from the development. This expectation is reflected in the 73 environmental permit conditions.

The potential for elevated total suspended solids and nickel loadings entering Lake Polvijärvi is considered significant given the permit conditions could well be exceeded by a small variance in the anticipated waste water loadings. Therefore, the potential for negative environmental monitoring results appears quite high and the situation may well arise where additional waste water treatment protocols will be required to achieve permit conditions.

A preliminary Mine Closure Plan has been generated and this is a valuable guide in determining the methodology for rehabilitation and management of waste materials from the start of mining activities. This is particularly relevant in terms of the proposed management of the TSF, where there appear to be some inconsistencies including:

- whether the TSF will be kept flooded to reduce the oxidation of sulphides contained in the tailings
- whether there is significant acid generating potential in the tailings
- whether the TSF will be capped and sealed as a final rehabilitation strategy.

The climatic conditions in Finland are such that Vulcan expects sulphide oxidation to be minimal.

Groundwater levels at 1.5 to 3 m below ground level will be impacted by the hydraulic load of the TSF and will lead to a rise in static groundwater levels or mounding of groundwater in close proximity to the TSF. It is anticipated that should groundwater saturate the base of the TSF then seepage of tailings liquor to groundwater will be enhanced. Depending upon the movement of groundwater in the project area catchment, this could lead to impaction of the off-site environment.

Comment was also made that the paste plant may utilise fly-ash as a substitute pozzolanic binder to cement. Although fly-ash is known to modify concrete properties to allow for greater flowability and reduced dehydration cracking, the use of fly-ash would be limited by the concentration of readily soluble heavy metals normally found in fly-ash. Well defined leachability criteria would be required prior to utilising fly-ash in what may well end up as a sulphate ion enriched, inundated underground storage repository. Sulphate attack on concrete could well result in liberation of heavy metals into the groundwater environment and may reduce the stability of backfilled stopes.

The close proximity of residential properties to the project area appears to present several potential on-going issues. The issues of noise, dust and mine run-off seem to be obvious and may well become problems to be resolved. The restriction of crushing hours, due to the requirement to limit noise impacts, may cause on-going production bottlenecks.

The transport of copper-gold concentrate 400 km across the country has the potential for impactation of ecosystems along the rail line, particularly in the event of a derailment. However, this is considered a minor risk. The transfer of loads at the Vuonos railhead has the potential for both the liberation of dust and the emission of odours (carbon disulphide), particularly during summer months. This could become an odour nuisance issue over time.

Overall, the Vulcan DFS has provided sufficient environmental management evidence which has allowed the permitting authority to grant an environmental permit for the commencement of mine development. Vulcan has also submitted a draft Mine Closure Plan.

The €400,000 bond required by the end of mine life sits at approximately 23% of estimated sustaining capital. The estimate of sustaining capital is consistent with the recommendations in the Finland Mine Closure Handbook 2008.

Waste water discharge to Lake Polvijärvi appears to be the most significant potential source for on-going liability arising from the development. The potential for liability arising from environmental damage, due to compliance with permit conditions, should be carefully reviewed and understood.

2.8 Transportation and logistics

As described above, transportation, whether by road and rail, is very easy within the region and within Finland as a whole. The nearest railhead is at Vuonos, 15 km to the west of Polvijärvi. Regular scheduled air services connect the town of Joensuu, 40 km to the southeast, with Helsinki and other regional destinations within Finland. It is not anticipated that seasonal snowfall will provide significant disruptions to transport, but notwithstanding this an allowance has been made in both the project schedule and the operational schedule for the effects of adverse weather.

2.9 Capital costs

A capital cost summary for the Kylylahti project is presented in Table 2.3. This assumes a Euro to USD conversion rate of 0.74 (0.63 in mid-July). The contingency rates vary from 1.8% for mining up to 65% for the Owners costs, but average at just under 14%, which is well within the range of error for projects such as this. Of note is that the contingency on the main capital item, the concentrator, is about 9%, which may be considered a little low.

Table 2.3 Capital cost summary

Capital costs	€(M)	US\$ (M)
Pre-production mining, decline, etc	28.64	43.72
Concentrator, utilities and infrastructure	49.47	75.53
Tailings storage facility and paste plant	7.98	12.18
Owners costs	10.38	15.85
Subtotal	84.80	147.28
EPCM	11.84	18.08
Contingency	13.23	20.20
Total	121.54	185.56

The total sustaining capital for the life of the mine is estimated at about €15.8M, of which 72% is associated with the mine. Another large item (€2.1M) is associated with the second lift of the TSF, due in 2016.

Snowden views the sustaining capital cost estimates as reasonable.

2.10 Operating costs

A summary of operating costs is provided in Table 2.4.

Table 2.4 Operating cost summary

Operating costs	€ (per t)	US\$ (per t)
Mining	20.88	25.06
Processing	11.39	13.67
Overhead and administration	3.12	3.70
Concentrate transport	4.94	5.93
Total	40.33	48.36

The largest component of the operating costs is the mining cost. This cost has been developed on the basis of utilising local contractors for all waste development, including ground support, and all loading and hauling of waste and ore. Vulcan employees will undertake ore development (excluding mucking), ground support and drill and blast activities. Snowden's view is that the mining cost model, which has been developed by SRK from first principles, including quotations, has a high degree of precision.

The largest component of the processing operating cost is the power cost and another significant component is labour costs. Snowden is of the view that these costs are reasonable, and may be expected to stay reasonably stable as the majority of the power is generated by hydro-electric schemes, and, moreover, there is currently a shortage of employment in the local region which will help to keep labour costs fairly constant.

Overhead and administration cost contribute 8% of the total operating costs and provides for personnel (remuneration and salaries), insurances (e.g. material damage, business interruption, marine cargo, public liability etc.) and general administrative items and allowances.

Concentrate transport equates to just over 12% of total unit cost and includes the transport, handling and storage of copper-gold concentrate and bulk concentrate to the rail sidings and ultimate destinations.

2.11 Technical inputs to cost model

Snowden has assessed the capital and operating cost inputs to the overall project cost model and finds that they have been estimated with a medium to high degree of diligence. The other technical inputs to the cost model are the mine and process plant physicals. These have been commented upon in other sections of this report, but in summary Snowden finds the mine schedule to have sufficient flexibility and to be reasonable and achievable, and the processing schedule also presents a moderate to high degree of flexibility in terms of changing ore blends and ramp-up time.

Snowden notes that other inputs to the cost model, such as estimates of inflation, exchange rates and commodity prices, are beyond the scope of the ITE.

2.12 Estimate of cost overrun

The assessment of the potential for cost overrun during project execution focussed on the uncertainty in the base capital cost estimates for individual cost items. These figures relate to the uncertainties in assumptions used in arriving at representative numbers for:

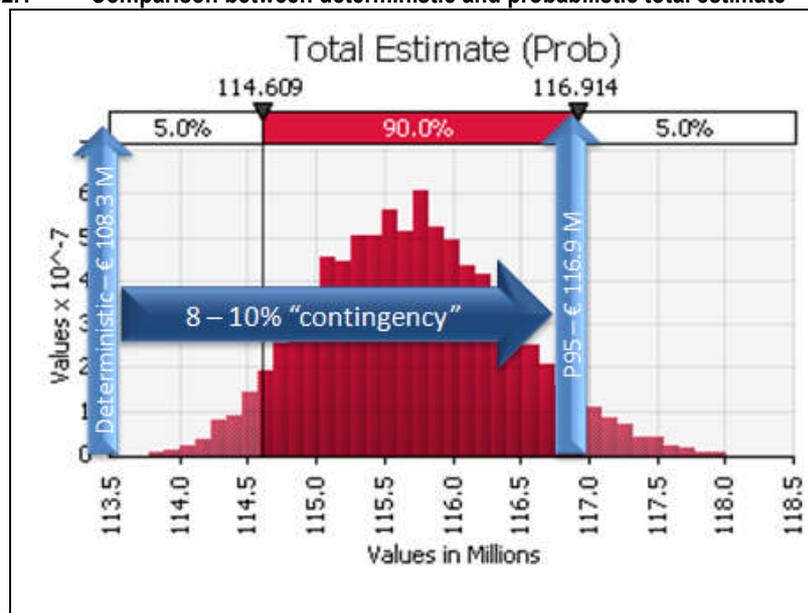
- “off-the-shelf” equipment (e.g. mills or crushers)
- quantities (e.g. m² of plate work, m³ of concrete, lengths of pipe or conveyor belt)
- unit costs (cost per unit used) of bulk commodities, and
- installation time and labour costs inherent in the construction and implementation phase.

The assessment did **not** address the cost implication of schedule risks due to delays, or uncertain effects of inflationary escalation between the time of estimation and the actual dates of procurement and acquisition. The likely effects of these considerations should be added to the contingency or overrun value derived through this analysis.

Snowden focused on quantifying the uncertainty in the base cost estimates by using a modified version of the contractor’s cost accuracy indicators. Subsequent to setting up statistical distributions and converting deterministic models to probabilistic models, Monte Carlo simulations were run and the resultant variability in the capital estimation total derived.

The results of the simulation have shown that the variability in the total estimate when compared to the deterministic total estimate amounts to 8% when compared to the probabilistic model at the 95th percentile. Snowden therefore recommends that a cost overrun or contingency range of **8% to 10%** be considered. This can be visualised in Figure 2.1.

Figure 2.1 Comparison between deterministic and probabilistic total estimate



The potential effects of inflationary escalation and schedule delays have not been quantified and should be added to the 8 % to 10% range in order to arrive at a total overrun or contingency value.

Snowden is of the opinion that in relation to cost overruns on engineering projects as a general trend, a benchmark of 2% to 5% of additional cost can be considering in addition to the values quantified above. Snowden believes however that due to the relatively low likelihood of overruns related to labour in this case, as well as relatively high confidence in timely delivery of critical capital equipment fabricated locally (in Finland), the likelihood for cost over run would rather tend to the lower than the higher end of this range.

Considering the above reasoning thus, a total cost over run range of 10% to 15% is proposed with a preferred value of 12%.

3 Geology and resource estimation

3.1 Regional and local geology

The Kylylahti project is located approximately 22 km northeast of the historic mining town of Outokumpu in the Karelia district of Finland. The Keretti copper-cobalt-zinc-gold deposit, which underlies the modern town of Outokumpu, was discovered in 1910 following the location of the source of a massive sulphide glacial boulder found some 50 km to the southeast. The mine operated continuously between 1914 and 1989, producing some 28 Mt of ore with a grade of 3.8% copper, 0.24% cobalt, 1% zinc and 0.8 g/t gold. In 1965 the Vuonos deposit was discovered about 10 km to the northeast of the Keretti mine, and produced 5.5 Mt of copper-cobalt-zinc-nickel-gold ore until closure in 1985. In all, and excluding the Kylylahti deposit, approximately 48 Mt of copper-cobalt-zinc-nickel-gold ore was produced or delineated from more than 40 occurrences in the Outokumpu area up to 1986 (Kontinen, 1985). The Kylylahti deposit was discovered in 1984 by Outokumpu geologists using diamond drilling following the application of geophysical techniques within rocks which form part of the Outokumpu trend.

The Outokumpu mining district (including the Kylylahti deposit) is situated within the North Karelia Schist Belt. This is a structurally complex package of amphibolite to granulite facies metasedimentary rocks located at a major crustal tectonic boundary between the Proterozoic Svecofennian belt of rocks to the southwest and the Archaean Karelian Craton to the northeast. The Outokumpu district is dominated by northeast-southwest striking isoclinal folds with subvertical fold limbs. The regional geology and main Outokumpu-type deposits are shown in Figure 3.1. The numbers represent the copper-cobalt-zinc-gold deposits of (1) Outokumpu, (2) Luikonlahti, (3) Vuonos, (4) Saramäki, (5) Kylylahti, (6) Perttilahti, (7) Riihilahti, (8) Hietajärvi, (9) Kettukumpu, (10) Hoikka, (11) Sola, and the nickel-cobalt deposits and prospects of (12) Outokumpu, (13) Vuonos, (14) Kokka, (15) Poskijärvet and (16) Petäinen.

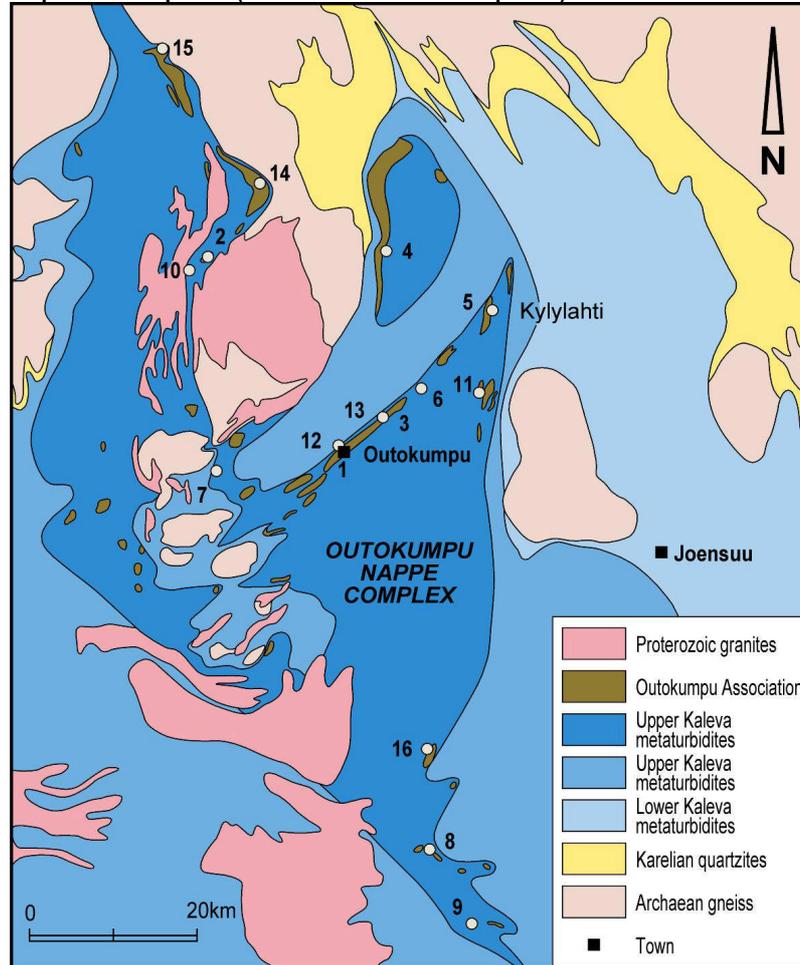
The Outokumpu style deposits are currently thought to have formed as a result of complex multiphase processes involving remobilisation of copper-cobalt-zinc sulphide rich rocks deposited on the ancient sea floor and their interaction with nickel sulphide rich rocks within the Outokumpu assemblage. Multiple stages of deformation coupled with this interaction have led to the thin but laterally extensive rocks (up to 4 km at Keretti) forming the lenses of mineralisation which have been mined since 1914.

3.2 Deposit geology

The Kylylahti deposit is hosted within a package of serpentinite, talc-carbonate, tremolite-quartz and quartz-sulphide rocks which form a distinctive association (the Outokumpu Association) within the district. The ultramafic rocks have been metasomatically altered to a tremolite-quartz-sulphide assemblage in close proximity to the Outokumpu deposits. This assemblage is termed a skarn throughout the Outokumpu region, including the Kylylahti area. Kylylahti mineralisation is hosted at the contact of these rocks and black sulphidic shales. Outokumpu Association rocks occur as pods or lenses of serpentinite, altered to talc-carbonate on the margins, which have been complexly deformed and elongated within the black shales. The sequence of ultramafic rocks and enclosing shales is interpreted to have been thrust into a thick sequence of mica schists and gneisses (Figure 3.2). The folded belt of Outokumpu Association rocks can be traced over a strike length of 300 km. At Kylylahti these rocks show multiple phases of deformation and strong

foliations within a tight synformal fold structure, with the mineralisation located along the near vertical eastern limb.

Figure 3.1 Regional geology of the Outokumpu area showing the location of serpentinite deposits (see text for names of deposits)



The location of a cross-section (Figure 3.3) is shown on Figure 3.2, and illustrates the podiform nature of the Kylylahti mineralisation, along with a recognisable coarse-grained core containing semi-massive sulphides surrounded by disseminated sulphides on the foliated black schist contact. The semi-massive mineralisation comprises 40% to 60% sulphide (predominantly pyrrhotite, pyrite and chalcopyrite, with subordinate local accumulations of cobalt-rich pentlandite, sphalerite, cobaltite and gold), and ranges in thickness from 5 m up to 20 m.

The disseminated zone contains medium to coarse grained sulphides (5% to 40% sulphides) and veinlets, with pyrrhotite predominating and lesser amounts of chalcopyrite, pyrite, cobalt-rich pentlandite and sphalerite. The disseminated zone is locally gold-rich, with grades of up to 20 g/t gold. The semi-massive zone grades sharply into the disseminated ore over one to two metres, although isolated pods of semi-massive mineralisation may occur entirely within the disseminated zone.

Mineralisation at Kylylahti occurs in two elongated lenses which strike to the northeast, dip near vertically to the northwest and plunge at between 25° and 40° to the southwest. The total length of the mineralised corridor as currently defined by Vulcan is 1.5 km and the orebody is open at depth. It is of note that the

mineralised systems at Keretti and Vuonos are both 3 to 4 km in length. Vulcan has named the two lenses or zones Wallaby and Wombat.

Figure 3.4, showing the semi-massive zones in red and the disseminated zones in brown (the grid represents 100 m cubes). Recent drilling has focussed on filling in the gap between Wallaby and Wombat, and has had some success in this regard. Vulcan is employing directional drilling and sophisticated electromagnetic ('EM') prospecting techniques for the first time in Finland to explore the limits of the Wallaby zone. A portion of the upper part of the orebody sits outside and to the north of Vulcan's lease on ground owned by Mondo Minerals Oy (a talc mining and processing company). Vulcan has an agreement with Mondo Minerals giving it rights to the base metals on the Mondo ground.

Snowden comment

The Kylylahti deposit lies within a region and in a geological environment which has a long history of mining. As such the nature of the mineralisation and the metallogeny are well known, although the local controls on ore localisation are becoming apparent through deposit-scale drilling. The mineralisation can be identified visually, leading to low risks associated with the identification and delineation of the orebodies. In Snowden's opinion there is considerable upside in the quantity of ore to be discovered, particularly in the down-dip area of the Wombat zone and in the somewhat complex area of overlap of the Wallaby and Wombat zones.

Figure 3.2 Surface geology of the Kylylahti area after Kontinen (2005)

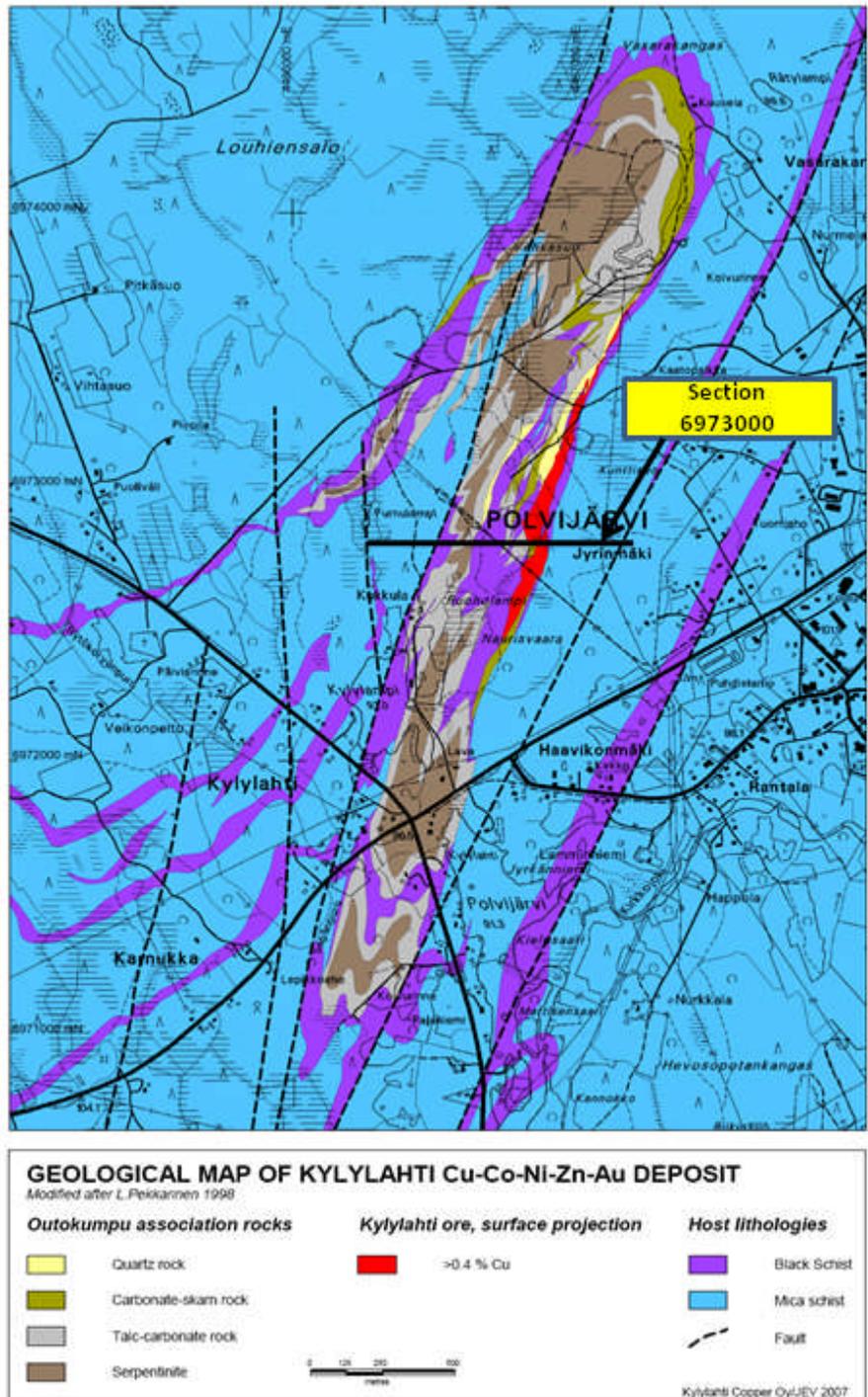


Figure 3.3 Cross-section at 6973000N through Kylylahti

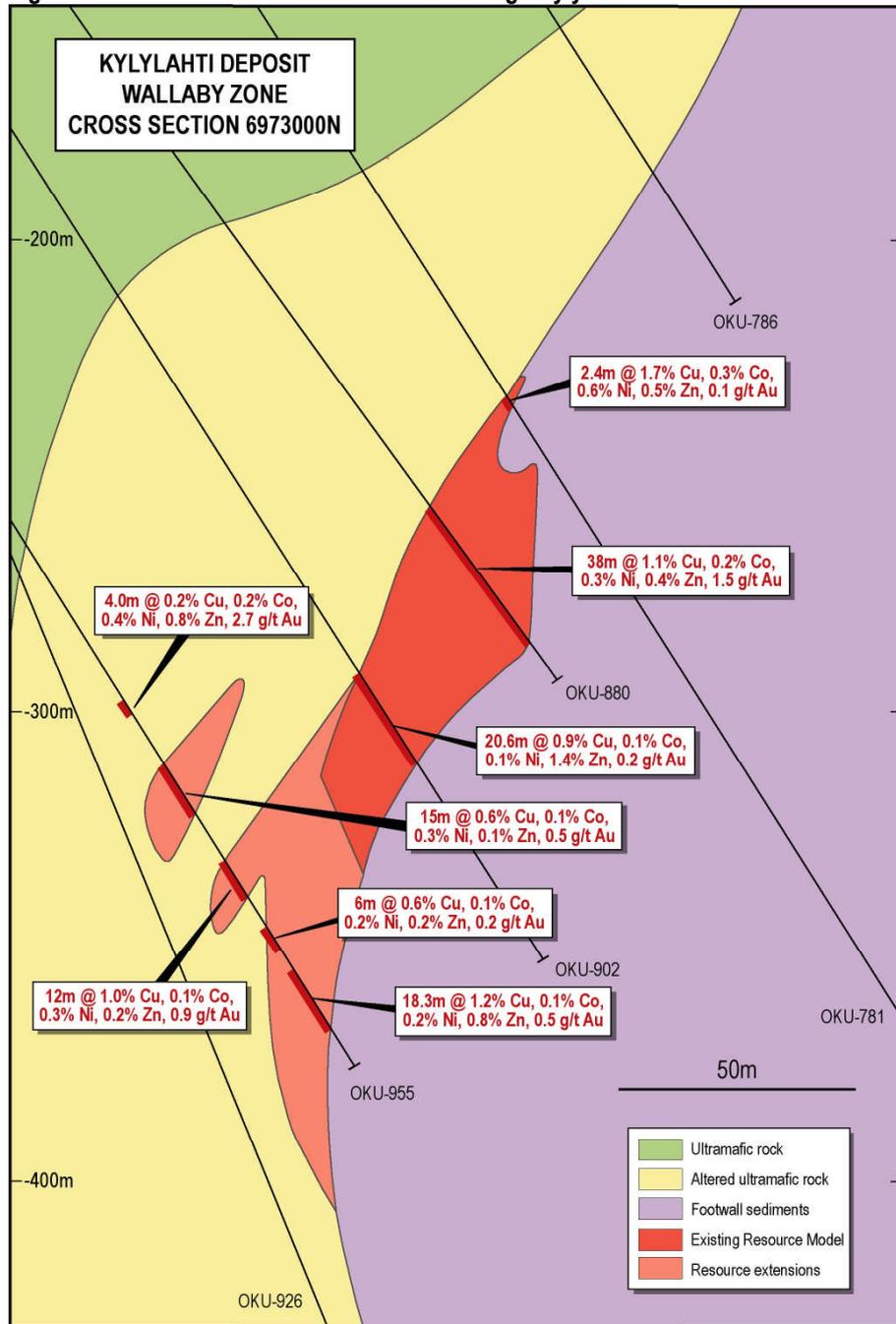
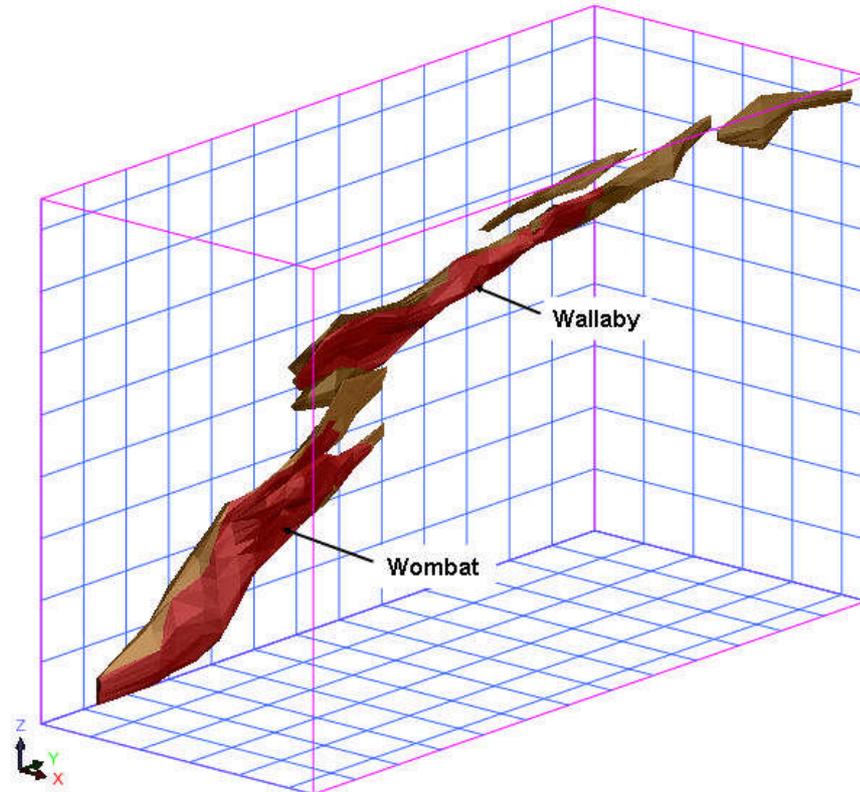


Figure 3.4 Three dimensional view of the Kylylahti orebody looking northwest, showing the distribution of semi-massive (red) and disseminated sulphides (brown)



3.3 June 2007 Mineral Resource estimate

3.3.1 Quality assurance and quality control

One of the key areas of any resource estimate is the certification of the underlying data quality. This depends in part on good quality assurance and quality control ('QAQC') procedures. QAQC in its broadest sense encompasses drillhole collar surveys, downhole surveys, data entry and density determination in addition to the more traditional meaning of assay quality assurance.

Vulcan has devoted considerable effort into ensuring the quality of the drilling data, particularly the pre-Vulcan (Outokumpu) drilling. Quality checks in this area include re-surveying of drillhole collars and downhole inclinations. In the area of assay quality control a number of the Outokumpu intersections were re-analysed and compared with the original values, with favourable results.

For its own drilling Vulcan introduced industry-standard QAQC procedures, including the use of duplicates, assays and blanks. This has ensured that the data collected by Vulcan is of high quality.

Snowden comment

Quantitative Geoscience ('QG') has reviewed Vulcan's QAQC procedures and verification of the Outokumpu work and has endorsed it. Snowden concurs with QG that the QAQC is of a good industry standard.

3.3.2 Overview of geological modelling

In June 2007, the Kylylahti Mineral Resource estimate was updated by Vulcan with the majority of the technical work and the review processes being performed by QG, which has a long-standing relationship with Vulcan and has previously provided modelling advice and review. The June 2007 estimate, an update to the

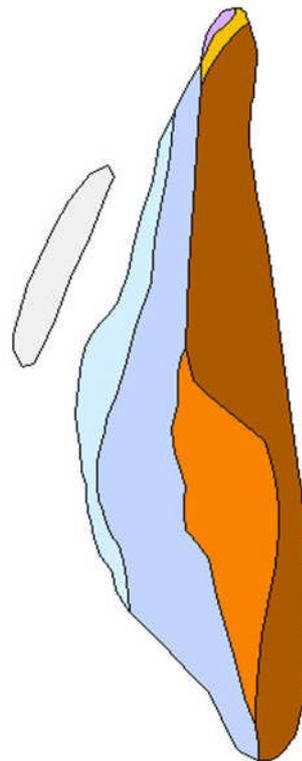
previous August 2006 Kylylahti Mineral Resource estimate includes additional drilling and a new geological interpretation inside the mineralised envelope. Vulcan supplied interpretations of the new geological domains and additional data to QG, and QG updated the assay top cut values, variogram models and kriging neighbourhood analysis. QG used this updated information to estimate into the geological block model, and validated and classified the resultant Mineral Resource model according to the guidelines in the JORC Code, 2004. Snowden carried out an independent validation of the resource block model estimates against the input drillhole sample data for the purposes of this document.

Mineralised areas of differing geological and statistical characteristics generally require the determination of separate geological domains for resource estimation purposes. The validated drillhole database for the Kylylahti project was used to define three dimensional domain shapes within the Surpac mining software package based on assay values and associated geological observations.

In terms of the definition of resource estimation domains, three different styles of mineralisation have been classified at Kylylahti; these are represented by the semi-massive sulphide (SMS), disseminated sulphide (DISS) and hangingwall domains. Both the DISS and SMS domains have further been subdivided into two separate areas, referred to as the Wallaby and Wombat zones. Both zones have similar orientations and mineralisation styles, whereby the SMS is hosted inside of the DISS domain. The largest zone of hangingwall mineralisation runs parallel to the Wombat zone, but forms a series of discontinuous pods to the west of the main orebody in both zones. The size and orientation of these domains are similar to those used for the previous resource model but some additional domaining has been performed to better separate copper and cobalt mineralisation, in recognition of the fact that mineralisation controls on copper and cobalt are slightly different.

In both the Wallaby and Wombat zones copper grades greater than 0.4% were used to define the DISS domain, which is specific to copper (CUDOM). A further copper grade threshold, above 1.0%, resulted in the transition to the SMS domain. Cobalt mineralisation does not always coincide with copper mineralisation, and a separate cobalt only domain was defined (CODOM). The cobalt DISS envelope was based on all values above 0.1% Co and a further increase to 0.3% defined the cobalt SMS domain. A third domain, encompassing both of the mineral specific domains, was created to define the envelope for the remaining minerals of interest (AUDOM). If one or both of CUDOM or CODOM was defined as DISS, then AUDOM was assumed to be coincident. A similar process was used to define the SMS domain. CUDOM and CODOM were constructed so that there would not be a mixing between the DISS and SMS domains. Figure 3.5 shows a schematic cross-section to explain how the domains have been coded; the field of view is approximately 50 m from top to bottom.

Figure 3.5 Schematic cross-section demonstrating the domain definitions



	Ore type	Copper (%)	Cobalt (%)	CUDOM	CODOM	AUDOM
	Hanging Wall			30	30	30
	Disseminated	Cu<0.4	0.1<Co<0.3	-	12, 17	13, 18
	Disseminated	0.4<Cu<1.0	0.1<Co<0.3	11, 16	12, 17	13, 18
	Disseminated	0.4<Cu<1.0	Co<0.1	11, 16	-	13, 18
	Semi-massive	Cu>1.0	Co<0.1	21, 26	-	23, 28
	Semi-massive	Cu>1.0	0.1<Co<0.3	21, 26	12, 17	23, 28
	Semi-massive	Cu>1.0	Co>0.3	21, 26	22, 27	23, 28

The geological codes for the domains were applied to the drillhole database and compositing was performed using Datamine Studio software, with a target composite length of 2 m inside of each domain. The sample length of the original or raw data was typically 1 m, with only a limited number of samples with a length greater than 2 m. After compositing, samples with a length less than 1 m were added back to the previous composite to try and preserve the metal content in the domain. Some of the latest drilling was performed to obtain longer samples of high grade material (up to 15 m) for metallurgical testing and it was decided that compositing these sample to 2 m would bias the global statistics. QG proposed removing these samples from the database but Vulcan requested them to be used, and a compromise was achieved by treating these long intercepts as a single sample.

