TECHNICAL REPORT FOR THE KEVITSA CU-NI-PGE MINE, FINLAND

Prepared For

Boliden Kevitsa Mining Oy

Report Prepared by

srk consulting

SRK Consulting (Finland) Oy

UK30559

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EXECUTIVE SUMMARY
TECHNICAL REPORT FOR THE KEVITSA CU-NI-PGE MINE, FINLAND

1 INTRODUCTION

SRK Consulting (Finland) Oy ("SRK") has been requested by Boliden Kevitsa Mining Oy ("Boliden" or the "Company"), to undertake reviews and produce audited statements of Mineral Resources and Mineral Reserves for their Kevitsa copper-nickel-platinum group element mine ("Kevitsa Cu-Ni-PGE mine", "Kevitsa" or the "Project") located in Finland. This report will be used to contribute to public annual reporting in relation to the Company's listing on the NASDAQ OMX Stockholm Exchange.

2 KEVITSA DESCRIPTION AND CURRENT STATUS

2.1 General

Kevitsa is located in Lapland, northern Finland some 142 km north-northeast of Rovaniemi. The previous owner, First Quantum Minerals Ltd ("FQM"), started mine production in 2012 before Boliden purchased the mine in 2016. A total of 54 Mt of ore has been mined from Kevitsa up to end-December 2019.

2.2 Geology

Kevitsa is a magmatic, layered-intrusive, Cu-Ni-PGE deposit of Precambrian age (circa 2 Ga). The mineralisation at Kevitsa is disseminated in style, while having some minor massive sulphide veins. The mineralisation is dominantly comprised of chalcopyrite ($CuFeS_2$) and pentlandite ($(Fe,Ni)_9S_8$). PGE carrying minerals, which are related to sulphides, occur mostly on sulphide grain boundaries.

2.3 Exploration

Exploration has been carried out on the property by various owners and the Geological Survey of Finland since the 1970s. FQM, the previous owners, , undertook significant exploration prior to commencing operations in 2012. Boliden has continued to undertake near-mine exploration in addition to grade control drilling since taking ownership in 2016.

The sampling and assaying procedures in place during the various exploration campaigns are reported by FQM to have been undertaken using industry best practice. SRK has not identified any issues with data quality.

2.4 Mineral Resource Estimate

The latest Mineral Resource estimate ("MRE") for Kevitsa was the 2018 Mineral Resource completed by Lion GeoConsulting Ltd ("LGC").

The MRE was completed using conventional wireframing and grade interpolation (Ordinary Kriging) techniques and was based on exploration data including 518 diamond core drillholes. SRK reviewed the MRE and did not identify any fatal flaws with the process undertaken or resulting end-2018 Mineral Resource statement. SRK has therefore used this MRE as the basis to report the 31 December 2019 Mineral Resource and Mineral Reserve statements herein.

2.5 Mining

Kevitsa is an established, large-scale conventional open pit mine, in production since August 2012. Historically, since 2014, the mine was able to achieve 6.9 to 8.3 Mtpa ore mining and total mining tonnage of 21 to 42.5 Mtpa with average yearly stripping ratios of 3.1 to 4.6.

To define the Mineral Reserves, Mineral Resource was established through the Whittle 4D Pit optimisation process and tested in a strategic mine plan. From the pit optimisation, a final optimal pit shell was selected, along with interim pushbacks shells and stages incorporating ramps and bench geometry according to the geotechnical criteria. The total inventory within the pit designs was comparable with the pit shell within a 7% margin for ore tonnes and metal content.

The final designs and pushbacks were scheduled in Deswik's interactive scheduler ("Deswik.IS") to produce a Life of Mine plan ("LoMp"). The LoMp forecasts high waste stripping between 2020 and 2022 (35 Mtpa) which is significantly reduced in 2023 down to 15 Mtpa, and further decreases from 2027 onwards. Recent expansions in the concentrator plant will see a ramp-up in ore mining production in 2020 to achieve 10 Mtpa of ore from 2021 onwards.

Based on the LoMp, the primary equipment fleet requirements were estimated from first principles on which the mining budget cost estimation was completed. In total, 17 additional Komatsu 830-E were acquired in 2019 as well as one new CAT 6060 Face shovel in 2019 and an additional CAT 6060 which will be commissioned in 2020 to supplement the increase in production.

The definition of ore in the LoMp and subsequent Mineral Reserves included the usage of an NSR formula which was used as an operational cut-off. An NSR cut-off of >EUR15/t was used to report Mineral Reserves in the LoMp which relates to the combined unit costs for processing and mining. The NSR formula considers factors for processing recoveries, metal prices, payability, treatment, and refinement charges. For the 2020 budget plan, the NSR calculation included an escalation in the Ni price which inadvertantly implied a lowered cut-off NSR and the inclusion of marginal ore in the budget plan.

2.6 Processing

Large run of mine ("RoM") stockpiles, nominally 1 Mt, are used to try to maintain reasonably consistent feed in terms of metal grades and ore hardness. The stockpiles represent 6 to 7 weeks plant feed. The RoM stockpiles are managed by the geology department.

Ore is blended on the RoM stockpiles considering Cu and Ni grades, chalcopyrite:cubanite ratio, pyrrhotite:pentlandite ratio, and ore hardness.

High talc ores (talc up to 30%) are stockpiled separately and processed in batches since this material has a detrimental effect of flotation, in particular copper flotation. The reduction in copper recovery is dependent on the level of talc in the feed and, in extreme cases, the recovery can be reduced by up to 4%.

The process flowsheet can be considered conventional and incorporates:

- primary crushing and screening via a single gyratory crusher;
- secondary and tertiary crushing;
- stockpiling;
- grinding to 76 to 78% -75 µm in two autogenous mills operating in parallel;
- AG mill pebble recycling to the tertiary crusher, a single pebble mill operating in closed circuit;
- sequential copper and nickel flotation;
- pyrite flotation and concentrate dewatering by thickening and filtration; and
- tailings are pumped to a dam and the high sulphur concentrate is stored separately in a lined pond.

All equipment installed in the plant is industry standard and is from high quality suppliers; there are no significant issues reported. The plant incorporates a very high degree of instrumentation and control and has demonstrated a throughput of 7.6 to 7.9 Mtpa over the period 2017 to 2018.

Historically, the Cu concentrate grade is typically 22 to 24% Cu and the average Cu recovery to the Cu concentrate is 82.8%. The Ni content of the Cu concentrate is typically 0.8 to 0.9% Ni representing around 5% Ni recovery. Gold recovery to Cu concentrate is 42 to 46%. Platinum and palladium recoveries to Cu concentrate are typically 29 to 34% and 36 to 44%, respectively.

Historically, the Ni grade of Ni concentrate is 8.4 to 10.0%, typically around 9.2% Ni at a recovery of around 70.8%. Cu content of the Ni concentrate is typically 1.2% Cu representing around 9% Cu recovery. The Co content of the Ni concentrate is typically 0.40% Co representing between 60 to 79% Co recovery. Gold recovery to Ni concentrate is around 9%. Platinum recovery to nickel concentrate is 23 to 24% at a grade of 2.4 g/t Pt in concentrate. Palladium recovery to nickel concentrate is 26 to 28% at grade of 3.4 g/t Pd in concentrate.

Lower Cu feed grades in 2019 have impacted copper recoveries to the Cu concentrate. Lower copper concentrate grades have to be targeted as soon as possible to maximise copper recoveries.

Operating figures show a recent fall in Cu and Ni feed grades and an increase in throughput is required to maintain copper and Ni concentrate production levels. The concentrator expansion project will address this issue.

The concentrator expansion project is nearing completion and is designed to increase the plant capacity up to 10 Mtpa. Based on the study, the new mill will increase throughput capacity up to 3.6 Mtpa. SRK does not have any reservations regarding the design and this capacity should be achievable. This project is planned to come on-line in Q1 2020.

2.7 Tailings Management

The existing tailings storage facility ("TSF") is forecast to reach design capacity during early 2030. An alternative TSF location will be required to store the current shortfall of approximately 39 Mt of tailings which are forecast to be generated during years 2030 to 2034 inclusive (2034 is the end of the current LoMP). As such, there is currently no design or environmental permit in place covering a new standalone TSF for this period.

Detailed deposition planning (including 3D modelling) is undertaken for the existing facility, to ensure that tailings deposition is occurring in the correct sector and that construction of embankment raises can scheduled within the short 7-month summer construction season each year. SRK considers this approach to be in line with international best practice and allows the Company to effectively plan over the short term.

The increased tailings production rate (9.5 Mpta) following plant expansion, will increase the observed rate of rise of the TSF. Additional analysis is required to ensure that each embankment raise can be installed as designed, on potential contractive tailings materials (which may be prone to static liquefaction). There is a risk that the scheduling of embankment raises will be impacted. Whilst the current construction method/scheduling has been tested, this needs to be checked against the future rate of rise of the facility. In addition, the capacity of for both contact and stormwater storage on the TSF is forecast to diminish towards the final years of TSF operation. Additional studies are required to adequately quantify the above risks and devise remediation measures as appropriate.

SRK considers the operating management system ("OMS") documents to be systematic and detailed. The documentation meets the requirements set out in the Mining Association of Canada ("MAC") Guidelines.

An annual monitoring report is produced by Golder Associates ("Golder"), which summarises collated data and any deviations recorded. All collected data are checked against 'trigger levels', which have been defined by Golder through stability analysis. No significant exceedances were recorded during 2018 and hence the facility was operated within anticipated parameters. SRK considers the number and location of instruments to be suitable for a facility of this size.

SRK recommends that design work should be progressed as a matter of priority for a new TSF, such that an optimised solution for tailings storage post 2030 can be realised. The permitting status and timeline for the new TSF locations should be checked in line with the project implementation schedule to ensure that no delays will be incurred to the project as a result of government approvals.

SRK has adjusted the financial model to make provision for the estimated shortfall in tailings storage between 2030 and 2034. An additional EUR 10 M for starter embankment construction (2028 to 2029) and EUR 4 M per annum thereafter (for subsequent embankment raises) has been estimated to ensure there is adequate provision for life of mine tailings storage.

2.8 Environment and Social

Groundwater monitoring has identified contamination of groundwater in the vicinity of the TSF, which suggests TSF water seepage through the peat/bentonite liner at this facility. A corrective action plan is being developed with the assistance of Boliden's external consultants. SRK understands that an additional TSF will be required to meet the waste storage requirements for the full LoMp. The groundwater contamination will likely make the permitting of an additional TSF a challenge with additional scrutiny of site selection, design and closure plans. This could prove to be on the critical path for the operation to achieve production out to the current plan of 2034 and subsequently beyond this date.

The occurrence of a rare moss species in the area of the planned waste rock dump ("WRD") extension is putting the approved location and design of this facility at risk. Boliden has conducted several surveys during the course of 2018 and 2019 and have identified multiple additional sites where the moss occurs A rare frog species has also been identified at the site. As with the moss, Boliden has sponsored a number of additional surveys which have identified multiple additional sites where the species occurs outside the proposed WRD footprint. SRK understands that this provides options for the potential relocation of both moss and frog populations. The additional habitat areas also provide options should groundwater drawdown associated with the pit extensions impact on wetland areas that form part of the frog habitats. Boliden has stated that they see no constraints in term of land availability to relocate and expand the WRD should this be required. This may imply a longer haul distance.

The 2018 environmental monitoring report concluded that elevated heavy metals were observed in some bioindicators (such as soil, humus and moss) collected from around the open pit and TSF areas. The report stated that the levels observed were higher than 2015 and probably due to dust deposition from blasting, traffic and tailings deposition. Higher dust concentrations were also observed further away from the mine and this was attributed to 2018 being generally a drier, warmer year contributing to the wider dispersion of dust. All other parameters (water discharges, surface water, air quality and noise) remained in line with previous year's results. SRK is not aware of any actions required are a result of the findings of the 2018 monitoring. Boliden will conduct further surveys in 2021 to determine whether the elevated levels continue to trend higher.

2.9 Economics

The Kevitsa Mineral Reserve LoMp returns a positive NPV, with sufficient margin to cover higher mining costs (more in line with those historically achieved) than those assumed by Boliden. If the cost assumptions proposed by SRK are agreed to be more appropriate by Boliden, SRK recommends that the Company re-assesses its methodology to forecast longer term operating costs, using appropriate cost drivers. If mining costs are higher than those currently estimated by the Company (and possibly more in line with historically achieved unit costs, although SRK acknowledges the recent purchase of modern more efficient equipment and other initiatives to reduce mining cost), the Company may need to re-assess its currently applied NSR cut-off of EUR 15/t for Mineral Reserves. The marginal cut-off NSR of EUR 10/t, as applied to the Mineral Resource, seems appropriate in the opinion of SRK.

3 MINERAL RESOURCE AND MINERAL RESERVE STATEMENTS

3.1.1 2019 Mineral Resource Statement

The 31 December 2019 Mineral Resource statement for Kevitsa produced by SRK on behalf of Boliden is presented in Table ES 1 (inclusive of Mineral Reserves) and Table ES 2 (exclusive of Mineral Reserves) with notes explaining the reporting procedure provided underneath.

Table ES 1: Mineral Resource Statement (inclusive of Mineral Reserves) effective of 31 December 2019*

Mineral Resource Category	Tonnes (Mt)	Sulphide Nickel (%)	Total Copper (%)	Gold (g/t)	Platinum (g/t)	Palladium (g/t)	Sulphide Cobalt (%)
Measured	88.2	0.24	0.35	0.10	0.20	0.13	0.01
Indicated	189.5	0.25	0.34	0.09	0.19	0.12	0.01
Meas+Ind	**277.7**	**0.25**	**0.34**	**0.10**	**0.19**	**0.12**	**0.01**
Inferred	19.2	0.22	0.33	0.06	0.13	0.09	0.01

**In reporting the Mineral Resource Statement, SRK notes the following:*

- *Mineral Resources have an effective date of 31 December 2019*
- *Competent Person for the declaration of Mineral Resources is Dr Lucy Roberts, an employee of SRK.*
- *Reported Mineral Resources are below the mined topography, dated 31 December 2019.*
- *Mineral Resources are reported inclusive of Mineral Reserves.*
- *Mineral Resources are reported as undiluted, with no mining recovery applied in the Statement. Assumptions for mining factors (mining and selling costs, mining recovery and dilution, pit slope angles) and processing factors (metal recovery, processing costs), during the optimisation process only.*
- *SRK considers there to be reasonable prospects for economic extraction by constraining within an optimised open pit shell constructed using long term market forecast commodity prices.*
- *Mineral Resources are reported above the optimised pit shell and above a Net Smelter Return ("NSR") marginal cut-off of EUR 10/t, which reflects the economic and technical parameters,*
- *Tonnages are reported in metric units, grades in percent or grams per tonne (g/t). Tonnages and grades are rounded appropriately.*
- *Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.*

Table ES 2: Mineral Resource Statement (exclusive of Mineral Reserves) effective of 31 December 2019*

Mineral Resource Category	Tonnes (Mt)	Sulphide Nickel (%)	Total Copper (%)	Gold (g/t)	Platinum (g/t)	Palladium (g/t)	Sulphide Cobalt (%)
Measured	26.5	0.23	0.33	0.08	0.16	0.10	0.01
Indicated	112.9	0.23	0.34	0.08	0.14	0.09	0.01
Meas+Ind	**139.4**	**0.23**	**0.34**	**0.08**	**0.15**	**0.09**	**0.01**
Inferred	17.8	0.22	0.33	0.06	0.13	0.08	0.01

**In reporting the Mineral Resource Statement, SRK notes the following:*

- *Mineral Resources have an effective date of 31 December 2019*
- *Competent Person for the declaration of Mineral Resources is Dr Lucy Roberts, an employee of SRK.*
- *Reported Mineral Resources are below the mined topography, dated 31 December 2019.*
- *Mineral Resources are reported exclusive of Mineral Reserves.*
- *Mineral Resources are reported as undiluted, with no mining recovery applied in the Statement. Assumptions for mining factors (mining and selling costs, mining recovery and dilution, pit slope angles) and processing factors (metal recovery, processing costs), during the optimisation process only.*
- *SRK considers there to be reasonable prospects for economic extraction by constraining within an optimised open pit shell constructed using long term market forecast commodity prices.*
- *Mineral Resources are reported above the optimised pit shell and above a Net Smelter Return ("NSR") marginal cut-off of EUR 10/t, which reflects the economic and technical parameters, and below the mine design pit shell used to report the Mineral Reserve.*
- *Tonnages are reported in metric units, grades in percent or grams per tonne (g/t). Tonnages and grades are rounded appropriately.*
- *Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.*

3.1.1 2019 Mineral Reserve Statement

The 31 December 2019 Mineral Reserve statement produced by SRK, on behalf of Boliden, is presented in Table ES 3 with notes explaining the reporting procedure provided underneath.

Table ES 3: Mineral Reserve Statement effective of 31 December 2019*

Mineral Reserve Category	Tonnage (Mt)	Sulphide Nickel (%)	Total Copper (%)	Gold (g/t)	Platinum (g/t)	Palladium (g/t)	Sulphide Cobalt (%)
Proved	62	0.25	0.33	0.10	0.19	0.12	0.01
Probable	78	0.23	0.31	0.11	0.24	0.16	0.01
Prov+Prob	**140**	**0.24**	**0.32**	**0.10**	**0.21**	**0.14**	**0.01**

**In reporting the Mineral Reserve Statement, SRK notes the following:*

- *Mineral Reserve statement has an effective date of 31 December 2019.*
- *Competent Person for the declaration of Mineral Reserves is Mr Hanno Buys, an employee of SRK and professional member of The Institute of Materials, Minerals and Mining ("IOMMM") in the United Kingdom and registered as a Professional Mining Engineer ("Pr.Eng") with the Engineering Council of South-Africa.*
- *Reported Mineral Reserves are below the actual mined topography, dated 31 December 2019 and above the final stage 4 pit design "kev_stage4_28052019.dtm" (based on recommended pit slope angles), and are all contained within the pit shell used for the Mineral Resource Statement.*
- *Mineral Reserves are reported inclusive of mining modifying factors which are based historical reconciliation results, a 7% dilution and a 93% mining recovery are applied in the statement.*
- *Mineral Reserves are inclusive of a 0.153 Mt of RoM stockpile at 31 December 2019.*
- *A life of mine plan production schedule along with mining factors (mining recovery and dilution), processing factors (Recovery and Processing costs) and revenue factors (metal prices, selling costs) were incorporated in a financial model and economic analysis by which SRK determined the Mineral Reserves to be currently economic.*
- *Mineral Reserves are reported within the pit design at a Net Smelter Return ("NSR") operational cut-off of EUR 15/tonne ore.*
- *Tonnages are reported in metric units, grades in percent (%) or grams per tonne (g/t). Tonnages and grades are rounded appropriately.*
- *Mineral Reserves include 40 Mt of Ore to be mined at the last four years of the LoM (years 2030-2034) for which current TSF capacity is insufficient. These Mineral Reserves are dependent on Kevitsa identifying a suitable location, designing and obtaining relevant permits for additional TSF capacity within the next 10 years - prior to the tailings deposition.*
- *Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.*

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TECHNICAL REPORT FOR THE KEVITSA CU-NI-PGE MINE, FINLAND

1 INTRODUCTION

1.1 Background

SRK Consulting (Finland) Oy ("SRK") has been requested by Boliden Kevitsa Mining Oy ("Boliden" or the "Company"), to undertake reviews and produce audited statements of Mineral Resources and Mineral Reserves for its Kevitsa copper-nickel-platinum group element mine ("Kevitsa Cu-Ni-PGE mine", "Kevitsa" or the "Project") located in Finland.

1.2 Purpose of Report

This report includes independently audited Mineral Resource and Mineral Reserve statements. This report will be used to contribute to public annual reporting in relation to the Company's listing on the NASDAQ OMX Stockholm Exchange.

1.3 Basis of Review

The report, as presented herein, has been based on:

- inspection visits by SRK to the mine operation, plant facilities and surface infrastructure in November 2019;
- access to Kevitsa key personnel for discussion, verification and enquiry;
- review of Boliden's internal estimates and classification of Mineral Resources and Mineral Reserves, including its methodologies;
- review of Boliden's mine plans and technical-economic models for each operation;
- review of site environmental conditions, closure costs and environmental objectives; and
- review of health and safety at the mine and plant, and from a corporate perspective.

In summary, SRK has prepared Mineral Resource and Mineral Reserve statements based on a review of the mine plan and technical economic model, mining licence and the methodologies applied for the estimation and classification of Mineral Resources and Mineral Reserves.

1.4 Capability and Independence of Consultant

This report was prepared on behalf of SRK by the persons whose qualifications and experience are set out in Table 1-1 below.

SRK is an independent consulting engineering organisation, wholly owned by its employees, that has been active in the mining and natural resources industries for nearly 40 years. The group operates globally and currently employs approximately 1,500 professionals in 48 offices worldwide. SRK has a demonstrated track record in undertaking independent assessments of resources and reserves, project evaluations and audits and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide.

This technical report has been prepared based on a technical and economic review by a team of consultants sourced from SRK's Group offices in the United Kingdom and Finland.

Neither SRK, nor any of its employees and associates employed in the preparation of this report, has any material present or contingent interest in the outcome of this report or in any of the Assets being assessed. Nor do they have any pecuniary or other interest that could be reasonably regarded as being capable of affecting their independence or that of SRK. SRK will be paid a fee for the preparation of this report in accordance with normal consulting practice.

The individuals who have provided input to this report, and who are listed below, have extensive experience in the mining industry and are members in good standing of appropriate professional institutions.

Table 1-1: Professional Qualifications of SRK Consulting (UK and Finland) Staff

Name	Professional Qualifications and Affiliations	Discipline and Role
Rick Skelton	Chartered Engineer (CEng), Professional Member of the Institute of Materials, Mining & Metallurgy (MIMMM); Member of Southern African Institute of Mining and Metallurgy (MSAIMM); MSc Mining Engineering; BSc (Hons) Mining; DIC	Project reviewer, Mineral Reserves
Tim McGurk	Chartered Engineer (CEng), Fellow IMMM (FIMMM), BEng (Hons) Mining Engineering,	Project reviewer, SRK Finland signatory
Guy Dishaw	Association of Professional Engineers and Geoscientists of Saskatchewan (P.Geo), Citation Program in Applied Geostatistics (CPAG), BSc. (Hons) Geology.	Project reviewer, Mineral Resources
Ben Lepley	Chartered Geologist of Geological Society of London (CGeol); MESci (Hons) Geology	Project manager, geology and Mineral Resources
Lucy Roberts	Member and Chartered Professional with the Australian Institute of Mining and Metallurgy (MAusIMM(CP)); BSc (Hons) Geology; MSc (Distinction) Mineral Resources; PhD Applied Geostatistics	Geology and Mineral Resources (Competent Person for Mineral Resource statement)
Hanno Buys	Professional Member of the IMMM (MIMMM), Member of Southern African Institute of Mining and Metallurgy (MSAIMM); Professional Engineer with Engineering Council of South Africa (Pr.Eng); MEng (Distinction) Mining Engineering; BEng (Hons) Mining Engineering	Mining engineering and Mineral Reserves (Competent Person for Mineral Reserve statement)
Michael Di Giovinazzo	Member AusIMM (MAusIMM); GCertEng Mining Geomechanics; BSc Applied Geology	Geotechnical engineering
David Pattinson	Chartered Engineer (CEng) and Member of the Institute of Materials, Mining & Metallurgy (MIMMM); BSc (Hons) Minerals Engineering; PhD Minerals Engineering	Processing engineering and metallurgy
William Harding	Member of the National Groundwater Association (UK); Fellow of the Geological Society of London (FGS); MSc Hydrogeology; BSc (Hons) Earth Sciences	Hydrogeology
John Merry	MPhil Environmental Risk; BSc Environmental Science	Environment and social
Jamie Spiers	Chartered Engineer (CEng); Professional Member of the IMMM (MIMMM); MSc DIC Environmental Technology; BSc (Hons) Geology and Physical Geography	Tailings engineering
Inge Moors	MSc Mining Engineering; Professional Member of the AusIMM (MAusIMM)	Financial modelling

1.2 Scope of Work, Materiality, Limitations and Exclusions

1.2.1 General

SRK has independently assessed the supporting data, including that relating to resources, reserves, equipment and manpower requirements, environmental, rehabilitation and abandonment issues and the future plans relating to productivity and production including projected costs and revenues for the Kevitsa mine. All opinions, findings and conclusions expressed in this report are those of SRK.

SRK's opinion contained herein is effective as of 31 December 2019 with regards to the Mineral Resource and Mineral Reserve Statements and the review of the mine plan. In addition, SRK has briefly reviewed the ore production, and where relevant ore processing volumes, updated mine plans and other significant issues that have occurred up to the time of the site visit. SRK's opinion is based on information provided by the Company throughout the course of SRK's investigations, which, in turn, reflects various technical conditions at the time of writing. These conditions can change significantly over relatively short periods of time. The achievability of the technical-economic plans is neither warranted nor guaranteed by SRK.