Snowden comment

Procedurally Snowden is in agreement with how the samples were composited but notes some discrepancies in the database provided. Summary statistics show that three samples with lengths less than 1 m are still contained in the database; however, these samples cannot be combined with an adjacent sample due to domaining or the lack of an adjacent sample. Working with a target composite length of 2 m should result in the majority of composites being contained within a reasonable range around this value. For each domain, Snowden looked at how many samples fell between an interval of 1.5 m and 2.5 m, and all but one domain

have more than 85% of the sample lengths within this range, with about half over 90%. These values are a little on the low side but reasonable for all domains except for the cobalt SMS domain, where only 76% of the 98 sample lengths fall inside this range. Some additional work on compositing may be beneficial but is expected to have little impact on the quality of the estimates.

3.3.3 Use of top cuts

The distributions of some of the variables are positively skewed and top cuts are one method of preventing these extreme values from overly influencing the resource model. Top cuts were applied to gold, zinc and arsenic in the 2007 resource update, and the gold and zinc top cut values are similar to those used in the August 2006 resource. Limited information was provided on the process used for determining the top cut values, but the August 2006 report provided sufficient details on this process.

Snowden comment

Snowden accepts the methodology previously described. The chosen top cut values may not match the ones that Snowden would recommend but they are sufficiently close to make little or no difference to the resource estimates.

3.3.4 Boundaries and spatial continuity

Once the domaining and compositing was completed, a check on stationarity was performed for each domain by QG; this involved tests to see that there were no gross trends within each domain which may invalidate the estimation. These checks showed that there were some small trends or inconsistencies in the spatial location of the average grades. These slight discrepancies do not require any further separation into sub-domains and will be accommodated by the chosen estimation technique.

Furthermore, an assessment was made to determine how quickly the grade changes took place between the domains, i.e. the boundary conditions. The previous work determined that grade changes were rapid and that the boundaries between the domains should be hard so that only samples situated inside of the domain being estimated would be used for block grade estimation. This work was not repeated for the 2007 update, and the previous work is assumed to be still valid since the changes to the domains have been limited.

The grade continuity was investigated by QG by calculating experimental variograms and variogram models were fitted to these points. The variograms were generated using composited but untransformed data, in alignment with good estimation practice. Due to the limited drillhole data, the Wallaby and Wombat zones were combined for all domains but the hangingwall zone was kept separate, since the grades are generally lower. Variograms were constructed for all variables and domains; copper and cobalt were calculated using their respective domain data, and the nickel, sulphur, zinc, gold, arsenic, iron and density variograms were constructed based on the AUDOM codes. Direction specific variograms were first created inside the plane of maximum continuity where this plane coincides with the general plane of mineralisation. The directional variograms were generally of poor quality and difficult to interpret. The additional drilling since the August 2006 resource resulted in some minor improvement to the copper and cobalt directional variograms for the DISS and SMS domains, but all other variables and domains continued to use the omni-directional variograms (average over all directions) in the plane of maximum continuity. Modelling of the variograms was achieved using two or three spherical structures. The copper and cobalt DISS variograms reach the sill at over 240 m and around 125 m for the major and semi-major directions respectively. The remaining DISS variograms achieve maximum variability between 75 m and 740 m, with half of the variables exhibiting a range below 100 m. For the

SMS domain, the copper variogram has ranges of 150 m and 85 m in the two principal directions, and the cobalt variogram is omni-directional, with a range of 95 m. The remaining SMS variograms achieve maximum variability between 65 m and 350 m, with only one variable showing a range over 150 m. The hangingwall variograms are all omni-directional and exhibit a maximum range between 30 m and 300 m, with copper and cobalt ranges of 50 m and 33 m respectively.

Snowden comment

Snowden endorses the decisions made on the boundary conditions and on the modelling of the variograms. The ranges of continuity are significant, suggesting that subsequent modelling is likely to produce blocks which have a moderate to good local precision.

3.3.5 Model estimation

The block model size of 5 mE by 25 mN by 5 mRL was maintained from the August 2006 resource but the subcelling was increased significantly, with a minimum subcell size of 1/10th the parent cell in each direction. QG used this level of subcelling after a request from Vulcan, but warns that subcells of this size imply an unrealistic level of precision in the domain definitions. The block model was constructed in the Datamine Studio mining package and then reblocked, with domain proportions, to the parent cell size for import into ISATIS software for grade estimation.

The estimation of grades into the domained block model was achieved by the ordinary kriging method in ISATIS software. The search strategy was optimised using the Quantitative Kriging Neighbourhood Analysis ('QKNA') process to determine the size of the search and the number of samples to be used for estimation. An elliptical search was used for all variables and domains with a range of 200 m by 100 m by 50 m in the major, semi-major, and minor directions respectively.

The minimum and maximum number of samples has changed significantly from the August 2006 resource, where most variables and domains used between 8 and 32 samples. For the copper domains, the minimum number of samples varies between 8 and 16 and the maximum ranges between 12 and 48. Not every block in the model was estimated using the stated parameters so a second pass was performed where the search ellipsoid was increased to 300 m by 300 m by 100 m and the sample number decreased to a minimum of 4 and a maximum of 16. This second pass was able to estimate all remaining blocks in every domain and for all variables. The Wombat SMS domain required the most additional blocks to be estimated with the second pass, and number of estimates was less than 2% of the total number of blocks.

Snowden comment

Snowden agrees that the level of subcelling is extreme and implies an unrealistic level of precision in the domains.

Snowden does have a small concern about the limited range of sample values used for the DISS domains. The Wallaby DISS has a minimum and maximum value of 8 and 12 respectively, and the Wombat DISS has the same value of 12 for both variables. The cobalt and nickel values have similar minimum and maximum numbers as used for the copper domains. Similar restrictive ranges were observed for the Wallaby SMS, with values of 16 and 20, and the Wombat DISS with values of 12 and 16 for the minimum and maximum respectively. The nickel domains values, which have been applied to all other remaining variables, typically have a minimum of 8 and a maximum of 40 samples, which seems reasonable. Notwithstanding these apparently small numbers, Snowden understands that the minimum and maximum numbers of samples were defined as the result of kriging

neighbourhood analysis (including due consideration of the percentage of negative weights) by QG, and is therefore probably of a low risk.

3.3.6 Validation of resource estimate

Confidence in the quality of the estimate was achieved by validating the block model against the components of its construction. The first check was to validate that estimation reproduced the mean composite grade, after declustering, for each variable. The volume weighted average estimation grade was calculated for each domain and variable, and these values were then compared with the composite average grade provided by QG. For copper and cobalt the average grade reproduction was reasonable, with less than a 10% difference for all domains. The remaining variables typically showed similar levels of quality in the average estimation grade, except for arsenic and gold. The arsenic difference from the domain average grade reached a maximum deviation of 25%, but reporting precision is a factor since the target average grade is reported to only two decimals. For gold, the maximum difference was 19%, but all significant gold deviations were in domains where the average estimated grade was less than the target average. Visual inspection of the provided plots of drilling overlain upon the block model shows that model grades compare reasonably well with the input composite grades.

Moving window trend graphs allow an assessment as to how well the local estimation mean compares against the local sample mean inside slices generated through each domain. The slicing was performed on 50 m increments in the northing and 25 m increment for elevation. Due to the orientation of the zones, slicing in the easting may provide limited information but it is recommended to include them for completeness. The grade trends are generally honoured in both directions but limited sample data makes comparison difficult for some variables and domains, specifically all iron results and the Wombat SMS and hangingwall domains.

Snowden comment

Snowden's validation of the QG model indicates that it honours the input data at the domain level and at the local (sub-domain) level. However, Snowden notes that the estimate is fairly smooth due to the lack of data and this is supported with visual comparisons in an area of close spaced drilling which show that there is significant local variability in the drillhole data which is not reflected in the estimate, although on the whole, broad trends are honoured.

3.3.7 Resource classification

The classification of the resource in 2007 is based on the same methodology as the August 2006 resource model, which is largely based upon the search criteria. Blocks that were informed in the first search pass have been classified as Indicated, while the remainder of the model has been classified as Inferred. Generally the drill spacing is 50 m by 50 m or better with reasonable continuity. The additional drilling and larger searches adopted in the 2007 update have resulted in higher tonnages in the Measured and Indicated categories. A small proportion of the overall tonnage is classified as Inferred but this cannot be compared to the previous resource model since no information was reported for the category. QG applied the classification, with direction from Vulcan, in accordance with the JORC Code, 2004.

Snowden comment

Snowden endorses the Kylylahti resource classification and the resource estimate as being a reasonable to good representation of the mineral inventory of the Kylylahti deposit.

4 Ore Reserves and mine design

4.1 Introduction

In formulating this review of the Kylylahti project Ore Reserve and mining plan, Snowden has reviewed the following documents:

SRK, 2008. Kylylahti Copper Project Mining Feasibility Study, March 2008.

Vulcan, 2008. Kylylahti Copper-Cobalt-Nickel Project, Definitive Feasibility Study Review and Optimisation Report, April 2008.

4.2 Mining method

The mining methods proposed for Kylylahti are:

- Wallaby zone – Longitudinal Open Stopping ('LOS')
- Wombat zone – Transverse Open Stopping (TOS)

Each zone is accessed from a spiral decline, located initially in the black schist in the footwall of the orebody to access the upper Wallaby zone, before changing to the more competent hangingwall skarn to access the Wombat zone. This approach supports design features of the mining methods adopted for each zone. SRK (2008) also reports that in the Wombat zone, the decline is located in better ground conditions in the hangingwall, compared to those of the Wallaby zone footwall location..

4.2.1 Longitudinal open stopping

LOS has been selected for the upper, narrower Wallaby zone as it allows full width extraction of the orebody using stable stope shapes and hydraulic radii, as determined by the geotechnical investigation programs. Stope strike lengths are 25 m and sublevel intervals are 30 m, with up to two sublevel lifts extracted in single stopes below 15 Level, about midway down the zone. Above 15 Level, the orebody plunge and thin vertical thickness allows only single lift stopes. Stopping is conducted in a retreat sequence from the extremities of the orebody, retreating towards a central access cross-cut. Stope widths are such that up to two strike drives on 12.5 m centres are usually accommodated. Stopes are planned to be sequentially backfilled with high strength paste fill as extraction proceeds towards the access cross-cut. A combination of up-hole and down-hole drilling of 76 mm blast holes will be used to extract the stopes.

Ore will be taken from the stope draw-points and either stockpiled or loaded directly into diesel haul trucks at a loading bay adjacent to a Return Air Rise ('RAR') for haulage to surface.

4.2.2 Transverse open stopping

TOS has been selected for the wide Wombat zone as multiple stopes are required across-dip to ensure geotechnical design parameters are honoured. Stope dimensions are limited to 25 m along strike with across-dip dimensions defined by domaining separate semi-massive and disseminated resource zones. Spans exceeding 60 m are exposed in some instances. The semi-massive sulphide domain sits adjacent to the footwall of the orebody and the layout allows it to be mined discretely where practical to take advantage of its relatively high grade. Stopes are accessed by single cross-cuts developed at 12.5 m centres from a hangingwall drive to the footwall of the orebody, enabling the high grade semi-massive ore to be extracted first with the disseminated ore extracted after filling the high grade stope. A combination of high strength paste and low strength hydraulic fill will be used to

enable the complete extraction of the orebody with no requirement for leaving remnant pillars.

Drilling and blasting and ore handling will be the same as for the Wallaby LOS operations.

Snowden comment

The approach to locating the decline is sound. The Wallaby zone portion is contained in the footwall, somewhat distant from the potentially destabilising effects of Wallaby stoping. The Wombat zone portion is located in the hangingwall which facilitates the sequenced extraction of high grade footwall primary semi-massive stopes followed by lower grade disseminated stopes.

The selected methods for mining the Wallaby and Wombat zones of Kylylahti are well established and appropriate given the spatial distribution of the resource and subject to the resolution of the various geotechnical matters discussed below.

4.3 Ore Reserve

Stope designs were prepared by taking vertical slices through the resource wireframe prepared at a 0.4% Cu cut-off at 15 m intervals, or 7.5 m intervals for more complex geometries. The minimum mining width was 3 m. Sublevel intervals were 30 m and the draw point spacing 12.5 m. Resource classified as Inferred was not included in the mine design.

Dilution and ore loss were provided for by factoring the resource within the stope shape in accordance with the modifying factors in Table 4.1, to produce a potential mining inventory. Note that dilution and ore loss factors have been combined into single tonnage and grade modifiers.

Table 4.1 Stope resource modifying factors

Stopes	Tonnage factor	Dilution grade	Applies
Wallaby	1.05	0.4% Cu	100% Wallaby (95 stopes)
Wombat Primary	1.05	0.1% Co 0.2% Ni 0.4% Zn 0.5 g/t Au	52% Wombat (54 stopes)
Wombat Secondary	1.08	0%	48% Wombat (50 stopes)

Vulcan reports that the grade of the diluting material is around that of the cut-off grade used to establish the resource wireframe.

The conversion of stope resources to potential mining inventory is shown in Table 4.2 (from SRK (2008), Table 6).

Table 4.2 Conversion of stope resource to potential inventory

Item	Unit	Resource	Potential inventory
Tonnage	T	7,854,192	6,945,809
	% Cu	1.17	1.17
	% Co	0.24	0.24
Grade	% Ni	0.22	0.20
	% Zn	0.49	0.49
	g/t Au	0.70	0.70

The stopes comprising the potential inventory were assessed for operating NSR and those with an NSR < -2.00 €/t were excluded from the inventory by SRK, in recognition of the variability and sensitivity of the results to many of the input parameters used in this calculation. The ten excluded stopes were all of disseminated material. The final Ore Reserve reported in Vulcan (2008) is shown in Table 4.3.

Table 4.3 Final Ore Reserve as reported in Vulcan (2008)

	Tonnes M	Cu %	Co %	Ni %	Zn %	Au g/t
Proved and Probable reserves	6.946	1.17	0.24	0.20	0.49	0.70

Snowden comment

The Ore Reserve estimation process appears to address the matters required by the JORC Code, 2004 guidelines. It considers, amongst other things,

- mining practicalities, including design and operability
- modifying factors
- economic factors.

The 0.4% Cu cut-off was not determined on an economic basis, but rather as a natural geologic boundary, beyond which the grade rapidly drops to approximately 0.1% Cu. The economic assessment of stopes designed within the cut-off boundary on an individual basis serves to further refine the ore reserve and exclude uneconomic stopes. This approach is reasonable.

SRK's approach to delete stopes providing an NSR < -2 00€/t removes the majority of unprofitable potential ore reserve material. However, stopes with an NSR between 0.00 and -2.00€/t remain in the ore reserves. Table 4.4 shows the inventory of these unprofitable stopes. All could be considered immaterial and within reasonable limits of accuracy, except stopes K33DISS_STP4 and W43DISS_STPA1302 which, combined, contain 66 kt of rock, 548 t of copper and lose €131k. The NPV analysis identifies unprofitable stopes under the assumptions applied, and by definition, these should not be included as part of the ore reserve. Vulcan has advised Snowden that these stopes become profitable when short term price forecasts are considered in conjunction with the scheduled timing of production, justifying their inclusion in the ore reserve.

Table 4.4 Stope inventories with NSR between 0.00 and -2.00 €/t

Stope	Tonnes t	Grade % Cu	Cu t	NSR/t €/t	NSR €k	Revenue €k	NSR/Revenue %
K15DISS_STP8	514	0.67	3	-0.76	-0.4	33.9	-1.15%
K18SM_STP7	12,772	0.95	121	-0.06	-0.8	923.8	-0.09%
K33DISS_STP4	9,969	0.50	50	-1.92	-19.1	650.0	-2.94%
K9DISS_STP22	4,592	0.81	37	-1.10	-5.1	297.1	-1.70%
W43DISS_STPA1302	55,987	0.89	498	-2.00	-111.8	3,989.8	-2.80%
Total	83,834	0.85	710	-1.64	-137.2	5,894.6	-2.33%

Snowden recommends that, for consistency of process, all stopes providing an NSR <0.00 €/t should be deleted from the ore reserve unless a supporting justification for inclusion is documented.

Vulcan also advises that it has modified the profitability calculation model since publication of the DFS and anticipates a reduction in the number of deleted stopes (increasing the mining inventory, by about 200 kt) when the ore reserve is next updated.

Based on examination of the design wireframes supplied as part of the project Mine2-4D graphical schedule, Snowden considers the designs to be generally appropriate, although some inconsistencies in detail are apparent:

- Stopes which form the basement at each level have troughs designed to direct ore into the draw point, typical of conventional open stopes, maximising the recovery of ore. However, stopes designed on upper levels have flat bottoms, from which full recovery of the designed stope volume is unlikely. In some instances, this ore may be fully recovered by undercutting the stope by a lower stope. In other instances, the upper stope will be filled before it can be undercut, losing some recovery of ore. As a result, it is unlikely that similar recoveries will be achieved from basement stopes and upper stopes. Operating experience will be required to accurately quantify these effects.
- In some instances, extraction of an upper stope is scheduled after extraction of its lower counterpart. The extraction horizon will have been undermined completely and will be supported by fill. The effectiveness of this support will be limited by the degree to which the fill can be tight packed. There may be deterioration of the mucking horizon of the upper stope between mining and filling. Heavy ground support may be required to minimise the potential for ore loss.
- Some Wallaby stope designs include leaving a narrow pendant of uneconomic material in the crown. This has potential for failure and to cause poor fragmentation reporting to the draw-points. Similarly, a thin rib is designed to be left between some stopes in the upper part of Wombat, where the orebody bifurcates. Subject to the matters raised in Section 4.2.2 above, this rib may have a high risk of failure. Infill drilling is likely to provide additional information to enable design refinements to reduce this risk.
- Dilution and ore loss characteristics for only two stope types, primary and secondary, are specified in SRK (2008). The definitions of primary and secondary stopes are not provided. There are a number of stope types, as

defined by wall exposures (of ore, waste and fill), which possess unique attributes defining their modifying factor characteristics. The most stable stopes will have only rock wall exposures. The least stable will have fill exposures on three sides, and include a fill crown. There are various degrees of instability characteristic between the most and least stable designs. It is doubtful that the two classes of modifying factors reported in SRK (2008) adequately reflect the range of dilution and ore loss amounts that could be expected in practice. In particular, most Wallaby stopes have at least one wall of fill, although all are characterised as primary stopes with a low dilution sourced from cut-off grade material. Early operational experience should be used to calibrate dilution and ore loss characteristics for the full range of stope types to be encountered.

- The grade of primary stope dilution, being the design cut-off grade, is questionable since the dilution can come from hangingwall or footwall waste, along strike ore, and possibly from adjacent fill. A more rigorous and detailed estimate of diluting grades could be established based on primary stope configuration.

Depending on the provenance of the stopes removed from the mining inventory after the NPV assessment, a reconsideration of the source and grade of dilution, and the classification of some stopes as primary or secondary, may be warranted. Ongoing refinement of designs should seek to optimise this aspect.

It is not clear if the addition of dilution to primary stopes is reflected as a depletion or ore loss from adjacent secondary stopes. Vulcan has since advised that dilution was planned to come only from the footwall, hangingwall or adjacent backfill at a combined grade equivalent to the cut-off grade. Vulcan considers this approach to have produced a low risk estimate. Snowden agrees that the factors applied will produce a conservative and low risk estimate of the impact of dilution on primary stope inventories. However, Snowden notes that this approach ignores the potential for dilution from adjacent ore exposures, and consequently the risk of ore loss from those adjacent exposures. In practice, Vulcan may find over-recovery from primary stopes and under-recovery from secondary stopes. This will impact short term predictability of production and this aspect should be investigated as operational results become available.

Some inconsistencies in reporting the mining inventory were noted in SRK (2008) Table 6, Table 8 and Table 9. Vulcan has confirmed that the final mining inventory, and effectively the project ore reserve, should be 6,945,809 t, as reported in Vulcan (2008) and in Table 4.3, above.

Note that the JORC Code, 2004 provides for the reporting of 'Proved', not 'Proven' Ore Reserves, as reported in Vulcan (2008).

SRK (2008) reports in Table 5 that there were 199 stopes included in the potential mining inventory. SRK Table 9 reports that, after the removal of 10 stopes based on negative NPV, a total of 209 stopes remain. Vulcan has confirmed that Table 5 should indicate 209 stopes were included in the potential mining inventory prior to deletion based on NSR.

SRK (2008) provides no discussion of the potential for dilution from fill having a deleterious impact on processing performance, particularly flotation recovery. This issue is not unknown in operations with similar features and warrants specific redress. This risk should be investigated as operational results become available.

4.4 Ventilation

The ventilation system is designed as a typical push/pull system. Fresh air is forced into the mine by downcast fans mounted in intake shafts, while exhaust air is drawn by upcast fans mounted on exhaust raises. Fan capacities have been selected to provide 290 m³/sec of fresh air with 260 m³/sec for upcast in exhaust airway and upcast in the access decline of 60 m³/sec.

The primary intake airway will be raise-bored at 5 m diameter over 165 m. The primary exhaust airway will be of 4 m diameter and 153 m long. Extensions to the system will be generated by developing internal fresh air and exhaust raises between levels as the mine develops to depth. The internal raises will be developed using drill and blast methods at 5.0 m and 4.0 m square dimensions, respectively.

The intake air will be heated using a 15 MW gas fired heater. The heating is required to ensure ice does not build up and obstruct the intake airway, while improving the temperature of the underground working environment.

Snowden comment

The ventilation plan is reasonable and typical of mines with similar design features. The plan has been developed to a high degree of detail.

One aspect not reported on is the potential for the development of fog in the decline. The risk of condensation from the warmed exhaust air in the main decline as it approaches the surface should be assessed to confirm that the risk of any 'fogging' hazard will be manageable with normal operational practices.

4.5 Schedule

The project schedule has developed through a number of stages, with the ultimate one being described in SRK (2008) and Vulcan (2008). Earlier versions of the schedule were driven by a target bulk concentrate production rate of 220 ktpa which required a mining rate of 650 ktpa. Production was constrained by reliance on capital intensive on-site concentrate processing facilities. An alternative concentrate processing strategy (sale to third party for off-site processing) was developed and the mining production constraint was lifted. Opportunities were identified which allowed mine production to reach 800 ktpa, and the mine schedule was formulated accordingly.

SRK concluded (SRK (2008)) that: *'The interaction of production hauling operations with personnel and consumable movement in the decline is believed to be the limiting factor with respect to the ore production rate.'*

The schedule has been compiled to gain the best advantage from developing the high grade Wombat semi-massive stopes as early in the mine life as possible to deliver the highest achievable grade for processing.

SRK summarised the scheduling aims as

- establish the primary ventilation system prior to stope production
- commence production mining as soon as possible
- maximise semi-massive ore as soon as practicable
- achieve and sustain a steady state mining rate of 800 ktpa
- commence production at the lowest mining elevation as soon as possible, mining in a bottom up sequence
- avoid mining under previously mined and filled stopes as much as possible
- maintain a primary and secondary stope sequence in the Wombat Zone.

The main activity scheduling constraints adopted by SRK are shown in Table 4.5.

Table 4.5 Scheduling constraints

Activity	Constraint
Decline development	80 m/month for first 100 m of decline
	120 m/month for second 100 m of decline
	180 m/month from 200 m to -120 mRL, about 640 m of decline
Level development	140 m/month below -120 mRL
	12 m/week in any one heading Each jumbo capable of 250 m/month
Production drilling	10 t/m drilled
	200 m/day
Production loading rate	1,200 t/day/stope
Filling rate	1,000 t/day
Fill cure time	28 days before start of mining in adjacent stope

The project LOM schedule is summarised in Table 4.6.

Table 4.6 LOM summary schedule

Item	Units	Years										
		2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019
Development	m	2,858	5,409	3,726	4,976	2,486	1,791	987	-	-	-	-
Wallaby ore (incl devt)	kt	9,944	223,998	555,903	346,053	155,068	-	59,052	150,460	284,388	531,160	230,575
Wombat ore (incl devt)	kt	-	-	91,637	460,357	647,103	800,331	744,619	651,461	517,155	270,072	216,474
Total ore	kt	9,944	223,998	647,540	806,410	802,171	800,331	803,670	801,922	801,543	801,232	447,049
Cu grade	%	0.64	1.27	1.14	1.29	1.42	1.36	1.41	1.09	0.94	0.82	0.89
Co grade	%	0.16	0.26	0.26	0.25	0.25	0.24	0.26	0.23	0.21	0.20	0.20
Ni grade	%	0.26	0.20	0.21	0.18	0.16	0.20	0.17	0.20	0.23	0.23	0.21
Zn grade	%	0.30	0.50	0.51	0.49	0.51	0.61	0.62	0.52	0.39	0.34	0.37
Au grade	g/t	0.44	0.70	0.66	0.61	0.70	0.92	0.79	0.77	0.64	0.57	0.54

Snowden comment

The mine schedule is based on reasonable assumptions and presents an achievable project outcome. In particular, achieving 800 ktpa from a 700 m deep decline access, mechanised bulk mining operation, with multiple open stope sources is routinely achievable.

In achieving the scheduling aims, however, constraints and complexities are introduced into the schedule. As noted by SRK, "The number of active headings will be high...management must communicate clear heading priorities to ensure the correct amount of metres are developed from the correct headings as this is fundamental to ensure the selected higher grade stopes are mined on time." There are clear dependencies between critical activities which must be satisfied so the schedule can be achieved. The critical activities are:

- development
- drill and blast
- production load and haul
- filling.

Although the critical scheduled activities are considered to be achievable, Vulcan should investigate the risk to, or sensitivity of, the project to non-achievement of critical schedule items, and the subsequent impact on the overall project. By doing this, a cost/risk/benefit analysis can be performed to assess the need for the complex scheduling the project is subject to. A simpler lower risk schedule with minimal project impact may be acceptable. The types of schedule risks which should be considered are, for example

- development not following critical sequence (out of schedule access to semi-massive stopes)
- development rate falling short of target (delayed access to production sources)
- stope load-out rates not achieved in secondary stopes due to oversized dilution or fill fall-off (delayed access for filling and mining adjacent stopes)
- decline congestion exceeding prediction (below target haulage rate).

SRK (2008) reports that congestion is the rate determining aspect of mine production. The use of a smaller fleet of larger capacity haul trucks, instead of the eight 21 t trucks currently considered, may provide an opportunity to remove this constraint and allow an increase in production. Simulation of the haulage function using appropriate software (e.g. Arena), could provide valuable insight into the potential for improvement.

The schedule is development intensive early in the mine life, with development completed during Year 7 of the project (2015). This comes at a high net present cost, although it is aimed at returning high net present value. Vulcan advises that numerous analyses and schedule iterations have demonstrated the benefit to the business model of this approach.

It is noted that there is no apparent provision for cable-bolting stope walls, which, as discussed in Section 5.2, appears to be a requirement of the geotechnical design of the mine to ensure high confidence stability in all instances.

4.6 Equipment

SRK (2008) lists the required equipment fleet for Kylylahti as included in Table 4.7.

Table 4.7 Mining equipment fleet

Equipment type	Number of units
Twin boom jumbo	2
Bolting jumbo	1
Mechanical scaler	1
Longhole drill	2
Development loader (14.5 t)	1
Production loader (10 t)	4
Haul truck (21 t)	8
Service equipment: washdown truck, development charge wagon, stope charge truck, IT, grader, shotcrete machine, diamond drill	1 of each
Light vehicles	13

All major equipment, excluding service equipment and light vehicles, would be leased from the manufacturer or supplied as part of the mining contract.

Snowden comment

Snowden has not undertaken a detailed fleet evaluation but considers the equipment types described in SRK (2008) to be adequate. There may be opportunities for cost and efficiency improvements by considering larger capacity underground haul trucks.

It is noted that cable-bolting of stope footwalls does not appear to have been included in activity schedules. If cable-bolting of stope footwalls is fully embraced, an additional contractor supplied long-hole rig may be required.

It is also unlikely that the single rock-bolting jumbo listed in Table 4.7 will be adequate to support both the Vulcan and contractor development activities.

4.7 Mine services

Vulcan's plans for mine services are described in SRK (2008), and include descriptions for provisions for pumping and groundwater management, electrical reticulation, compressed air and water supply, communications, refuge chambers and emergency egress.

Snowden comment

The provisions described in SRK (2008) are deemed adequate for the purposes of the study.

5 Geotechnical work

Snowden has reviewed the following documents as the basis of this section:

- SRK, 2007. Kylylahti Copper-Cobalt-Nickel. Project Definitive Feasibility Study. Vol 3 Part 2 Geotechnical Report and Appendices A to M, October 2007.
- SRK 2008. Kylylahti Copper Project Mining Feasibility Study. Section 3 Geotechnical Assessment Summary, March 2008.

SRK Consulting has provided additional comment on stope design issues by letter dated 9 September, 2008. Based on this information Snowden has modified the comments made on the DFS.

The following geotechnical work programmes has been reported for Kylylahti:

5.1 Investigation programme

- The investigation programme included geotechnical logging of 4,925 m of core from 33 drillholes. The logged intervals are within and adjacent to proposed development and production mining areas.
- Structural logging of orientated core was undertaken. Also, some limited mapping of surface outcrops was carried out.
- A geomechanical laboratory testing programme was undertaken, including compressive, triaxial and tensile strength tests of drill core samples.
- Major fault structures have been located and investigated.
- Zones of deep weathering within the bedrock in the decline portal area have been identified by geophysics surveys.

Snowden comment

The investigation programme has included all essential activities for feasibility level studies. The work has been conducted in accordance with industry-standard methods, and based upon a review of core photographs, the standard of the drilling and logging work is considered to be good.

The key geotechnical risks to the project have been identified.

Issues of concern are:

- The relative paucity of information for the Wombat zone, particularly for its lower section, and for the upper section of the Wallaby zone. The holes drilled and geotechnically logged by Vulcan appear to be clustered where the Wallaby and Wombat zones overlap. Hence at the time of the DFS there was an uneven distribution of geotechnical data through the deposit, leading to a risk of the data being spatially biased. Snowden understands that further drilling is being carried out to address this issue.
- The DFS contains references to a number of mine-scale fault structures that may affect stope and pillar stability. The lack of a model of such structures means that their effects cannot be assessed at present.
- The minimal assessment of in-situ stress conditions.

5.2 Design programme

- Standard rock mass characterisation and classification methodologies were used to assess geotechnical conditions, viz. Laubscher RMR and NGI Q.
- Basic, broad geotechnical domains were identified.
- Support designs are based on Q rock mass classification.
- Slope stability assessments have been undertaken with a standard empirical method, i.e. Mathews Stability Chart.

Snowden comment

Geotechnical design has used industry standard methods. The design decisions arrived at in order to deal with the main hazards appear rational.

Portal and decline designs appear suitable for the geotechnical conditions. The principal geotechnical risks to the mine access have been identified and assessed, and the portal and decline have been sited to minimise these risks.

Open-stope mining methods and the use of paste fill are both appropriate to the geotechnical conditions and orebody dimensions.

The use of the lower quartile (25 percentile) Q values for design purposes is a valid approach to addressing the typical variability of geotechnical conditions.

The development support designs are reasonable for the anticipated conditions, and are possibly slightly conservative (the Excavation Support Ratio ('ESR') used in the DFS is low for permanent development). However, Vulcan may be able to achieve more efficient designs if rock-mass classes were modified; for example, Class 2 contains a broad a range of ground conditions, some requiring surface containment (shotcrete) and some only requiring spot bolting. The use of mesh rather than shotcrete in some conditions may also provide cost savings.

Issues of concern are:

- The effects of in-situ stress may have been under-estimated. Although data from nearby sites (Outokumpu area mines) has been sourced, it was obtained with a now-obsolete method and has low reliability. The regional stress field data for Finland (freely available on the World Stress Map) shows a "thrust" stress regime, comprising a high major principal stress aligned horizontally and approximately WNW, i.e. normal to the orebodies, an intermediate principal stress aligned horizontally and approximately NNE, and a minor principal stress aligned vertically and equal to gravitational loads.
- In the skarn and ore domains, there appears to be two distinct populations of material with widely different geotechnical ratings. Snowden has reviewed the geotechnical core logging database and noticed that the Joint Set Number (Jn) used in the calculations of Q and Q' is based on the number of structural sets observed in the individual logging intervals, and not the number of structural sets present in the geotechnical domain. Snowden believes this procedure is not correct and lead to overestimating the Q' and resulting N values.
- Snowden also noticed that the estimated rock strength values in the geotechnical logs frequently are substantially higher than the mean values of the laboratory test results. Provided there are a reasonable number of representative samples tested, the laboratory test results should reflect the intact rock strength better than field estimates; hence the test data should be

used in rock mass classifications. This will affect rock mass classifications derived using the Laubscher or Bieniawski methods.

- The stability of stopes is likely to be controlled by footwall conditions as the black schist has markedly lower Q values than the ore and skarn zones. Snowden has undertaken a design check on the stability of stope hangingwalls and footwalls using the strength and rock mass characterisation data provided in the DFS (Vol 3 Part 2) The results are presented in Table 5-1. If stopes with footwalls formed in black schist have dimensions greater than those shown **they are** likely to require support in the form of pattern cable-bolting.