This report contains technical information which may have been used in subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding which consequently introduces margins of error. Where these occur, SRK does not consider them to be material to the purpose or use of this report.

1.2.2 Compliance and Reporting Standard

The international reporting code used for the reporting of Mineral Resource and Mineral Reserve statements is the Pan-European Standard for Reporting of Exploration Results, Mineral Resources and Reserves ("PERC", or the “PERC Standard”). The PERC Code is a reporting code which has been aligned with the Committee for Mineral Reserves International Reporting Standards (“CRIRSCO”) reporting template. Accordingly, SRK considers the PERC Code to be an internationally recognised reporting standard which is recognised and adopted world-wide for market-related reporting and financial investment.

1.2.3 Limitations

SRK has no reason to believe that any material facts have been withheld and the Company believes it has provided all material information.

The achievability of the projections of technical-economic parameters as included in this Report are neither warranted nor guaranteed by SRK. The projections as presented and discussed herein have been proposed by the Company's management and adjusted where appropriate by SRK and cannot be assured; they are necessarily based on economic assumptions, many of which are beyond the control of the Company. Future cashflows and profits derived from such forecasts are inherently uncertain and actual results may be significantly more or less favourable.

Unless otherwise expressly stated all the opinions and conclusions expressed in this Report are those of SRK.

1.2.4 Reliance on Information

SRK believes that its opinion must be considered as a whole and that selecting portions of the analysis or factors considered by it, without considering all factors and analyses together, could create a misleading view of the process underlying the opinions presented in the Report.

SRK's assessment of Mineral Resources and Mineral Reserves, and technical-economic forecasts, are based on information provided by the Company throughout the course of SRK's investigations, which in turn reflect various technical-economic conditions prevailing at the date of this Report. In particular, the Mineral Resources and Mineral Reserves, and the technical-economic models are based on expectations regarding the commodity prices and exchange rates prevailing at the date of this report. These projections can change significantly over relatively short periods. Should these change materially the projections could be materially different. Furthermore, SRK has no obligation or undertaking to advise any person of any change in circumstances which comes to its attention after the date of this Report or to review, revise or update the Report or opinion.

1.2.5 Declaration

SRK will receive a fee for the preparation of this report in accordance with normal professional consulting practice. This fee is not contingent on the outcome of any applications made by the Company and SRK will receive no other benefit for the preparation of this report. SRK does not have any pecuniary or other interests that could reasonably be regarded as capable of affecting its ability to provide an unbiased opinion in relation to the Mineral Resources and Mineral Reserves, and the projections and assumptions included in the various technical studies completed by the Company, opined upon by SRK and reported herein.

Neither SRK, the SRK professional staff responsible for authoring this Report, nor any Directors of SRK, have at the date of this report, nor have had within the previous two years, any shareholding in the Company, the Assets or advisors of the Company. Consequently SRK, the SRK Competent Persons and the Directors of SRK considers themselves to be independent of the Company.

In this Report, SRK provides assurances to the Board of Directors of the Company that the technical-economic models, including production profiles, operating expenditures and capital expenditures, of the Assets as provided to SRK by the Company and reviewed and where appropriate modified by SRK is reasonable, given the information currently available.

1.2.6 Copyright

Copyright of all text and other matter in this document, including the manner of presentation, is the exclusive property of SRK. It is an offence to publish this document or any part of the document under a different cover, or to reproduce and/or use, without written consent, any technical procedure and/or technique contained in this document. The intellectual property reflected in the contents resides with SRK and shall not be used for any activity that does not involve SRK, without the written consent of SRK.

1.3 Inherent Risks

Mining and processing are carried out in an environment where not all events are predictable. Whilst an effective management team can identify the known risks and take measures to manage and mitigate these risks, there is still the possibility for unexpected and unpredictable events to occur. It is not possible therefore to totally remove all risks or state with certainty that an event that may have a material impact on the operation of a mine will not occur. Similar considerations apply to the marketing of the minerals.

2 RELIANCE ON OTHER EXPERTS

SRK's opinion contained herein is based on information provided by Boliden technical staff throughout the course of the investigations.

SRK has relied upon the technical work of consultants in the project areas in support of this technical report.

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

Aarne Perälä, Chief Mine Geologist at Kevitsa, has reviewed this report.

3 PROJECT DESCRIPTION AND LOCATION

3.1 Location

The location of the Kevitsa Mine is shown in Figure 3-1 and Figure 3-2. It is located some 142 km north-northeast of Rovaniemi, the capital of Finnish Lapland, and approximately 140 km north of the Arctic Circle in the Municipality of Sodankylä. Sodankylä is located approximately 40 km south of the mine by road and the nearest village Petkula is located 8 km west of the property.

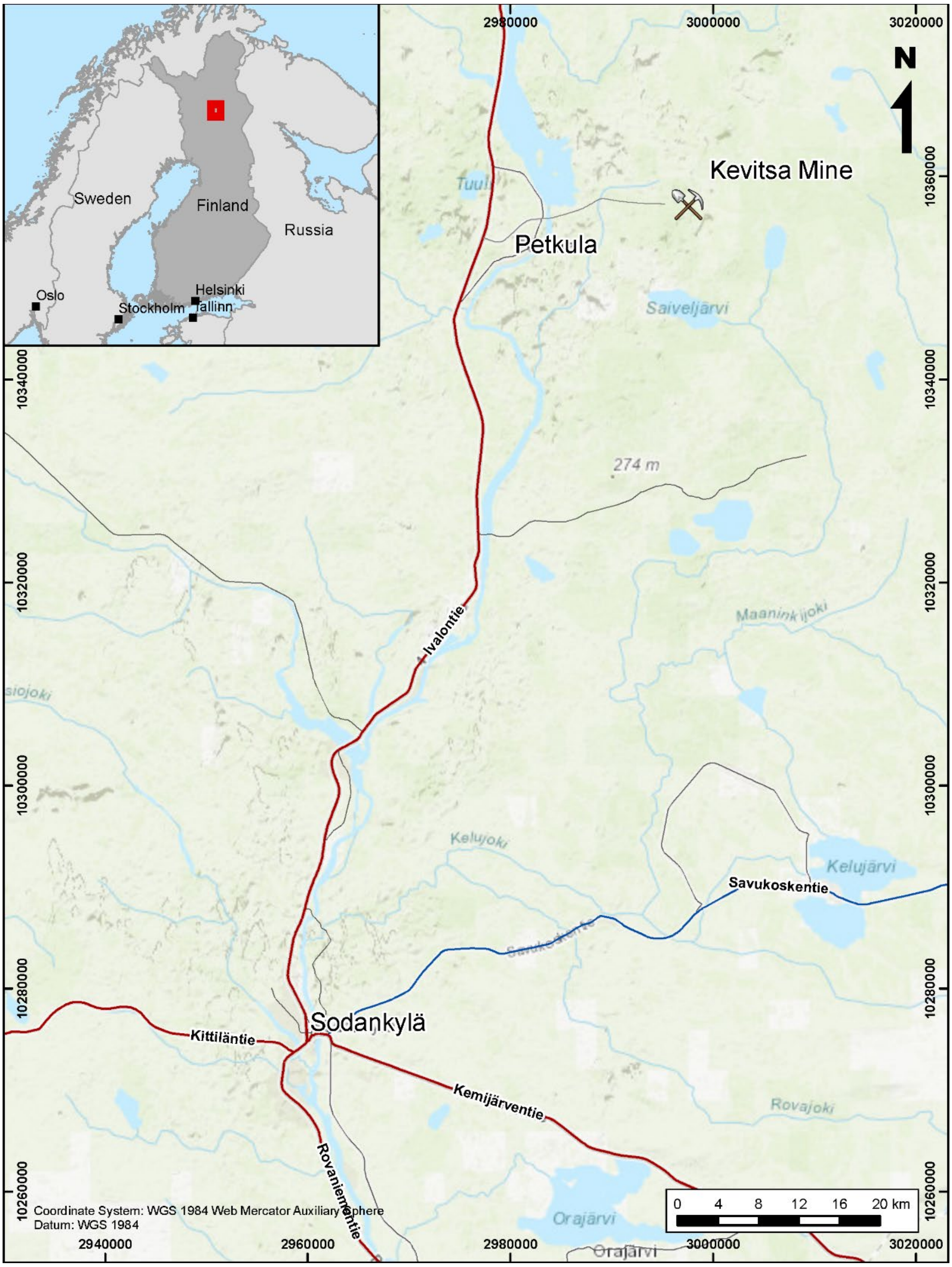


Figure 3-1: Kevitsa Mine location in northern Finland (Source: SRK, 2019)

Figure 3-2: Kevitsa Mine satellite image (Source: SRK, 2019)

3.2 Licences and Permits

The site operating entity is Boliden Kevitsa Mining Oy. The Ministry of Economic Affairs and Employment of Finland originally granted mining concession No. 7140 to FQM Kevitsa Mining Oy (owned by FQM) on 28 September 2009. The Company has also applied for an expansion of the mining concession for the potential requirement of building new infrastructure around the mine area.

Around the mining concession, the Company has nine valid exploration permits granted by Finnish Safety and Chemicals Agency ("TUKES"). Two of those permits are awaiting the three-year validity extension. The Company has also two new exploration permit applications.

A separate exploration company, Boliden FinnEx Oy, operates exploration in these permit areas and holds three valid exploration permits (one is waiting the three-year validity extension). Boliden FinnEx Oy also has three exploration permit applications around the near mine area.

Figure 3-3 shows the valid and pending mining concessions and the surrounding exploration permits.

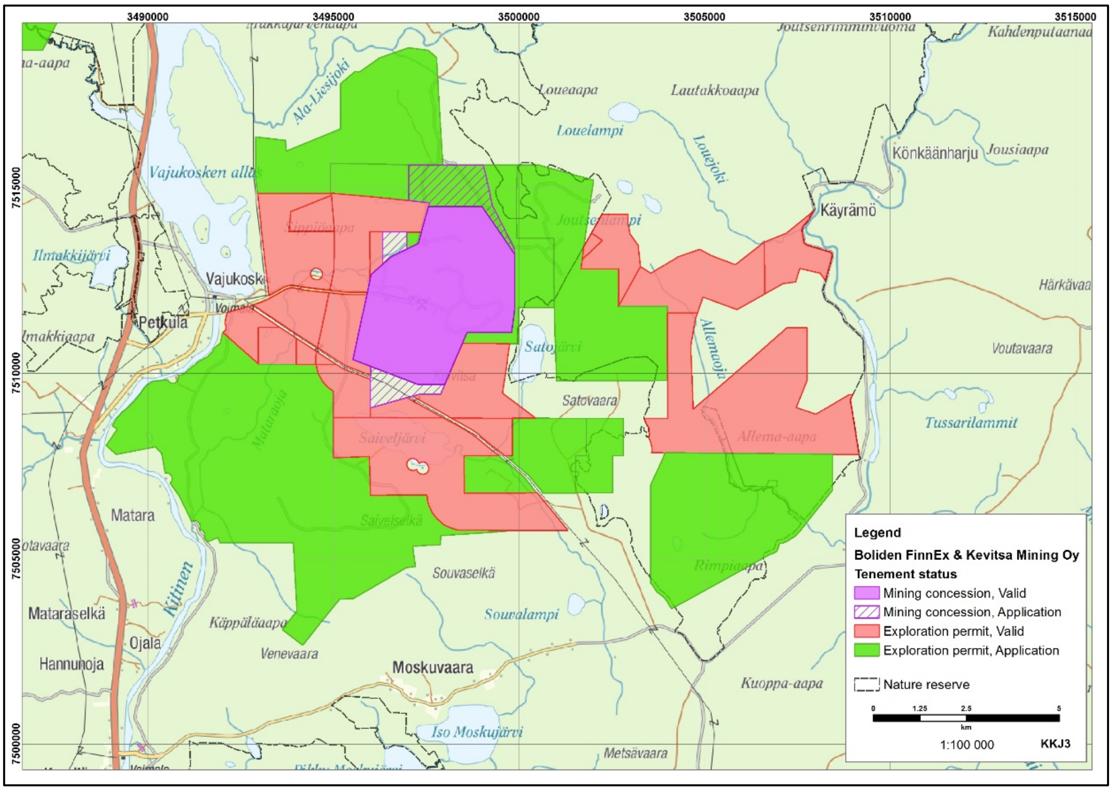


Figure 3-3: Valid Boliden exploration permits and mining concessions around the Kevitsa mine (Source: Boliden, 2019)

3.3 Environmental Permits and Royalties

A number of permits relating to environmental and social monitoring are in place and monitored by Boliden via an online system; more detail is provided in Section 17.3.