Table 5.1 Design check on stope dimensions

Orebody	Wall	Rock type	N'	Hydraulic Radius	Stope Height	Maximum Strike Dimension* (Unsupported)
Wallaby	HW	Skarn	39.4	9	30 m	45
	FW	Black schist	11.8	6	30 m	20
Wombat	HW	Skarn	15.8	6.5	60 m	16.6
	FW	Black schist	4.7	4	60 m	9.2

- Although pattern cable-bolting is likely to be required to stabilise a substantial proportion of stope footwalls in the black schist, no conceptual designs were provided in the DFS. A preliminary assessment by Snowden indicates that where cable-bolts are needed the typical spacing should be 3.0 m by 2.5 m. This will be very difficult to achieve with the proposed level spacing (30 m) and stope development layouts.
- No consideration has been given to geotechnical issues related to the extraction sequencing of the Wombat stopes and pillars, the possible need for sill or rib pillars to control abutment stability or the potential for high induced stresses to develop in pillars and abutments.
- No kinematic stability assessment has been undertaken for the proposed 25 m wide Wombat stope/pillar backs. The structure data indicates that a joint set with low to moderate dip is prevalent in the orebody; these features may adversely affect the stability of stope backs.
- No consideration has been given to bogging level stability issues during mining of the second lift of Wombat pillars and wider sections of Wallaby longitudinal stopes. The current mining plan will create wide, unsupported spans and brows beneath which bogging will have to take place. Cable-bolt support may be required to mitigate the risk caused by formation of large unstable wedges.

The geotechnical investigations are adequate for feasibility level mine studies, although there are some risks to the mining plans associated with the limited spread of drillhole locations, the limited assessment of in-situ stress conditions and lack of a mine-scale geological structure model.

There are some issues with processing the geotechnical classification values, that when corrected may lead to design modifications for stopes and pillars, and for particular extraction sequences to be adopted.

These issues impose risk on the project, as they could impact on mining conditions, support requirements, production rates and costs. However, given their nature, these risks can be mitigated by appropriate design modifications and are not likely to prove to be “showstoppers”.

5.3 Recommendations for further work

- The Q and Q' values should be re-calculated with Jn values appropriate to the geotechnical domains, not individual logging intervals.
- Slope stability assessments and support designs should be re-evaluated once revised Q and Q' values are available. Conceptual cable-bolting plans are required.
- In-situ stresses should be measured before commencing production. The AE method can measure in-situ stresses using reliably orientated HQ3 core that may currently be available on site.
- The potential influence of mine-scale structures on stability needs to be evaluated once a model of these structures is available.
- The operational risks associated with the proposed stope and pillar mining method need to be re-assessed, in particular the creation of wide unsupported bogging levels during pillar recovery.

6 Hydrology work

There are two principal hydrological units in the project area; water hosted by unconsolidated till and peat and water hosted in the underlying bedrock.

The main investigation into the local hydrology was conducted by Golder Associates ('Golder') in January 2007 as part of the Kylylahti DFS. The purpose of the study was to survey the existing watercourses and to measure water flow rates and quality where appropriate.

Golder's main conclusions from this work were:

- The nearest major water bodies in the local area are Lake Kylylampi (6.96 ha), lake Polvijärvi (22.31 ha), Purnulampi Pond, and two flooded quarries in the Vasarakangas mining district (Large Mondo Pit (4.66 ha) and the Small Mondo Pit (1.56 ha)).
- All surface water from the mining license area flows into Lake Polvijärvi which is located one km south-southeast of the project area. The water quality of the lake is poor and very eutrophic (nutrient-rich, leading to a proliferation of algal growth).
- The mining lease area is not located on or in the vicinity of any classified groundwater areas or nature protection areas.
- The results of surface water analysis from the Kylylahti area show elevated concentrations of iron, manganese and aluminium that consistently exceed the Finish drinking water standard. The quality of the groundwater is much poorer in the bedrock than in the overlying soil.
- Golder believes, based on its modelling of the groundwater formation, that there is no significant connection between the soil and bedrock groundwater. However, this relationship may change as a consequence of the blasting work which would be conducted as part of the mine operations.

Water encountered during mining operations is expected to be sufficient to support the Kylylahti operations under normal operating conditions. If additional water is required it will be drawn from the nearby Vasarakangas open pits.

Excess water will be discharged into Lake Polvijärvi via an underground pipeline. This is expected to raise the overall water level and reduce the seasonal variations in the water level. Both impacts are seen as a positive change by the regulatory authorities and the local stakeholders. However, it is possible that the discharged water may need to be extensively treated to meet the local environmental requirements.

The chosen water sources and discharge options were selected to minimise the impact on the local bodies. Mining operations are not expected to have a significant impact on the local ground water quality; however, monitoring will be conducted on a regular basis throughout the life of the project.

Snowden comment

Snowden endorses and supports the hydrology work and the findings of Golder and sees little risk in the supply and management of project water. The use of local water sources is addressed in some detail in the conditions of the environmental permit, which are summarised in Section 11.

The risks associated with the planned water use are addressed more fully in the environmental and permitting review (Section 11.4).

7 Process engineering

7.1 Terms of reference

The work for this section has been carried out by Mineral Engineering Technical Services Pty Ltd ('METS'), specifically Mr Jeff West, Senior Metallurgist, and Mr Damian Connelly, Principal Consulting Engineer. METS provided review services under the auspices of Snowden's appointment as ITE and Snowden takes responsibility for METS findings.

7.2 Process mineralogy

7.2.1 Kylylahti mineralogy

Mineralogical studies of the Kylylahti ore by Outokumpu and later G&T laboratories have been key to understanding the process behaviour of the ores and have been critical in developing a suitable process flowsheet.

The Kylylahti deposit is a typical Outokumpu-type mineral sulphide ore containing copper, cobalt, zinc, and gold occurring in association with pyrite and pyrrhotite. The gangue is quartz with accessory carbonates and talc. As previously described, there are two main mineralogical and metallurgical zones:

- The high grade semi-massive sulphide ore (SMS), dominated by sulphides such as pyrite.
- Lower grade disseminated ore (DISS), dominated by the presence of sulphides such as pyrrhotite.

The distinction between the two zones is on the degree of mineralisation. SMS ore typically has more than 1% copper, around 20% sulphur and has high pyrite and pyrrhotite contents. The DISS ore has lower copper grades, lower sulphur content (10%) and more pyrrhotite than pyrite.

The SMS ore sits in the ore footwall with the DISS ore further away. The sequence from hangingwall to footwall is serpentinite grading into talc-magnesite rocks, to tremolite-carbonate-quartz rocks to quartz-tremolite rocks to sulphide bearing quartz-tremolite rocks. The hangingwall consists mainly of poorly mineralised disseminated ore. Footwall material is hard black schist primarily derived from black sandy metapelites and typically remains fine grained after being metamorphosed.

Metallurgically, there are three ore types; Kyly A, Kyly B and Kyly C (See Table 7.1). The influence the mineralogy has on the metallurgy is now well understood.

Table 7.1 Typical mineralogical analysis of the main Kylylahti ore types

Ore - Type Sample No.	Kyly-A 9553467	Kyly-B 9553469	Kyly-C 9553468
Sulphides			
Pyrite	14.5	31.1	2.6
Pyrrhotite	21.7	4.9	15.3
Pyrite + Pyrrhotite	36.2	36	17.9
Cobalt -pentlandite	0.5	0.2	
Cobaltite	0.1		0.1
Chalcopyrite	7.2	6.9	1.8
Sphalerite	1.2	0.9	0.5
Pentlandite			1
Siegenite		<0.1	-
Sub -Total	45.2	44	21.3
Non - Sulphides			
Quartz	30	37	42
Talc	11	1	10
Biotite	3	2	2
Amphibole	4	11	18
Carbonates	8	6	7
Plagioclase	<1	<1	<1
Sub -Total	56	57	79
TOTAL	101.2	101	100.3

7.2.2 Variability

The proportion of iron sulphides, pyrite and pyrrhotite, varies widely among the orebodies. In the pyrrhotite rich areas cobalt pentlandite is the main cobalt mineral whereas in the pyrite rich areas it is the main cobalt carrier. The metallurgical process is mineralogically driven and the solid solution nature of some of the metals and the low grade of others (e.g. zinc) means individual concentrates of cobalt, nickel and zinc are not possible. Variations in the mineralogy result in characteristic changes to the nature of the bulk concentrate produced as a result. Metallurgically the identification of the ore zones is important. When cobalt pentlandite is present it is possible to produce a higher grade cobalt concentrate than when pyrite is the main cobalt phase. The mass flows will vary accordingly, and performance of the concentrator could be impacted if the mine schedule of feed to the concentrator is not managed to achieve optimum outcomes. Vulcan has used assaying and chemical mass balancing to model the ore species in 3D and therefore manage the potential problem.

7.2.3 Geometallurgical approach

The main sulphide minerals are pyrite, pyrrhotite, chalcopyrite, cobalt pentlandite and sphalerite. Accessory minerals include pentlandite, cobaltite and siegenite. Vulcan has used wet chemical assays for copper, sulphur, zinc and arsenic. Cobalt, nickel and iron values have been obtained by bromine methanol dissolution. Vulcan used this and other methods to estimate the spatial distribution of mineral species in the ore blocks so that the distribution could be visualised in 3D.

The geometallurgical approach taken by Vulcan is excellent as this technique results in a minimisation of risk and unwanted surprises during mineral processing operations.

Geometallurgy at the feasibility stage

Vulcan has adopted a geometallurgical approach during the feasibility studies. The improved ore characterisation combined with the spatial modelling of critical physical characteristics that forms the basis of a geometallurgical approach provides a much improved basis for operational and mineral processing plant design. This approach reduces the risk associated with developing a new operation. By increasing orebody knowledge and then using it to design the entire operational flowsheet (from in-situ rock to end product) a more reliable and realistic model of the production capacity of the system can be developed. Key bottlenecks, product constraints and low-cost/high value opportunities can be identified. The result can be increased total metal recovery and improved asset utilisation. This in turn allows better decision making, in terms of capital expenditure, to optimise project economics.

Snowden comment - risk reduction

Vulcan has directly reduced the risks associated with meeting production targets by applying geometallurgical modelling techniques. Geometallurgy has the potential to act on both the consequences and likelihood axes to decrease risk. The spatial distribution results have been included in the mine schedule.

Snowden comment - roasting and leaching

This has not been addressed as it is no longer a part of the process flowsheet despite the fact that a significant amount of work was undertaken in this area. In Snowden's opinion the decision to export a bulk concentrate has significantly reduced the risk profile for the project.

7.3 Metallurgical testwork

7.3.1 Prefeasibility testwork

Vulcan acquired the rights to the Kylylahti deposit in 2004 and in 2005 embarked upon a series of studies that included a review of historical reports, processing options studies and engineering cost studies. Further drilling increased the defined resource and marketing studies were conducted for the concentrates. These studies culminated in the publication of a prefeasibility study which was published in October 2005. During the prefeasibility study Vulcan selected a processing route whereby ore from Kylylahti would be processed through a concentrator to produce a saleable copper-gold concentrate and a bulk concentrate containing cobalt, nickel, copper and zinc, which would be roasted and leached to recover the value metals.

Although the bulk concentrate processing route was poorly defined, sufficient potential was apparent to proceed with a definitive feasibility study.

7.3.2 Feasibility study testwork

Outokumpu Oy has conducted detailed investigations since discovering the project in 1984. The Outokumpu testwork proposed a two stage flowsheet to produce a marketable copper-gold concentrate and a bulk zinc-cobalt-nickel-copper concentrate for offsite processing.

Vulcan has commissioned metallurgical testwork at GTI in Canada and at AMMTEC and SGS in Perth. This has been defined by Vulcan as Stage 1, Stage 2 and Stage 3 in the overall testwork programme. The programme (see Figure 7.1) comprised the following:

- Evaluation of preliminary flotation properties and roast-leach testwork.
- Selection of appropriate samples of the different ore types for further testing.
- Further development and testing of an appropriate concentrator flowsheet.

- Testing of samples for comminution properties.
- Production of a bulk concentrate for further downstream and ancillary testing.

A metallurgical and mineralogical classification of the deposit was undertaken to ensure the representativeness of samples used in testwork. Extensive prior mineralogical and metallurgical testwork by Outokumpu Oy also informed the testwork programme. Flotation testwork was undertaken at AMMTEC Perth and Outotec completed roasting and leaching testwork on bulk concentrates as part of the DFS.

Physical testwork was undertaken at AMMTEC and the recommendation for the milling circuit was completed by Orway Mineral Consultants. A single stage fully autogenous grinding mill with pebble crusher has been adopted as a result of the review work. SNC recommended two stage grinding; however, single stage grinding was the practice at the former Outokumpu operations in the area.

A standard flotation circuit was proposed by SNC in the DFS with a copper removal stage followed by cleaning of the tail to generate a bulk concentrate. The copper-gold concentrate is optimised for grade and recovery to maximise payments from smelters and the bulk concentrate is optimised to maximise sulphur and metal recovery rather than payable metal grades. Sulphur is a key driver for bulk concentrate processors. Minor changes were made to SNC's flowsheets in the review.

Sample selection

Vulcan's first sample was termed the Stage 1 composite and this was followed by Master Composite 1 (MC1, as depicted in Figure 7.1), predominantly SMS ore from the Wallaby zone representing Years 2 to 5. Master Composite 2 (MC2) represented predominantly DSS ore and low grade production from years 1 and 6 to 12 from the Wombat zone. In addition there were a number of variability samples selected.

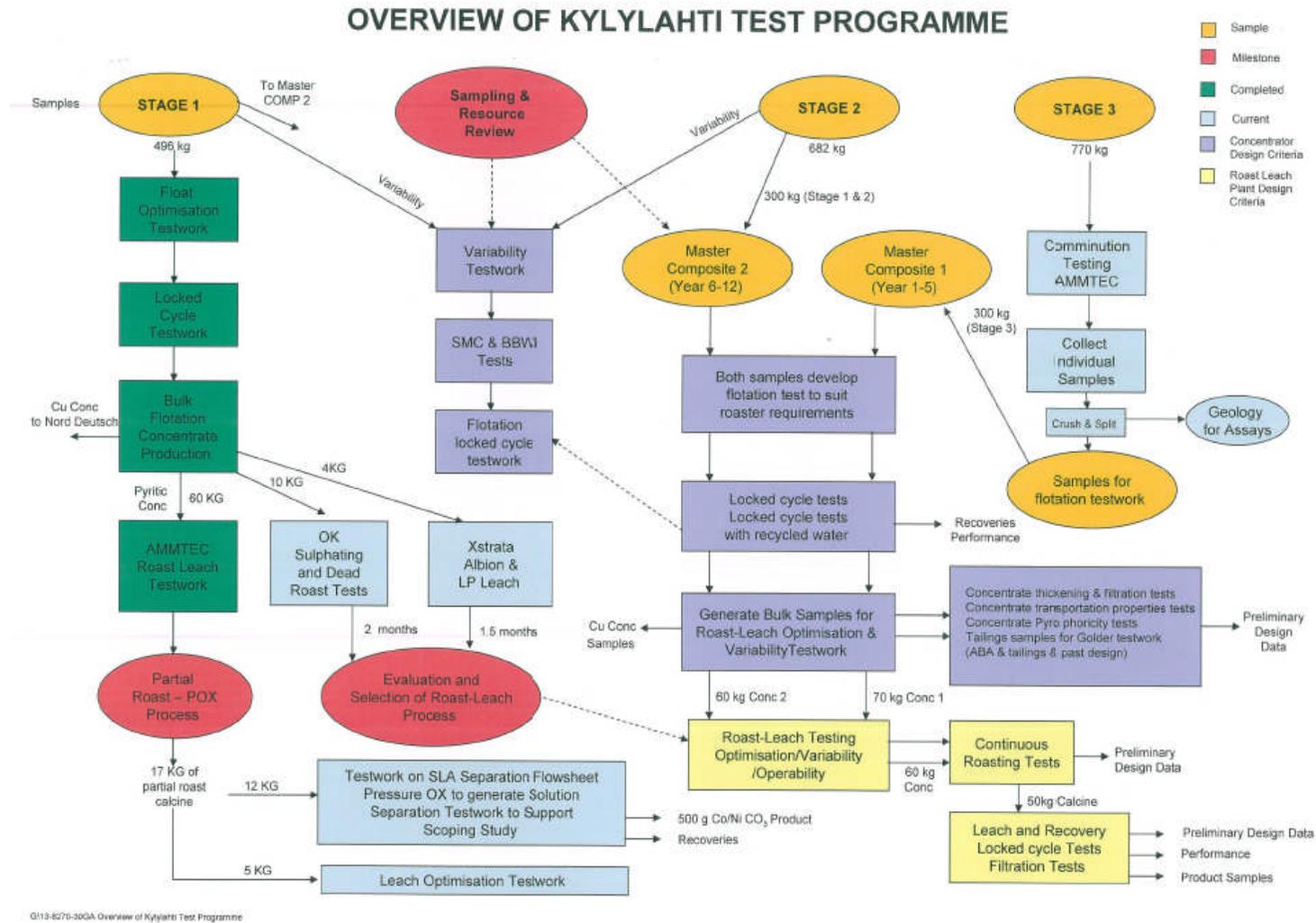
Comminution testwork

A number of comminution circuits (fully autogenous grinding ('FAG'), semi-autogenous grinding (SAG) and SAG plus a ball mill) were considered with respect to capital cost, operability/climate, and downstream processes.

Testwork on representative samples included bulk samples of core, variability samples on core, using the following tests:

- Autogenous Media Competency
- Impact Crushing Work Index
- Unconfined Compressive Strength
- Abrasion Index
- SMC tests on variability samples
- Bond BRod & Ball Mill Work Index.

Figure 7.1 Details of Stage 1, 2 and 3 testwork



Results of the testwork are summarised in Table 7.2. The conclusions of the testwork are that the ore is amenable to autogenous milling. The final comminution circuit selected is a single stage FAG mill with a recycle crusher. The targeted grind size is P80 106 μm . The circuit has been designed with some excess capacity and a lot of flexibility, such as being able to add steel to the mill if required. The mill motor is variable speed. The basis of the testwork and the design of the comminution circuit is:

- representative ore samples
- extensive mineralogical analysis
- comminution physical characteristic testwork as described above
- independent checking of the circuit by OMC using JKSimMet software.

Table 7.2 Comminution design parameters

Parameter		Units	Value
SMS	CW1	kWh/t	21.4
	RW1	kWh/t	13.7
	BW1	kWh/t	13.9
	A x b		48.8
	SG		3.47
Footwall	CW1	kWh/t	-
	RW1	kWh/t	-
	BW1	kWh/t	30.2
	A x b		34.3
	SG		2.87
DISS	CW1	kWh/t	24.0
	RW1	kWh/t	13.4
	BW1	kWh/t	14.6
	A x b		51.2
	SG		3.07

Note: CWI: Crushing Work Index

RWI: Rod Mill Work Index

BWI: Ball Mill Work Index

A*bJK SAG: Milling amenability parameters

SG: Ore specific gravity

Snowden comment

The comminution circuit is conservatively sized and is flexible with respect to a variety of operating scenarios. Steel can be added if required, the pebble crusher can process 100% of the feed and the grate and pebble size can be optimised. There are a number of examples of this type of circuit operating successfully. The mill will have all of the SACADA type process control loops typically used for such a circuit, and thus is endorsed by Snowden.

Flotation testwork

Site water is excellent, and although Perth tap water was used for all flotation tests the impact of site water has been tested and resulted in a suitable local water source being selected.

Snowden comment - copper flotation

The primary grind size of P80 106 μm is coarser than normal in this type of circuit. The requirement here is to achieve a grade suitable for sale to smelters. There is flexibility with grade and recovery. This is a robust simple float and not technically demanding. Testwork has shown that the production of a saleable concentrate is possible from all three ore types; this is possible due to the liberated form of the chalcopyrite in the ore. The circuit has a tower mill to regrind composite particles and ensure the copper grade is as high as possible.

Bulk Flotation

Extensive bench scale and locked cycle flotation testwork was carried out by G&T Canada and thereafter by AMMTEC. The process is robust and aimed at maximising metal recovery and sulphur recovery. Technically this is a less demanding flotation regime than one normally finds because grade is not a limitation.

Cobalt recovery is important from a revenue perspective and the cobalt recovery is dependent on the ore type. Of the three ore types described above (Section 7.2.1), Kyly B is most affected by cobalt inclusions in pyrite, and also has a lower nickel recovery (nickel associated with pyrrhotite). Kyly A is less affected and achieves high recoveries. The DISS ore offers the best potential to make a higher grade cobalt concentrate, albeit at slower flotation rates.

The variable recovery of cobalt reflects the differing mineralogy in the ore types.

Snowden comment – blending of ore types

It is important to note that it is relatively easy from a mining viewpoint to blend the two main ore types, DISS and SMS, and that the concentrator is designed to handle varying proportions of these two types of ore throughout the mine life. In fact the recovery of the plant is not sensitive to the relative proportions of DISS and SMS, which provides more flexibility to the mining schedule.

Thickening and filtration

Extensive thickening and filtration testing has been undertaken on samples of concentrates and tailings. In addition the transportable moisture limit ("TML") has been determined. Drying of the concentrates is not required. The TML dictates the moisture level above which the concentrate cannot be shipped.

Pilot plant testwork

The pilot plant failed to achieve bulk concentrate sulphur grade other than the last day of operation. As a result the bulk flotation had to be repeated; however, notwithstanding the short pilot plant operating time the plant did demonstrate the ability to achieve high grade copper concentrates with little cobalt and a bulk concentrate with high cobalt recoveries.

Snowden comment

There will be a learning curve for the plant and the three month ramp-up period will allow fine tuning of the process. While the suggestion to do more piloting is a simple one this may not achieve anything worthwhile. Evidence from Radio Hill selective flotation (copper from nickel) is that a circuit which could not be made to work at AMMTEC was in fact successful on site. The technical resources for

process metallurgy should be put in place during commissioning and should be available for the first six months of operation.

Predicted concentrator recoveries

Flotation testwork was conducted at AMMTEC using the MC1 and MC2 composites described above, and using both open and locked cycle tests. The locked cycle tests best reflect the estimated recovery from the concentrator. A further nine variability tests were conducted using composites from the Wallaby and Wombat zones. Composite MC1 was regarded as representative of the SMS ore and MC2 representative of the DISS ore.

Snowden comment

The flotation circuit is conventional and flexible. The copper-gold concentrate is optimised for grade to maximise smelter returns while the bulk concentrate is optimised for sulphur and metal recovery rather than grade.

Expected recoveries

Flotation recoveries vary through the project life and are dependent upon the proportion of pyrite and pyrrhotite rich sulphides.

Recoveries to the bulk concentrate are:

- Cobalt 95-96%
- Nickel 86-94%
- Zinc 86-92%
- Copper 14-23%.

Recoveries to copper-gold concentrate are:

- Copper 75-86%
- Gold 62-64%.

Copper-gold concentrate grades are:

- Copper 27-28%
- Gold 10-14 g/t.

Snowden comment - recoveries

There are limitations to the method used and a number of assumptions and qualifications apply. A range of recoveries have been quoted to reflect this. Under the circumstances Snowden believes that the approach used is valid.

Optimisation testwork

As part of the optimisation process carried out by Vulcan following the release of the DFS a certain amount of testwork was carried out. This includes the following:

- A report by Orway Mineral Consultants to review the testwork and recommend a size for equipment to perform the comminution duty with an autogenous and recycle pebble crusher.
- A review and FEED performed by Turnberry RPA to facilitate capital and operating costs to be derived operation of the paste backfill facility.
- Filtration testwork by FL Smidth to determine the filtration rate for equipment sizing in relation to the filter utilised in the paste backfill facility.

Snowden comment

In Snowden's opinion the extra testwork commissioned by Vulcan was fully justified and has added to the body of knowledge related to the project and to the optimisation changes detailed below and in Section 12.1.3.

7.4 Process flowsheet description

7.4.1 Introduction and overview

The Kylylahti project process flowsheet is conventional and employs proven technology. It is considered to be robust and flexible from an operational perspective. This is a typical conventional Outokumpu process flowsheet and is similar to those employed at the Keretti and Vuonos mines (Figure 7.2). The process flowsheet for this type of ore is well established by Outokumpu. The annual ore throughput rate will be 800,000 tpa. The design availability of the plant is 91.3%, reflecting climatic conditions. The copper-gold concentrate will represent 3.5% w/w of the feed, the bulk concentrate will be 32% w/w of the feed and the balance will be tailings. A flowsheet schematic is given in Figure 7.3.

Figure 7.2 Location of Kylylahti mine and concentrator

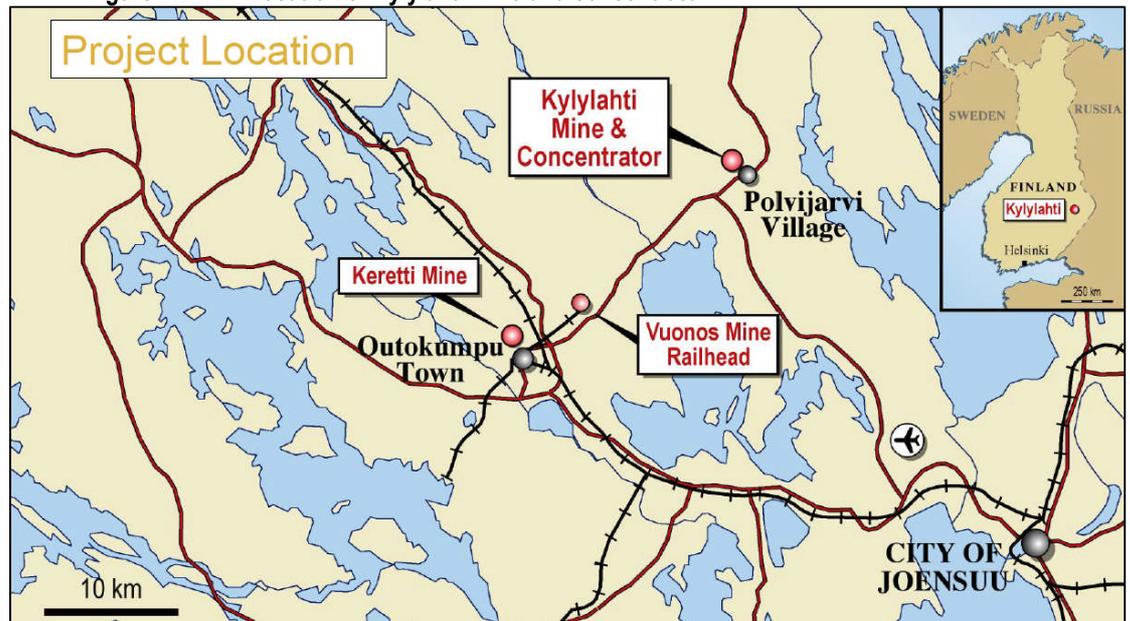
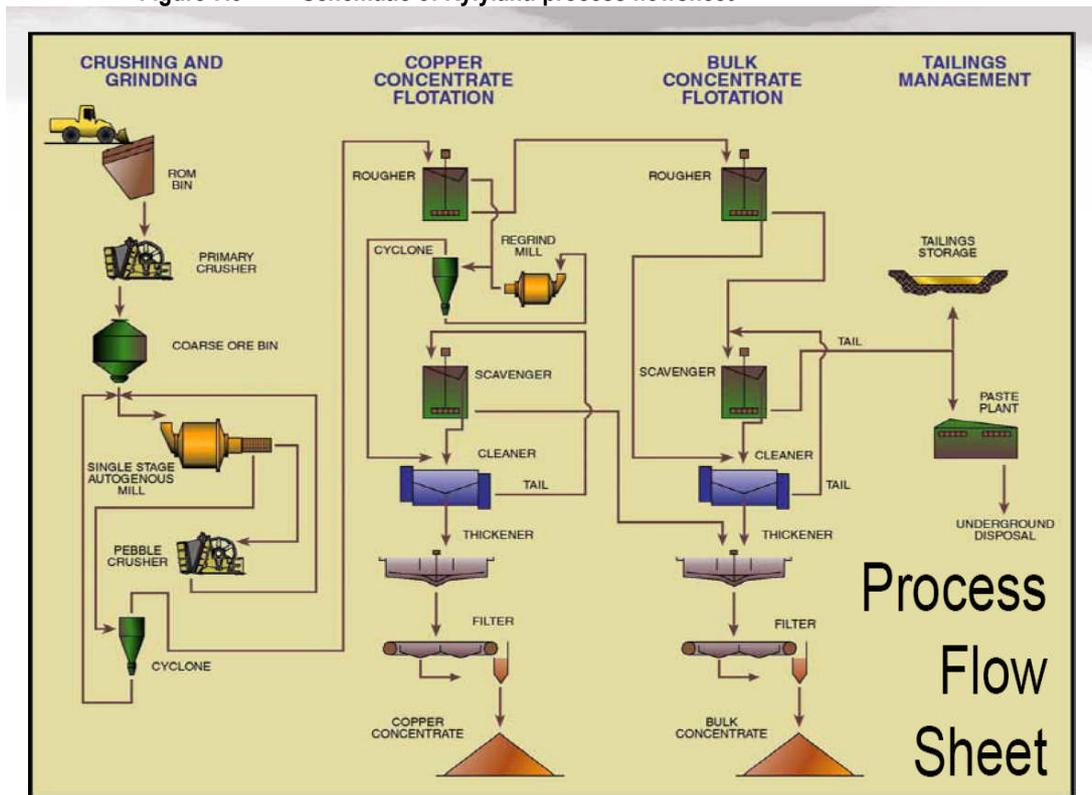


Figure 7.3 Schematic of Kylylahti process flowsheet



7.4.2 Crushing and ore storage

The run of mine ('ROM') pad will have three days ore storage. Ore will be reclaimed by front end loader ('FEL') from the ROM fingers into a ROM bin, which has a variable speed apron feeder to a single toggle jaw crusher with a nominal product size of P80 155 mm and conveyed to a coarse ore bin ('COB'). A rock breaker has been included to break oversize and clear blockages on the grizzly.

The COB has a live capacity of 1,000 t or 10 hours at the design capacity. The ore is competent, and there is no oxide or clay ore which could cause materials handling problems. The crushing circuit has excess capacity above design.

7.4.3 Grinding and classification

The Kylylahti sulphide ore is conveyed from the COB via an apron feeder to the primary FAG mill (6.1 m by 6.1 m, 2,400 kW) with a recycle pebble crusher. The product grind size is P80 106 μm . The mill is variable speed and the crushing circuit can accommodate 100% of the feed. The mill motor can accommodate a 6% ball charge should this be necessary. The mill will be lined with Polymet type liners, which are a steel rubber composite liner.

The trommel oversize (+6 mm) from the mill will be conveyed to the pebble crusher and returned to the mill feed. A bypass facility is included for crusher maintenance and tramp iron protection using a magnet. The trommel undersize will report to the mill discharge sump and be pumped to the cyclone cluster. Cyclone underflow will be returned to the mill feed while the cyclone overflow will report to the copper flotation feed.

The mill will have the typical automatic control systems usually employed for such a grinding circuit. The milling circuit has excess capacity above design.

7.4.4 Copper gold flotation

The flowsheet employed is typical of those employed in the past to process for Outokumpu type deposits and is considered conventional in respect of stage and equipment selection. Tank cells will be installed in the roughing scavenger and for larger cleaning duties and smaller tank or trough cells will be used for the cleaning duties. Key points are:

- Ore grades and mineralogy will vary due to the variable proportions of semi-massive and disseminated ore mined over the mine life.
- Ore ageing (oxidation) has not been investigated, although Vulcan points out that this is not likely to be a major issue due to the climate, the dry conditions underground, and the relatively short surface stockpile residence time.
- Masses recovered by the flotation circuit vary over the mine life due to declining sulphur grades over the life of the mine.
- No detrimental effects are anticipated from dissolved ions in the water taken into the circuit from mine dewatering.

Copper flotation will incorporate roughing, cleaning and cleaner scavenging stages, and operate in conjunction with the grinding circuit for a total plant availability of 91.3%. Copper flotation reagents are added to the milling circuit where there is sufficient time for conditioning. The reagents added are lime for pH control, Dextrin for pyrite depression in the rougher circuit and zinc sulphate as a depressant in the cleaning circuit. Aerophine 3418A is used as the collector. Frother is added at the start of the copper flotation circuit to promote stable froth conditions.

The primary cyclone underflow will report to the copper rougher flotation cells. Five tank cells in series will recover the majority of the contained copper into a combined rougher concentrate, which will report to the copper cleaner circuit. Copper rougher tailings will report to the bulk flotation conditioning tank. Total roughing residence time will be approximately 12 minutes, representing a scale up factor from testwork of two times. Copper rougher concentrate will be pumped to a set of hydrocyclones for classification to achieve a cyclone overflow of P80 of 40 µm. Cyclone overflow will be directed to the first cleaner cell feed box. Cyclone underflow will be reground in the regrind mill, which is a Japanese Tower mill.

Copper cleaning will be carried out in three stages with final cleaner concentrate being directed to the copper concentrate thickener. The first cleaner tail is rejected to the copper cleaner scavengers and the tail from this cell reports to the bulk sulphide thickener (copper cleaner tails). The first cleaner concentrate is fed to the head of the second cleaner concentrate bank. The second cleaner concentrate is fed to the head of the third cleaner cells. The final cleaner concentrate reports to the copper concentrate thickener.

Cell levels will be controlled by means of pinch valves and level transmitters installed in each tank cell and on each bank of trough cells. Air control in all cases will be manual, via valves located on the air inlet line to each cell.

Slurry pH will be measured on line by pH meters installed in the feed box of the first rougher cell, cleaner bank and recleaner bank and will be used to control lime addition to the various dosing points.

7.4.5 Bulk concentrate flotation

Bulk flotation will incorporate roughing, scavenging, and cleaning stages for the production of final concentrate. Copper rougher flotation tailings from the thickener will be pumped to the bulk sulphide conditioning tank, where they will be

mixed with sulphuric acid for pH adjustment, with copper sulphate for activation and Potassium Amyl Xanthate ('PAX') as a collector. The bulk concentrate conditioning tank will be provided to allow addition of the reagents and, based on laboratory testwork, will have a total residence time of three minutes.

Slurry will overflow the bulk sulphide conditioning tank and flow under gravity to the first of two rougher tank cells in series. Total roughing residence time will be approximately 16 minutes. Frother will be added to the feed box of the first roughing cell and slurry pH will be measured by means of an online pH probe to control the addition of acid to the conditioning tank.

Bulk sulphide rougher concentrate from all cells will combine in a concentrate hopper and will be pumped to the bulk sulphide cleaner flotation circuit. Bulk sulphide cleaning will take place in a bank of four tank cells with a total residence time of 33 minutes. Bulk sulphide cleaner tailings will be combined with the rougher tails and will report to bulk sulphide rougher scavenger flotation. Residence times for the scavenging duties will be 20 minutes. Bulk sulphide scavenger tailings will be pumped either to the TFS or the paste plant at a nominal level of 35% solids by weight.

Cell levels will be controlled by means of pinch valves and level transmitters installed in each tank cell and on each bank of trough cells. Air control in all cases will be manual via valves located on the air inlet line to each cell. Slurry pH will be measured on line by pH meters installed in the feed box of the first rougher cell and the cleaner, recleaner and finisher banks and will be used to control lime addition to the various dosing points. Provision will be made to add frother to the cleaner bank and xanthate to the cleaner and recleaner banks. The final bulk sulphide concentrate will be pumped to the bulk sulphide concentrate thickener for partial dewatering.

The copper and bulk sulphide flotation circuits will be laid out adjacent to each other and will be joined with a common platform to facilitate ease of operator access. This area will also be enclosed in an insulated and heated building and serviced with an overhead travelling crane.

Flotation circuit spillage will be contained by banded concrete slabs separated into copper and bulk sulphide areas by a dividing wall and recovered to the process via vertical spindle sump pumps. Sufficient fall will be provided on flotation circuit floors to allow for the high specific gravity minerals to be washed to sumps. Specialised froth pumps, designed with industry standard froth factors, will be employed for all concentrate duties. Variable speed drives will be installed on appropriate flotation feed and tailings pumps to allow control of overall flow rates through the circuit.

7.4.6 Sampling, assaying and metallurgical accounting

An eight stream XRF Analyser will be installed to provide real time assays and measure the flotation performance. Shift samples will be generated from the analyser and used for metallurgical accounting purposes and shift reporting.

7.4.7 Copper-gold concentrate handling

The concentrate handling circuit will produce separate dewatered concentrates with moisture levels suitable for road and rail transport. The moisture limits for transportation have been determined via testwork to be 11% for the copper-gold concentrate and 9.7% for the bulk sulphide concentrate.

The concentrate handling circuit will operate in conjunction with the flotation circuit 365 days per year, 24 hours per day, for a total availability of 91.3%. Concentrate is to be discharged into a storage facility. Concentrate filters will be housed in the same enclosed building as the flotation circuit.

The final copper-gold concentrate will be pumped to an eight metre diameter high rate thickener fitted with a self-diluting feedwell to allow clarified overflow solution to dilute the incoming feed to 15% solids. A solids flux rate of 0.25 t/h/m² has been arrived at via testwork. Diluted flocculant will be added to the thickener feed well to increase particle settling rates. Water sprays will be installed on the thickener to assist in froth breakdown.

Thickened underflow slurry will be withdrawn from the thickener at 70% solids and pumped to a 40 m³ agitated concentrate storage tank providing six hours storage capacity. Thickener overflow will report to the process water circuit for reuse. Flocculant addition will be controlled on the basis of the thickener bed level. A ball float and reed switch bed level element will continuously measure the interface between settled solids and supernatant liquor, the result of which will be used to control the speed of the flocculant dosing pump.

Thickener underflow density will be controlled by varying the rate of withdrawal. Bed pressure will be monitored by a pressure transmitter installed on the underflow cone of the thickener. Increasing pressure will indicate an increase in settled solids pulp density, calling for an increase in the speed of the underflow pump. Decreasing pressure will require a decrease in underflow pump speed to maintain underflow density. Peristaltic hose pumps have been selected for the thickener underflow duties in order to handle high pulp densities and associated viscosities. Provision has been made for underflow recycle to prevent consolidation of the bed during plant shutdowns. Thickener concentrate will be pumped from the concentrate storage tank to a 30m² ceramic capillary vacuum filter and filter cake will discharge onto a fixed speed conveyor for transfer to a storage shed.

7.4.8 Bulk concentrate handling

The final bulk sulphide concentrate will be pumped to 14 m diameter high rate thickener fitted with a self-diluting feedwell to allow clarified overflow solution to dilute the incoming feed to 15% solids. A solids flux rate of 0.25 t/h/m² has been arrived at via testwork. Diluted flocculant will be added to the thickener feedwell to increase particle settling rates. Water sprays will be installed on the thickener to assist in froth breakdown. Froth followers and de-aerators may be fitted in future should they be required.

Thickened underflow slurry will be withdrawn from the thickener at 70% solids and pumped to a 100 m³ agitated concentrate storage tank providing three hours storage capacity. Thickener overflow will report to the process water circuit for reuse.

Flocculant addition will be controlled on the basis of thickener bed level. A ball float and reed switch bed level element will continuously measure the interface between settled solids and supernatant liquor, the result of which will be used to control the speed of the flocculant dosing pump. Thickener underflow density will be controlled by varying the rate of withdrawal. Bed pressure will be monitored by a pressure transmitter installed on the underflow cone of the thickener. Increasing pressure will indicate an increase in settled solids pulp density, calling for an increase in the speed of the underflow pump. Decreasing pressure will require a decrease in underflow pump speed to maintain underflow density. As with the copper-gold concentrate, thickener concentrate will be pumped from the concentrate storage tank to a 60 m² ceramic capillary vacuum filter and filter cake will discharge onto a fixed speed conveyor for transfer to a storage shed.

7.4.9 Tailings disposal

The tailings will be disposed of in both a conventional paddock style conventional TSF and pumped underground as paste fill.

7.4.10 Reagents

The processing plant will employ the following reagents:

- Hydrated lime (pH modifier)
- Sulphuric acid (pH modifier)
- Copper sulphate (CuSO₄, bulk sulphide activator)
- Zinc sulphate (ZnSO₄, pyrite depressant)
- Frother
- Dextrin (pyrite depressant)
- Aerophine 3418A (418A, copper collector)
- Potassium amyl xanthate (PAX, bulk sulphide collector)
- Flocculant.

Reagents will be stored in accordance with statutory regulations in a dedicated building.

7.4.11 Power and water

The power supply will be from the local grid. There is no shortage of good quality water at the site. The water circuit includes raw, potable, process and tailings return water. Process water will be sourced from mine dewatering whilst potable water will come from the town supply.

7.4.12 Occupational health and safety (OH&S) at the process plant

Vulcan has developed high standard Safety Policies to be implemented in the process plant. The environmental and health issues of airborne dust being carried into residential areas and workplaces have been addressed by the use of enclosed buildings and the installation of dust collection units in the crushing plant and coarse ore bin. In addition, the conveyors transferring material are covered and the intermediate ore storage is an enclosed bin rather than a covered stockpile. Water sprays will not be used due to the potential hazards presented by winter freezing.

The presence of tremolite, a fibrous mineral, in the disseminated ore means that this will have to be engineered to a high standard. Monitoring will ensure that there is no risk to the operators or townsfolk.

Within the concentrator there are no unusual chemicals outside those normally used and therefore no specific OH&S risks.

7.5 Process optimisation

The process design is well advanced as a result of the Optimisation Study. The detailed design phase will allow further improvement with the process design. A design availability of 92% has been used as the basis for design. The ore has a high specific gravity of 3.0 to 3.3 t/m³. The SMS ore is abrasive but more amenable to SAG milling. The Bond Ball Mill work Indices are 13.1 kWhr/t for the SMS ore and 15.3 kWhr/t for the DSS ore, which is not in the range normally considered for 'hard' ore. The Rod Mill Work Indices are similar. The copper ore flotation laboratory residence time is 12 minutes (scale up 2.5) with a mass recovery between 6 and 14.5% wt/wt. The 2.5 scale up is an industry convention backed up by detailed and published studies. The regrind mill produces a product size of P80 40 µm. The bulk laboratory flotation residence time is 8 minutes roughing, with 10 minutes scavenging (scale up 2.5), with a mass yield of concentrate between 21 and 33% by weight. The copper-gold concentrate thickening is 0.25 t/m²/hr with a thickener underflow density of 70% wt/wt. The specific filtration rate is 650

kg/h/m², with a cake moisture of 9% wt/wt. The bulk concentrate thickening is 0.25 t/m²/hr with a thickener underflow density of 70% wt/wt. The specific filtration rate is 780 to 800 kg/h/m² with a cake moisture of 8% wt/wt.

Snowden comment

The above data does not indicate any unusual or abnormal figures; in fact all of the numbers are within the range of what may be anticipated from such an orebody.

7.5.1 Process plant optimisation details

The following changes have been made to the equipment selected for the plant during the DFS optimisation process:

- The grizzly feeder and plate work have been excluded.
- New dry dust collectors will be added at the crusher and ore storage areas.
- An air receiver has been added for the crushing area.
- A new sump pump has been added to the crushing area .
- Insulated cladding has been added to crusher building.
- The SAG mill and Ball mill have been replaced by a FAG mill and pebble crusher.
- Pebble crusher feed, discharge and re-cycle conveyors including a tramp magnet have been added.
- The trash screen to the flotation circuit has been deleted.
- The regrind 'tower' mill has been replaced with a conventional mill.
- There has been a rationalisation of the hoppers, tanks and pumps associated with conditioning and slurry transfer in the flotation circuits.
- The thickener and pumps relating to a separate water circuit for each flotation duty have been deleted.
- New pumps have been added and existing pumps have been altered to pump unthickened tails.
- Hoppers for copper cleaner concentrate streams have been added.
- Filter circuits have been re-configured and transfer conveyors for concentrates to new storage building have been added.
- Flocculant mixing has been considered as vendor supplied packages.
- Added lime mixing and storage tanks have been added.
- There has been a rationalisation of the sizes of reagent mixing and distribution tanks.

Snowden comment – flowsheet changes

The DFS process flowsheet has undergone a number of changes from the DFS configuration. The change from two mills to one FAG mill is significant in reducing the capital cost. It is a higher risk option; however, it has been shown to work on this type of ore. In addition the recycle pebble crusher and variable speed mill will mitigate the FAG risk. Moreover, the mill has been sized to allow steel addition. Experience from many projects is that if throughput is mill limited the addition of steel will draw more power and increase throughput.

7.5.2 Tailings management

The DFS optimisation has seen a number of changes, as shown below, with the purpose of operational and environmental improvements. The paste plant has been relocated to allow paste to be distributed underground at an inclination of 60 degrees. The tails thickener has been relocated to the paste plant from the concentrator. A High Density Polyethylene ('HDPE') liner with bentonite bedding has been allowed under the TSF in order to ensure that the environmental permit condition of one metre of compact/compacted moraine with permeability of 1×10^{-8} m/s (K-value) or better at the base can be achieved.

7.5.3 Site infrastructure, water management and services

Changes to the DFS as detailed below are project refinements:

- The grizzly feeder and plate work have been excluded.
- The administration and fixed plant workshop has been combined in a single building, and the vehicle workshop has been deleted.
- A store has been added for general storage and spares and consumables.
- A reagent store has been added.
- Raw water tanks have been removed, along with other similar equipment by recycling process water for all process duties other than flocculant mixing with potable water.
- HV power distribution has been changed to 20kV from 11kV in line with common Finnish practice.
- An optimised HV distribution system and single-line diagram has been developed.
- Power correction equipment has been added.
- Transformers for voltages less than 690V have been added.
- Switch rooms and motor control centres have been optimised.
- Emergency power generators have been removed.
- The process control system has been simplified.
- Miscellaneous lighting, earthing equipment, cable trays, welding outlets and junction boxes have all been optimised.
- Overhead power facilities for TSF decant pumps have been added.
- Bulk earthworks and road works have been rationalised.
- An allowance for further geotechnical investigation has been added.
- Locations of infrastructure elements have been optimised to reduce cost and comply with recommended separations from the 110kV power line.

7.6 Concentrator operational issues

7.6.1 Management

The Process Manager will be responsible for efficient running of the Kylylahti Concentrator. Among the key personnel reporting to the Process Manager will be the Maintenance Manager. The Processing Manager will also be responsible for recruiting and developing the processing team for the Kylylahti Concentrator; this includes the Maintenance Manager and the maintenance planning team.

KCO staff will run the concentrator. The feasibility study assumes that KCO staff will also undertake maintenance; however, a number of organisations in the Joensuu – Outokumpu region are able to carry out ongoing maintenance on plants such as the Kylylahti Concentrator. Once the study is in place, discussions with these groups will determine the financial advantage to KCO of this approach.

7.6.2 Operators

KCO intends to employ its own operators for all positions in the Kylylahti concentrator and, where appropriate, in the underground operation.

With high levels of unemployment in the region, the local and national government has committed to providing significant training subsidies to KCO so that personnel can be trained to work in the mine and the Kylylahti concentrator facility. Given the history of Outokumpu (mining), other labour intensive industries (forestry) and high-tech industries (electronics) in the region it is expected that the skills of applicants will be appropriate to converting to mining and metallurgical roles.

The Kylylahti concentrator has been designed to use a minimal amount of labour. Equipment such as an on-line analyser has been installed to reduce labour required to undertake ongoing sample collection throughout the plant.

Hiring of skilled tradesmen for the mine, concentrator and hydrometallurgical facility is expected to be straight forward.

Snowden comment – constraints to production

The mill throughput is conservatively designed and includes the flexibility to add balls should this be necessary and increase the pebble port size and crush more tonnes.

Even if the ore hardness were to increase or the characteristics changed there is a very low risk that the mill will not achieve design throughput.

With respect to metal recoveries and grade, if the design grade was not achieved the transport and refining cost would increase but the risk here is considered low. With regards to metal recoveries the testwork and pilot plant work have demonstrated the flexibility and ability to adjust along the grade recovery curve. The previous history of processing these orebodies also provides assurance that targets will be met.

7.7 Commissioning and handover

This section describes the plan Vulcan and its engineer would use for starting up the project process facilities. It defines the various stages of commissioning and identifies responsibilities for activities conducted during those stages. It has formed the basis for the study and outlines the methodology to be followed to assist with project scheduling, as well as preparation for commissioning.

Vulcan has deemed two Phases: Phase 1 includes pre-commissioning activities and Phase 2 covers planning and wet commissioning.

Following pre-commissioning is wet commissioning followed by process commissioning, where items of equipment grouped together into plant modules are run as close as practically possible to normal plant operating conditions, first without the introduction of ore or reagents then under plant operating conditions. There are many terms for these operational descriptions but for consistency this document will use the terms above.

Pre-commissioning

Pre-commissioning is the verification that the facility, or a section thereof, has been constructed in accordance with the relevant design and vendor documents and that each item of equipment is operationally acceptable and installed.

Pre-commissioning consists of two sequential activities, construction and installation testing, followed by dry testing of all equipment without load. When pre-commissioning is complete, the installation is said to have achieved mechanical completion.

Construction and installation testing would typically include hydrostatic pressure tests, flushing of lines, alignment checks, electrical point to point checks, merger tests and component identification checks. Dry testing includes motor direction tests on all motors prior to drive connection, all drives run, valves stroked open and closed, conveyors run and tracked, instruments checked, control system verified and facility sequence testing. By the conclusion of pre-commissioning all equipment and systems must be cleaned out.

The Engineer's Construction Manager, via the Construction Supervisors, is responsible for managing all pre-commissioning activities, along with recording and approval of results. The testing will be conducted by the appropriate contractor. The Commissioning Manager will assist with co-ordinating the dry testing phase of pre-commissioning.

Vulcan will advise the Engineer of any tests that they are required to be witnessed and ensure the availability of suitable witnesses for those tests.

Mechanical completion

Mechanical completion of a section of the plant is achieved when pre-commissioning is complete and that plant section meets all requirements with respect to design, safety, physical operability, specifications and the relevant module is ready for extended operation and/or the introduction of ore/process fluids.

Wet commissioning

A commissioning plan will be developed along with a suitable commissioning schedule. Wet commissioning consists of successfully testing and operating the equipment grouped together into systems or modules, on an inert material, generally water or air, but without ore, reagents or other process material. At the successful conclusion of wet commissioning, ore/process fluids are introduced into the circuit and process commissioning commences.

The Commissioning Manager is responsible for planning and execution of all wet commissioning activities. The commissioning team will conduct wet commissioning with assistance from the client's operating personnel. This phase forms an important part of plant familiarisation and training for the plant operators.

Process commissioning

Process commissioning follows the successful completion of wet commissioning. During this time the initial introduction of ore and reagents to the process will occur. The circuit will be operated to achieve nominal throughput and metallurgical performance.

Process commissioning will be the responsibility of Vulcan under the guidance of the Project Engineer.

Documentation and handover

Detailed commissioning requirements will be incorporated into all contracts including methodology and requirements. At the completion of Phase 1 and 2 all documentation will be handed over to Vulcan.

7.8 Ramp-up

Snowden comment

The process plant is not a complex or sophisticated chemical plant. We would expect Vulcan to achieve design production and efficiency within a three month time frame. With a well designed plant incorporating local knowledge and provided adequate training programmes are in place we cannot see why this would not be achieved.

Additional technical resources should be engaged during this period to fine tune the process as soon as possible.

Table 7.3 Suggested ramp-up time for concentrator

Month	% Design throughput
1	50
2	75
3	100

8 Infrastructure

8.1 Introduction

The infrastructure in Finland is excellent. There is an extensive and well-maintained road system plus a low cost reticulated power supply to most areas in the country, which is a mixture of hydroelectric, coal-fired and nuclear. The government is very supportive of new mining projects and will assist with infrastructure to facilitate a project if there are alternate use benefits. There are bountiful supplies of very good quality water. Similarly, the availability of all weather shipping ports (icebreakers have kept Finnish ports ice free for more than 60 years) and infrastructure such as rail is excellent.

The Kylylahti mine and concentrator site at Polvijärvi is accessible via a 400 m long road from national highway 502. Access to the Finnish rail system is available 15 km from Polvijärvi at Vuonos. The existing main access road requires upgrading to a sealed road, and new unsealed roads are required within the project area. Figure 8.1 is an aerial view of the mine and processing plant site location.

Figure 8.1 Aerial view of Kylylahti site location



8.2 Site layout

Figure 8.2 shows the current design for the site layout, which has not changed significantly from the DFS to the optimisation stage. Access will be through a main security gate to the Administration car park. Traffic will be minimised by keeping Administration and other buildings away from the operating areas.

8.3 Power

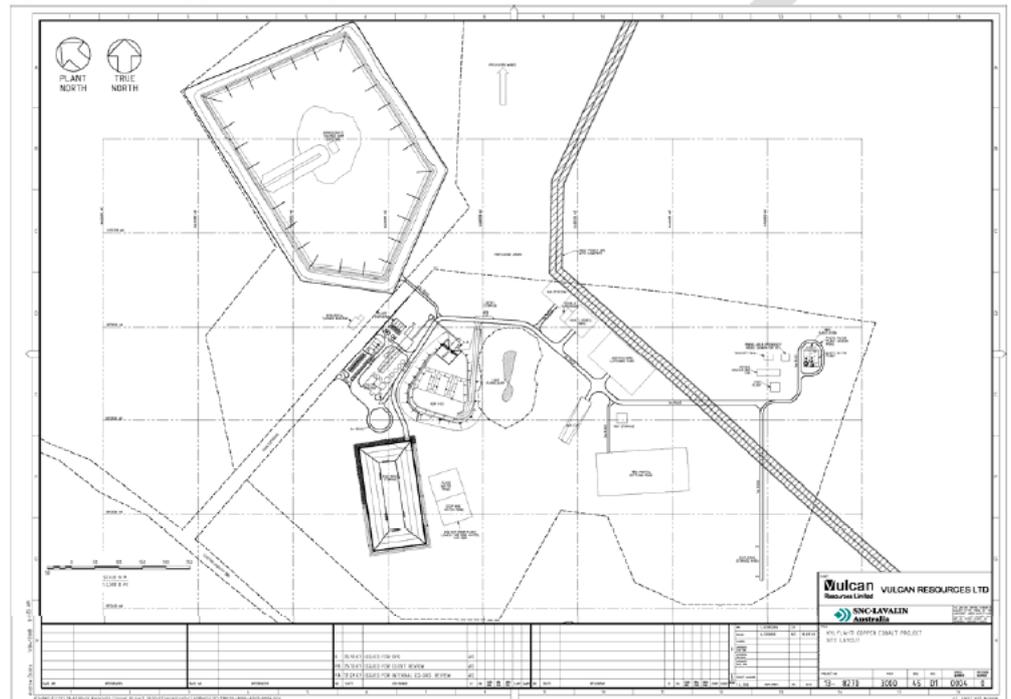
The power supply to the mine and the plant will be from an 110kV overhead supply via the transmission line that runs through the mining lease.

The maximum power requirements for the process plant, mine and associated facilities at the 800,000 tpa rate have been estimated as follows:

- A connected (total draw) load of 11,827 kW
- A maximum (utilised) demand of 9,268 kW.

A 110kV/20kV main substation will be constructed on the lease and will contain all the necessary circuit breakers, current transformers, voltage transformers and primary step down transformer for the operations. This facility will be the origin of the 20kV distribution system to the mine and paste plant. The capital cost for the system augmentation to provide power at site is included in the study capital cost. The power cost is estimated at €0.06/kWhr which provides a significant advantage for a major cost item.

Figure 8.2 Kylylahti site layout schematic



8.3.1 Power reticulation

The main substation will include a 20kV switch room and this will contain the necessary circuit breakers and equipment for distribution. A feeder will deliver power via a buried cable to the concentrator 20kV switch room. A separate feeder will deliver power to the mining operations 20kV switch room via a buried cable.

The concentrator 20kV switch room will distribute power to the concentrator, administration and service facilities. The mining operations switch room will distribute power to the surface mining infrastructure as well as deliver power to the underground operations at 20kV. Low voltage power for the operations will be 690V.

8.3.2 Emergency power generation

Emergency power generation has been deleted as part of the study optimisation following discussions with the power supplier and consideration of the reliability of the service provided.

Snowden comments

Snowden sees the lack of an emergency power system as surprising and somewhat of a risk as power outages do occur during heavy snowfall in other countries having a similar climate. Vulcan has provided further background to this decision, and

notes that the critical power requirement for key mine and plant items is minimal from a safety and critical operability consideration. Vulcan notes that a range of power rental options are available from Vuonos or from Outokumpu at relatively short notice, but also that the purchase of a small back-up generator is being considered. Snowden does not believe that this will impact duly on the capital estimates.

8.4 Water

The mine is expected to supply water in excess of the requirements of the concentrator and paste plant. Water for the project will be sourced principally from mine dewatering, harvested rainfall and snowmelt. Upper mine water is not expected to be contaminated and is expected to be able to be discharged directly to the environment after settling suspended solids. The control of mine water inflow will be by face pumps discharging to sumps located on each mining level, from which the water will be lifted in stages to a main pumping station 170 m below ground level. Water will be pumped from there to a sedimentation pond for the settling of solids. In addition to settling of solids, the mine water may require treatment prior to discharge to the environment and a project water treatment plant has been costed for that purpose.

The water quality is excellent.

8.5 Infrastructure optimisation

Changes between the DFS and the optimisation study to the infrastructure area have been detailed in Section 7.5.3.

8.6 Transport logistics

There is a very good arterial road system in Finland. Transportation to/from site of major items will be as follows:

- Site access - there will be a main site access road for all vehicles, and employees will park in the adjacent car park.
- Concentrates - all concentrate movements will be by carried out by independent contractors.
- Fuels - fuel delivery systems will be supplied and maintained by the fuel vendors.
- Quicklime will be trucked to Kylylahti from Jouhi in southeastern Finland.
- Consumables - all will be delivered to site by trucks.

8.7 Logistics for transport of concentrate

Vulcan proposes to move the copper-gold concentrate from the mine to the railhead at Vuonos by truck, and thence by rail to a smelter. The bulk concentrate (which is much greater in mass) is proposed to be moved entirely by truck. The trucks will have a 40 t capacity, and the concentrate within the trucks will be loose although the truck will be covered. At the Vuonos railhead a concentrate shed will be constructed, with concentrate loaded from the shed into rail cars by open loader. The rail cars themselves will be covered with a removable solid roof, thus eliminating wind dispersion of concentrate.

Finland has a very good arterial road system which sees much heavy haulage traffic (from forestry and from other mining operations); similarly the rail system is

extensive and efficient, and there is likely to be good availability of rolling stock for concentrate transport.

Snowden comment

Snowden sees very little risk in the proposed logistics for concentrate transport. Community consultation has raised a concern about potential noise pollution from traffic; Vulcan has addressed this via a traffic modelling study which shows the impacts of additional concentrate truck traffic to be largely insignificant. In addition, the local municipality at Polvijärvi will be constructing a new pedestrian/cycle path for safe transit. The movement of bulk material by road is in compliance with Finnish practice and should cause no concerns. The only point at which some care needs to be exercised is the transition from the closed shed to the rail cars, which will be effected by an open loader. Snowden suggests that Vulcan closely monitors this area of the supply chain and exercises caution in the presence of strong winds.

8.8 Accommodation

The workforce will be recruited locally and will live in the township of Polvijärvi. It is anticipated that affordable housing for the workforce will be freely available. There are reasonably priced houses available in Polvijärvi, and the major town Joensuu is only 35-40 km from the project. The Polvijärvi municipality has many vacant houses available to rent or sell, and is willing to renovate the houses to suit KCO's needs. People in Finland are generally used to travelling long distances to work. It is the case that every operating mine usually has a group of people who live in the mine region on a temporary basis, and travel every weekend to their home town. The availability of reasonably-priced rental accommodation in Polvijärvi will help attract these people. Travelling costs are personally tax deductible.

A local workforce can, and will, be used during construction and operation of the mine. There are also experienced miners living in Outokumpi-Polvijärvi-Joensuu region who are willing to work in this area if job opportunities exist.

9 Operational issues

9.1 Management structure

Project management will be carried out by a team comprising Vulcan representatives together with Finnish engineers. The basic structure of the hierarchy is shown in Figure 9.1. Key positions on the Project team are the Project Manager (now termed Project Director), the Engineering Manager, the Process Manager, the Administration Manager, the Construction Manager, the Project Engineer and the Project Accountant.

Vulcan's two main representatives are John Brodziak, Project Director, and Brad Brown, Engineering Manager, both of whom have considerable mining and project management experience. Vulcan's representatives will work closely with Ahma Engineers, which is Finland's largest independent and privately-owned full service project management company. Ahma will provide the other key roles in the organisation chart, with services in construction management, procurement and project controls engineering, along with site supervision, including the provision of a number of statutory personnel required under Finnish law.

In addition to the Project team, the owner's operations group will have staff assigned to the project. These staff members will be progressively recruited during the development stage so as to be sufficiently exposed to the plant, infrastructure and mine pending the commissioning of their areas of responsibility.

The project will be developed and owned by Vulcan's 100% owned Finnish subsidiary KCO. Figure 9.1 details the organisational chart – yellow boxes represent Vulcan employees, the remainder are KCO employees or Finnish contractors from Ahma.

Project management services comprise:

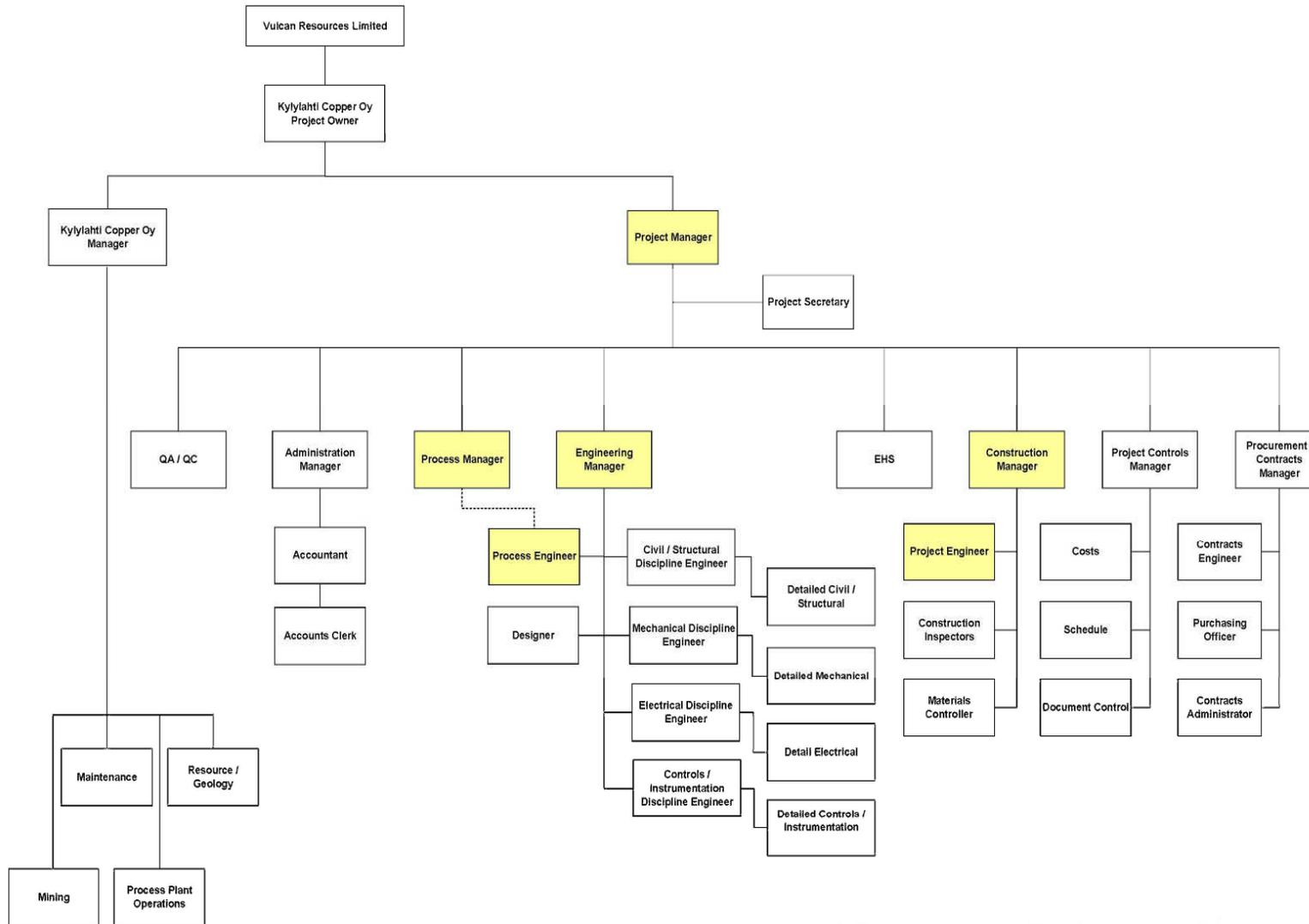
- Preparation of project procedures.
- Control and monitoring of the EPCM teams and the services provided in regard to safety, quality, cost, budget and schedule.
- Production of periodic reports.
- Management of the commercial administration of the project.

Hazard and Operability Analysis ('Hazop') reviews will still be required during the development stage with similar experienced consultants as well as Vulcan employees as they are recruited.

Snowden comment

Snowden views the proposed management structure as being appropriate for a project of this size. The project sits in an area with a long mining history and many of the key employees will have significant mining experience. A number of the key contracts have already been awarded, as described below.

Figure 9.1 Organisational chart for the Kylylahti project



9.2 Engineering services

Vulcan has elected to use Australian mine and plant engineering expertise. The main design contractors in Australia are Arcon (process and front end engineering and BEC Engineering (electrical engineering). Arcon is a relatively new company but its principals were involved in the foundation of Minproc Engineers, which is a well-respected company. Arcon has already won a number of other major contracts which suggests that it has good credentials. BEC Engineering ('BEC') also has a core of experienced principals and has an impressive track record on small to major mining projects.

Engineering management services will comprise:

- review of the DFS and supplemental optimisation
- review and develop project engineering design criteria
- co-ordinate and monitor the engineering of design concepts
- perform risk assessment, operability and maintainability studies on plant equipment, systems and services
- establish specifications for the supply and erection of plant, equipment, systems and services
- planning, scheduling and monitoring of engineering and drafting services including the reporting of progress on deliverables
- document control, including the timely issue of drawing revisions, specifications and data sheets
- preparation of technical input into procurement packages
- technical tender evaluation
- provision of technical advice to procurement and construction management
- review and approval of operating and maintenance manuals.

Process and basic engineering

Following on from the Optimisation Study the project definition is now clearly established. The process and basic engineering of the process plant and infrastructure will follow on from the current level of preliminary engineering which has been carried out as part of the DFS, and which has been of sufficient detail to identify the process flowsheet and fundamental layout. The preliminary engineering has also been used to generate equipment lists, capital expenditure estimates, power demand, reagent usages and operating costs.

Process and basic engineering will be carried out in Perth by Arcon, working together with BEC, and will involve an assessment of the results of all comminution and metallurgical testwork and the development and confirmation of the process flowsheet for the project scope. Alternatives identified during the DFS and the project review will be assessed by developing sufficient detail to allow comparative capital and operating cost estimates to be produced and for capital and operating costs to be reviewed and updated. Basic engineering will also make reference to Vulcan's Finnish staff and consultants for advice with respect to local standards and preferred modes of construction.

Completion of this phase will result in the finalisation of the process design, plant layouts, general arrangements and updated equipment lists. The basic engineering phase will also involve the identification and specification of long lead-time equipment for early procurement.

Detailed engineering

Detailed engineering will be carried out in Finland, and there will be a transition period towards the end of the basic engineering when the engineering focus will shift to the Finnish detailed engineering consultants. The transition will be undertaken to ensure a timely commencement of the detailed engineering. A small number of the Perth engineering team will transfer temporarily to Finland to ensure that the detailed engineering information is transferred effectively.