In Finland, the mining permit holder pays an annual compensation (excavation fee) to the owners of land included in the mining concession. The Company owns the land inside the mining concession and pays no royalties.

4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

4.1 Physiography

Kevitsa is situated in Finnish Lapland. The area has a gently undulating terrain and is a plateau at between 220 and 240 m above sea level ("masl"), with local hills rising to 350 masl. The Kevitsa deposit is located at the watershed between the streams Mataraoja draining towards northwest and west and Viivajoki draining towards east and southeast. The flat terrain creates extensive areas of bog land alternating with slightly raised terrain with pine forest. The original forest at Kevitsa was cut down decades ago by commercial logging.

Bedrock outcrops on the hills but is generally covered by a 1 to 5 m thin layer of clay and/or sandy till. In boggy land, a 1 to 5 m thick peat layer is developed over the till.

Within the Project area, Kevitsa Hill is the highest point with an elevation 310 masl, the lowest areas being 212 masl, and on western boundary of applied mining concession the average elevation is 230 masl in the main resource area.

The geographic coordinates of the property are 67°41' 51.09″ N, 26°58' 18.35″ E.

4.2 Access

Access to the mine site is via well-maintained all-weather sealed roads.

The main road from Rovaniemi is the E75. It connects to the village of Petkula via a tarmac surfaced local road to the Vajukoski hydropower station and dam. A new road and bridge were constructed to access the mine, which included the construction of two bridges over the Kitinen River and the Mataraoja stream.

Port facilities are available at Kemi Harbour, which is approximately 290 km from the Project by road.

4.3 Climate

According to the Köppen climate classification, Finland, with the exception of the southern coastal region, belongs entirely to the continental subarctic climate zone without dry season and cold summers. The coldest month averages below 0 °C and 1 to 3 months average above 10°C. The climate is typical of northern Fennoscandia with temperate summers and cold winters. During the summer months (June – August) temperatures are mostly between 10°C to 25°C. October to April have negative temperatures, with January being the coldest period averaging at -13.4°C. Snow covers the terrain on an average of 180 days in the year with a maximum snow thickness varying from 0.6 to 1.2 m in March. Bogs, lakes and rivers are frozen for four to five months of the year. Exploration work can be conducted year-round, including the winter by taking advantage of the frozen bogs. The average precipitation is 544 mm per year.

As the Kevitsa area is 140 km north of the Arctic Circle, part of the winter is the period known as the polar night, when the sun does not rise above the horizon. In the northernmost extremity of Finland, the polar night lasts for 51 days. In the Sodankylä area the polar night is four days.

4.4 Infrastructure

All infrastructure required by the mine is in place including sealed roads, power lines and substations, process plant, site offices, workshops, tailings and waste storage facilities. For the purpose of this MRE report, further information regarding infrastructure is considered irrelevant.

4.5 Local Resources

Lapland is relatively sparsely populated; however, Boliden endeavours to employ local personnel and contractors where possible.

5 HISTORY

5.1 Discovery and Exploration

Mafic and ultramafic rocks in the Kevitsa area have been known since early geological observations made by Erkki Mikkola in the 1920s and 1930s. During the late 1960s the area was first covered by airborne surveys as part of the Geological Survey of Finland ("GTK") national mapping campaign. During the early 1970s, Outokumpu initiated exploration areas adjacent to Kevitsansarvi by applying ground electromagnetic and magnetic methods combined with shallow pitting. The first phase of systematic mapping over the larger area, including the Kevitsa igneous complex, was completed in 1980 as part of GTK 1:100 000 geological mapping programs. Systematic and persistent work conducted by GTK, and in particular by Dr Tapani Mutanen, over the main Kevitsa intrusion, led to the discovery of the mineralisation in 1987.

5.2 Ownership

The mine area has been explored by several companies since the GTK surveying and drilling in the 1990s. The following shows the ownership changes of the licence:

- GTK: 1987-1994;
- Outokumpu: 1996-1998;
- Scandinavian Minerals Limited: 2003-2008;
- First Quantum Minerals Ltd: 2008-2016; and
- Boliden: 2016 - present.

5.3 Previous Compliant Mineral Resource Estimates

Previous MRE were produced by Lappalainen & White, 2010, effective December 2010, Gray, Cameron, & Briggs, 2016 effective March 2016 on behalf of FQM and an update in November 2018 (Degen *et al.,* 2018) by Lion GeoConsulting Ltd ("LGC") on behalf of the Company.

The December 2010 MRE was completed by Qualified Persons, Mr Galen White of CSA Global (UK) Limited ("CSA") and Mr Markku Lappalainen of FQM's Kevitsa Mining Oy. At the time of reporting, mining had not yet started at Kevitsa. The Mineral Resource statement was reported in compliance with the reporting requirements of the National Instrument 43-101: 'Standards of Disclosure for Mineral Projects' of the Canadian Securities Administrators and in turn complied with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM Guidelines, 2005). The Mineral Resource statement (Table 5-1) was reported using a (total) nickel cut-off grade of 0.1%.

The March 2016 MRE (note: reporting date 31 December 2015) was completed by Qualified Person, Mr David Gray of FQM Kevitsa Mining Oy, in compliance with CIM and NI 43-101.

Compared to the 2010 estimate, the 2016 MRE increased total available nickel sulphide (“NiS”) metal by 6% and increased total available copper (Cu) metal by 4%. Metal increases resulted from expanded mineralised volumes which have been guided by:

- on-mine reconciliation data of the 23 Mt mined since the 2010 MRE;
- additional reverse circulation (“RC”) drilling, which improved domain delineation and geology detail;
- in-pit mapping with improved geology and deposit understanding;
- updated 3D seismic structural interpretations;
- a comprehensive data set of XRD data supporting mineralisation, lithology and alteration definition; and
- alignment and improvement of the employed estimation methods to improved geology model, mineralisation domains and added data.

In addition, the Mineral Resource statement (Table 5-2) was reported using a nickel sulphide equivalent cut-off grade of 0.22% using Equation 5-1.

Equation 5-1: 2016 Nickel sulphide equivalent cut-off calculation

$$2016\ \mathrm{Ni(S)Eq} = \mathrm{Ni(S)}\ \% + (0.722 * \mathrm{CuS}\ \%) + (0.081 * \mathrm{Pt\ ppm}) + (0.056 * \mathrm{Pd\ ppm}) + (0.219 * \mathrm{Au\ ppm})$$

The November 2018 MRE was completed by Qualified Person, Mr Christian Degen of LGC, in compliance with the PERC Code.

Compared with the 2016 estimate, the 2018 updated Mineral Resource statement (Table 5-3) increased the quantity of Measured and Indicated Mineral Resources by 7%, while the tonnage of the Inferred category has been decreased by 30 Mt (65%). The sum of Measured, Indicated and Inferred Mineral Resources has not changed. This is including a reduction in tonnage due to mining between 31 December 2015 and 30 September 2018 of approximately 22 Mt.

The previous 2016 Mineral Resource estimate was reported as in-situ mineralised tonnes and grades and had not been tested for and reported with “reasonable prospects for eventual economic extraction (“RPEEE”)”. The RPEEE test was conducted as part of the 2018 study and while marginal volumes of mineralisation were excluded as not demonstrating RPEEE (below a conceptual pit shell) the density estimate yielded higher dry tonnages and slightly higher Ni(S) grades. In addition, the 2018 statement was reported using a Ni(S) equivalent cut-off grade of 0.16% using the Equation 5-2:

Equation 5-2: 2018 Nickel sulphide equivalent cut-off calculation

$$2018\ \mathrm{Ni(S)Eq}\ (\%) = \mathrm{Ni(S)}\ (\%) + 0.60\ \mathrm{Cu}\ (\%)$$

Table 5-1: Kevitsa Mineral Resource statement effective 21 October 2010

Mineral Resource Category	Tonnes (Mt)	Sulphide Nickel (%)	Total Copper (%)	Gold (g/t)	Platinum (g/t)	Palladium (g/t)
Measured	89.3	0.26	0.40	0.12	0.23	0.17
Indicated	150.8	0.29	0.42	0.11	0.19	0.14
Meas+Ind	**240.1**	**0.28**	**0.41**	**0.11**	**0.21**	**0.15**
Inferred	34.7	0.27	0.36	0.09	0.14	0.10

Table 5-2: Kevitsa Mineral Resource statement effective 31 December 2015

Mineral Resource Category	Tonnes (Mt)	Sulphide Nickel (%)	Total Copper (%)	Gold (g/t)	Platinum (g/t)	Palladium (g/t)
Measured	108.7	0.23	0.34	0.10	0.21	0.13
Indicated	167.0	0.24	0.34	0.09	0.16	0.11
Meas+Ind	**275.7**	**0.23**	**0.34**	**0.09**	**0.18**	**0.12**
Inferred	57.1	0.20	0.31	0.06	0.12	0.07

Table 5-3: Kevitsa Mineral Resource statement effective 30 September 2018

Mineral Resource Category	Tonnes (Mt)	Sulphide Nickel (%)	Total Copper (%)	Gold (g/t)	Platinum (g/t)	Palladium (g/t)
Measured	99.3	0.24	0.34	0.10	0.20	0.13
Indicated	195.2	0.25	0.33	0.09	0.19	0.12
Meas+Ind	**294.5**	**0.24**	**0.34**	**0.10**	**0.19**	**0.12**
Inferred	20.2	0.22	0.32	0.06	0.13	0.09

5.4 Previous Compliant Mineral Reserve Estimates

The most recent Mineral Reserve estimate was completed by FQM effective as of December 2015 (Gray *et al.*, 2016) based on the 2016 MRE block model, which is provided in Table 5-4. Since this estimate, the annual Mineral Reserve statements for 2016, 2017 and 2018 have been based on the 2016 MRE block model. The 2015 statement, produced by Tony Cameron of FQM, was based on a 0.22% Ni(S)Eq cut-off grade (2016 Ni(S)Eq calculation, as above) and within the 2016 designed final pit shell.

Table 5-4: Kevitsa Mineral Reserve statement effective 31 December 2015

Mineral Resource Category	Tonnes (Mt)	Sulphide Nickel (%)	Total Copper (%)	Gold (g/t)	Platinum (g/t)	Palladium (g/t)
Proved	88.2	0.23	0.36	0.11	0.22	0.14
Probable	66.3	0.25	0.36	0.11	0.22	0.15
Prov+Prob	**154.5**	**0.24**	**0.36**	**0.11**	**0.22**	**0.15**

The latest previous Mineral Reserve statement dated 31 December 2018 is provided in Table 5-5. The changes between the 2015 and 2018 statements are principally due to mine depletion but were also impacted by a re-design of the final pit and changes to the NSR calculation. The 2018 statement, produced by Naomi Fogden of Optiro Pt Pld ("Optiro"), was based on a net smelter return ("NSR) in Euro/tonne cut-off (details below) and within the 2018 designed final pit shell.

Two NSR calculations were used to report the 2018 statement due to differences in the prices used for different years. For blocks in the model due to be mined in 2019 and 2020, an NSR cut-off of EUR 16 / t according to Equation 5-3 (further details provided in Section 14.3; note: CoS was not reported in 2018):

Equation 5-3: NSR for blocks planned to be mined in 2019

$$\text{NSR\_BUD} = (72.11 \times \text{NiS}) + (38.83 \times \text{Cu}) + (7.96 \times \text{Pt}) + (12.64 \times \text{Pd}) + (12.51 \times \text{Au}) + (44.93 \times \text{CoS})$$

Beyond 2020 for the remainder of the LoMP, the NSR cut-off based on long-term forecast metal prices is shown in Equation 5-4.

Equation 5-4: NSR for blocks planned to be mined from 2020 onwards

$$\text{NSR\_LTP} = (64.47 \times \text{NiS}) + (43.83 \times \text{Cu}) + (6.80 \times \text{Pt}) + (9.18 \times \text{Pd}) + (8.97 \times \text{Au}) + (68.32 \times \text{CoS})$$

Table 5-5: Kevitsa Mineral Reserve statement effective 31 December 2018

Mineral Resource Category	Tonnes (Mt)	Sulphide Nickel (%)	Total Copper (%)	Gold (g/t)	Platinum (g/t)	Palladium (g/t)
Proved	62.5	0.21	0.35	0.09	0.18	0.12
Probable	66.1	0.24	0.34	0.10	0.21	0.14
Prov+Prob	**128.6**	**0.22**	**0.34**	**0.09**	**0.19**	**0.13**

5.5 Mine Production

Mining movement has increased each year since commencement of operations in 2012. The total 2019 ore and waste production was 40 Mt, including 7.5 Mt of ore (Table 5-6).