Arcon and BEC will be supported by Finnish engineers who will provide the valuable experience of Finnish standards and climatic conditions. SWECO Industry Oy, a major consulting engineering company in Finland, will be responsible for all detailed engineering on the project. Basic engineering design will be progressively handed over to SWECO by Arcon and BEC. Liaison between the engineers carrying out the basic design and those completing the detailed design will continue throughout the design process. Engineering design will be essentially complete prior to major works commencing on site. The SWECO engineers and designers allocated to the project have just completed a similar mining project in Finland which will provide valuable experience for Kylylahti.

Detailed engineering tasks are:

- process engineering to the extent of reviews of design from metallurgical and operability viewpoint, and commissioning
- civil engineering
- concrete and structural engineering
- mechanical and piping engineering
- electrical engineering
- instrumentation
- process control.

9.3 Construction management

As described above, construction management services will be provided by Finnish group Ahma Engineering, working closely with Vulcan's Project Director and Engineering Manager. The following activities will be carried out by Ahma as part of the EPCM team:

- establishment and management of procedures for all site based activities
- management of all contractors
- establishment and management of a materials control procedure for free-issued equipment and materials.
- arrange for the handover of spare parts and special tools or equipment purchased by Vulcan in time for commissioning and early operation.
- monitoring construction work for compliance with drawings and specification
- establishing and managing the site inspection and test procedures and maintain records.
- review, approve and monitor detailed work plans of contractors and sub-contractors.
- managing and co-ordinating the field engineering capability on site provided by the detailed engineering consultant.

- ensure compliance with and management of, the health, safety and environmental requirements of Vulcan.
- establishing and managing all temporary site construction facilities necessary for the works.
- managing all associated logistics and transport
- carry out all field purchasing, expediting and inspection services
- carry out all site construction administration and general office activities
- collate and report production, safety and workforce statistics
- managing the scheduling of contractors works to comply with the project construction schedule.
- co-ordinating and managing site industrial relations.

Snowden comment – engineering appointments and personnel

Vulcan has now awarded the key contracts for the project development. The engineers appointed will bring to the project process engineering skills, plant design and layout skills, expertise in tailoring engineering for Finnish climatic and operating conditions and current mine project experience in Finland. This has resulted in a mix of Australian and Finnish engineering groups and Finnish procurement and construction management expertise.

Snowden is of the opinion that Vulcan has assembled a strong team which combines Australian design expertise and mining project experience with Finnish construction, management, procurement and local knowledge.

9.4 Mining contract

Award of the mining contract is not as far advanced as the plant and services construction contracts. However, Vulcan has generated a short list of Finnish and other Scandinavian mining contracting companies which will be invited to tender. The initial contract, for the decline development and the first few levels, has undergone legal review. The tender documents have been generated in Finnish and are being translated into English for final review.

9.5 Employee relations

Illegal strikes are uncommon in Finland. There is a detailed process set up for both unions and employers to follow with respect to negotiations between employees and employers. There are several employee (unions) and employer organizations in Finland, covering the main industry branches. Those organizations tend to negotiate long-term labour agreements for the industries, and agree to main points on labour relations. The main issues are:

- salary increases for the period.
- changes in shift structure, underground working hours and other compensations.
- changes in working hours.

The mine union 'Metalliliitto' is the employee union, and technology industry 'Teknologia Teollisuus' is the employer organization. All the major mines in Finland follow those labour agreements, and are members of those organizations.

For representation during these agreements the mine workers select a representative from among themselves. This person will be allowed to spend some work time to handle the local union issues. The amount of time is agreed in the labour

agreement. This representative signs local agreements on behalf of the workers, and will handle labour-related issues as advised in the labour agreement.

The Finnish Ministry of Labour has provided and will continue to provide English translations to all agreements and labour laws.

9.6 Safety management

A Health and Safety Management Plan ('HSMP') has been prepared for the project. It provides an outline of health and safety issues associated with the project and corresponding procedures and guidelines for successful management of these risks.

Both Vulcan and KCO are committed to protecting the health, safety and wellbeing of their workers and aim to achieve a high level of safe work practices through development and implementation of the HSMP. Provisions of the HSMP apply equally to employees of KCO and direct employees of Vulcan.

The disseminated orebodies contain tremolite which is a fibrous asbestiform mineral. Within the process plant dust mitigation has been addressed.

Snowden comment

The overall impact of any fibrous minerals on the project is unclear at this stage. Vulcan should, wherever possible (and being mindful of the fact that OHS legislation in the past is less rigorous than at present) draw upon the experience in this area from the other Outokumpu mines which have similar geology to the Kylylahti project.

9.6.1 Training and competency

The local training centre (Pohjois Karjalan Aikuiskoulutuskeskus, based in Outokumpu) has expertise to train underground operators, underground maintenance personnel, concentrator operators and technicians. They have recently trained personnel for several operations in Finland, including the Talvivaara Mine (open pit) and Hitura Mine (underground). The programme lasts usually for 9-12 months, and 50 % of which time is spent in on the job training.

The course to become an underground operator involves:

- explosives handling licence
- long hole drilling
- jumbo drilling
- mechanized bolting
- mucking
- some maintenance.

The participants will tend to specialise in certain areas of expertise. The common procedure in labour training is to advertise job opportunities available via the training programme in the appropriate media. Applicants will also be screened by KCO and the training centre.

At the completion of the programme at least 80% of the participants must be employed by the mining company (this is part of the subsidy agreement between the company and government). The government will provide 40-70% of the training costs. The training programme (before compensations) for 15-20 persons costs approximately €165,000. This cost is paid by KCO which covers the wages paid to the trainees and the cost of the training course. At the completion of the course 40-70% of the cost is refunded by the government to KCO.

9.6.2 Operating procedures

Standard Operating Procedures have been and will continue to be developed by KCO to manage the critical risks identified in the project hazard and risk management process. Procedures will be developed either as KCO General Risk Management Procedures or as site-specific procedures. Development of procedures is an ongoing process whereby new activities are identified and old procedures improved as part of risk assessment and job safety analysis.

KCO will develop detailed procedures for the main operational activities and for additional activities that are identified during the course of operations and in the pre-operational risk assessment.

9.6.3 Continuous improvement

The KCO Health and Safety Management Plan works on a system of continual improvement. This will be achieved through the development of Key Performance Indicators and by management review.

9.7 Project implementation

9.7.1 Project schedule

The project schedule outlines the high level tasks required for project development. The schedule shows a total development duration of 36 months from the decision to proceed until full production. A ramp-up period of three months has been allowed for the concentrator to reach full production.

The key activities after the decision to proceed and which define the schedule are:

- completion of bulk concentrate sales agreement and copper-gold concentrate sales agreements.
- funding arrangements
- basic engineering and site establishment
- mine development
- basic and detailed design of the process plant
- procurement of process equipment
- plant construction
- plant commissioning.

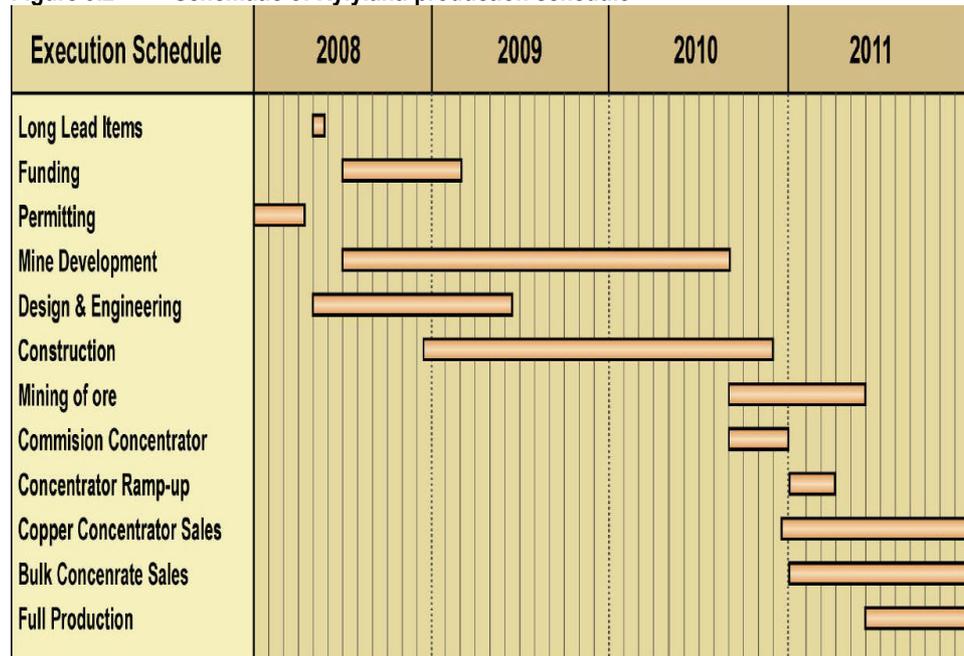
The timing of these key milestones for the project is as depicted in Table 9.1.

Table 9.1 Key project milestones

Project item	Date of completion
Completion of study optimisation and value engineering	March 2008
Front end engineering commences	April 2008
Order of grinding mills/long lead items	June 2008
Completion of off take agreements	October 2008
Start decline	December 2008
Plant commissioning	October 2010
Commence production	December 2010
Full production	April 2011

This is summarised graphically in Figure 9.2:

Figure 9.2 Schematic of Kylylahti production schedule



Snowden comment

For a base metals copper (plus other metals) concentrator the project schedule is very long. In Australia similar projects would be completed in a much shorter period. Significant differences with Australia are the climatic influences in Finland and the long lead time required for mills. Snowden thus believes that the project schedule is realistic and achievable, covering 36 months to full production.

Vulcan advises that the mill specifications have been completed and are being issued to procurement agents for finalisation of the bid documentation prior to tenders being called in mid-August 2008.

Snowden believes that the timing of the mine schedule and the overall plan for the EPCM, using a mixture of Australian and Finnish expertise, are all realistic, appropriate and achievable. The lead time for the mill is long but this allows Vulcan the time to fine tune the various parts of the EPCM package for optimal risk-free execution.

9.7.2 Project controls

Scheduling

Project scheduling services will include:

- preparation of the baseline master project schedule
- monitoring and reporting actual progress against baseline and recommend schedule recovery plans in the case of variance.
- maintaining and supporting schedule activities for engineering, procurement, construction and commissioning.
- providing schedule data and control requirements for all procurement packages
- co-ordinating with cost control to ascertain the cost impact of scope changes.

Cost control

Project cost control services will include:

- preparation of project control budget for direct and indirect costs
- measuring progress and monitoring and reporting actual performance against project and providing a recovery plan where appropriate.
- managing change and forecasting costs at completion for direct and indirect costs.
- producing periodic cost status reports based on man-hours and money.

Accounts payable services

Accounts payable services will include:

- managing the receipt of all accounts of all accounts payable, security deposits, performance guarantees and insurance provisions for suppliers and service suppliers.
- evaluating and certifying payment including the submission to KCO for approval to pay invoices.
- pay invoices from KCO's project account
- providing to KCO a cash drawdown assessment for the purpose of forward planning.
- hand over to KCO all suppliers securities outstanding after delivery of goods and services.

10 Tailings storage and paste fill plant design

10.1 Tailings storage facility

10.1.1 Overview of facility

The site plan for Kylylahti (Figure 7.2) shows the location of the tailings storage facility (TSF). The original TSF work in the DFS was carried out by Golder. Key points from the DFS TSF design were:

- the TSF is to be constructed in two stages, with a total capacity of approximately 1.48 Mt.
- approximately 66% of concentrator tailings would be delivered back underground as paste fill – the remainder would be discharged to the TSF.
- the TSF would be unlined – this decision was made as a consequence of testing which showed a low probability that material to be dumped on the TSF was acid generating.
- material at 65% solids would be pumped to the TSF
- a site has been identified which was flat, seismically stable and which required some tree clearing.
- the initial excavation would be 1.5 m below the ground surface in moraine
- the final dam wall height of 11 m was achieved with a predicted geotechnical factor of safety of about 1.7: in other words a low risk of tailings wall failure.

Since the DFS was delivered to Vulcan the environmental permit was granted, along with a large number of conditions. Key conditions pertaining to the TSF are:

- any water flows from the TSF must be treated such that the environmental load of the discharge water is minimal, and water treatment facilities must be operational before any discharge of water to Lake Polvijärvi.
- material used for the construction of the TSF must be non-acid forming and must not include significant levels of environmentally hazardous materials.
- the TSF is considered as normal waste landfill
- generation of dust from the side walls of the TSF must be prevented by keeping the TSF walls damp or by other means.
- the company responsible for the project must take care of the TSF rehabilitation and monitoring for at least 30 years after mine closure.

Of note is that there is no requirement for the TSF to be lined and that it is to be considered as normal waste landfill.

10.1.2 Material balance for TSF

The current life of mine production schedule for the operation sees the mining of some 6.9 Mt ore, generating 4.5 Mt of tailings. The life of mine backfill requirement is some 3.3 Mt of ore, leaving a net discharge to the TSF of about 1.15 Mt. The current design capacity of the TSF is 1.48 Mt, leaving an excess of capacity of 22% over current tailings tonnage predictions.

Snowden comment

It appears likely that the Kylylahti deposit will be bigger and longer-lived than the current DFS optimisation forecast. While the magnitude of the expansion is not

known, the 22% safety margin may potentially be under threat. Vulcan should ensure that expansion of the TSF, if required in the latter years of the (extended) mine life, can be achieved either through a third lift on the TSF (unlikely) or by an areal expansion of the TSF. While not yet requiring a definitive study, Vulcan should at least determine that one of these options is at least technically feasible.

10.1.3 Optimisation changes to the TSF plan

As part of the optimisation process Vulcan has made a number of improvements to the TSF plan. The first of these is to change the constitution of the material delivered to the TSF from 65% solids to 35% solids. This gives more flexibility with respect to the paste fill plant and means that the tailing thickener will be installed at the paste fill end. This reduces the risk with the filling of the TSF (35% solids being somewhat easier to manage than 65% solids) and lends more flexibility to the paste fill plant. It also means that much more mine water will be discharged to the TSF than previously planned. The other major change, in conjunction with the alteration of the solids density, is the decision to line the TSF at an additional capital cost of €1.59M. This decision has been taken as the risk of seepage from the TSF is much greater with the extra water being initially discharged; in fact Vulcan sees the TSF as a primary water storage area in order to manage inconsistent supply from underground and seasonal variations. The other advantage of running more water through the TSF is that capture of potentially damaging suspended solids becomes easier, thus reducing the load on the water treatment plant.

Snowden comment

The twin changes of changing the solids capacity to 35% and lining the TSF are seen by Snowden as sound risk mitigating factors with respect to the water balance and the successful operation of the paste fill plant (see below). However, Vulcan needs to be confident that when the TSF nears full capacity towards the end of the mine life (notwithstanding a potentially increased mine life and total production) that the additional liquids that are delivered to the TSF through the 35% solids will not cause overflow. Snowden suggests some preliminary modelling of this eventuality.

10.2 Paste fill plant

10.2.1 Introduction and sources of reference

This section presents the findings of a review of the current designs and reports defining the paste backfill system for the project.

The following studies and reports have been reviewed by Snowden:

- SRK, 2008. Kylylahti Copper Project Mining Feasibility Study, March 2008.
- Turnberry, 2008. Kylylahti Copper Project Paste fill Plant Front End Process Design, Turnberry RPA Pty Ltd, March 2008.
- Golder, 2007. Feasibility Study for a thickened tailings surface disposal and paste backfill system – Kylylahti Mine, October 2008.

Snowden's review addressed the following:

- Capacity of the tailings supply and backfill placement system to meet production schedules.
- Adequacy of the designed system to meet the project back fill requirements.
- Adequacy of the capital and operating cost estimates for the backfill supply system.

10.2.2 Mining and paste fill system

The underground mining of the Kylylahti deposit uses a combination of transverse and longitudinal mining methods depending on the geometry of the mining area. Paste fill is an integral part of these mining methods to provide stope stability and a means of disposing of surface tailings.

Mining production is generated from stopes that have typical dimensions of 20 m (w) by 25 m (l) by 30 m (h) containing approximately 48,000 t of ore each. The stopes are sequentially extracted from the bottom of the orebody upwards. Once each stope has been extracted, the void is filled with paste fill. Mining then continues adjacent to the filled void once the fill has set and cured for an appropriate period. Paste filled voids that are to be exposed by subsequent stopes will require cement as a binder to improve the strength of the fill so as to limit fall-off, dilution and instability.

The paste fill plant will be constructed on the surface, with reticulation through boreholes and pipelines to the appropriate stope requiring the backfill.

10.3 Production and back filling schedule

The following is an extract from the SRK Mining Feasibility Study (2008):

“The scheduling of the Paste Fill requirements is a direct relation to the generation of voids while mining. ... The initial Paste Fill requirement during the Semi Massive stope production period is between 140,000 m³ to 170,000 m³ per annum. Once the tonnage rate is increased during the production of the Disseminated Stopes the paste fill requirement reaches 210,000 m³ per annum. It has been assumed that there will be no lag of Paste Fill. If a void inventory is created (Paste Fill not able to fill as fast as mining progresses) the production schedule will be compromised as adjacent stopes to voids cannot be mined until they have been filled.

The stope cycle consists of drilling, firing, mucking and paste filling. To simplify the scheduling at this level of study an average production rate over the entire stope life was used. When a detailed mine schedule is compiled the intricacies of the Stope cycle interaction must be incorporated within the scheduling. Based on the simplification of the stoping sequence used in this study there is expected to be no major disconnect to what is attainable when the mine is in production”

The mining method is a bottom-up mining method with mining activities on top of previously filled voids. Thus the complete mined void will have to be filled except for the last stopes at the end of the mine life.

The concentrate produced by the Kylylahti process plant has a reasonably low upgrade factor with an average mass pull to concentrate of 34.4%. This ratio results in a solid mass in tailings of 65.6% of ore tonnes mined. At a production rate of 800,000 t per annum this equates to a maximum tailings mass of 525,000 t per annum.

The available tailings can produce up to 287,000 m³ per annum of paste fill. The total tailings produced are in surplus to paste fill requirements of approximately 14%. The surplus tailings are expected to be deposited on an appropriate tailings storage facility.

Snowden comment

Snowden's estimate of paste fill requirements differs from that stated in the SRK study text. The average void to be filled on an annual basis at an ore production rate of 800,000 t per annum, with an in-situ ore density of 3.15 t/m³, is 256,000 m³ per

annum. Snowden notes that the SRK cost model contains annual paste fill volumes of up to 260,000 m³ which does not match the text of the study but does closely aligns with Snowden's estimate.

Table 10.1 Production and paste fill profile

	Unit	Total	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019
Wallaby ore	kt	2,547	10	224	556	346	155	0	59	150	284	531	231
Wombat ore	kt	4,399	0	0	92	460	647	800	745	651	517	270	216
Milled tonnes	kt	6,946	10	224	648	806	802	800	804	802	802	801	447
Avg ore in-situ density	t/m ³	3.15											
Mined void	k.m ³	2,205	3	71	206	256	255	254	255	255	254	254	142
Avg Cu-Au concentrate mass pull	%	3.5											
Avg bulk concentrate mass pull	%	32.1											
Concentrate mass	kt	2,395	3	77	223	278	277	276	277	276	276	276	154
Tailings mass	kt	4,551	7	147	424	528	526	524	527	525	525	525	293
Tailings ratio (t solid / m ³ paste)	t/m ³	1.5											
Fill placed density (incl water 25%)	t/m ³	1.97											
Paste fill volume available	k.m ³	3,034	4	98	283	352	350	350	351	350	350	350	195
Paste fill requirement	kt	3,308	5	107	308	384	382	381	383	382	382	382	213
Mined void	k.m ³	2,205	3	71	206	256	255	254	255	255	254	254	142
Tailings surplus / shortfall (+/-)	k.m ³	829	1	27	77	96	96	96	96	96	96	96	53
Tailings surplus / shortfall (+/-)	kt	1,244	2	40	116	144	144	143	144	144	144	143	80
Cum tailings surplus / shortfall	k.m ³	38%	1	28	105	201	297	393	489	584	680	776	829

10.3.1 Tailings feed rate

The process plant is designed to process ore at a peak of 100 t per hour for 8,000 hour per annum at peak production rates. At the average mass pull, the feed rate of tailings for paste fill production is an average of 66 t per hour.

The rate of backfill placement depends on the availability of the backfill plant, the complex underground distribution system and mined voids ready for backfill placement. It is common practice for a stope to be filled to ¼ capacity, paused until the fill has partially set (2 to 5 days) and then filled to completion. (This limits the hydraulic pressure on the backfill barricade.) These factors would result in the paste fill plant operating for less than the 8,000 hours per annum of the processing plant.

Snowden comment

Turnberry has estimated the paste fill plant utilisation to be 70% (Snowden assumes this is a combination of availability and utilisation). Snowden's estimate of effective paste fill plant operating hours is 6,300 hours per annum (72% availability and utilisation) as described below (Table 10.2).

Table 10.2 Paste fill plant operation

Availability	90%
Days per year available	329
Utilisation	80%
Hours per day in operation	19.2
Hours per year in operation	6307
Average paste fill feed rate (tph)	60

Snowden's summary of the paste fill plant operation is as follows. The average required paste fill feed rate during operation of the plant is thus 60 t per hour. This implies that for approximately 1,700 hours per annum, the processing plant will be diverting tailings to a tailings storage facility. On average, during operation of the paste fill plant, the feed from the processing plant will be 66 t per hour (solids). This will be sufficient, on average, to meet the instantaneous need of the paste fill system with 10% extra capacity if required to meet surges in demand. The details of this arrangement can only be assessed with a more detailed mining and filling schedule.

10.4 Paste fill plant design

Unthickened flotation tailings (35% solids) will be pumped some 950 m from the process plant to the paste fill plant. When the paste fill plant is not operating, the tailings will be diverted to the tailings dam, the design of which has been recently modified to accommodate unthickened tailings deposition.

The tailings thickener will be located adjacent to the paste fill plant with a capacity to feed up to 71 tph along with a reasonable surge/peak allowance.

The tailings thickener will feed a filter feed tank with a live capacity of 255 m³. This will provide a surge capacity of between four to five hours in the paste fill placement process. Should delays longer than this occur, tailings will be diverted to the tailings dam.

Filtration of the thickened tailings will be with a vacuum disc filter. Further value engineering options study would provide final confirmation of the choice of a disc filter over a belt filter.

Snowden comment

Snowden views the paste fill plant design as appropriate for the specified requirements of the Kylylahti operation.

10.4.1 Paste fill reticulation

Turnberry has recommended an increase in the pipe size initially recommended by Golder.

Snowden comment

Snowden concurs that the 150 mm internal diameter Schedule 80 pipe recommended by Turnberry is an improvement with regards to ultimate system friction loss on the 100 mm recommended by Golder. The galvanised schedule 80 pipe specified for the majority of the reticulation system is appropriate. The use of lower pressure and lower cost HDPE piping for the last 200 m before the discharge point is appropriate.

The flow modelling and pressure profiling analysis completed by Turnberry shows that the paste fill can be delivered through the reticulation system by gravity.

Snowden agrees with the methodology used to analyse the system and the results expressed are reasonable.

10.4.2 Further work

Snowden notes that Turnberry recommends further paste fill strength analysis to determine likely cement consumptions rates to achieve the desired strength characteristics. This could affect the costings for the paste fill plant, but only in a positive sense. It will also lend more flexibility to the mining schedule.

11 Environmental and social impacts and permitting

11.1 Background

KCO has received Environmental Permit approval from the Eastern Finland Environmental Permit Authority (EFEPA) for development of the four square kilometre Kylylahti project in the Polvijärvi municipality of the Joensuu district (380 km northeast of Helsinki). The 11 year mine life project comprises four mining leases (181 ha) and four mineral claims (234 ha) on land owned by both the Polvijärvi municipality and private landholders.

The mine and plant are in close proximity to major transport infrastructure (road and rail) and can be supplied with Finnish grid electricity, potable scheme water and municipal sewage treatment.

11.2 Regulatory setting

Environmental policy in Finland is largely based on regulations adopted by the European Union. Responsibility for the environment is incorporated into the Constitution of Finland (Section 20). Environmental policy is governed, at the national level, by the Ministry of the Environment. At the regional level, environmental law is enforced by the Regional Environment Centre ('REC') and the municipality. Environmental permits are issued by the Environmental Permit Authorities and by the RECs and municipalities. The majority of decisions made by the environmental authorities are public and require a degree of stakeholder interaction to ensure information is available for stakeholders to make an informed decision. Finland is a signatory to the Aarhus Convention (access to information in environmental matters). Persons impacted by the permit, the municipality, the REC and registered environmental associations may appeal against environmental permits.

Pursuant to the Environmental Protection ('EP') Act (86/2000) environmental permits are required for certain operations listed in the Environmental Protection Decree ('EPD' - 169/2000). Environmental permits are operation specific, not operator specific. Developments listed in the EPD or that may have a significant impact on the environment (ore extraction > 550,000 tpa) require preparation of an Environmental Impact Assessment ('EIA'). Environmental impact assessment is governed by the Environmental Impact Assessment Act (468/1994) and the Decree on Environmental Impact Assessment (269/1999).

The extraction of metallic ores is regulated under the Mining Act (503/1965). The Act requires that an exploration permit be granted from the Ministry of Employment and the Economy. The exploration permit does not grant a right to exploit a mineral deposit. On-site activity can only commence after the preparation and submission of an Environmental Impact Assessment, in compliance with the Environmental Impact Assessment Act (468/1994) and the granting of an environmental permit under the Environmental Protection Act (86/2000) Section 28 (1). An EIA document for the project was submitted to regulatory authorities with the EIA process being successfully concluded on 4 October 2006. Subsequent to completion of the EIA process, KCO submitted a permit application and was granted an Environmental Permit on 19 December 2007 by EFEPA. This gave KCO a permit under Section 7 of the Water Act to construct discharge pipeline into Lake Polvijarvi and to pump groundwater as required to dewater the mine. Ongoing monitoring shall be regulated by the North Karelia Environment Centre ('NKEC') under the 73 permit conditions (see Section 11.6). The Permit conditions, including requirements for monitoring, reporting and preparation of a

Mine Closure Plan are summarised in Section 11.6.9. The Permit conditions address all of the areas of potential concern with the development and set appropriate environmental bonds for the development with the potential to reduce outstanding bond commitments by continuous rehabilitation.

The EFEP was required to issue the environmental permit (EP Act – Section 31) due to the activity being referred to in the EP Act Section 28 (2) and the requirement for a permit to abstract groundwater (> 3 Gigalitres per annum) and discharge treated water to Lake Polvijärvi under the Water Act (264/1961) Chapter 9 and 11.

Under the EP Act, the contamination of soil and groundwater is prohibited at all times. Government authorities must be informed if contamination enters the soil or groundwater. The Waste Act (1072/1993) requires that waste deposited to an on-site landfill (TSF) must not adversely impact upon the environment or human health.

A breach of permit conditions can result in strict liability for damages under the Compensation for Environmental Damage Act (737/1994) and may also result in action under the Penal Code (39A/1889), particularly for directors and officers of the corporation. In addition, environmental damage that results from activities within environmental permit conditions may also result in liability and if this may impact upon the company's solvency this is recommended to be noted in the company's financial statement. An investor/lender may become liable for environmental damage if they are effectively in charge of an operation. The purchaser of real estate may become secondarily liable for remediation costs if the purchaser knew or should have known about the contamination when acquiring the property. The seller is required to disclose all relevant information of activities that may have caused environmental harm. If no liable party can be found, the municipality may become liable for the remediation costs.

The REC can request further remediation even after a remediation programme has been agreed; however, damages for aesthetic damage are not possible.

11.3 Baseline data

11.3.1 Environment

The Polvijärvi area has an average annual rainfall of 751 mm and a dominant wind direction from the south (Finland Meteorological Institute, 2002).

The mine leases and mineral claims are on predominantly clayey, sand till (81%) and Carex peat (14%), indicative of the typical aapa mires occurring in the area. Water and nutrients drift to aapa mires as specific runoff from surrounding areas rather than as rainfall.

The Kylylampi pond, a natural surface water body, is located approximately 500 m south, southeast of the planned plant area. Lake Polvijärvi (catchment area approximately 27 km²) is located approximately 1 km south, southeast of the site. The Kylylahti mining district surface waters drain into Lake Polvijärvi via the Kirkkojoki River (from the north), the Kylylampi pond system (from the south, southeast) and the Sepänpuro creek (from the west). Lake Polvijärvi flows via the Viinijoki River to Lake Viinijärvi, located approximately 4.5 km southwest of the project area.

Lake Polvijärvi water quality has been monitored since 1988 (Savo-Karjala Environmental Studies Oy). The epilimnion layer has been very eutrophic ([P] ~ 0.08 mg/l; [O₂] ~ 7mg/l; pH ~ 6.8) and the hypolimnion is hypoxic ([P] ~ 1.5mg/l; [O₂] ~ 1.5 mg/l; pH ~ 6.5) during both winter and summer. The hypoxic conditions result in solution of heavy metals (chromium, cobalt and copper) from

lake sediments, although at concentrations less than maximum concentrations set by the Finland Ministry of Social Affairs and Health (Decision 961/2000). Similarly, the Kirkkojoki River is on average more eutrophic than the epilimnion in Lake Polvijärvi (partly due to the loading from the Polvijärvi municipality sewage treatment plant), while the Viinijoki River is equal to the epilimnion of Lake Polvijärvi and is classified as eutrophic or very eutrophic.

Surface waters in the Kylylahti mining district have naturally elevated concentrations of iron, aluminium and manganese¹ due to mineral deposits in the area, although concentrations are below the chemical parameters set by the Ministry of Social Affairs and Health. Stream sediments contain naturally elevated concentrations of cobalt, nickel and zinc.

The Räiskynkorpi 1 groundwater area (raw water source for the Polvijärvi municipality) is located approximately 800 m to the northeast of the project area and 1.8 km from the proposed tailings storage facility, but in a different catchment area to the project. Groundwater in the project area is typically shallow (1.5 to 3.0 m below ground level), acidic (pH 5.2 to 5.7) due to high humic content and contains elevated concentrations of copper, nickel and molybdenum, with nickel concentrations higher than the maximum concentrations set by the Ministry of Social Affairs and Health for potable water.

The North Karelian Tornimäki, a protected old growth forest, is located approximately 3.5 km northeast of the Kylylahti mining district. There are no nature reserves, as defined by the Nature Conservation Act, in the vicinity of the mining district.

Residential properties are located 500 m southwest and northeast of the planned plant area and 200 m west of the planned tailings storage facility. There are approximately 51 properties within a 1 km radius of the project area. Land use in the area is limited to small-scale forestry and unemployment in the region is high at approximately 17%. Polvijärvi is situated within a region with identified requirements for significant assistance for regional development and assisting projects that enhance the rate of employment within the private sector.

The close proximity of residential properties to the project area has required that surface crushing activities are restricted to 16 hours per day. The inherent design for the environmental conditions ensures that most equipment is installed indoors, reducing incidental noise impacts.

11.3.2 Consultation

The proposed Kylylahti mine is two kilometres from the town of Polvijärvi. Vulcan has undertaken a programme of community consultation with residents of Polvijärvi and the Polvijärvi Town Council. The first community consultation session, held in December 2005, resulted in Vulcan commissioning a Social Impact Study of the mine, by Jyväskylä University. The findings from this study were largely positive, indicating a significant benefit to the local community in terms of increased employment and indirect benefits. Overall, Jyväskylä University concluded that the project was 70% community positive.

Negative impacts were identified as increased traffic flows and the potential of groundwater contamination. The Social Impact Study predicted a 3% total increase in the traffic levels.

¹ Geological Survey of Finland, 2005

11.4 Environmental aspects of project development

A full project overview has been provided in preceding Sections of this document. Key aspects of the project which may impact on the environment are detailed and discussed below.

11.4.1 Fibrous minerals

Vulcan consulted with a local geological expert, Dr. Asko Kontinen, of the GTK (Finnish Geological Survey), regarding the potential for fibrous asbestiform minerals at Kylylahti. Kontinen's view was that there are minor zones of the orebody where fibrous minerals may occur, but that these are not generally within the main mining areas. Moreover, the potentially serious fibrous mineral anthophyllite has not been observed and is unlikely to occur due to the geological conditions. Vulcan subsequently submitted two samples of ultramafic to a testing laboratory and these revealed no asbestiform species. Visual examination by a geologist during core logging of 11 drillholes revealed some potentially 'fibrous looking' material, but this was neither confirmed nor eliminated by further testing. Snowden's overall conclusion is that the potential for fibrous minerals is low but has not yet been shown to be negligible.

11.4.2 Processing plant

- The planned reagents for the processing plant include calcium oxide (DG 8), potassium amyl xanthate (DG 4.2), copper sulphate pentahydrate (DG 9 – very toxic to aquatic environment), methylisobutyl carbinol frother (DG 3), and Aerophine 3418A promoter (sodium diisobutyl dithiophosphate).
- Sewage from the project will be pumped to the Polvijärvi municipality sewage treatment plant for treatment and disposal. All putrescible wastes will be disposed to the Polvijärvi municipality waste facility.
- Due to climatic considerations, no wash-down bays will be installed. The mining fleet will be washed underground.

11.4.3 TSF and mine waste rock

Disposed waste rock (sulphur content > 1%), flotation tailings and neutralisation sediments have been classified as non-hazardous waste. Crushed waste rock that does not contain significant metals or have acid generation potential (sulphur content < 1%) and is used in construction is not classified as waste.

Vulcan proposes to return tailings and potential acid forming waste rock (approximately 50,000 tpa) underground as cement paste or rock fill. Nominally, 500,000 tpa of tailings with stated 0.17 to 0.26%² sulphur concentration will be produced with a proposed 69% (345,000 tpa) used in paste fill and the balance (145,000 tpa) sent to the proposed 1,500,000 t (1,018,000 m³) capacity (TSF). The placement of tailings underground may act to minimise any residual generation of acidic solution from the surface tailings, although the tailings are indicatively non-acid forming³. The paste is proposed to be a blend of ore dressing pulp, pozzolanic material and potentially acid forming rock. The pozzolanic material may be cement or fly-ash.

The proposed 18 ha TSF is designed to hold up to three years of tailings (up to 150,000 t) before the placement of tailings underground can commence. Based on leaching test (EN 12457-3), leaching of nickel exceeds the parameters set for

² SNC-Lavalin Report 138270-0000-49RA-0003, Section 2.0

³ SNC-Lavalin Report 138270-0000-49RA-0003, Section 3.0

permanent waste (Finnish Government Decree 202/2006). Tailings are reported as being non-acid forming and the Environmental Permit does not require the TSF be lined, although a HDPE liner with bentonite bedding has been budgeted and committed to installation during construction of the TSF. The average hydraulic permeability of the ground moraine in the area designated for development of the TSF reportedly ranges from 1×10^{-8} to 5×10^{-10} m/s, with static groundwater levels approximately 2 to 3 metres below ground level (according to a GTK (Finnish Geological Survey) report). The environmental permit (see below for full details) requires 1 metre of compacted moraine with hydraulic permeability of 1×10^{-8} m/s at the base. Diversion channels will be constructed to prevent inundation of the tailings pond and surrounds in accordance with the Safety Technology Authority and the TSF is proposed to be sealed during rehabilitation works, although some ambiguity existed in the documentation (SNC 138270-EV-0008) on the basis of reducing harmful substance leachability by placement under water in the TSF (and neutralising with lime), or by sealed surface encapsulation; the two being mutually exclusive.

It is reported that the GTK has stated that tailings may be piled permanently in the underground areas or in the TSF, assuming the piling area includes intact rock depressions, dense peat and/or silt-clay sediment or fine till.

It is anticipated that 280,000 m³ of waste rock will be generated during development of the decline, with approximately 240,000 m³ assumed to be non-acid forming ('NAF') and approximately 40,000 m³ potentially acid forming (PAF). Mica schist is classified as PAF by the GTK, although this classification has been questioned. Waste rock generated during stoping will be mainly PAF and is proposed to be added to the paste fill circuit. Waste rock will be stored on compacted natural or built ground with collected rainwater diverted to the operations water circuit.

The GTK noted that waste rock may contain fibrous minerals (asbestos). An expert statement (Kontinen, 2006) is stated to have advised that although Kylylahti waste rock may possibly contain fibrous minerals (asbestos), it is unlikely that it actually does. It is stated that laboratory investigations of waste rock failed to show the presence of asbestos fibres.

11.4.4 Project water and tailings liquor

Approximately 0.6 ML of raw water will be required for operations per day (0.4 ML from the Vasarankangas talc ore old trial quarry, if required and the balance from the approximately 6.0 to 8.5 ML per day of mine dewatering water that will be generated. Mine dewatering water is anticipated to contain iron and nickel with a pH of 5.2 to 6.4 and will be pumped to the TSF, where process liquor alkalinity will act to neutralise the pH and precipitate metals. Tailings pond clarified water will be recirculated to the process plant at a rate of approximately 3.6 ML per day, with approximately 6.5 to 8.5 ML per day, depending upon plant requirements, diverted to Lake Polvijärvi. Waters entering Lake Polvijärvi have been reported to potentially contain an average nickel concentration of 0.01 to 0.04 mg/l (15-60 kg Ni/yr) during decline establishment and 0.05 to 0.5 mg/l (130-260 kg Ni/yr) during the operational phase⁴. This concentration is expected to increase as the mine proceeds deeper.

Total suspended solids ('TSS') discharged into Lake Polvijärvi are anticipated to average 1 to 5 mg/l during the mine development phase. This concentration is anticipated to remain similar during the operational phase, with peaks to 10 mg/l⁵, however, total environmental loading will increase due to increased discharge flows.

⁴ SNC Document No. 138270-EV-0008, p 66-67

⁵ SNC Document No. 138270-EV-0008, p67

With respect to nickel and suspended solids loadings, permit condition 8 (Section 11.6.2) requires that total nickel and suspended solids loadings do not exceed 200kg/yr and 7,000 kg/yr respectively. At the nominal operational phase discharge volume of 6,500-8,500 m³ per day⁶ and the nominal operational discharge concentrations for nickel and total suspended solids, the potential loadings can be assessed as follows:

Table 11.1 Potential loadings of nickel and Total Suspended Solids

Concentration	Discharge Volumes	
	6,500 m ³ per day 2,373 ML per year equivalent	8,500 m ³ per day 2,373 ML per year equivalent
Nickel		
0.05 mg/L	118 kg/year	155 kg/year
0.1 mg/L	237 kg/year	310 kg/year
Total Suspended Solids		
1 mg/L	2,372 kg/year	3,102 kg/year
5 mg/L	11,860 kg/year	15,510 kg/year

Interpolating from the calculated results above, it can be seen that nickel may breach permit annual loadings if either:

- >11,000 m³ per day (29% increase in discharge volume) is discharged at 0.05 mg/L Ni; or
- >5,500 m³ per day is discharged at 0.1 mg/L Ni,

or various permutations of the above.

It can further be seen that TSS may breach permit annual loadings if either:

- 6,500 m³ per day is discharges at > 2.95 mg/L TSS; or
- 8,500 m³ per day is discharged at > 2.25 mg/L TSS,

or various permutations of the above.

Tailings return liquors may contain elevated levels of cadmium, nickel, manganese, antimony and uranium. The parameters for harmless concentrations for aquatic environments⁷ are potentially exceeded by cobalt, molybdenum, selenium and zinc. However, the project benchmark has been set by the permit conditions (see Section 0).

Metal concentrations (copper, cobalt and nickel) in the top soil surrounding the ROM pad, crushing circuit and concentrate loading area are anticipated to increase to above naturally occurring concentrations. These are developed areas and will allow for collection of contaminated run-off water in the sedimentation ponds.

11.4.5 Dust generation

Dust generated during mine level development (explosions), ore movements, waste rock handling and during site earthworks and construction will largely be underground and/or will be restricted by the dominant weather conditions. Dust particles have been stated to be predominantly >30 µm and are stated to settle

⁶ SNC Document No. 138270-EV-0008, p59

mostly within 100-250 m from the emission source, although on occasions, particularly for particle sizes <10 µm this may increase to 400-500 m and potentially up to 1 km. Dust is proposed to be managed by soil and waste rock wetting and salting of roads.

11.4.6 Concentrate transport

Copper-gold concentrate will be transported by road to Vuonos (ore holding shed), then by rail to a smelter. Transport of the concentrate will be at the risk of KCO for the 400 km road and rail journey.

11.5 Rehabilitation costs

An overview of project capital and operating costs, including rehabilitation costs (detailed under Sustaining Capital), is provided in Section 12.1.8. Costs specific to the rehabilitation of the project area and surrounding land are described here.

Mine life is anticipated at 11 years followed by infrastructure demolition and site rehabilitation requiring 1 to 2 years. The site will then be reclaimed for forestry use. Sustaining Capital has been designated for 2020 and is budgeted within the mining budget (€156,500/year) with the following anticipated expenditure:

- TSF rehabilitation €748,000
- plant site rehabilitation €400,000
- mine infrastructure rehabilitation €125,000
- paste plant site rehabilitation €50,000
- waste dumps rehabilitation €350,000
- site roads, infrastructure rehabilitation €36,000
- vuonos railhead facility rehabilitation €12,000.

This equates to a total cost of €1.72 M. A rough estimate indicates that only 10% of costs are realised while the mine is operational, 50% to 60% of costs are realised in the two years after mine closure and the remaining 30% to 40% of costs are distributed over the several years post rehabilitation.

Snowden comment - costs

The largest cost is normally related to rehabilitation of the TSF, with costs of €5,000 to €25,000 per hectare, although this is largely influenced by cyclical fluctuations in the construction market and by the mine location. Demolition of a concentrator is estimated to be at least €100,000 to €300,000. Mine infrastructure demolition and rehabilitation is estimated at €100,000 to €400,000. The covering and stabilisation of waste rock piles may contribute €1,000 to €20,000 per hectare. Site rehabilitation is anticipated to cost €100,000 to €200,000.⁸

The requirement for additional environmental verification studies and closure permits may add €50,000 to €100,000 to costs. Aftercare and monitoring depends upon the impact of surface water and groundwater contamination, the impacts of water flows across contoured landforms and the general safety of the site. The annual cost can be €5,000 to €20,000 per year for some time after mine closure.

KCO have prepared a Preliminary Decommissioning and Closure Plan. This will be reviewed on commencement of the project. Decommissioning and closure costs will be estimated annually and reflected in internal project accounting.

⁸ Outokumpu Oyj, Mine Closure Handbook, Environmental Techniques for the Extractive Industries, Chapter 8 – reserve Procedure, Pledging A Guarantee and Cost Estimates in Mine Closure, 2008

11.6 Summary of permit conditions

The Eastern Finland Environmental Permitting Authority has granted a permit under the Environmental Protection Act (86/2000), a permit for abstraction of groundwater and disposal of treated process water to Lake Polvijärvi, under the Water Act (264/1961) for starting construction of the mine and concentrator. The permit specifies 73 conditions as follows:

11.6.1 Environmental contamination

- Notification to the North Karelia Environment Centre (NKEC) and Polvijärvi municipality environment authority before commencing construction.
- Appointment of a Project Liaison Officer
- Avoidance of unnecessary land clearing
- Prevention of natural surface water flows to the TSF and PAF waste rock Storage areas.
- Discharges to Lake Polvijärvi must be minimised

11.6.2 Emissions to water and soil

- All contaminated water flows (lay down area, TSF, underground mine, ROM pad, PAF waste rock storage area, constructed areas and process liquor) to be treated to reduce environmental loads prior to discharge. A water collection, recycling and treatment plan is to be submitted to the NKEC prior to discharge to Lake Polvijärvi.
- Monitoring of water treatment facility condition and operation must be in place.
- The discharge water quality must achieve
 - pH 6.0 to 8.5
 - nickel 0.5 mg/l and 200 kg annual load limit
 - soluble cadmium 0.01 mg/l
 - solid matter 10 mg/l and 7,000 kg annual load limit.
- All municipal waste water (ablutions) must be pumped to the Polvijärvi municipality waste water treatment facility or treated to achieve reduction of BOD7 (90%), total phosphorus (85%) and total nitrogen (40%).
- Rock and soil used for construction must be NAF and contain reduced concentrations of environmentally hazardous metals.
- The base for the ROM pad is to be constructed from low hydraulic permeability ($< 1 \times 10^{-8}$ m/s).

11.6.3 Emissions to air

- Crushing, screening and conveyors to be encapsulated and water used to minimise dust emissions to < 10 mg/Nm³ air.
- Dust from roads, waste rock storage and TSF to be minimised.
- Sulphur content of heavy fuel oils $< 1.0\%$ w/w and for light fuel oil $< 0.1\%$ w/w.

11.6.4 Noise and vibration

- Noise and vibration to nearest households is to be minimised.

- Blasting is to be undertaken at times notified to locals.
- Equivalent noise levels at nearest household for day time (07:00 to 22:00) < 55dB and for night time (22:00 to 07:00) < 50dB.
- Mobile crushing equipment (construction phase) should be located to minimise noise levels at the nearest household.
- A mobile crushing equipment (construction phase) noise model is to be submitted to NKEC and Polvijärvi municipality as soon as possible.
- Above-ground earth construction (within 500 m radius of nearest household) is limited to the day time (08:00 to 18:00). Underground working is not time limited.

11.6.5 Waste management

- Produced waste fraction as defined by the Waste Act includes:
 - (top)soil
 - non-acid forming (NAF) waste rock
 - potentially acid forming (PAF) waste rock
 - tailings.
- Material not classified as waste includes NAF rock or soil with limited metal concentrations, used for construction purposes.
- Tailings can be located to TSF or as backfill to underground mine. TSF is considered as normal waste landfill (Finland Cabinet landfill decision).
- Base and embankment walls of the TSF must equal or exceed intensity and thickness demands of 1 m of compacted moraine with hydraulic permeability < 1×10^{-8} m/s.
- Dust must be minimised from the TSF by keeping tailings damp.
- PAF can be temporarily stored in PAF storage area with compacted base moraine of hydraulic permeability < 1×10^{-8} m/s. PAF waste rock to be placed underground during decommissioning.
- NAF rock to be used in earth construction works. NAF waste rock to be stored on compacted moraine.
- Excavated top soil must be stored on-site for use in rehabilitation during decommissioning and closure activities.
- The company responsible for the project (Environmental Permit) must rehabilitate and monitor the TSF for a minimum period of thirty years after closure.
- Waste management must be guaranteed by placement of a bond equivalent to €100,000 at commencement, increasing by €25,000 annually to a sum of €400,000 after 12 years of operation.
- The bond, as bank guarantee or deposit, is in favour of the North Karelia Environment Centre. The bond is reduced by accomplished rehabilitation. €80,000 must be preserved for rehabilitation and on-going monitoring.
- The operation cannot be commenced before lodgement of the bond.
- Unused explosives and covers must be incinerated or disposed in a proper manner. Hazardous waste must be handled properly.

11.6.6 Chemicals and fuel

- Safety guides (Australian MSDS equivalent) for concentrator reagents must be lodged with NKEC three months prior to operations commencement.
- The MSDS must be updated regularly.
- Hazard assessment of reagents is to be submitted to NKEC within three years of operations commencement. The assessment must include
 - the use of reagents
 - emissions of reagents
 - details of reagents with potential environmental toxicity or sustainability impacts.
- Reagents, process fluids and process wastes must be contained to prevent potential hazards. Sumps to incorporate covers and cut-off valves.
- Fuels to be stored in doubled skinned tanks or in a manner to prevent contamination of soil or ground water. Lubricants and waste oil storage must be covered. Rainwater in bunded areas must be removed regularly.
- Fuel and chemical loading areas must be designed to prevent emission to soil.
- Chemical storage areas must be monitored regularly to reduce possible contamination of soil. Absorbent materials must be stored in close proximity to reagent storage areas.
- A reagent storage plan including segregation requirements must be submitted to NKEC 3 months prior to commencing operations.

11.6.7 Incidents and reporting

- Events that may impact upon soil or water must be notified to the NKEC and Polvijärvi municipality.
- A bioremediation pad must be provided for hydrocarbon contaminated soil.
- An emergency response plan must be prepared and annual environmental incidents reported to the authorities.
- The Permit holder must monitor operations, emissions and environmental impacts to surface waters (EC, Sb, Co, Cu, Cr, Mo, Zn, Fe and NO₃), groundwater, soil, flora, noise, vibration and the TSF area.
- The discharge point at Lake Polvijärvi must monitor across the vertical profile (Sb, Co, Cu, Cr, Mo, Zn, Fe and NO₃), water level and outflow volumes.
- Operations monitoring must report:
 - production tonnes, hours and waste rock crushing
 - consumption of fuel and reagents
 - discharged water quality, quantity and mass load to Lake Polvijärvi
 - quantity, quality and handling of produced wastes
 - environmental protection measures
 - reporting of incidents and near misses.
- A Lake Polvijärvi Impact study must be prepared after three years of operations.

- Analysis of groundwater must be undertaken monthly (pH, E_c, SO₄, NO₃, Fe, Cu and Ni).
- Waste rock from decline development must be monitored to ensure appropriate management.
- Tailings quality must be monitored quarterly and sulphur concentration analysed weekly. In year three of operations, the NP/AP-ratio of the tailings must be determined. Results and an expert's statement of results must be submitted to the NKEC, Polvijärvi municipality and permitting authority for Permit updating.
- Equivalent and peak noise levels must be determined at the nearest house during box-cut development, decline waste rock crushing and during earth works within 500 m of the nearest house. Noise levels must be monitored annually during operations and the results reported to the NKEC, Polvijärvi municipality and permitting authority.
- During decline development, structural observations for vibration impacts must be conducted at the nearest household with results submitted to the NKEC and Polvijärvi municipality.
- PM₁₀ levels are to be measured continuously during the construction phase and a study should also be undertaken in the second year of operations at the nearest house and from allocation within the site. Results should reflect emissions during both summer and winter and be submitted to the NKEC, Polvijärvi municipality and permitting authority for Permit updating.
- All measurements, sampling and assaying must be performed to CEN, ISO, SFS or other national/international standard or in a manner accepted by the NKEC.
- The annual environmental report must be provided to the NKEC and Polvijärvi municipality by the end of February.

11.6.8 Fishery monitoring

- Fishery monitoring must be done in a manner approved by the North Karelia TE-centre (fishery centre).

11.6.9 Decommissioning and closure

- The Decommissioning and Closure Plan ('DCP') must be regularly updated. The plan must include:
 - original landform contours and embankment wall levels
 - quantity of tailings material deposited
 - sulphur and NP/AP-ratio
 - material used for rehabilitation of the TSF
 - evaluation of TSF hydraulic permeability
 - updated water management plan
 - determination of fuels, reagent, containers and raw materials disposal
 - determination of quantity location and rehabilitation of contaminated soil and waste rock

- earthworks causing generation of dust, noise, surface/groundwater emissions and methods to reduce impacts.
- An updated DCP must be included in the application for Permit renewal or provide to the NKEC and Polvijärvi municipality twelve months prior to closure of the operations.
- The purity of removed or relocated soil must be ensured.

11.6.10 Terms for water management (Water Act)

- The quantity of pumped mine dewatering water must be measured and the flow-rates recorded.
- The location and depth of the Lake Polvijärvi discharge pipeline must be recorded and ensure that discharge will not disturb sediment.
- The company must pay compensation to the owner of the water area (Sotkuma water area owner) of €400.

Snowden comment

The 63 permit conditions appear to be comprehensive and cover every aspect of the mining and processing operation, together with the interfaces to the environment. The majority of the conditions relate to waste management, specifically the use and storage of potentially acid forming (PAF) material in the tailings storage facility (TSF) and the construction and operation of the TSF. This is appropriate given the potentially sulphur-rich nature of the waste material. Environmental bonds are required before TSF construction can begin. Snowden does not see any major risks associated with the permitting process and believes that all of the requisite approvals are in place to allow construction to commence.

11.7 Compliance with Equator Principles

11.7.1 Summary of the Equator Principles

The Equator Principles (www.equator-principles.com) were adopted by the Equator Principles Financial Institutions (‘EPFI’) in 2003 and revised in 2006. Most of the major financial institutions worldwide have now adopted the Equator Principles to guide their lending. Azure and its partners have requested that Snowden assess Vulcan’s compliance with the Equator Principles with respect to the Kylylahti project.

In summary, the Equator Principles are as follows:

1. The project which is proposed for financing should be categorised according to its expected social impacts and risks in accordance with the environmental and social screening criteria of the International Finance Corporation (‘IFC’), that is as either Category A (potential significant adverse social or environmental impact), Category B (potential limited adverse social or environmental impact) or Category C (minimal or no social or environmental impact).
2. For each project classified as either Category A or B the borrower needs to conduct a Social and Environmental assessment in order to assess the project against a list of potential environmental or social issues. The assessment needs to detail potential mitigation processes for any issues identified.
3. The project, which is in a High-Income OECD country (Finland), must feature an Assessment which is compliant with local or national laws and

which must meet or exceed the IFC Performance Standards and Environment, Health, and Safety (‘EHS’) guidelines.

4. For all Category A or Category B projects the borrower must compile an Action Plan which addresses the relevant findings of the Assessment as carried out in 2 above. The Action Plan must feature a Social and Environmental Management System which addresses the management of those risks and impacts in compliance with the applicable host country social and environmental laws.
5. For Category A and Category B projects not located in High-Income OECD countries, the borrower must consult with the project affected communities in a structured and culturally appropriate manner. The project must adequately incorporate or address the concerns of affected communities.
6. For those projects identified in 5 above, there must be a grievance mechanism whereby the affected communities can raise concerns about the proposed social and/or environmental performance of the project.
7. For Category A and Category B projects, where appropriate, an independent social or environmental expert not associated with the borrower should review the Assessment, the Action Plan and the consultation process and should assess Equator Principles compliance.
8. For Category A and Category B projects the borrower must incorporate a number of covenants in financing documentation. These include a commitment to comply with the relevant host country social and environmental laws, regulations and permits, to comply with the Action Plan, to provide periodic reports to the lenders and to decommission the facilities in accordance with an agreed plan.
9. For all Category A and some Category B projects the lenders will require the appointment of an independent environmental and/or social expert to verify the ongoing compliance with the Action Plan.
10. The lenders which have adopted the Equator Principles must report annually regarding their Equator Principles implementation process and experience.

The IFC Procedure for Environmental and Social Review of Projects (IFC, 1988) states that ‘A proposed project is classified as Category B if its potential adverse environmental impacts on human populations or environmentally important areas - including wetlands, forests, grasslands, and other natural habitats - are less adverse than those of Category A projects. These impacts are site-specific; few if any of them are irreversible; and in most cases mitigatory measures can be designed more readily than for Category A projects’. Finland is a High-Income OECD country with well established environmental regulations, in strict compliance with European Union legislation. The Kylylahti project is a Category B project under the criteria of the IFC.

11.7.2 Vulcan compliance with Equator Principles

The following tabulation summarises Snowden’s opinion of Vulcan’s compliance with the Equator Principles with respect to the Kylylahti project.

Table 11.2 Summary of Vulcan’s compliance with Equator Principles

Principle	Summary of principle	Comments
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Number		
1	Review and categorisation	Review (by Snowden according to IDC principles) makes it clear that Kylylahti is a Category B project.
2	Social and environmental assessment	Vulcan has carried out baseline environmental studies as part of the EIA application. A socio-economic assessment was also commissioned.
3	Applicable social and environmental standards	Finland is classified as a High-Income OECD country, and as such its regulatory, permitting and public comment process exceeds the IFC Performance Standards. Vulcan has had to comply with these standards in order to obtain its permits.
4	Action plan and management system	Vulcan has in place an Environmental Management Plan, a Water Management Plan and a Health and Safety Management Plan, which will be updated throughout the life of the project.
5	Consultation and disclosure	The Finnish EIA application process requires extensive community consultation and disclosure of Vulcan's plans. Stakeholder submissions were called for and public consultation (via a town hall meeting) was carried out.
6	Grievance mechanism	Not applicable to Kylylahti as the project is located in a High-Income OECD country.
7	Independent review	Not applicable
8	Covenants	The terms of the EIA require bonds to be posted. A decommissioning plan is in place as part of the conditions of the granting of permits. Other covenants will need to be signed with the lenders.
9	Independent Monitoring	To be organised by lending institutions as part of the terms of the loan, as required.
10	EPFI	Does not apply to Vulcan.

Snowden comment

Snowden believes that Vulcan has substantially complied with the Equator Principles with respect to the Kylylahti project. The process of permitting and the EIA procedure has ensured that Principles 2, 3 and 5 are satisfied. Vulcan has the appropriate Action Plans in place, as required by the terms of the permit. Azure and its partners will need to instigate covenants and any independent monitoring as part of any loan facility as it wishes.

12 Capital and operating cost estimates

12.1 Capital costs

12.1.1 Summary of costs

The capital cost estimate is based on a treatment rate of 800,000 tpa. The DFS was completed and capital costs generated; however, a decision has been made by Vulcan to export the bulk concentrate and not proceed with the hydrometallurgical facility. The capital costs generated in the DFS were then used for the optimisation study to re-calculate the revised capital cost for the bulk concentrate sale option. The capital expenditure for the concentrator and infrastructure has been segregated into five areas in accordance with how Vulcan scoped the initial feasibility study costs.

The areas considered in detail are as follows:

- Pre-production mining
- Concentrator and infrastructure
- Paste plant
- Tailings storage facility
- Owner's costs.

This has been summarised in Table 12.1, with all costs presented in Euros.

Table 12.1 Capital cost summary

Description	Concentrator and infrastructure	Paste plant	Tailings storage facility	Mining	Owner's costs
Capital Cost	51,101,437	5,732,023	2,437,623	16,968,554	8,563,128
EPCM	9,065,153	1,357,040	943,588	471,125	
%	17.7	23.7	38.7	2.8	
Sub-total	60,166,590	7,089,063	3,381,211	17,439,679	8,563,128
Contingency	5,505,631	995,507	460,003	314,084	5,954,324
%	9.2	14.0	13.6	1.8	
Total	65,672,221	8,084,570	3,841,214	17,753,763	14,517,452

Vulcan notes that the actual cost contingency for the concentrator is 14.6% when the cost escalation contingency (which was grouped in the capital expenditure summary under Owner's costs) is apportioned over the concentrator, paste plant, tailings storage facility and mine.

12.1.2 Pre-production mining costs

The mining capital cost estimate reported in SRK (2008) includes capitalising certain activities such as decline and ventilation installations, and includes equipment purchases by Vulcan and proportional allocation of leased equipment where applicable. Replacement capital is included. The capital expenditure estimate is shown summarised in Table 12.2. The capital cost of the paste fill plant is not included here, rather in the cost for surface processing facilities.

Table 12.2 Mine capital expenditure estimate

Item	Cost €M
Capitalised mining development	18.72
Capitalised mining maintenance	1.39
Mobilisation/demobilisation costs	0.80
Light vehicles mine	0.44
Mobile equipment	0.95
Owners equipment	4.06
Minor equipment	0.91
Fixed plant and equipment	0.66
Grand Total	27.94

Over the life of the mine, capital expenditure costs €4.02 /t, with an average cost of €6.75 /t from Year 1 to Year 7, when the majority of the capital expenditure is incurred.

Snowden comment

The capital estimate is based, in the main, on tendered contract rates and quoted prices. It is estimated within the cost model discussed in Section 12, and is therefore estimated to a high level of detail and from first principles.

Within the limitations of gross and spot checks of the cost model, no errors in logic or inputs were identified relevant to the capital cost estimate. Therefore, high confidence can be placed on the capital estimate.

12.1.3 Concentrator and infrastructure capital costs

Snowden comment - overview of plant costs

The capital cost estimated for an 800,000 tpa concentrator is what would be expected for such a concentrator. The accuracy quoted is +/-15%.

Development of concentrator costs

The revised capital cost estimate for the concentrator has been developed using an approach that involved the following steps:

- Development of a new design criteria based on matching new mine production volumes, metallurgical processing requirements and mine and surface tailings disposal strategies.
- Preparation of specifications for major equipment that was changed as part of the review.
- Development of capital estimate spreadsheets that reflect the original consideration but identify with the revised requirements.
- Budget pricing of selected major capital equipment.
- Development of a revised electrical distribution system and infrastructure.

The costs utilised for major capital equipment in the study have been selected as the values that will allow the project to select from a number of reputable vendors during project implementation.

It should be noted that during the project implementation phase all new equipment listed will be re-tendered and the selection made at the time based on further

development of the technical specifications as a result of detailed design, technical and or process compliance to the enhanced specification, commercial conditions and an acceptable delivery to the project.

A comparison with the original DFS figures shows that the total cost for the concentrator and infrastructure as the result of the review is €65,672,221 compared to the original DFS of estimate €62,995,753.

During the detailed design phase there would be an opportunity to employ value engineering principles and improve the design. Reducing the capital cost may be difficult.

Changes to equipment during optimisation

For the purpose of the study equipment listed below has been reviewed and where required budget pricing obtained using technical specifications developed from metallurgical testwork, revised design criteria, technical information and process data developed within the study and/or available at the time of the review.

The selection criteria for capital expenditure estimation purposes were process compliance, commercial conditions and an acceptable delivery to the project. Each of the major cost items (the primary crusher, the apron feeders, the dust collectors, the flotation cells, the pebble crusher, the thickeners, the concentrate capillary feeders, the lime storage and mixing area, the flocculant mixing area, the peristaltic pumps, the administration and workshop buildings, the concentrate storage buildings, the power reticulation and the ROM pad construction) was re-tendered as part of the optimisation process. The tenderers were major suppliers such as Sandvik, FL Smidth, Metso, Outotec, Krebs, Larox and local and regional Finnish construction engineers.

Concentrator and infrastructure equipment not re-tendered

Pricing for the following major process equipment utilised in the concentrator was not considered to change by any substantial amount or if at all from the DFS, using the yardstick of a $\pm 15\%$ change in value:

- Air compressors
- Air driers
- Air receivers
- Self cleaning magnets and rectifier
- Centrifugal slurry pumps
- Sump pumps
- Reagent transfer and dosing pumps
- Overhead cranes and hoists
- Water and solution pumps
- Instrumentation
- Agitators
- Valves
- Piping

Bulk materials

Where required, new quantities were estimated for the following basic materials. The unit rates for the various materials were those established in the DFS:

- Bulk earthworks

- Civil works
- Structural steel
- Plate work
- Piping
- Electrical reticulation
- Building architecture
- Conveyors
- General bulk materials for equipment installation

Installation labour

The installation labour cost for the various disciplines was the same value utilised in the original DFS as well as the appropriate labour consumption and efficiencies for tasks.

Freight

Freight was not assessed in detail but the allowances covered in the DFS are considered more than sufficient for the deliverables.

12.1.4 Paste plant

The estimated paste fill capital cost according to the Golders report is described in Table 12.3.

Table 12.3 Paste fill capital cost estimate

Item	Cost (€M)
Plant mechanical cost (thickening plant)	0.69
Plant mechanical cost (paste plant)	2.67
Plant direct costs	5.57
Indirect costs	1.94
Overall plant cost	10.87

Snowden comment

Snowden's view is that the estimated €10.87M is a reasonable estimate for the capital cost of the paste fill plant, reticulation and boreholes.

12.1.5 EPCM

The EPCM figure for the concentrator and infrastructure, paste plant and TSF has been maintained at the rates contained in the DFS. A further allowance has been allowed for EPCM for the items in the mining capital expenditure for the design, supply and installation of equipment. The total EPCM allowance within the capital expenditure estimate is €11,836,906, representing 14% of the total of direct construction costs.

Snowden comment

The proportion of the total capital cost which the EPCM cost represents is high for a project such as this, with 10% being more common. In response, Vulcan notes that EPCM as average over the areas of expenditure is 13.7%. The proportion allocated to the concentrator is 18.3% (being the majority of the work). The average for the remainder is 21%, albeit not weighted by work value. Vulcan notes that the average accepted values in the industry are in the range of 15% to 20% and to some extent vary according to the capital cost of the works, given that increases

and reductions in plant size do not necessarily generate a proportional increase or decrease in management.

12.1.6 Tailings storage facility

The initial capital cost of the TSF is €3.8M, which includes a 13.6% contingency on the cost estimate of €3.38M. This cost has increased from the DFS estimate in order to reflect the cost of lining the dam, a decision which is a risk mitigating measure and which is associated with the move towards the use of 35% solids in the discharge rather than the 65% allowed for in the DFS. Note that the second lift of the tailings dam, with an associated cost of €1.93M, has been allowed for in sustaining capital, along with €0.8M in closure costs.

Snowden comment

If, as is a medium to high possibility, the mine ends up being larger than forecast, Vulcan will probably need to make a second sustaining capital forecast for either putting another lift on the TSF (unlikely) or extending the TSF (more likely). As this will be around Year 7 of the project the additional capital is unlikely to have a significant effect on the project NPV.

12.1.7 Owner's costs

Owner's project costs have been allowed as detailed below.

Mining fees and tenement costs

Annual tenement fees, the cost of property acquisition, compensation and income from forestry valuations, allowances for possible property damage during construction and the cost of compensations to the municipality have been allowed. The total cost of these items during project development is €558,000 after allowances for agreed subsidies.

Water and sewerage connections

A total of €220,000 has been allowed to pay the municipality after agreed subsidies.

Power capacity reservation fee

A power capacity reservation fee of €225,600 has been allowed based on the average power requirement of the project of 8MW. This commits the power provider to make this level of power available and is an initial and one-off payment.

Construction and project insurances

The following insurances during the construction phase of the project have been allowed based on a quotation from insurance brokers. The total premium for the construction period is €694,750. The premium covers the following:

- Construction All-Risk Material Damage Insurance to the limit of €5M for underground installations (excluding the decline), above ground insurable works to €55M and €2M for above ground mobile plant and machinery.
- Construction All-Risk delay in start-up insurance for above-ground losses to a limit of €10M.
- Ocean Marine Cargo for a total value of shipments of €25M.
- Ocean Marine Cargo delay in start-up insurance to a limit of €10M.
- Construction Public Liability for a total insured contract value of €55M with a limit of indemnity of €10M.
- miscellaneous insurance including motor vehicles, travel insurances, expatriate personnel insurances, etc.

Capital spares

The spares listing from the DFS was reviewed and only allowances relating to capital spares and commissioning were retained. Sums for operating consumables, are allowed for in operating costs. There is no list of what these spares are.

Owner's project management costs

All costs associated with project employees of the Owner, including salaries and allowances, all employment costs, travel, accommodation and expenses, vehicle hire, etc. have been allowed to a total of €3,187,879. This covers the entire project development period from approval until the completion of commissioning.

Owner's pre-production costs

All costs of the Owner's operations personnel, including salaries and allowances, recruitment costs, administration costs, relocation expenses and all associated costs of employing personnel from their time of recruitment until the end of commissioning of the process plant are allowed. Included in this allowance are all permanent employees of KCO in Finland during this period. The cost is €3,676,899.

Construction cost contingency

Construction costs have been estimated to an accuracy of +/- 15%. In the course of estimating, and considering the extent of engineering that has been completed, it can be estimated that costs will be incurred for items that have not been specifically identified, but will be incurred. The assessment of the quantum of this contingency is carried out as part of the estimating process, and varies according to the area of the project being estimated. It is also referred to as an accuracy allowance. In addition there is a contingency allowance attributed to potential cost increases. The total allowance in the capital cost estimate is €13,229,548, representing 13.7% of the total of direct, indirect and EPCM costs.

Snowden comment

Snowden's experience of estimating contingencies for engineering construction projects using probabilistic methods is that contingencies vary between 10% and 20%, with a mean around 15%. Thus the Kylylahti contingency is well within the range of expectations for this stage of the project.