Table 5-6: Mine production 2012-2019

Production	2012	2013	2014	2015	2016	2017	2018	2019	Total
Ore (Mt)	3.4	5.8	6.9	6.6	7.7	8.3	7.9	7.5	54.1
Ore Ni(S)%	0.22	0.22	0.23	0.20	0.22	0.25	0.26	0.19	0.23
Ore Cu%	0.31	0.28	0.30	0.29	0.31	0.42	0.39	0.30	0.33
Waste (Mt)	4.2	16.0	21.2	30.4	31.9	34.2	33.5	33.5	204.9
Total (Mt)	**7.6**	**21.8**	**28.1**	**37.0**	**39.6**	**42.5**	**41.4**	**40.0**	**258.0**

6 GEOLOGY

6.1 Introduction

The description of the geological setting and mineralisation are largely reproduced from Lappalainen and White (2010).

6.2 Regional Geology

The Kevitsa igneous complex lies within the Central Lapland Greenstone Belt ("CLGB"), a large area of volcanic and sedimentary rocks of Palaeoproterozoic age located within the Precambrian Fennoscandian Shield. The CLGB is divided into a number of volcano-sedimentary associations or stratigraphic groups (Räsänen *et al*., 1996), from oldest to youngest these are: Salla, Onkamo, Sodankylä, Savukoski, Kittilä, Lainio, and Kumpu Groups, with Kevitsa hosted within the Savukoski Group. Räsänen *et al*. (1996) stated these rocks are polyfolded and thrusted resulting in overturning and structural repetition of the stratigraphy. A regional geology map is provided in Figure 6-1.

The Salla Group is overlain by siliceous high-Mg basalts and mafic volcanic of the Onkamo Group. Volcanism was followed by widespread deposition of quartz-rich epiclastic and carbonate rocks of the Sodankylä Group. Age-dating of quartzite at several localities within the CLGB has failed to return any Proterozoic ages, instead an Archaean granitic source region for this sedimentary sequence is inferred.

The Savukoski Group represents a major marine transgression dominated by phyllites and carbonaceous schists which host the Kevitsa ultramafic complex. The only published age constraint on the Savukoski Group is a 'diorite' dyke at Kevitsa that describes very similar to the common rodingite veins and gives a U-Pb zircon age of 2054±5 Ma. Mutanen (2005) reports an unpublished age of 2.15 Ga for the schists in the Kevitsa-Satovaara area and these are considered a maximum age for the area. The contact between the Savukoski Group and the overlying mafic volcanic rocks of the Kittilä Group is described mainly as being allochthonous. Younger quartzites and conglomerates of the Lainio and Kumpu Groups unconformably overlie the aforementioned sequences and have a maximum age of 1.88 Ga.

Structural and metamorphic work in the CLGB has been described by Hölttä *et al.* (2007). The earliest recognized tectono-metamorphic event (D1) is characterized by bedding-parallel foliation preserved in F2 hinges and as inclusion trails within various porphyroblastic minerals. The most prominent structural feature in CLGB rocks is an S2 foliation that is axial planar to F2 folds and in most cases is sub-parallel with bedding. F2 folds are tight to isoclinal, recumbent or reclined folds, and inferred to result from northward directed thrust deformation (D2) that is opposite, yet sympathetic, with north-south-directed shortening associated with thrusting of the granulite allochthon over the CLGB (as proposed by Ward *et al*., 1989). The third phase of deformation D3 is characterized by E-W and N-S oriented F3 folds with axial planes ranging from horizontal to vertical, and shear zones of various orientations. Tectonic movements are considered complex and involve rotations between subjacent shear zones.

The level of metamorphism in the CLGB is based on observed mineral assemblages in mafic and pelitic rocks. The pattern of metamorphic zonation is complex due to interference between thermal aureoles associated with granite plutons in the west and south against a southwestwardly decreasing metamorphic gradient from two-pyroxene granulite to mid-upper greenschist (Hölttä *et al*., 2007).

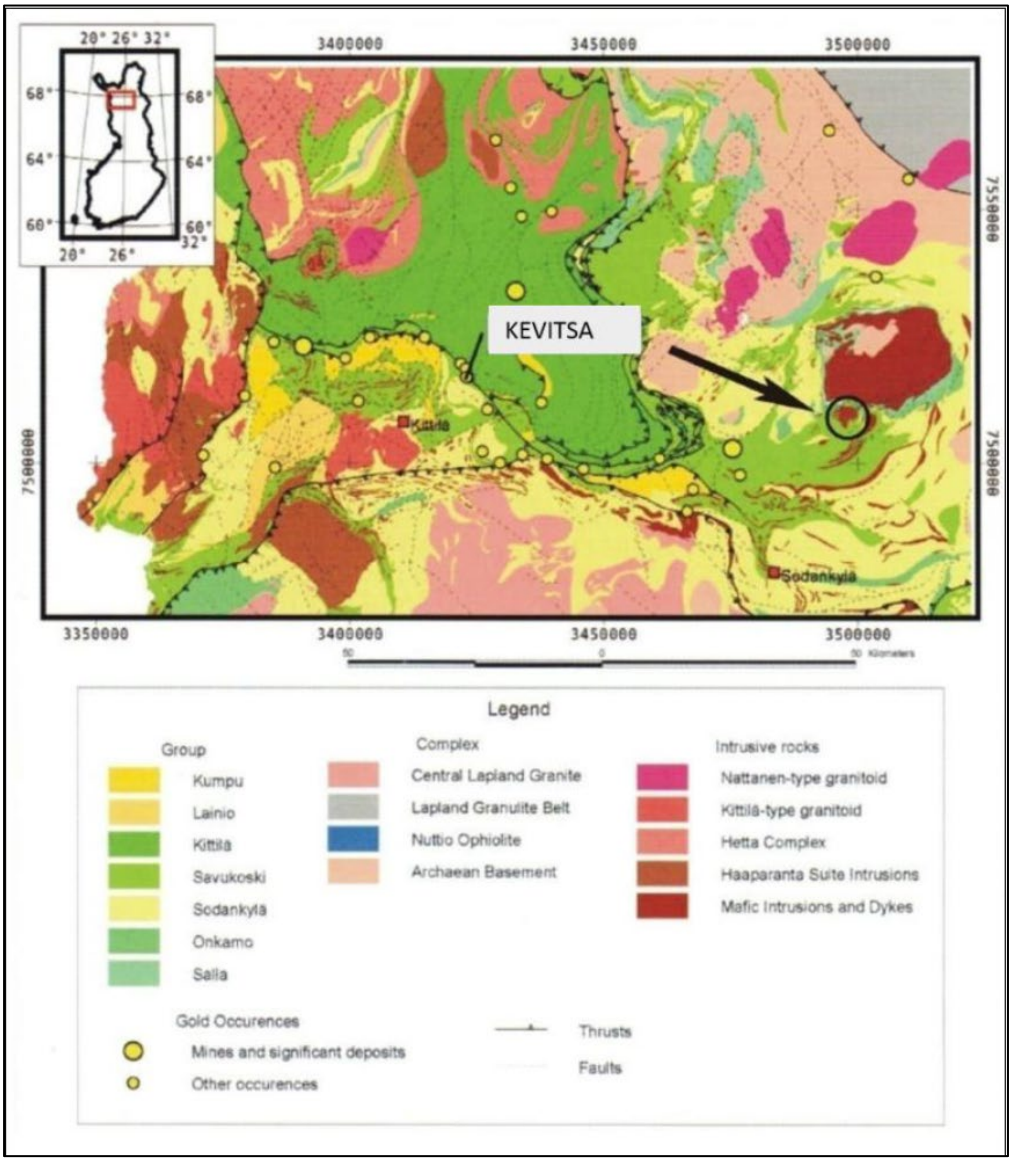


Figure 6-1: Regional geology map highlighting the position of Kevitsa igneous complex in relation to the Central Lapland Greenstone Belt geology (Source: Hölttä, *et al.*, 2007)

6.3 Property Scale Geology

The property geology is dominated by ultramafic and mafic intrusive rocks belonging to the Kevitsa intrusive complex enveloped by above described supracrustal rocks belonging to the Savukoski Group. A map of the local geology is shown in Figure 6-2 with the final planned pit limit shown in red. The local stratigraphic column is presented in Figure 6-3. The Kevitsa intrusive complex has pristine undeformed margins along its southern and northern contacts; however, its eastern margin has suffered significant structural modification by the Satovaara Fault. The nearby Kevitsa deposit is overprinted by many structures developed in association with the evolution of the Satovaara Fault.

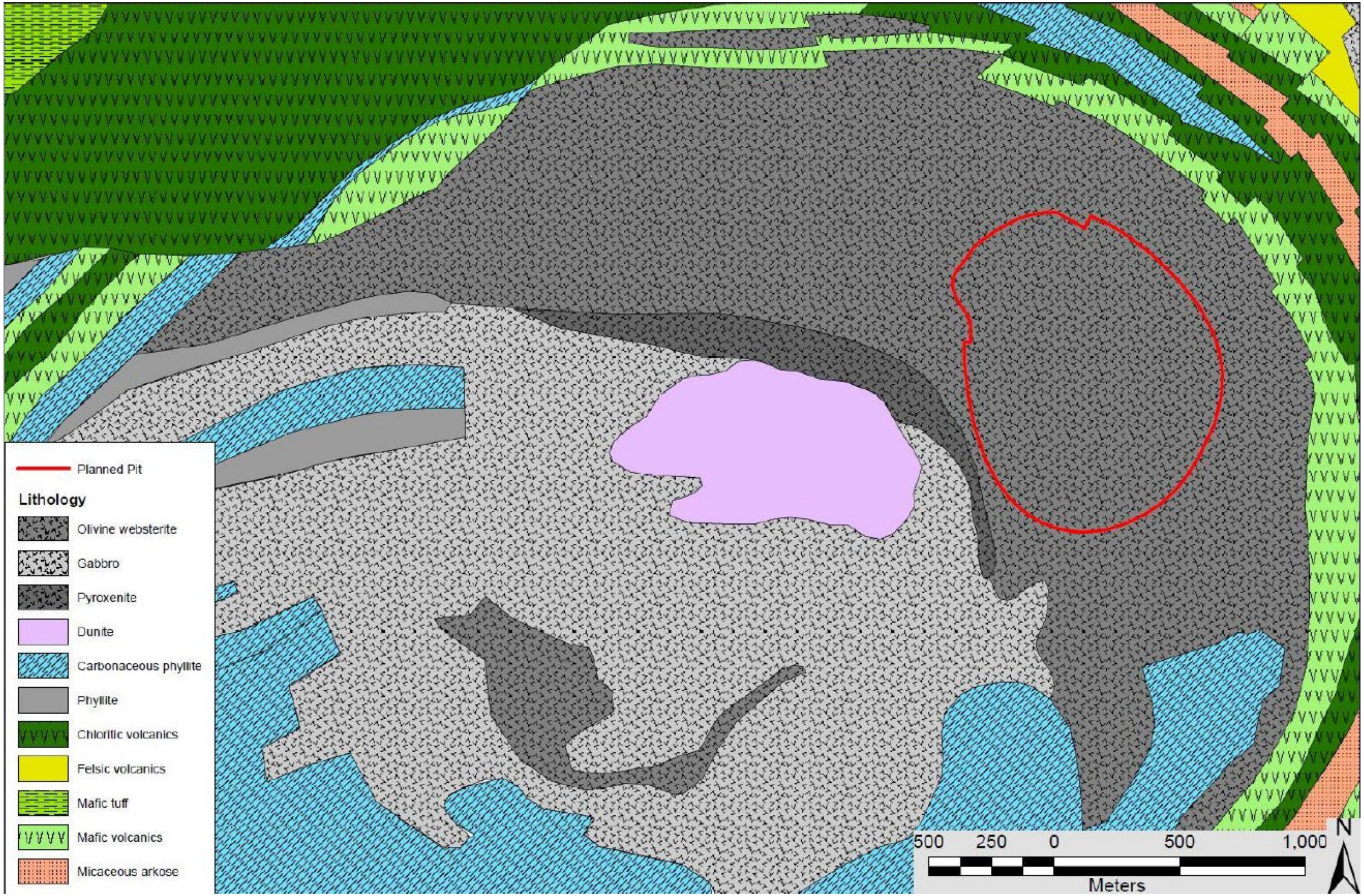


Figure 6-2: Local geology map (Source: Gray, *et al.*, 2016)

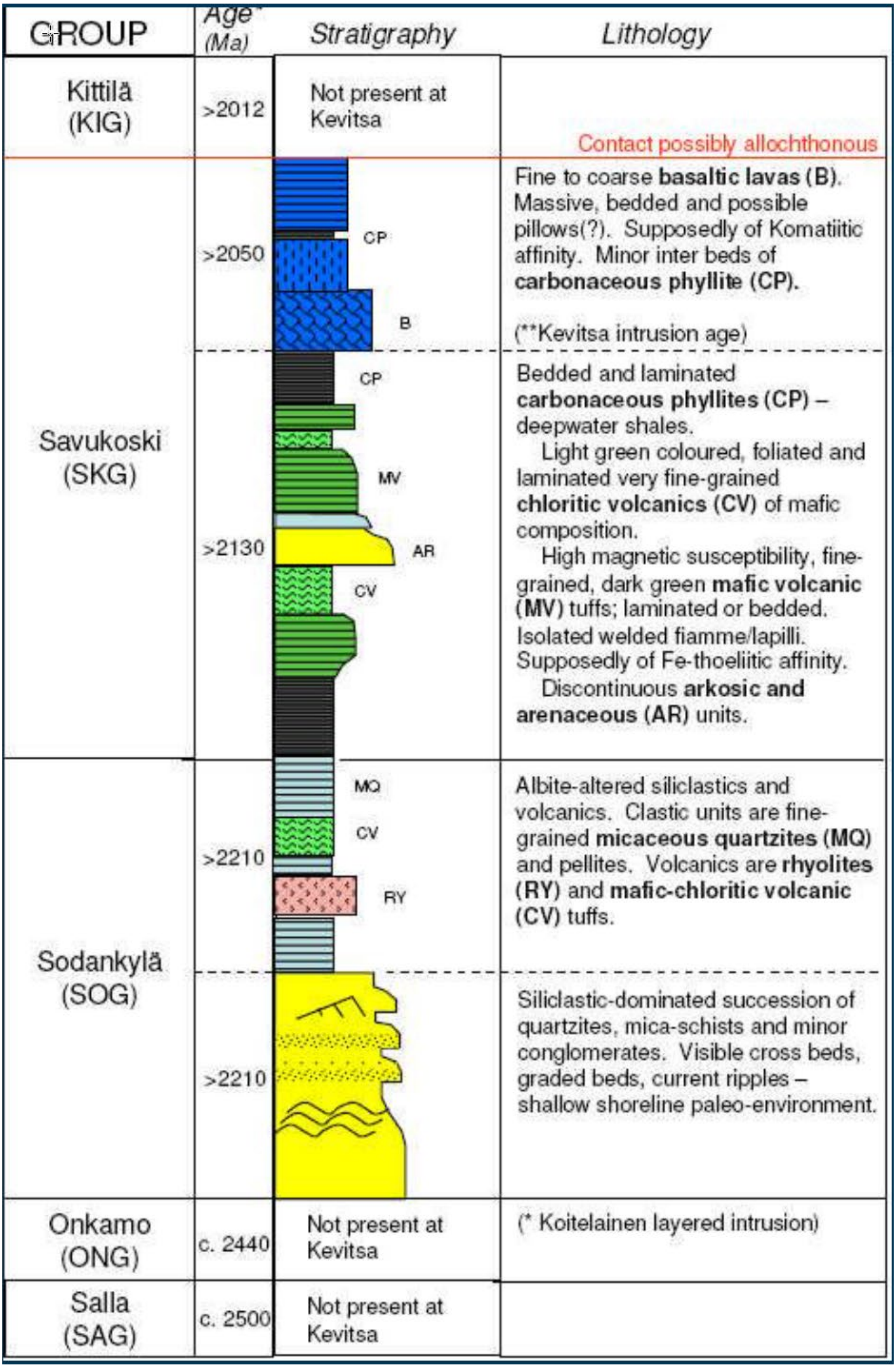


Figure 6-3: Stratigraphy around Kevitsa Igneous Complex (Source: Luolavirta, 2017)

6.4 Deposit Geology

Luolavirta (2018) contributes the construction of a geologic model for the origin of Kevitsa intrusive suite rocks, which is summarised in Figure 7-2. The model proposes a complex multi-stage magmatic evolution for the intrusion. At stage 1, olivine-chromite cumulates (Central Dunite) accumulated in a picritic magma conduit and were followed by intrusions of more evolved basaltic magma crystallizing olivine-pyroxene cumulates and enclosing rafts of stage 1 dunitic cumulates and country rock xenoliths (stages 2 and 3). The contrasting intrusive stratigraphy obtained from the Kevitsa mineralisation and the surrounding part of the intrusion is interpreted to reflect different emplacement histories. It is proposed that the Kevitsa magma chamber was initially filled by stable continuous flow ("single" input) of compositionally homogeneous basaltic magma followed by crystal fractionation in an at least nearly closed system (stage 2). At this stage, some sulphides precipitated at depth in the magmatic system, resulting in metal-poor magma precipitating false mineralised bodies in the Kevitsa magma chamber. At the following stage (stage 3), magmas were repeatedly emplaced into the hot interior of the intrusion in a dynamic (open) system, forming the inter-cumulus sulphide mineralised bodies. The formation of the Ni-Cu mineralised bodies may involve assimilation of proto-ores formed at stage 2.

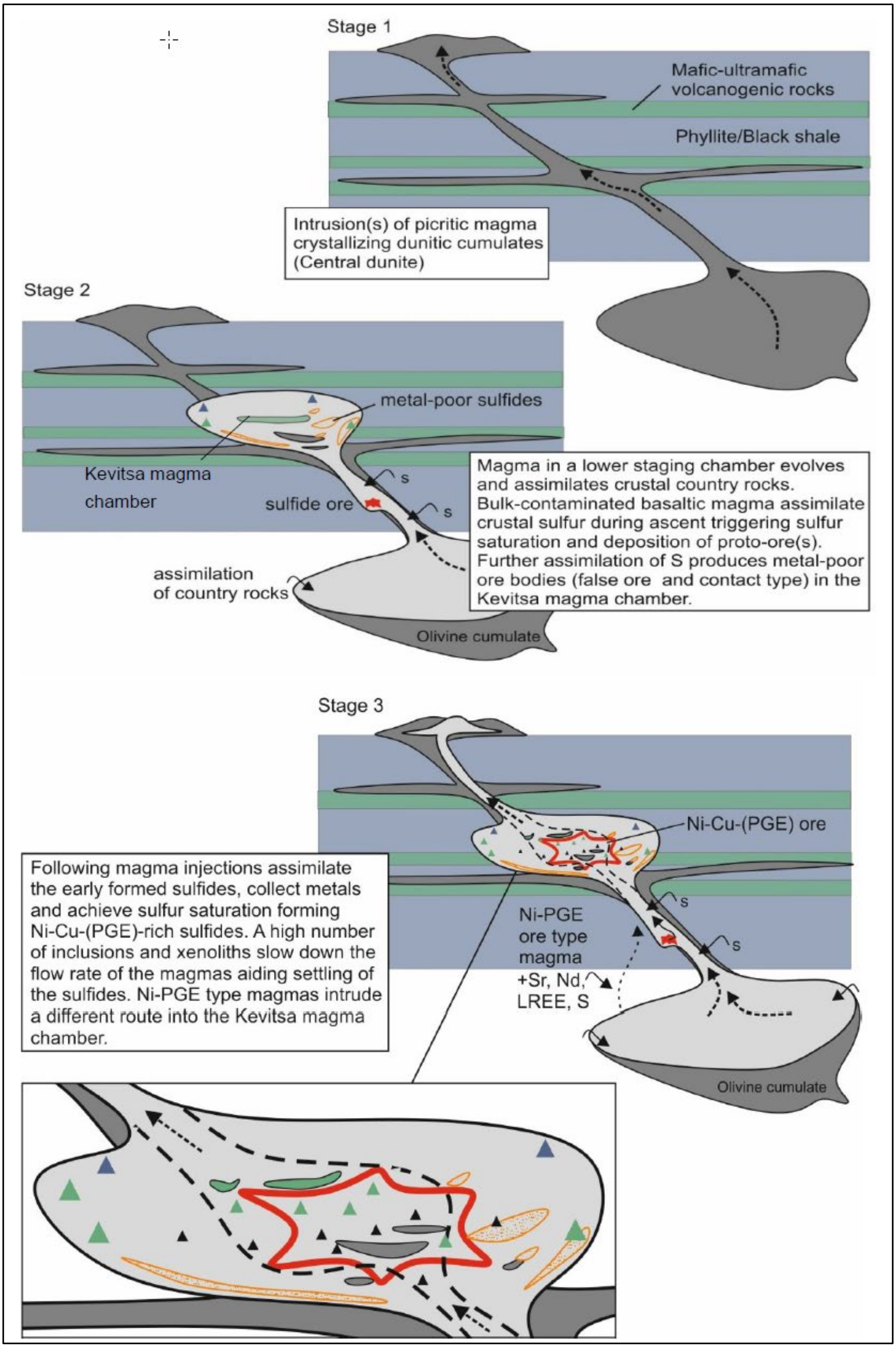


Figure 6-4: Schematic illustration of the emplacement of the Kevitsa intrusive suite rocks and formation of the Ni-Cu PGE deposit (Source: Luolavirta, 2018)

6.5 Rock Types

This chapter is reproduced from Santaguida *et al*. (2015).

The Kevitsa intrusion consists of an ultramafic lower part (approximately 1 km thick) overlain by gabbroic rocks. A large lherzolite body occurs in the central part of the intrusion but is not spatially associated with the mineral deposit. Compositional variations within the lower ultramafic portion are minor, but discrete lithological units can nevertheless be mapped. Layering is locally developed, particularly within the deposit, but in general, the contacts between rock types are diffuse. Alteration of pyroxene and olivine is intense in places, making primary rock types difficult to recognize and further complicating stratigraphic correlation.

Olivine websterite is the dominant rock type and host rock for the sulphide mineralisation, defined locally as containing more than 5% orthopyroxene. Olivine occurs as discrete grains or clusters a few millimetres in size. The rock has a poikilitic (heteradcumulate) texture, with orthopyroxene-forming oikocrysts. Typical accessory minerals include plagioclase, magnetite, sulphides, and apatite. Hornblende and phlogopite also occur locally.

Olivine pyroxenite resembles the olivine websterite in terms of texture but is devoid of orthopyroxene (<5%). Because pyroxene is susceptible to overprinting by amphibole, it is difficult to distinguish these two rock types; however, the **olivine clinopyroxenites** can be clearly identified in thin section and, in general, are more prevalent outside of the mineralisation.

Plagioclase-bearing (olivine) websterite occurs as discontinuous zones within the olivine websterite/clinopyroxenite. The plagioclase-bearing (ol) websterites show orthocumulate textures and contain visible plagioclase (>10%) as an intercumulus phase. Orthopyroxene oikocrysts are also more abundant (15–25%) than in typical olivine websterite. Olivine is absent or rare (<15%). Contacts with the olivine websterite/clinopyroxenite are mainly diffuse but can be locally quite sharp. In places, the plagioclase-bearing (olivine) websterite forms marker horizons characterized by magmatic layering, but in most cases, the layers are discontinuous and cannot be traced beyond a few hundred meters. Overall, these rocks are weakly mineralised and not found outside of the mineralised area.

Pyroxenite, with <5% olivine, forms the uppermost ultramafic cumulate unit below the gabbroic rocks, outside the mineralised area. The transition between olivine websterite and pyroxenite is highly gradational.

The **marginal rocks** of the intrusion are composed of pyroxenite (± minor olivine) and gabbro. Mutanen (1997) considered these rocks to be “microgabbros.” The contact between olivine websterite and the marginal rocks is gradational. In places, distinct layering or banding between pyroxene-rich and plagioclase-rich rocks is seen, but most commonly the marginal rocks are varitextured. The marginal rocks vary in thickness from a few meters to more than 50 m. In places, the marginal rocks are absent and faulting is inferred. Where the marginal rocks are sulphide mineralised, they form so-called ‘contact ore’, dominated by pyrrhotite, and thus are uneconomic. Fragments of country rock are also common within the marginal sequence. The immediate country rocks to the intrusion consist of mafic volcanic flows and epiclastic rocks, as well as micaceous phyllites and carbonaceous schists. In many places, the contact is sharp and intact, although faulting is prevalent at the southern margin of the intrusion.

The intrusion contains a number of distinct olivine-rich bodies and lenses that contain >50% olivine. They are of lherzolitic to wehrlitic composition but have been collectively termed **dunite** in Kevitsa mine terminology. The rocks are intensely serpentinised, particularly near the surface. A large lherzolite body occurs in the central portion of the intrusion but shows no spatial relationship with the mineralisation. Lherzolite has also been intersected by drilling below the deposit; whether this lherzolite is related to the previously mentioned lherzolite body is currently unknown. Lherzolite clasts occur throughout the mineralised zone, but their origin remains contentious (Mutanen, 1997; Yang *et al*., 2013). The clasts are highly variable in size, ranging from centimetres to traceable zones roughly tens of meters in thickness. Cumulate texture of olivine is locally preserved, although most clasts are foliated along serpentinised planes. The clasts may occur as discrete, rounded fragments, or, in places, lherzolite is intermingled with olivine websterite. Pyrrhotite is common within lherzolite, whereas pentlandite and chalcopyrite only occur locally. In general, the lherzolite clasts host the same sulphide assemblage as their surrounding olivine websterite/pyroxenite.