12.1.8 Sustaining capital

In addition to the capital expenditure during the project development period, there are ongoing costs during the life of the mine which, for accounting purposes, are treated as capital and not as operating expenses. The major costs in this category are related to mining development and replacement of capital items.

A total allowance over the LOM for sustaining capital is €15.763M, to be incurred between 2011 and 2020. Rehabilitation items are deemed to be carried out during 2020, being the year after the last full year of production.

The items in this category are:

- Concentrator plant replacement €0.510M (2016)
- Paste plant replacement €0.169M (2016)
- TSF wall lift and associated works €1.932M (2016)
- Mine development €7.633M (2011 – 2015)
- Mine maintenance during development €0.634M (2011 – 2015)
- Mine light vehicles €0.189M (2011 – 2018)
- Mine mobile equipment €0.473M (2014 – 2015)

- Ventilation fans €1.852M (2011)
- Mine minor equipment €0.546M (2011 – 2019)
- Mine fixed plant and equipment €0.101M (2011)
- TSF rehabilitation €0.748M (2020)
- Plant site rehabilitation €0.400M (2020)
- Paste Plant site rehabilitation €0.050M (2020)
- Mine infrastructure rehabilitation €0.125M (2020)
- Waste dumps rehabilitation €0.350M (2020)
- Site roads, Infrastructure rehabilitation €0.036M (2020)
- Vuonos facility rehabilitation €0.012M (2020)

12.1.9 Contracts and insurance

In the current climate Vulcan contracts going forward will need to reflect the following as they could have an impact on the final capital cost.

Process guarantees

The process guarantee should include:

- The level of insurance the engineer can arrange and the recovery of that insurance cost
- the ability of the engineer to support any guarantees from new equipment vendors for equipment supplied
- The process guarantees and results are limited to the fact that the ore must have the same characteristics as the BFS metallurgical testwork sample ore
- The quality and accuracy of the data and information provided by Vulcan
- The representivity of the DFS study test results.

The limit and extent of any guarantee will be to the extent of the engineer's fees. Vulcan can assist in obtaining project insurance above this level with the insurer to meet the specific needs of any project financier and the engineer. In addition to engineers and contractors insurances Vulcan intends to take out principal arranged insurance to increase cover in an optimised construction insurance package.

Cost overruns

It is common these days to incentivise the contract with 'pain'/'gain' incentives covering schedule and capital cost.

Performance delays

If the engineer is awarded the EPCM contract they must give a completion guarantee to achieve 'practical completion' after developing a project schedule, which clearly identifies the project critical path and the limitations of their responsibilities. Practical completion will be as defined in the tender documents.

Liquidated damages

Liquidated damages (also referred to as liquidated and ascertained damages) are damages in which the amount recoverable in the event of a specified breach (e.g., late performance) is agreed at the date of a contract. In such circumstances a liquidated damages provision will be included in the contract. When damages are not predetermined or assessed in advance then the amount recoverable is said to be 'at large' (to be agreed or determined by a court or tribunal in the event of breach).

If the project is late this will cost Vulcan increased interest charges.

Vendor warranties

The engineer will act as a part of the EPCM project where new equipment vendors independently guarantee the plant equipment with regard to capacity and duty, in addition to it being fit for intended purpose. Vendors will supply commissioning services and 12 month warranty for the equipment they supply. Extended warranties can also be obtained.

Recommended spares

The contract and equipment enquiries should include request for recommended spares. In addition Vulcan should conduct an independent critical spares review.

Snowden comment

At the time of writing Vulcan is in the process of negotiating the major construction and engineering contracts for the Kylylahti project. It is therefore not timely to assess these contracts in this report as they have not been finalised. However, Snowden believes that Vulcan has made suitable contingency provisions in the EPCM and other areas (see Section 12.1) to cover any contractual or other delays. This aspect is also discussed in the estimate of the cost overrun (Section 13).

12.1.10 Accuracy of capital cost estimate

The capital cost estimate has been maintained to an accuracy of $\pm 15\%$. The major pieces of equipment will be more accurate than this, and areas which have a high labour component, civil works for example, will be less accurate.

12.2 Operating costs

12.2.1 Mine costs

The project has been costed on the basis of utilising local contractors for all waste development, including ground support, and all loading and hauling of waste and ore. Vulcan employees would undertake ore development (excluding mucking), ground support and drill and blast activities.

The cost estimate was compiled by SRK and was supplied to Snowden in the spreadsheet 'SRK Kylylahti Cost Model (Yearly) Iteration 15.3 excel 2003.xls'. The project mining costs are summarised in Table 12.4, with the unit cost representing cost per tonne of ore delivered to the process plant.

Table 12.4 Summary of mining operating and capital costs

Item	Cost €M	Unit cost €/t
Operating cost	145.0	20.88
Capital cost	27.9	4.02
Total cost	172.9	24.90

The capital cost includes capitalising certain activities such as decline and ventilation installations. It includes equipment purchases and proportional allocation of equipment use where applicable.

Snowden comment

SRK's cost model is comprehensive and works from first principles at a very high level of detail. In many instances it is populated with quoted prices for consumables and services.

A complete audit of the cost model was not undertaken; however, gross and spot checks of inputs and calculation logic were completed.

The computational logic underpinning the cost model was reviewed and found to be sound, though some errors in inputs were found.

Further development of the cost model reported in the feasibility study has been reported in a modified model (SRK Kylylahti Cost Model (Yearly) Iteration 15.8 excel 2003.xls) which results in a life of mine operating cost of €20.68 /t.

Although the cost model includes provision for cablebolting development intersections, there is no inclusion for cablebolting stope walls. The project geotechnical assessment indicates that cablebolting may be required in some instances to ensure high confidence in stope wall stability. If this cablebolting is to be undertaken, resources will need to be allocated, and depending on the extent of the requirement, a significant cost increase could result. An accurate appreciation of the potential cost impact of this activity can only be gained by updating the mine design, schedule and cost estimate incorporating a planned level of this activity.

12.2.2 Paste fill plant

The operating costs of the backfill system comprise three main areas:

- Paste fill plant operation.
- Underground reticulation and instrumentation.
- Backfill placement and bulkhead construction.

Paste fill plant operation

The paste fill plant operating costs estimated by Turnberry are tabled below (Table 12.5).

Table 12.5 Paste fill plant operating costs

Unit operating costs over life of mine	€/t backfill	€/m ³ backfill	%
Power cost	0.46	0.67	6.1
Cement cost (delivered to site)	3.78	5.56	50.0
Flocculant cost	0.04	0.07	0.6
Filter bag changes	0.28	0.41	3.7
Disc filter sector replacements	0.15	0.23	2.0
Lubricants	0.02	0.04	0.3
Spares	0.61	0.90	8.1
Lab/testwork/consultant costs	0.38	0.56	5.0
Surface Labour costs	1.83	2.68	24.2
Surface unit operating cost	7.56	11.11	100.0

Snowden comment

Snowden views the operating cost as reasonable, but slightly below those experienced in Australia. The lower cost is identified and is justified in the Turnberry report by lower Finnish unit labour, power and cement costs. Snowden notes that this updated estimate by Turnberry is 20% higher than in SRK's study cost model.

Underground reticulation

The SRK estimate for reticulation operating costs is €0.45 / m³ consisting of:

- Replacement of 10% of the installed schedule 80 pipe metres per year.
- Replacement of all 96 installed elbows every year.
- 18 m of HDPE pipe per stope.
- Two reticulation operators and an electrician and boilermaker as required.

Snowden comment - reticulation

Snowden views the reticulation operating cost to be reasonable and notes that the labour component of the reticulation operating cost is 77% of the total. The other reticulation operating costs appear to be reasonable and even if substantially underestimated will not be a significant increase in actual paste fill operating costs.

Snowden comment – backfill placement and barricades

Snowden views the SRK estimate of €4,148 per barricade to be reasonable. The estimated cost for construction of backfill barricades with bricks and mortar could be rationalised if the use of shotcrete proves to be viable.

12.2.3 Plant costs

Manning levels

The manning levels assumed for the concentrator are detailed in Table 12.6.

Remuneration for all salary staff has been based on figures supplied by the Vulcan management group based in Finland as being representative of allowances to secure the necessary personnel.

Remuneration for all wages staff has been based on utilising an hourly rate, applying values for shift allowances, etc and then applying this hourly rate into a shift pattern to provide for operation of the facility on a continual basis.

Finnish regulations require that as far as practicable personnel do not work more than 1,689 hours per calendar year. To meet this requirement a shift rotation utilising five panels has been considered in the operating costs.

In the pay system labour agreements will classify all current jobs (TKO) according to how demanding those jobs are and a classification number is provided (1-9).

The employees are ranked according to their skills in relation to different (HKO) classes (3-20% on top of the base wage). Together with the basis minimum rate, these components form the minimum salaries for the jobs.

The manning level appears to be more than adequate for the process plant. The skill level and operating in Finland is very high compared to say Africa or South America.

Table 12.6 Process plant manning levels

Position	Per crew	Number
Management and Technical		
Process Superintendent		1
Plant Metallurgist		1
Graduate Metallurgist		1
Production Foreman		1
Maintenance Superintendent		1
Electrical Foreman		1
Mechanical Foreman		1
Maintenance Planner		1
Shift Operators		
Shift Supervisor	1	5
ROM Pad Loader Operator	1	5
Crusher Operator	1	5
Control Room Operator	1	5
Mill/Flotation Operator	1	5
Thickener/Filtration Operator	1	5
Concentrate Loader Operator	1	5
Day Crew		
Fitter/Welder		3
Electrician/Installer		3
Total		49

Plant operating cost summary

The plant operating costs are summarised in Table 12.7.

Table 12.7 Summary of key operating cost items for the plant

Area	Total cost €/ yr	Unit cost €/ t	Fixed component (€)	Variable component (€)
Personnel	2,873,373	3.59	2,873,373	
Laboratory	667,644	0.83	667,644	
Maintenance	865,101	1.08		1.08
Reagents	1,116,736	1.40		1.40
Consumables	776,915	0.97		0.97
Power (fixed)	456,331	2.42	456,331	
Power (variable)	1,483,030			1.85
Utilities and fuel	430,286	0.54	430,286	
Mobile equipment	445,034	0.56	445,034	
Total	9,114,450	11.39	4,872,668	5.30

Snowden comment

The total power cost per tonne of ore is consistent and typical for a concentrator of this throughput. A higher cost should be used for the ramp-up period until full design throughput is achieved.

Power consumed has been calculated from the drive list, in most cases by multiplying 80% of the installed power for the drive by the design operating utilisation. The labour cost is relatively high but typical of the Nordic countries. Over time the manning levels may be reduced. Power is a major cost in such projects and the low power cost in Finland is a distinct advantage over diesel fired power supply. In some instances (e.g. mill drives) a higher utilisation of the installed power has been used, while in other cases to reflect operational characteristics (e.g. sump pumps and gantry cranes) a much lower (< 10%) utilisation has been used.

A unit power cost of €61/MW/h has been used. This comprises an energy portion and a transmission portion. The energy portion is purchased from the Nordic Electricity Market (‘Nordpool’) and the transmission cost is based on delivery at 110kV from the local power infrastructure operator PKS.

The reagent cost is also a major cost and these are based on budget pricing. PAX and copper sulphate represent 47% of the reagent cost.

Snowden does not expect a significant escalation in operating costs for the following reasons:

- the expected power cost is stable
- labour costs are stable given the high unemployment levels in Finland
- inflation is under control in country
- no fly in fly out or accommodation is required.

Laboratory

A sample preparation facility will be located at Kylylahti and Vulcan intends to use a commercial laboratory (Labtium) for plant and mine assays.

Snowden comment

The key issue with new projects with respect to achieving design operating costs is to ramp up quickly and achieve design throughput. The fixed cost component impacts strongly on the operating cost if design throughput is not achieved. Snowden is very comfortable with the mill flexibility and flotation capacity, and expects that the detailed design will also support these objectives.

12.2.4 Administration and Overheads

The operating cost estimates for administration and overheads have been provided for as in Table 12.8.

Table 12.8 Operating Cost Summary – Administration and Overheads

Area	Total Cost €/yr	Unit Cost €/t
Personnel	730,000	0.91
Insurances	647,457	0.81
Administration Costs	1,119,394	1.40
Total	2,496,851	3.12

The operating cost estimates have been derived using a data and information derived from sources including vendor budget prices and supplier quotes.

Personnel

Remuneration for all salary staff has been based on figures supplied by the Project management group based in Finland as being representative of allowances to secure the necessary personnel.

Personnel considered in the administration costs are as shown in Table 12.9.

Table 12.9 Administration Staff

Position	Number
General Manager	1
Administration Manager	1
Environmental Scientist	1
Purchasing Officer	1
Accountant	1
Accounts Clerk	1
Payroll Clerk	1
Administration and Secretarial Assistant	1
Receptionist	1
Stores Personnel	1
Total	10

Insurances

Provisions made for various forms of insurance are as show in Table 12.10

Table 12.10 Insurances

Item	Annual Cost €
Industrial Special Risks – Material Damage	260,000
Business Interruption	225,000
Marine Cargo	43,253
Public Liability	69,204
Miscellaneous Insurances (vehicles, travel, etc.)	50,000

Administration Costs and Allowances

The costs outlined below in Table 12.11 cover allowances and charges that provide the administration function for the Project. Where services or provisions for costs cover both the mining and processing functions these costs are combined as an overhead and covered in the following area.

As such the combined mining operating cost, concentrator operating costs and administration operating costs represent the on-site costs for the Project up to the point of concentrate sales nominated.

Table 12.11 Administration Costs

Item	Unit	Allowance (€)
Administration Vehicles (GM)	Month	2,000
Telecommunications	Month	3,000
Electric Power Transfer fee	Month	1,473
Stationary	Month	1,050
Postage Couriers and Light Freight	Month	1,250
Computers and Software	Month	14,000
First Aid Costs	Month	5,000
Entertainment	Month	2,500
Safety Clothing	Annually	22,400
Meal Subsidy (Staff only)	Month	2,289
Training	Annually	25,000
Travel and Accommodation		
- International Airfares	10 Off	8,000
- Domestic Airfares	15 Off	500
- Staff Travel Accommodation	150 Off	150
- Staff Travel Expense	210 Off	100
Recruiting/Relocation	Annually	50,000
Environmental Permit Costs	Annually	50,000
Tenement Fees	Annually	3,000
Mining Fee	Annually	50,000
Community Support	Annually	50,000
Security Services	Month	2,000
Contract cleaning & laundry	Month	6,250
Canteen Ancillaries	Month	2,000
Snow Removal	Month (7 Months only)	3,750
Rubbish and Waste Disposal	Month	1,000
Consultants	Month	10,000
Miscellaneous	Month	15,000

Snowden comment

The admin cost and allowances are reasonable for this phase of the project.

12.2.5 Concentrate Transport

The average annual cost of concentrate transport as detailed below is €3,953,322 at the rate of €4.94 per tonne of ore.

Copper-Gold Concentrate

Transport costs of copper-gold concentrate are based on product being sold to the New Boliden Harjavalta smelter. The point of sale is FIS Harjavalta smelter site.

The transport task includes trucking to the covered storage facility at the Vuonos rail siding, the unloading and stacking of the concentrate in storage, the re-loading into trains comprising approximately 25 bulk wagons, each with a capacity of 50 tonnes, and the transport by rail to the New Boliden rail siding at Harjavalta.

The costs are quoted as shown in Table 12.12

Table 12.12 Copper-Gold concentrate transport

Route	Unit Cost (€ /wet tonne)
Road (Kylylahti to Vuonos)	2.45
Road to Rail Transfer (Vuonos)	0.57
Train (Vuonos to Harjavalta)	15.00
Total	18.02

The average annual cost for the transport of 27,125 tonnes (dry) of copper-gold concentrate at a moisture content of 8% is €527,896.

Bulk Concentrate

Transport costs of bulk concentrate are based on product being transferred to the Talvivaara site facility. The point of sale is FIS Talvivaara mine site.

The transport task comprises the transport by road in truck and trailer configuration (40 tonnes per load) from site at Kylylahti to the receival facility at Talvivaara and equates to €12.80 /wet tonne.

The average annual cost for the transport of 247,716 tonnes (dry) of bulk sulphide concentrate at a moisture content of 8% is €3,424,427.

Snowden comment

The transport cost seems reasonable from a trucking/railing cost per t point of view.

13 Estimate of potential cost overrun

13.1 Introduction

Snowden was requested to provide an assessment on the likelihood and amount of cost overruns on the project. Due to the time frame Snowden elected only to focus on the potential cost overruns related to the accuracy and uncertainty in the base cost estimate(s). This uncertainty was associated with the basis or source of information on which the various individual cost estimates were made. Snowden has used the cost accuracy ranges provided by Vulcan as a basis for the derivation of statistical distributions.

Issues that were **not** addressed were:

- The potential of **schedule risks due to delays**. Please note that as recorded in Snowden's qualitative risk assessment results (Table 14.5) the two key risks in the project phase (EPCM) are the timely procurements of long lead items and personnel shortages in the construction industry. Both of these risks are rated medium to high in the qualitative risk assessment, which indicate that they could have significant impact on the successful delivery of the project. Both of these components will have an impact on the total cost incurred due to the delayed/extended construction that their realisation would imply. These aspects, although having been qualitatively identified and flagged, have not been assessed quantitatively for their likely effect.
- The uncertain effect of **inflationary escalation** between the time of estimation and the actual dates of procurement and acquisition has not been taken into account, and could lead to higher cost if compared to the original base estimates. It is of note that an assessment of the effects of inflation was outside of Snowden's scope.

The likely effects of these two considerations should be added to the "contingency" value derived through this analysis.

13.2 Methodology

Snowden has focused on quantifying the uncertainty in the base cost estimates by using a modified version of the contractor's cost accuracy indicators.

The capital expenditure for the concentrator and infrastructure has been segregated into five areas in accordance with Vulcan's subdivision of the initial feasibility study costs. The areas considered in detail are as follows:

- pre-production mining
- concentrator and infrastructure
- paste plant
- tailings storage facility
- owner's costs.

In this report the terms "deterministic" and "probabilistic" are used frequently. In a deterministic system no randomness is involved in the development of future states of the system. Deterministic models thus always generate the same output for a given starting condition. A probabilistic system, however, is a system in which randomness is involved in the development of future states of the system and which produces a range of outputs for a given starting condition.

The spreadsheets supplied by Vulcan show the break-up of the calculation of each of the deterministic cost estimates. The deterministic values themselves do not include estimation uncertainty. This uncertainty was introduced as a “contingency” component added to the deterministic total.

The deterministic estimates are summarised in Table 13.1. All costs are in €M.

Table 13.1 Deterministic cost estimates

Item	Cost estimate (including contingency)	% of total	Cost estimate (excluding contingency)
Pre-production mining	28.64	24%	28.64
Concentrator and infrastructure	45.17 (27.15 + 8.01)	37%	45.17 (27.15 + 18.01)
Paste plant	5.01	4%	5.01
Tailings storage facility	2.44	2%	2.44
Indirect	16.67	14%	16.67
Owner's costs	10.207	8%	10.207
Contingency	13.23	11%	0
Total	121.54		108.31

The six largest component groups of the capital cost estimate (each amounting to around €10M) are as follows:

- Ore Storage and grinding facilities (grouped under “Concentrator”) – €9.36M (7.7% of total).
- Capitalised mining development (grouped under “Mining”) – €11.09M (9.1% of total).
- Capitalised operating costs (grouped under “Mining”) – €11.67M (9.6% of total).
- Electrical power supply (grouped under “Infrastructure”) – €9.26M (7.6% of total).
- EPCM (grouped under “Directs”) – €11.84M (9.74% of total).
- Contingency (grouped under “Owner’s Costs”) – €13.23M (10.88% of total).

Snowden has removed the “contingency” components out of the total values for each of the five areas in order to prevent double accounting of the estimation risk. Snowden has assumed that the contingency component was included to make provision for the estimation accuracy aspect only, and that it was not used as a proxy for unforeseen or unexpected cost items.

In a typical cost estimate the basis for the cost estimate of individual cost items will have varying levels of uncertainty. Typically the level of uncertainty will decrease as the availability of information increases. Moreover, the uncertainty will relate to the level of confidence in estimates of both unit costs as well as quantities.

The relevant uncertainties related to cost items were derived from the information sources used in the base estimation of each of these items or item groups. Snowden assumes that cost estimators used a number of sources and approaches as the basis of their cost estimates. These range from sophisticated and relatively high

confidence sources such as “fixed and firm” estimates to “in-house” estimates and simple factorisation at the other extreme.

Table 13.2 is a summary of Vulcan’s cost estimation approaches applied to the various capex items.

Table 13.2 Vulcan’s information sources and accuracy ranges of estimate

Source for estimate	Variation down	Variation up
Budget : -0.05 / 0.05	-5%	5%
Budget : -0.1 / 0.1	-10%	10%
Budget : -0.05 / 0.15	-5%	15%
Budget : -0.15 / 0.15	-15%	15%
Budget : -0.05 / 0.1	-5%	10%
Budget : -0.1 / 0.1	-10%	10%
Budget : -0.05 / 0.1	-5%	10%
Budget : -0.1 / 0.15	-10%	15%
Design, Vendor Pricing : -0.15 / 0.15	-15%	15%
Design, MTO, Unit Pricing : -0.1 / 0.2	-10%	20%
Design, MTO, Unit Pricing : -0.1 / 0.15	-10%	15%
Estimate : -0.25 / 0.25	-25%	25%
Estimated Quantity off Vendor Fax Price : -0.15 / 0.15	-15%	15%
Fixed and firm : -0.05 / 0.05	-5%	5%
In-house Equipment Estimate : -0.15 / 0.2	-15%	20%
In-house Estimate : -0.15 / 0.2	-15%	20%
In-house Estimate : -0.25 / 0.25	-25%	25%
In-house Estimate : -0.15 / 0.15	-15%	15%
In-house Estimate : -0.05 / 0.15	-5%	15%
In-house Estimate, costs quoted : -0.1 / 0.2	-10%	20%
In-house Estimate, Vendor supplied : -0.1 / 0.2	-10%	20%
In-house Pricing : -0.15 / 0.2	-15%	20%
List Price : -0.1 / 0.1	-10%	10%
Project Pricing, Estimated Quantity : -0.15 / 0.2	-15%	20%
Quantity Calculated, Fee at Tariff : -0.05 / 0.15	-5%	15%
Quantity In-house Estimate, rates fax : -0.1 / 0.2	-10%	20%
Quantity In-house Estimate, rates fax : -0.25 / 0.25	-25%	25%
Unit Pricing, Vendor Supplied : -0.15 / 0.15	-15%	15%
Unit Pricing, Vendor Supplied, Factor internals : -0.2 / 0.2	-20%	20%
Vendor Design, MTO, Unit Pricing : -0.15 / 0.15	-15%	15%
Vendor Quantity of Vendor list Price : -0.15 / 0.2	-15%	20%
Vendor Quotes/%Factor : -0.1 / 0.2	-10%	20%

Snowden comment

Snowden believes that the relative confidence and accuracy levels associated with the sources used and quoted by the cost estimators can be used in assessing the relative uncertainties associated with various types of cost estimates and their assumptions. Snowden does not, however, concur with the actual values used. As can be seen from the table above, the estimators had introduced their accuracies in a symmetric or close to symmetric fashion; in other words, factoring in a similar upside and downside. Snowden's view, backed up by experience, is that vast majority of engineering projects end up being more expensive than the initial estimate. By examining completed projects and comparing the cost estimates with the actual recorded expenditure, history has shown that the likelihood of actual expenditure being lower than the initial estimate is much less than the inevitability of exceeding the original estimate.

13.2.1 Introduction of uncertainty

Statistical distributions, based on modification of the estimation accuracies, were introduced into the cost estimate sheets for each of the five major cost areas. The potential variability in estimates were defined as triangular distributions taking the parameters; minimum, most likely and maximum. The ranges were defined as percentages of the deterministic estimate values. Snowden thus converted the cost estimates from a deterministic to a probabilistic state. Table 13.3 shows the triangular distributions used by Snowden in comparison with the equivalent estimation accuracies used by the cost estimators. Note the adjustment that was made by Snowden in some cases in terms of the skewness in the modified distributions.

Snowden has taken the extreme (but realistic) view that costs will in all cases **not be less than the original deterministic values** (which translates to all of the "minimum" values being 0%, and therefore equal to the deterministic values).

The approach that was taken in transforming the estimation accuracy into the statistical distributions was as follows:

Snowden applied the uncertainty (as expressed by the cost estimators' accuracy ranges) on the total cost estimate per cost item. The individual input contributing to the cost items (i.e. equipment, bulk materials and labour) were not individually quantified in terms of their respective estimation base uncertainties, since the estimation accuracy which was applied by the cost estimators were applied per cost item as a whole.

Table 13.3 Estimate accuracy ranges and modified statistical distributions

Source for estimate	Vulcan		Snowden		
	MIN	MAX	MIN	ML	MAX
Budget : -0.05 / 0.05	-5%	5%	0%	3%	8%
Budget : -0.1 / 0.1	-10%	10%	0%	5%	15%
Budget : -0.05 / 0.15	-5%	15%	0%	3%	18%
Budget : -0.15 / 0.15	-15%	15%	0%	8%	23%
Budget : -0.05 / 0.1	-5%	10%	0%	3%	13%
Budget : -0.1 / 0.1	-10%	10%	0%	5%	15%
Budget : -0.05 / 0.1	-5%	10%	0%	3%	13%
Budget : -0.1 / 0.15	-10%	15%	0%	5%	20%
Budget + List Prices	-10%	10%	0%	5%	15%
Design, Vendor Pricing	-15%	15%	0%	8%	23%
Design, MTO, Unit Pricing	-10%	20%	0%	5%	25%
Design, MTO, Unit Pricing	-10%	15%	0%	5%	20%
Estimated Quantity off Vendor Fax Price	-15%	15%	0%	8%	23%
Fixed and firm : -0.05 / 0.05	-5%	5%	0%	3%	8%
In-house Estimate : -0.15 / 0.2	-15%	20%	0%	8%	28%
In-house Estimate : -0.25 / 0.25	-25%	25%	0%	13%	38%
In-house Estimate : -0.15 / 0.15	-15%	15%	0%	8%	23%
In-house Estimate : -0.05 / 0.15	-5%	15%	0%	3%	18%
In-house Estimate, costs quoted	-10%	20%	0%	5%	25%
In-house Estimate, Vendor supplied	-10%	20%	0%	5%	25%
List Price : -0.1 / 0.1	-10%	10%	0%	5%	15%
Project Pricing, Estimated Quantity	-15%	20%	0%	8%	28%
Quantity Calculated, Fee at Tariff	-5%	15%	0%	3%	18%
Quantity In-house Estimate, rates fax	-10%	20%	0%	5%	25%
Quantity In-house Estimate, rates fax	-25%	25%	0%	13%	38%
Unit Pricing, Vendor Supplied : -0.15 / 0.15	-15%	15%	0%	8%	23%
Unit Pricing, Vendor Supplied, Factor internals	-20%	20%	0%	10%	30%
Vendor Design, MTO, Unit Pricing	-15%	15%	0%	8%	23%
Vendor Quantity of Vendor list Price	-15%	20%	0%	8%	28%
Vendor Quotes/%Factor : -0.1 / 0.2	-10%	20%	0%	5%	25%

13.3 Modelling framework

Snowden used the Monte Carlo simulation method to model probabilistic outputs.

Monte Carlo simulation takes its name from Monte Carlo, Monaco, where the primary attractions are casinos containing games of chance. Such games of chance (roulette wheels, dice, and slot machines) exhibit random behaviour, which translates to risk.

Monte Carlo simulation is a sampling technique which repeatedly generates random values (from defined distributions) for uncertain variables in order to simulate a probabilistic model. The model is typically built using a spreadsheet which integrates all of the uncertain variables into a total cost estimate. “Simulation” in this context refers to an analytical method intended to imitate a real-life system.

Spreadsheet based financial or cost models generally only produce single outcomes (deterministic values), which are the most likely or average scenario. Monte Carlo risk analysis uses both a spreadsheet model and a simulation procedure to automatically analyse the effect of varying inputs on outputs of the modelled system. This is not the same as sensitivity analysis where input variables are varied individually (one at a time) along a linear scale.

In the construction of a Monte Carlo simulation model, values for uncertain variables (ones that have a range of possible outcomes) are typically defined through the use of probability distributions. The types of distributions selected are based on the conditions surrounding variables and may be normal, lognormal, triangular, and uniform or various other distributions. In this case all distributions were triangular.

In probabilistic cost estimation exercises, ranges around uncertain cost items are commonly defined using triangular statistical distributions. The triangular distribution is commonly used in business decision making, particularly in simulations where little is known about the distribution of an outcome. It has three parameters, the minimum and the maximum (which define the range of values), and the most likely value (the peak) which defines the central tendency. The distribution is skewed to the left (negatively) when the most likely value is close to the maximum and to the right (positively) when the most likely value is close to the minimum. It is a simple distribution that, as its name implies, has a triangular shape.

The probabilistic models were run through 10,000 iterations of Monte Carlo simulation using Palisade’s @Risk version 5.0.

13.4 Simulation results

The following figures (Figure 13.1 to Figure 13.7) summarise the results from the Monte Carlo simulation for each of the main capex areas as well as for the total estimate. The composite figures show the output distribution and well as the cumulative distribution in addition to some key statistics. The summary on the right of each diagram shows the percentage difference between the Vulcan deterministic estimate and the 95th percentile of the probabilistic estimates. The main findings are summarised in Table 13.4.