Gabbroic rocks occur on top of the ultramafic cumulates. They are particularly prominent in the southwestern portion of the intrusion. Plagioclase is the dominant mineral along with clinopyroxene and accessory olivine. Modally, the rocks are gabbros, olivine gabbros, and gabbronorites. Apatite, magnetite, and ilmenite are common accessory phases. Magnetite-rich hornblende gabbro is prominent along the southern portion of the intrusion. Drilling has shown that the gabbroic rocks form a relatively thin unit (<500 m) overlying the thick ultramafic portion of the intrusion. Overall, sulphide minerals are rare and consist mostly of pyrite.

Xenoliths of hornfelsed pelitic sediments and mafic volcanics are common throughout the intrusion but are particularly concentrated within the deposit area where they are spatially associated with lherzolite clasts. Xenoliths are concentrated in discrete zones that measure several meters in thickness and extend for several hundreds of meters in a north–south direction. Most xenoliths are pervasively altered to phlogopite.

Numerous **dykes** crosscut the intrusion and the mineralisation. Most are olivine gabbroic in composition. The coarse-grained dykes rarely contain sulphide minerals, although veins containing pyrrhotite-chalcopyrite ± pentlandite may form locally along the margins. Fine-grained gabbroic dykes cut the mineralised ultramafic rocks and often contain pyrrhotite-chalcopyrite ± pentlandite. These are altered to a chlorite-actinolite-magnetite assemblage. Felsic dykes consisting of feldspar, quartz, and minor amounts of mafic minerals also occur. Dykes are rarely traceable beyond a single drillhole, thus their orientations are not established.

Granophyre occurring along the southern margin of the intrusion has been described by Mutanen (1997). Despite extensive drilling, these rocks have not been encountered in the present exploration and mining operation. Instead, several tens of metres of albitised gabbroic rocks and dykes have been intersected in some drillholes along the southern margin of the intrusion. These rocks do not host mineralisation and so they have not been intensely studied nor are they shown on the most current geological maps.

A schematic rock type column after Luolavirta (2017) is given in Figure 7-3.

Lithogeochemistry has been useful for discriminating between rock types in the Kevitsa intrusion. Multi-element ICP analyses have been completed on selected drillholes to improve correlation within the lithological units. Specifically, Al/Cr ratios have been found to be reliable proxies to the presence of plagioclase even where alteration is prominent, allowing plagioclase-bearing (ol) websterites to be more confidently recognized. Olivine-rich rocks have an inherently low Al/Cr ratio compared to pyroxenite. Notably, high Cr typically reflects the presence of clinopyroxene, although Cr may locally be hosted by magnetite. High Al/Cr is characteristic of fine-grained gabbroic dykes, and these are easily recognized during logging. Magnesium correlates well with Cr and is low within the plagioclase-bearing rocks, corresponding to reduced olivine content rather than alteration.

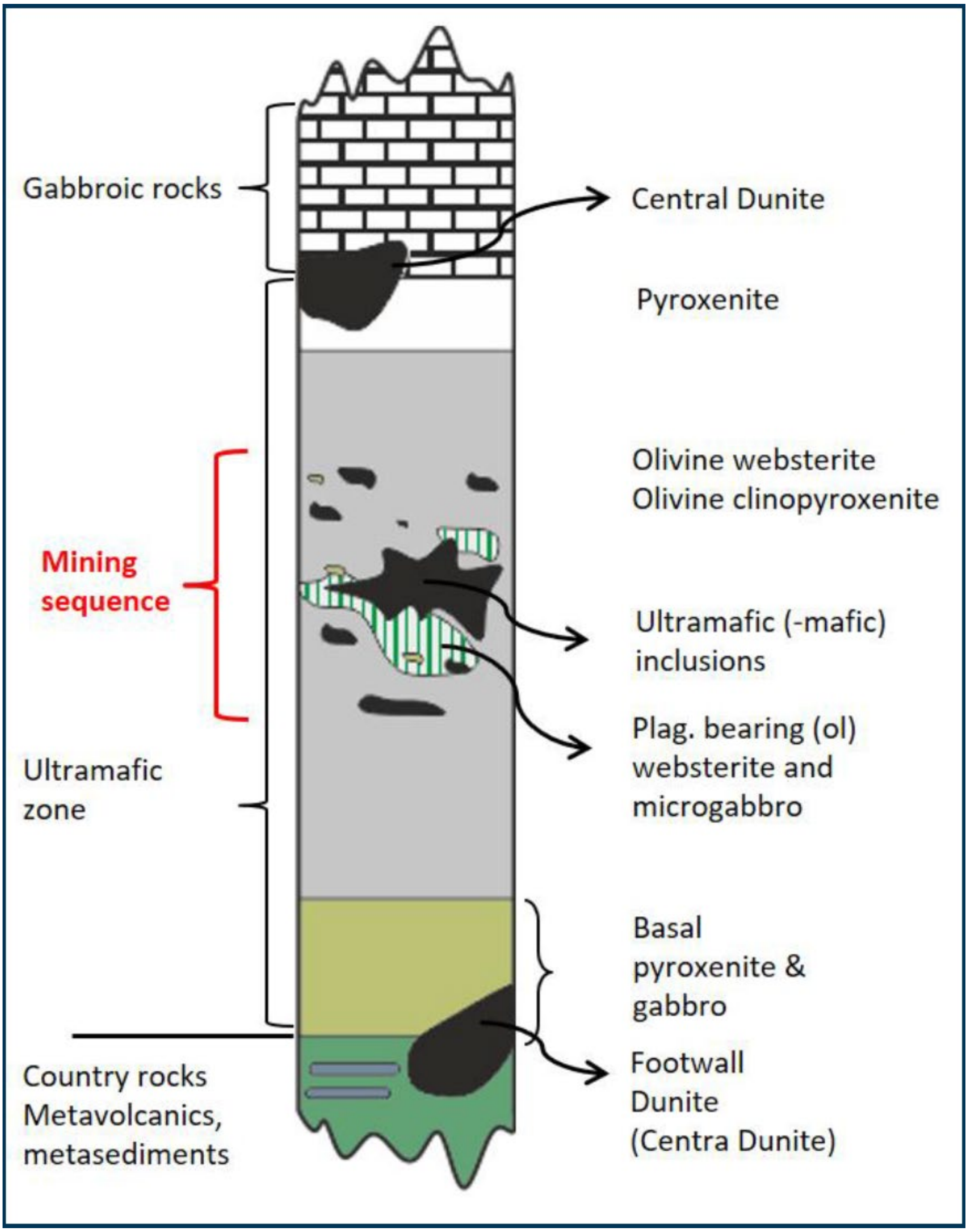


Figure 6-5: Schematic stratigraphic column of rock types (Source: after Luolavirta (2017))

6.6 Alteration

The orthopyroxene is usually the first mineral to be affected by amphibole alteration, meaning its colour is lost, but the crystal habit of the grain is often retained. The recognition of plagioclase is also important in understanding the stratigraphy, perhaps more so than orthopyroxene, because it is retained through weak to moderate alteration. It displays a fine grain size (1 to 2 mm), is white and usually present in anhedral-subhedral, intercumulus masses between pyroxene species and olivines. Phlogopite is present in trace amounts through most olivine pyroxenites at Kevitsa and is easily identifiable by its dark brown colour and coarse book-like habit. The phlogopite is believed to be a primary magmatic phase, with the potassium being derived from magma consumption of sedimentary xenoliths. Phlogopite is particularly common in the upper parts of the sequence, in proximity to a sedimentary or mixed xenolith zone.

Amphibole alteration of ferromagnesian minerals, olivine, orthopyroxene, clinopyroxene, is the most recognisable and widespread of the alteration styles within the Kevitsa area. Previously this alteration type was described by Mutanen (1997) and traditionally rock, which has gone through pervasive amphibole alteration, has been called as “Metaperidotite”. The alteration is usually pervasive and does not often “grade out”, instead having relatively sharp boundaries. These pervasively altered zones are often associated with millimetre to metre scale carbonate or carbonate-quartz veining. Carbonate alteration (calcite, dolomite) commonly accompanies amphibole in the selvedges of such veins. Rocks partially affected by the amphibole alteration tend to have a patchy or blotchy amphibole development rather than light, pervasive alteration.

Other typical alteration styles are **serpentine alteration** where olivine is replaced by dark green serpentine and appears to be the first alteration phenomenon. **Epidote alteration** is observed with rodingite (metasomatic) dykes at depth and seems to be linked to early serpentine alteration. Initially **magnetite** crystallized from magma throughout the olivine pyroxenite portion of the Kevitsa intrusion, but is also associated with various alteration, veining and metamorphic events. It is certainly present in the early serpentine veining and has possibly been upgraded in pervasively serpentinised rocks. Magnetite is also present in many carbonate±quartz veins associated with amphibole alteration, but not usually significant in the amphibole alteration itself. Magnetite is destroyed by the early silica-epidote alteration style. **Actinolite-chlorite alteration** is dominantly associated with structural features. Beyond the green actinolite it is common to see a broader, light green-grey amphibole alteration halo. Narrow actinolitic selvedges are also common on carbonate±quartz vein margins, but these wider, very green actinolite features are a distinctive vein set. **Talc and carbonate alteration** is also associated with late fractures and veins and indicates the presence of a CO_2-bearing fluid. The habit of this alteration style can range from selective replacement of ferromagnesian species to pervasive alteration of the rock. The most notable occurrence of this alteration style is at depth beneath the Kevitsa deposit, within and around a flat-lying shear zone and composite quartz-carbonate reef.

6.7 Mineralogy

Known mineralisation is disseminated in style, while having some minor massive sulphide veins. Mineralogically, pyrrhotite (iron sulphide; FeS) is the main sulphide mineral followed by chalcopyrite (copper-iron sulphide; $CuFeS_2$) and pentlandite (iron-nickel sulphide; $(Fe,Ni)_9S_8$). Sulphide grain size is typically fine to medium. In near surface parts of the mineralisation, pyrrhotite is partly replaced by pyrite and pentlandite replaced by millerite (nickel sulphide) and heazlewoodite (nickel sulphide) (Kojonen *et al.*, 2008).

According to the magnetite suspension staining tests and microprobe analyses, most of the pyrrhotite is hexagonal type or troilite, which are both non-magnetic. Pyrrhotite has on average 0.17% Ni in its lattice. Chalcopyrite occurs in the intercumulus sulphides as large anhedral grains, sometimes with cubanite, and as fine intergrowths within the gangue silicates. Pentlandite has coarse grained sub-euhedral grains, smaller intergranular grain bands between silicates and pyrrhotite, and “exolution flame” inclusions within pyrrhotite of very fine grain size (Kojonen & Laukkanen, 2004).

The sulphide grain aggregates are mostly interstitial to silicates. The sulphide/silicate grain boundaries are plain and smooth in unaltered rocks but irregular and serrated in amphibole altered rocks. Elevated grades of palladium (Pd) and platinum (Pt) are correlated to a specific Ni mineralisation phase which has significantly lower sulphur content compared to the majority of the mineralisation.

The melonite (nickel telluride; $NiTe_2$) contains a varying amount of Pd (0-15%) and Pt (0-15%) and it has a complete solid solution series to merenskyite (palladium telluride; $PdTe_2$), michenerite (palladium telluride-bismuthide; PdTeBi) and moncheite (palladium telluride; $(Pt,Pd)(Te,Bi)_2$), which are also common in the Kevitsa samples. Over half (54%) of PGE carrying minerals are as inclusions in amphibole, serpentine and chlorite, implying that PGE were mobile during the amphibole alteration event. Le Vaillant *et al*. (2016) studied the effects of hydrothermal alteration on the distribution of base and precious metals within the Kevitsa deposit and, in contrast to Gervilla & Kojonen (2002), argued that no significant mobilisation of Ni or PGE has occurred but Cu (and Au) may have been mobile. Although Le Vaillant *et al*. (2016) observed some decoupling between Pt, Pd and I-PGE (iridium-group platinum-group elements) in the Ni-PGE mineralisation type, which could be attributed to hydrothermal alteration and addition of Pd and Pt to the mineralisation type, the undisturbed magmatic correlation between Pt and Pd argue against any large-scale redistribution.’’ (Luolavirta (2018).