Figure 13.1 Output from Monte Carlo simulation – concentrator and infrastructure

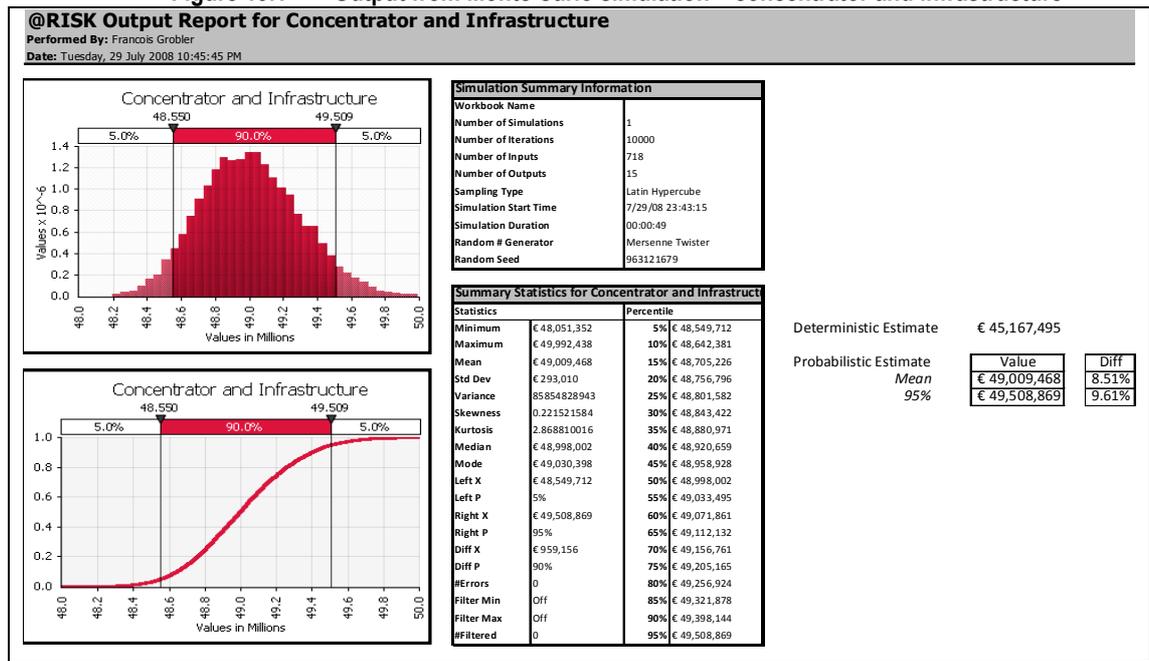


Figure 13.2 Output from Monte Carlo simulation – paste plant

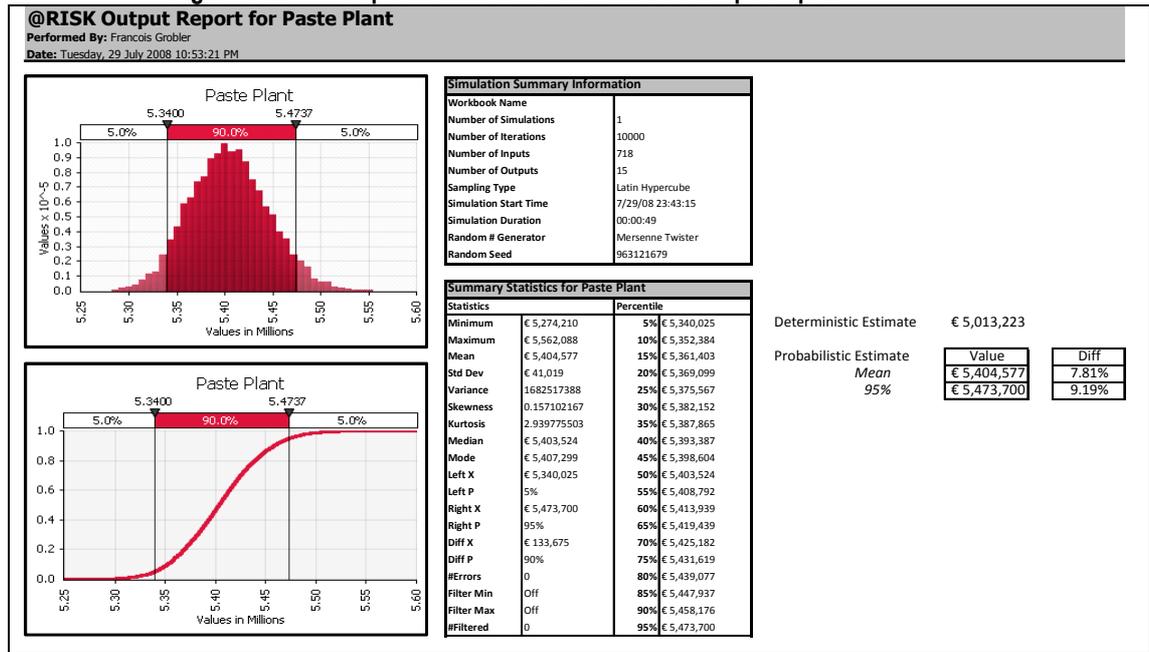


Figure 13.3 Output from Monte Carlo simulation – TSF

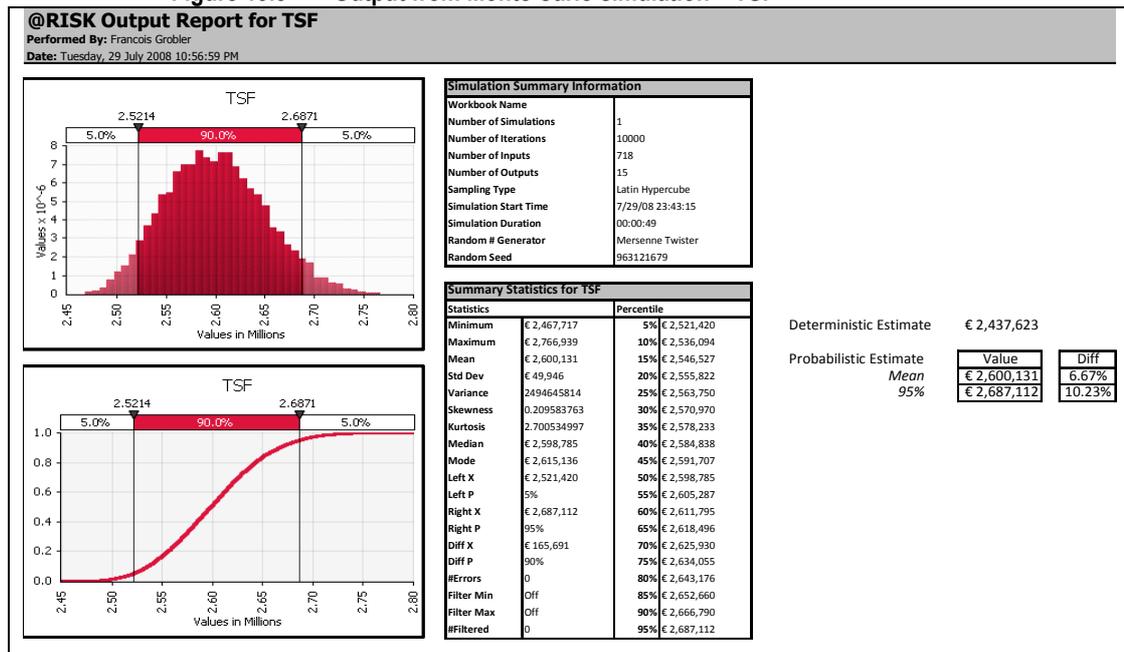


Figure 13.4 Output from Monte Carlo simulation – mining

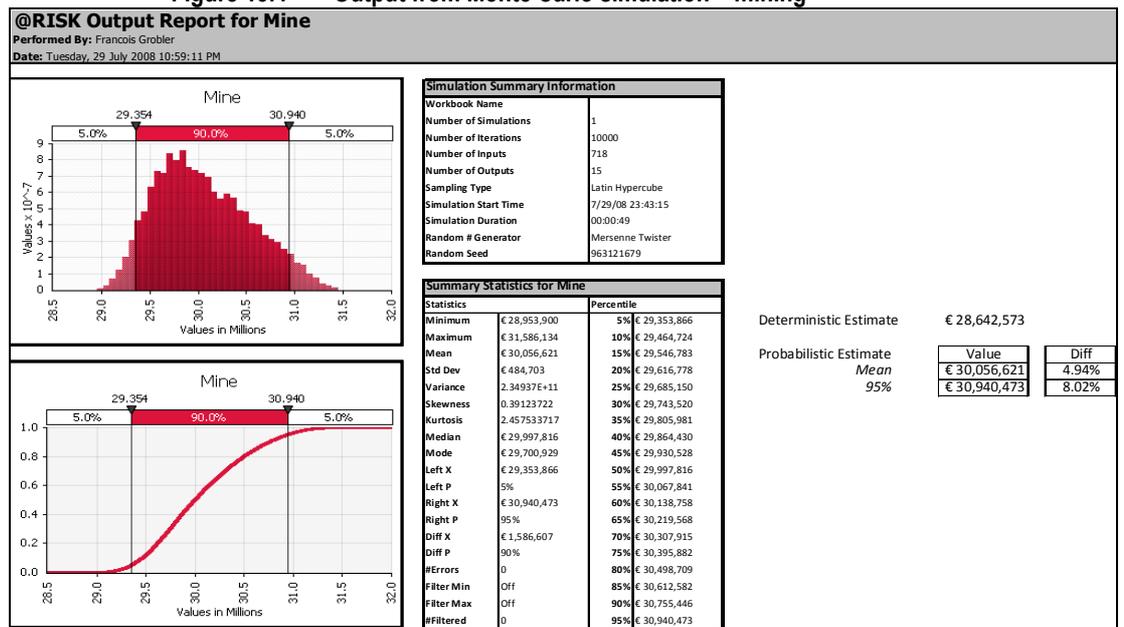


Figure 13.5 Output from Monte Carlo simulation – indirect

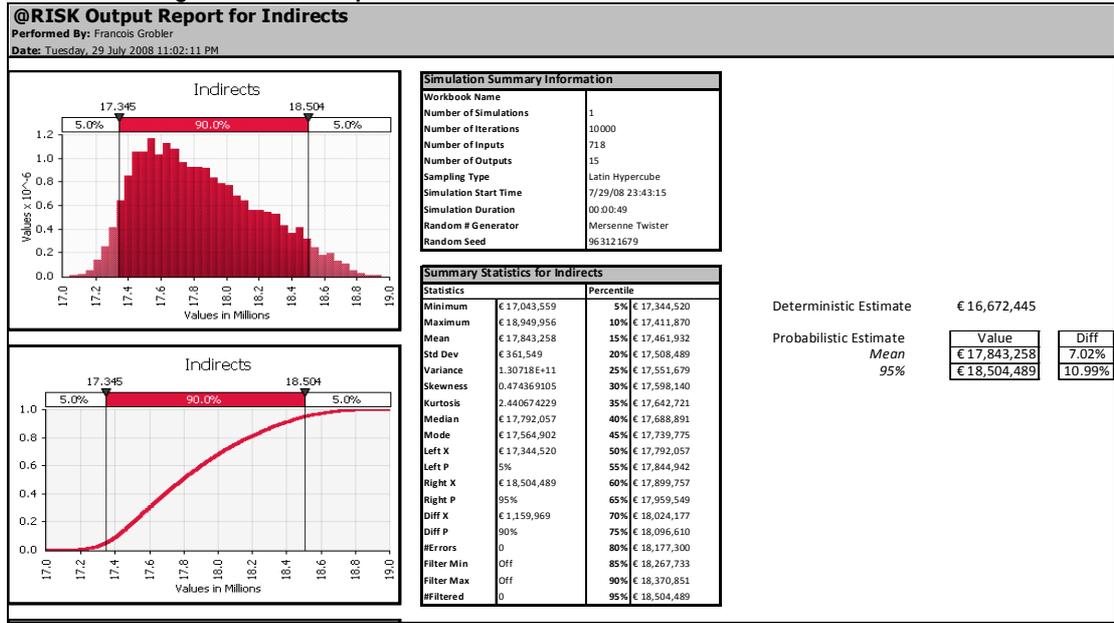


Figure 13.6 Output from Monte Carlo simulation – owner's cost

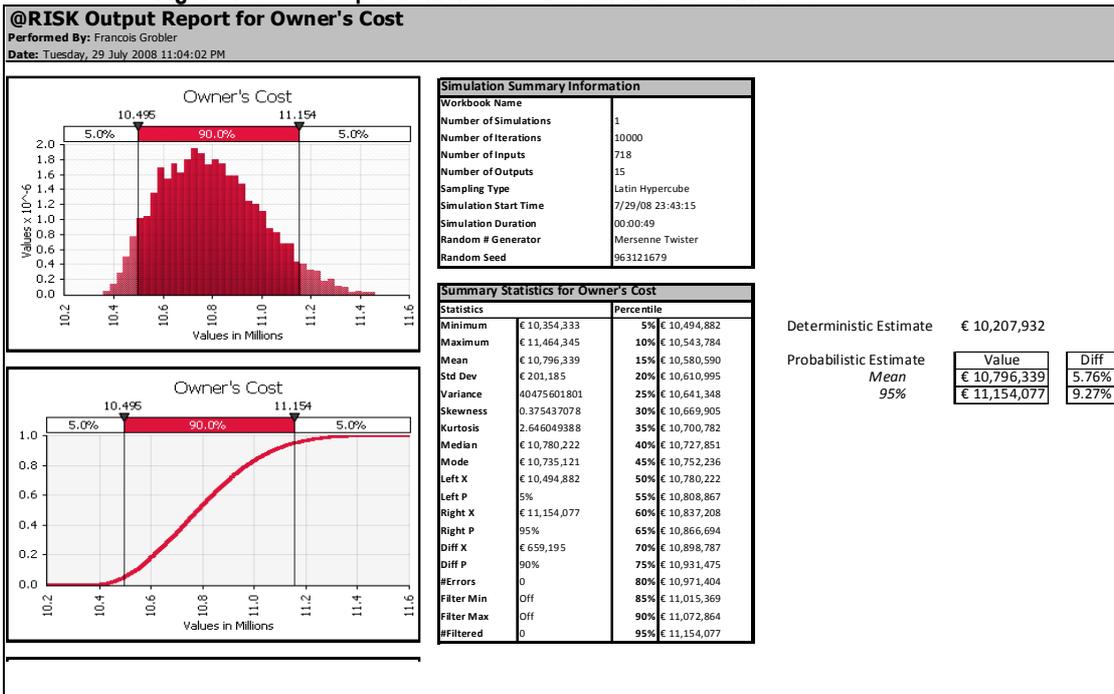


Figure 13.7 Output from Monte Carlo simulation – total estimate

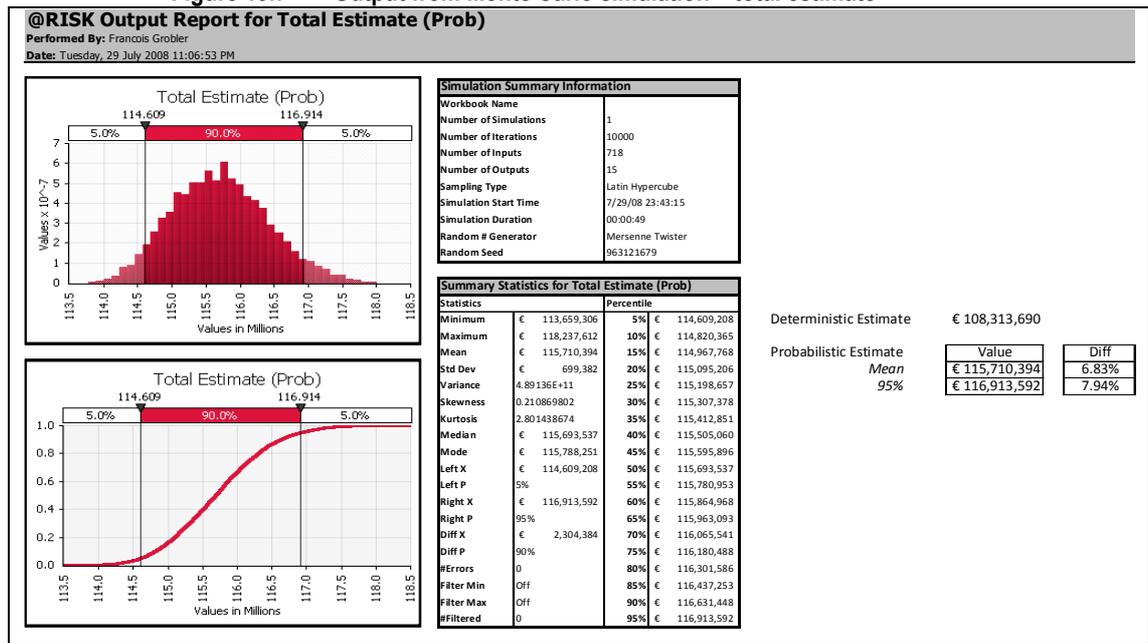


Table 13.4 Summary table with key statistics from Monte Carlo simulation

Component	Deterministic value	Probabilistic mean value	P95 value	% Difference ²
Concentrator and infrastructure	45.2	49.0	49.5	9.6
Paste plant	5.0	5.4	5.5	9.2
TSF	2.4	2.6	2.7	10.2
Mining	28.6	30.1	30.9	8.0
Indirect	16.7	17.8	18.5	11.0
Owner's cost	10.2 ¹	10.8	11.2	9.3
Total	108.3	115.7	116.9	8.0

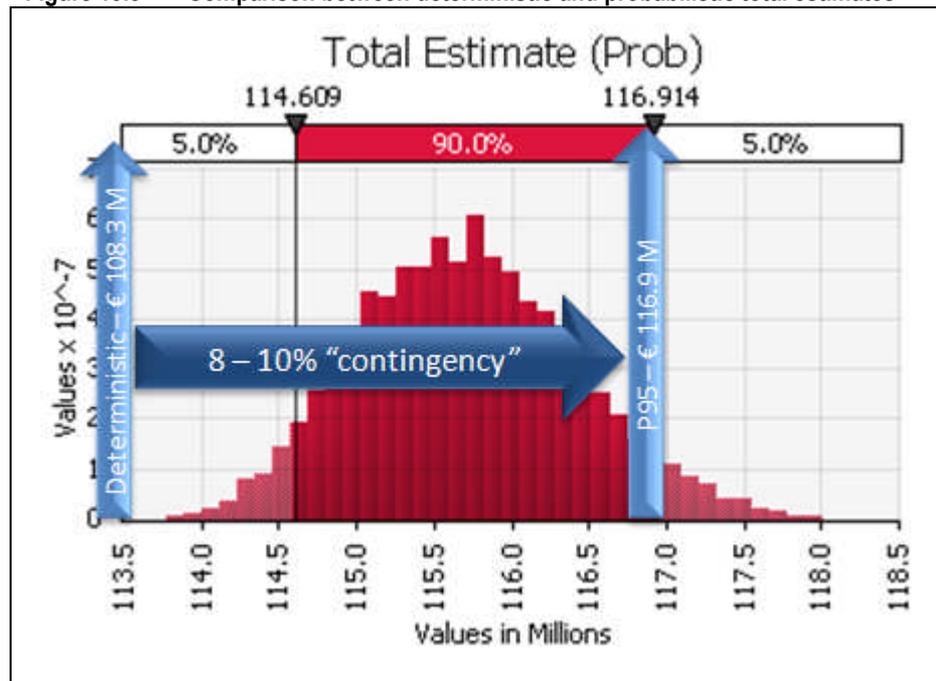
¹ Vulcan contingency excluded

² Differences between deterministic value and P95 value

13.5 Interpretation of results and overrun estimate

The Monte Carlo simulation has introduced the uncertainty related to the various cost estimates through statistical distributions derived from the estimation accuracies. The results of the simulation show that the variability in the total estimate (see Table 13.4) when compared to the deterministic total estimate amounts to 8% when compared to the probabilistic model at a 95th ('P95') percentile. Snowden therefore recommends that in considering the values at the P95 for each of the individual components a contingency range or overrun factor of 8% to 10% of the capital cost be considered. This can be visualised in Figure 13.8.

Figure 13.8 Comparison between deterministic and probabilistic total estimates



Snowden cautions that the figure derived for the estimate uncertainty is related to the confidence in the estimation of base costs comprising the estimation assumptions around quantities and unit costs for equipment, bulk materials and labour.

The potential effects of inflationary escalation and schedule delays have not been quantified and should be added to the 8% to 10% range in order to arrive at a total contingency value or overrun estimate. Snowden is of the opinion that in relation to cost over run on engineering projects as a general trend, a benchmark of 2% to 5% of additional cost can be considering in addition to the values quantified above. Snowden believes however that due to the relatively low likelihood of overruns related to labour, as well as relatively high confidence in timely delivery of critical capital equipment fabricated locally (in Finland), the likelihood for cost over run would rather tend to the lower than the higher end of this range.

Considering the above reasoning thus, a total cost over run range of 10% to 15% is proposed with a preferred value of 12%.

14 Risk summary

14.1 Introduction

Snowden's brief included a thorough risk assessment of the areas covered by the ITE, which are detailed below. Vulcan also has a Corporate Risk Management Plan and register, which is broader than the Kylylahti project. Vulcan's Finnish engineer, Sweco, is working with the Company to develop a Risk Management Strategy for construction and of the project that will be relevant to Finnish staff. Vulcan has also commenced developing a Risk Management Plan for mining and processing operations. As the project moves towards the commencement of construction of both the mine and the plant the Risk Management Plan will be solidified. Snowden is confident that Vulcan has sufficient lead time to both identify the risks (many of which are raised in this Section) and to implement suitable mitigation strategies.

14.2 Resource risks

Risks associated with the Mineral Resource estimate usually fall into three categories; risks associated with the quality of the geoscientific data and its collection, risks associated with the geological framework and the specific interpretation of the orebody and risks associated with the actual resource estimation technique. All of these issues are encapsulated in the Mineral Resource classification which has been assigned to the orebody by the relevant Competent Persons.

With respect to Kylylahti, Snowden notes from its site visit in 2006 that the geological data has been collected according to industry best practice. The vast majority of the data for the resource estimate is based upon diamond drilling, the most reliable form of geoscientific sampling, and the diamond core itself has a high recovery. Drillholes have been surveyed (both collar and downhole) according to good industry practice and a full assay quality control programme has been embarked upon. Vulcan has generated a good database of density measurements with which to derive tonnage factors. Thus Snowden believes that the overall risk associated with geoscientific data and its collection is low.

The geological model and the geological interpretation builds upon local knowledge of the Outokumpu style orebodies; however, each ore environment is different. Having said this, the two main zones at Kylylahti have been reasonably well defined and the two ore-type approach (i.e. the semi-massive sulphide and the disseminated sulphide) is easy to apply. There is some risk associated with the connection between the Wallaby and the Wombat zones, and drilling carried out since the DFS optimisation has illustrated that there is potential upside in the resource estimate. Thus Snowden believes that there is risk, but risk of upside. The key issue is whether the mining method adopted can deal with any upside, and Snowden believes that it has this flexibility.

The final risk is associated with the estimation methodology. Vulcan has engaged respected consultants QG to carry out the resource estimate, and the method that QG has adopted (ordinary block kriging) is an industry standard and has a low risk.

The combination of these three areas is reflected in the overall resource classification which has been applied to the various parts of the deposit. QG, together with Vulcan, have classified the Kylylahti orebody as a combination of Measured, Indicated and Inferred Resource, according to the guidelines in the JORC Code, 2004. Snowden believes that this classification fairly reflects the knowledge of the orebody, the quality of the data and the low risk in the estimation method.

14.3 Reserve and geotechnical risks

Reserve and mine design risks

The main risks to the reserve and achieving the mine plan come from:

- poorly mitigated geotechnical risk (see below for further discussion)
- non-achievement of planned modifying factors (dilution and ore loss)
- failing to achieve the mine schedule
- exceeding planned costs.

Geotechnical risks

The main geotechnical risks to the project are:

- The relative paucity of information for the Wombat zone, particularly for its lower section, and for the upper section of the Wallaby zone. There is not an even distribution of geotechnical data through the deposit, leading to a risk that the data is biased.
- The lack of a model of mine-scale structures, in particular the faults shown bounding the ore zones, means that the effects of such structures on stope and pillar stability have not been assessed.
- Insufficient assessment of operational issues such as stope/pillar layouts, extraction sequences, and support requirements.

These issues impose risk on the project, as they could impact adversely on mining conditions, support requirements, production rates and costs. However, given their nature, these risks are not likely to prove to be fatal flaws.

Modifying factors

Successful mitigation of the identified geotechnical risks will go some way to reducing the risk associated with the modifying factors for dilution and ore loss. Regardless of the geotechnical risk, the application of modifying factors in the reserve estimate is rudimentary and anticipates only two configurations, Wallaby/Primary Wombat and Wombat Secondary stopes. There are a number of stope types, as defined by wall exposures (of ore, waste and fill), which possess unique attributes defining their modifying factor characteristics. The most stable stopes will have only rock wall exposures, the least stable will have fill exposures on three sides, and include a fill crown, and there are a number of variants between the extremes. The use of high strength cemented paste fill reduces this risk somewhat. The nature of potential dilution varies for each stope type.

There is a risk that the modifying factors employed under-predict actual conditions, particularly with regard to the stopes with inherently reduced stability. The risk to the reserve is that stopes with a low profit margin may become uneconomic when increased dilution or ore loss are encountered through unmitigated geotechnical risk or under-estimated modifying factors. SRK (2008), Figure 16 shows that 30 stopes (comprising 1.0 Mt, or 14% of the reserve) make a low contribution to NSR of < 10 €/t, compared to the average of 29 €/t. The sensitivity of the reserve to modifying factors should be determined so that appropriate mitigation strategies can be developed.

Snowden considers that it is possible that this risk might materialise at some time. Although the impact or consequence of it occurring may be major in a short term time frame of up to six months, it will be minor over a mine life relevant time frame, when the effects will be localised and mitigation strategies are likely to be

developed and implemented. The risk rating is therefore Low in accordance with **Error! Reference source not found.**

Schedule

The mine schedule is complex and relies on effectively providing for the many interdependencies between stopes and within the mining activities, including development, drilling and blasting, loading and hauling and filling. SRK (2008) identified the criticality of achieving all scheduled activities, with the plan relying on concurrent and sequential production from a number of small interdependent sources. SRK identifies congestion as being the overall rate determining factor for overall mine production.

There is a risk that underperformance of any particular activity will have downstream impacts and compromise the availability of stopes for production or filling when required. There is also a risk that the geotechnical and modifying factor risks will impact the schedule by slowing the rate of production or increasing the tonnage to be removed from some stopes, extending the duration of those stope lives and further compromising the availability of stopes for vital functions. The risks described can have a compounding effect and may impact the overall productive capability of the mine. The sensitivity of achieving the overall mine schedule to these risks should be determined so mitigation strategies can be developed accordingly.

Snowden considers it possible that this risk will materialise at some time. Although the impact or consequence of it occurring may be major in a short term time frame, it will be moderate over the life of mine timeframe, as there may be mine wide impacts. The risk rating over the life of the mine is considered Medium.

Cost

Snowden considers the estimation of capital and operating costs for the mining operation to be comprehensive and having been carried out to a high level of detail. The operating cost estimate comprises activity variable costs to produce tonnes of ore, and time based fixed costs for certain support functions. The greatest risks to costs are the effects of the modifying factors and the schedule risks. A true breakdown of fixed and variable costs is not known; however, fixed costs may typically represent more than 60% of mining costs.

The impact of the modifying factor risks on costs is to increase the duration of an activity, resulting in an increase in the fixed costs attributable to that activity, and increasing the quantum of variable costs expended. Where schedule dependencies are critical, as for Kylylahti, the whole mine production rate may be negatively impacted, extending the mine life with the variable and fixed costs compounding, for no real gain in metal production. The sensitivity of costs to the modifying factors and schedule should be determined.

Snowden considers it is possible that this risk will materialise at some time though most likely as a result of the risks associated with the modifying factors and schedule. The impact or consequence of it occurring will be similar to that of the schedule risk, major in a short term time frame, and moderate over the life of the mine timeframe. Over a mine life timeframe the risk rating is Medium.

14.4 Process risks

14.4.1 General comments and summary

The process route as proposed appears to be reasonable. The concentrator producing copper-gold concentrate and bulk concentrate is relatively low risk.

A substantial amount of investigative technical work has been conducted by Outokumpu Oy at the VTT technical centres and later by Outokumpu Technology. This work has been followed up by technical review of the reports and further bench scale testing on selected drillhole composites. These results confirm the viability of the two concentrate concept. Assuming that the samples tested are representative of the orebody, indications are that the proposed concept is viable. Variability scale bench testing, followed by a bulk or pilot scale test, has been undertaken. There is no oxide or transition zone and this reduces process risk significantly.

The project currently has a short life but could this could be extended if additional resources can be proven and given the style of orebody and the history of the nearby Keretti mine this may happen.

The process risk has been greatly reduced with the decision to forego downstream processing. Vulcan has greatly simplified the project by deciding to export concentrates.

The process risks are, in summary:

- metallurgy-recovery/ concentrate quality/ technical management
- mineralogy and variability
- fine tuning the flotation process
- testwork sufficient to cover process risks
- the process operability maintainability
- cold climate issues
- cultural and design issues
- the availability of good management
- staffing - skill and experience.

14.4.2 Discussion of specific process risk issues

Mineralogy and talc issues

In general the talc is in the waste and not the ore. There will inevitably be some dilution, but the talc extracted should not be significant or pose a serious problem.

The level of talc in the copper-gold concentrate and bulk concentrate from the testwork is approximately 1%.

Metallurgical testwork

Outokumpu originally undertook testwork on the orebody. Vulcan has undertaken a significant amount of comminution testwork and bench scale flotation testwork. Variability testwork and geometallurgical spatial analysis of the orebodies has been undertaken. In addition piloting has been undertaken at AMMTEC and the results have confirmed the prior bench scale tests conducted previously.

Climatic issues

The cold climate will require that process equipment is inside buildings which have internal heating. Piping will require lagging and heat tracing or to be buried and local knowledge will be important; fortunately local knowledge, both mining and processing, is in abundant supply. Further to this, Vulcan has appointed or has shortlisted a number of EPCM contractors and design engineers which have extensive knowledge of and experience in extreme Finnish conditions (see Section 9.3).

Cross cultural and design codes issues

Each country has its own design codes and the design will need to be undertaken in Finland to ensure the design codes are met. This means that the detailed design will be undertaken in Finland, and Vulcan has already started to implement this.

Bulk concentrate production

The flotation of Outokumpu-style ores is well developed and there is a history of successful flotation at Keretti. Snowden expects that this part of the process flowsheet will ramp up quickly, and provided that skilled flotation operators are employed there should be no unusual issues. Neither Vulcan nor Snowden see any obstacles in obtaining the necessary technical resources for the flotation fine tuning.

14.5 Semi-quantitative risk assessment

Snowden has adopted a semi-quantitative approach to the risk assessment of the Kylylahti project, using the approach recommended in Australian Standard 4360 – Risk Management. This approach examines risk using a two-dimensional matrix. The risk assessment contains a judgement of the probability or likelihood of issues associated with the project detailed being realised (or not being realised, depending upon the item), the impact of that realisation (or non-realisation) and an overall risk rating based on the combination of both.

Measures of likelihood or probability are as defined in Table 14.1.

Table 14.1 Risk descriptors - likelihood or probability

Risk likelihood or probability		
Level	Descriptor	Description
E	Almost certain (1:1)	Is expected to occur in most circumstances
D	Likely (1:10)	Will probably occur in most circumstances
C	Possible (1:100)	Might occur at some time
B	Unlikely (1:1,000)	Could occur at some time
A	Rare (1:10,000)	May occur only in exceptional circumstances

Measures of consequence or impact on the project are as defined in Table 14.2. Impact can be defined in many ways – in terms of financial impact, in terms of project delays, or in terms of safety, to name a few. To avoid ambiguity a general description of consequence has been adopted here. It should be apparent which dimension is being considered when considering the respective line items.

Table 14.2 Risk descriptors – consequence or impact

Consequence or impact on the project	
Level	Descriptor
1	Insignificant
2	Minor
3	Moderate
4	Major
5	Catastrophic

Vulcan has provided its own quantitative assessment of the impact descriptors in Table 14.2 in terms of the impact on the NPV of the Kylylahti project. This is reproduced in Table 14.3. Snowden accepts this as a reasonable set of criteria and has cast its impact assessments in this light.

Table 14.3 Financial dimension of impact risk descriptors

Consequence or impact on the project in financial terms		
Level	Descriptor	
1	Insignificant	No material impact
2	Minor	Project economics impacted by less than 20% of NPV
3	Moderate	Project economics materially impacted, NPV reduction by more than 20%
4	Major	Project becomes uneconomic on an NPV basis but can continue
5	Catastrophic	Project fails or shuts down

The likelihood and impact dimensions of risk have been combined to provide an overall risk rating as per Table 14.1. The overall semi-quantitative risk assessment for the key project aspects is provided in Table 14.5.

Table 14.4 Combination of likelihood and consequence to provide overall risk ratings

		Consequence				
		Insignificant	Minor	Moderate	Major	Catastrophic
Likelihood		1	2	3	4	5
Almost certain	E	L	L	M	H	H
Likely	D	L	L	M	H	H
Possible	C	VL	L	M	H	H
Unlikely	B	VL	VL	L	M	H
Rare	A	VL	VL	L	M	H

Note: VL = Very Low, L = Low, M = Medium, H = High

Table 14.5 Semi-quantitative risk assessment for key aspects of the Kylylahti project

Specific Issue	Detail	Probability	Impact	Risk
Mineral Resources				
Data quality	Reliability of resource data	B	2	VL
Geological model	Overall orebody understanding	C	2	L
Estimation error	Accuracy of estimation	C	2	L
Resource classification	Suitability of resource categories	B	2	VL
Ore Reserves				
Modifying factors	Inadequate provision for dilution or ore loss	C	2	L
Schedule	Achievable schedule	C	3	M
Costs	Costs exceed estimate	C	3	M
Geotechnical Considerations				
Geotechnical model	Incorrect processing of data	D	2	L
Ground conditions	Inadequate geological and structural models	D	2	L
Mine design parameters	Inadequate assessment of influence of geotechnical conditions on stope/pillar layout and extraction sequences	D	2	L
Metallurgy				
Cu/Au/Co/Ni recovery	Ore variability and blending problems	C	3	M
Concentrate quality	Contaminant levels exceeded	C	2	L
Mineralogy changes	Variability/complexity	D	2	L
Accounting issues	Metal balance problems	B	2	VL
Technical management	Key personnel leave	D	3	M
Optimisation flotation	Insufficient technical resources to fine tune the flotation	B	3	L
Operational				
Production targets	Lower than forecast tonnages	C	3	M
Cost control	Higher costs than budget	C	3	M
Manpower/Training	Shortages of skilled operators	C	3	M
Availability	Mechanical plant breakdown	C	2	L
Maintenance	Lack of critical spares and personnel	D	2	L
Climatic issues	Cold weather freezing difficulties	D	2	L
Cross culture issues	Communications difficulties	D	2	L

Specific Issue	Detail	Probability	Impact	Risk
Environment				
Tailings(lined)	Infiltration of contaminated water into soil	B	2	VL
Tailings(lined)	Infiltration of groundwater into aquifers	B	2	VL
Tailings	Residual organic reagents	C	3	M
Tailings	Discharge of contaminated groundwater through ruptured pipes	C	2	L
Emissions	Dust and particulate matter emissions	C	2	L
Emissions	Discharge of dust from transport and crushing facility	C	2	L
Processing	Presence of fibrous minerals in mining or processing material	C	2	L
Waste rock stockpile run-off	Discharge into surface waters	C	2	L
Infrastructure				
Power outages	Loss of power supply cold weather	C	2	L
Process water shortages	Cold weather	D	3	M
Potable water shortages	Cold weather	A	4	M
Shipping issues	Boats late or not available	E	1	L
Tailings dam	Lack of tailings capacity/difficulties	D	3	M
Transport logistics	Vehicle breakdowns	D	1	L
Services				
Assay	Assay accuracy problems	B	1	VL
Offsite support	Technical, mechanical supply problems	C	3	M
EPCM				
Cost control	Delays and cost escalation	D	3	M
Engineering	Codes and local knowledge	D	3	M
Procurement	Long lead items	D	3	L
Project Management	Shortage of management	C	3	M
Construction	Labour shortages	B	4	M
Commissioning	Personnel shortages	C	3	M

15 References

- Golder, 2007 Kylylahti Project, Finland, Mine Dewatering Strategy, Golder Associates, January 2007.
- Golder, 2007 Feasibility Study for a thickened tailings surface disposal and paste backfill system – Kylylahti Mine, October 2008.
- JORC, 2004 Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code), 2004.
- Kontinen, 1987 An early Proterozoic ophiolite- the Jormua mafic ultramafic complex, northern Finland. Precambrian Research 35, Kontinen, A., 1987, p 313-341.
- ME, 2005 Ministry of the Environment, Aquatically harmful and Dangerous Substances, 2005.
- Outokumpu, 2008 Outokumpu Oy, Mine Closure Handbook, Environmental Techniques for the Extractive Industries, 2008.
- SRK, 2008 Kylylahti Copper Project Mining Feasibility Study, March 2008.
- SRK, 2007 Kylylahti Copper-Cobalt-Nickel Project Definitive Feasibility Study, Vol. 1 – 8, SRK Consulting, October 2007.
- SNC, 2007 SNC-Lavalin Report 138270-0000-49RA-0003, Tailings and Supernatant Testing Results, 25/05/07.
- SNC, 2007(2) SNC Document No. 138270-EV-0008, Kylylahti Water Management Plan, 05/04/07.
- Turnberry, 2008 Kylylahti Copper Project Pastefill Plant Front End Process Design, Turnberry RPA Pty Ltd, March 2008.
- Vulcan, 2008 Kylylahti Copper-Cobalt-Nickel Project, Definitive Feasibility Study Review and Optimisation Report, April 2008.