PGE carrying minerals which are related to sulphides occur mostly on sulphide grain boundaries (38%), inclusions in sulphides (6%) or in late fracture fillings in pentlandite (2%) (Kojonen *et al.*, 2008).

An example of ‘Normal Ore’ low-grade mineralisation is shown in Figure 6-6. Examples of high-grade mineralisation are shown in Figure 6-7 and Figure 6-8.

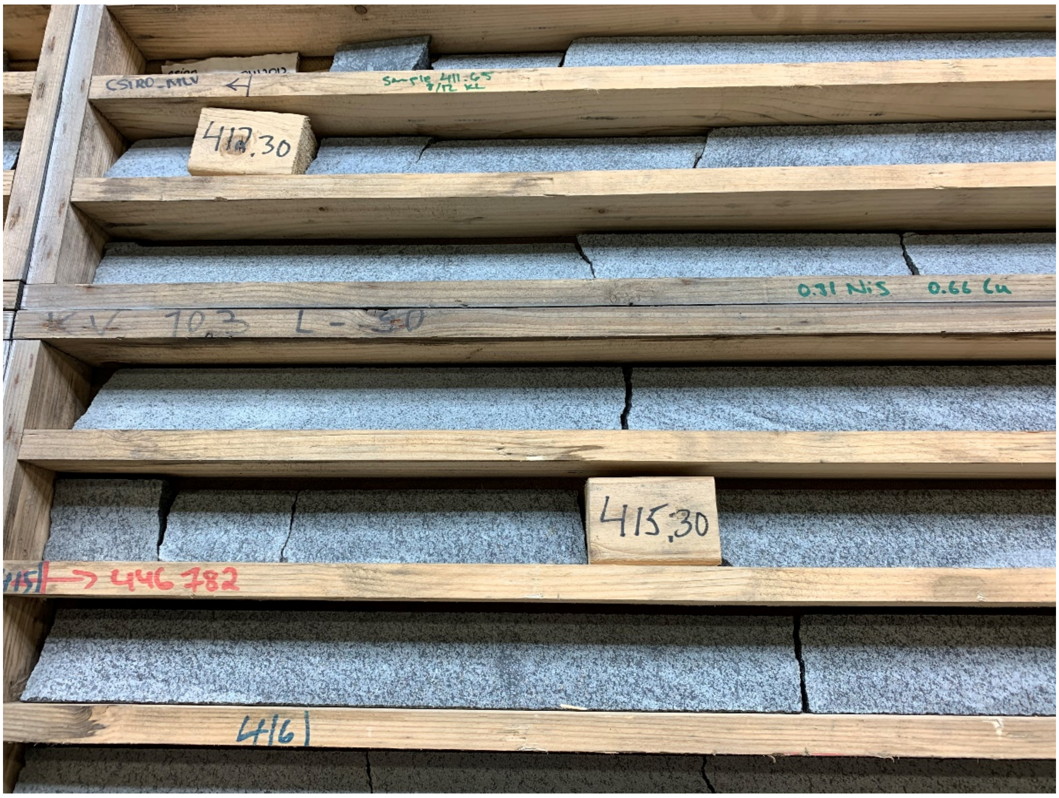


Figure 6-6: **Core displaying low-grade 'Normal Ore' mineralisation with fine grained disseminated sulphides (Source: SRK site visit, 2019)**

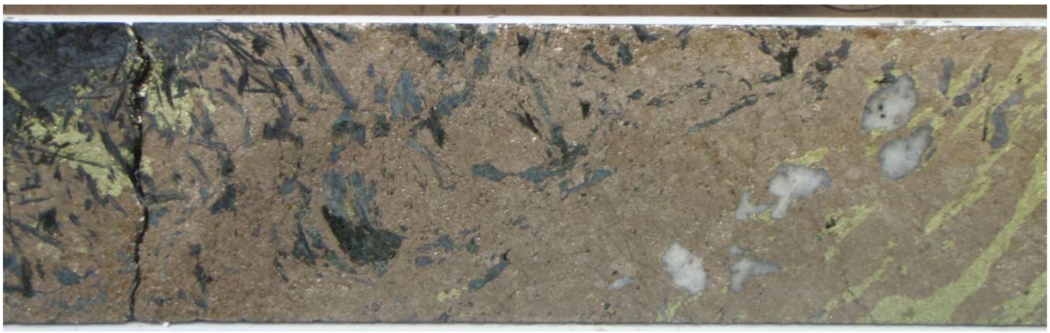


Figure 6-7: **Core displaying pyrrhotite and chalcopyrite vein with fibrous actinolite wallrock alteration (Source: Standing, 2009)**

Figure 6-8: **Core displaying pyrrhotite+pentlandite vein with angular clasts of biotite(?) (Source: Standing, 2009)**

6.8 Mineralisation Geometry and Structures

Before mining commenced, mineralisation noted to sub-crop across an area of approximately 13.3 hectares. In general, mineralisation dips and plunges in multiple directions. The main mineralisation extends at least to 800 m depth below the surface and has confirmed strike length of 1,250 m.

The following is re-produced from Le Vaillant *et al*. (2017):

Mine geologists have separated the mineralisation into different mineralisation types, mainly on the basis of their Ni-PGE grades. Low grade mineralisation which forms near the base of the intrusion and along the margins of the Cu-Ni mineralisation, but are also found internally, are classified as ‘‘false ore”. This pyrrhotite-rich mineralisation, dominantly disseminated but locally net-textured and semi-massive at the decimetre scale, is often associated with country rock xenoliths.

The ‘Normal ore’ (or Cu–Ni mineralisation type) represents the bulk (>90%) of the Mineral Resource and is characterised by 2 to 6% of sulphides (pyrrhotite, pentlandite, and chalcopyrite) and average Ni and Cu ore-grades of 0.3 and 0.4% respectively (Santaguida *et al.*, 2015). Finally, the ‘Ni–PGE ore’, which occurs more locally, has a similar sulphide content to that of the ‘Normal ore’, but the sulphides are predominantly pentlandite, pyrite and millerite, and the mineralisation has higher and more variable Ni grades, lower Cu grades (Ni/Cu = 1.5 to 15), and extreme Ni tenors in excess of 30%.

Extremely high Ni contents are developed within olivine grains in the ‘Ni-PGE ore’ (Yang *et al*., 2013). It is likely that these Ni-PGE ores constitute another example of extremely high Ni grade mineralisation related to formation from Ni-enriched magmas at high values of silicate to sulphide mass ratio (R factor) in the presence of olivine (Barnes *et a*l., 2013).

The following is re-produced from Standing *et al*. (2009):

A diffuse internal stratigraphy has now been identified at Kevitsa. Critically, within the deposit area there is clear evidence of multiple olivine pyroxenite magma pulses:

- pulses may be differentiated, both visually (mineralogically) and geochemically (using MgO and Al_2O_3 trends);
- key feature of these differentiated pulses within the deposit area is their lateral discontinuity, and apparent high aspect ratios;
- mineralisation appears to be associated with each magma pulse, in effect making each pulse a different mineralising event; and
- southwest-dipping Cu-rich mineralisation (basically all included in the current pit design) are interpreted to be a result of the repetitive magma pulsing and have a more continuous spatial distribution associated with the internal stratigraphy of the Kevitsa intrusive.

Another important outcome from the work by Standing *et al.* (2009) was the recognition of the significance of large sections of dunitic material, which appear to form part of a north-plunging body, discordant to the igneous layering. The geometry of this dunitic unit, discordantly cutting the broad igneous stratigraphy, suggests that the rock has a different provenance to the layering in the olivine pyroxenite. The north-plunging high-Ni mineralisation is interpreted to be intimately associated with this north plunging discordant dunite intrusive.

A steep series of reverse shears vertically offset the main mineralisation of the Kevitsa deposit. These shears trend approximately north-south (and variants thereof) and dismember and similarly offset the igneous layering. Steep east-west structures have also been observed at outcrop and possibly affect the mineralisation to a lesser extent. At present, these structures are limited to the observation controls at outcrop.

The Kevitsa deposit is overprinted by many structures developed in association with the evolution of the Satovaara Fault. The impact of this structural modification is manifest as veins arrays (many overprinted by simple shear) shear zones and faults at Kevitsa. A comparison between high-grade Cu and Ni zones against low-grade Cu and Ni zones shows no difference in the geometry of structures or veins between them, indicating that main role of structure at Kevitsa is modification rather than any cryptic primary control; however, the role of crustal-scale structure may be very important for explaining the location of the Kevitsa deposit.

6.9 Nickel in Silicate Minerals

Silicate minerals such as olivine, pyroxene, and tremolite can also carry nickel within the crystal lattice. This nickel which is in silicate and cannot be recovered with flotation methods and is “contaminating” analytical results. When a strong digestion method is applied, such as Aqua Regia, or XRF methods are used, results include total nickel in sample and do not specify or quantify amount of sulphide nickel.

To estimate recoverable nickel, that is, sulphide nickel, FQM, and subsequently Boliden, applied a sulphide selective leach analytical method based on ammonium citrate hydrogen peroxide leach together with normal digestion methods to estimate total nickel (Ni) and sulphide nickel (Ni(S)) in samples.

7 DEPOSIT TYPES

Kevitsa is a magmatic, layered-intrusive, Cu-Ni-PGE deposit. Layered intrusions are the host for various types of mineralisation such as; chrome, PGE, copper and nickel deposits. Typically, a single intrusion contains several types of mineralisation as distinctive layers. The Central Lapland Greenstone Belt hosts a number of mafic-ultramafic intrusions, some of which are mineralised such as Sakatti Cu-Ni-PGE deposit located 20 km to the southwest of Kevitsa.

Layered intrusions are rare worldwide but typically occur at cratonic margins. They are formed by fractional crystallisation. In these systems, minerals accumulate in an order determined predominantly by mineral density and size. Heavy minerals such as olivine accumulate first in lower parts of the magma chamber while plagioclase, being lighter than magma, floats settling within the upper parts of a magma chamber. Simultaneous with mineral crystallisation the chemical composition of the rock changes from MgO-rich to Al_2O_3-rich. The primary magmatic cumulate texture is poikilitic with orthopyroxene forming oikocrysts. Typical inter-cumulus minerals are plagioclase, hornblende, sulphides, and magnetite. The widespread alteration of the host rocks makes identification of primary cumulate textures and magmatic layering very difficult.

8 EXPLORATION

8.1 Introduction

Exploration has been carried out on the property by various owners and the GTK since the 1970s. This report item is divided into two chapters describing Exploration work carried out before and following the acquisition of the Project by Boliden in June 2016. Exploration from 1978 until March 2016 has been initially reported by Gray *et al.*, (2016), as described below.

8.2 Exploration from 1978 until March 2016

Exploration work, completed prior to June 2011, focused predominantly on geophysical methods and a range of geophysical datasets have been compiled over the years by both FQM and the GTK. These geophysical datasets include:

- **Magnetic:**
  - aeromagnetic data from the GTK national mapping program at 200 m line spacing and 30 m flight height;
  - numerous ground magnetic surveys from 1984 to 2007 and 2012 at various line spacing.
- **Radiometric:**
  - airborne radiometric from the GTK national mapping program at 200 m line spacing and 30 m flight height.
- **Gravity:**
  - ground gravity surveys from 1978, 1982, and 1984 on a 100 x 20 m grid on various orientations.
- **Electromagnetic:**
  - airborne single-frequency electromagnetic ("EM") from the GTK national mapping program at 200 m line spacing and 30 m flight height;
  - airborne electromagnetic survey ("VTEM") conducted in 2009 covering 470 line-kilometres at 200 m spacing, and reduced to 100 m spacing over the Kevitsa-Satovaara Igneous Complex;
  - horizontal loop, frequency ground EM (Slingram 1984 and Maxmin 1987) and VLF at different frequencies from 1993 to 1995; and
  - local ground-based EM in 2012.
- **Electrical:**
  - induced Polarization and Resistivity from 1989 and self-potential from 1994;
  - surface mise-a-la-masse ("MAM") from 1994 and down-hole MAM from 2008; and
  - titan-24 survey combining Tensor Magnetotelluric ("MT") Resistivity, Galvanic Direct Current ("DC") Resistivity, and Induced Polarization ("IP") conducted 2008.
- **Seismic:**

  - 2D reflection seismic from 2009 covering 33.6 line kilometres, using Common Mid-Point (CMP) with symmetrical split-spread goniometry, 402 active channels at 12.5 m interval spacing and maximum receiver offset of 2,502 m; and
  - 3D reflection seismic from 2010 (Seistronix and Sercel).
- **Down-hole Logging**:
  - density, magnetic susceptibility, Induced Polarization, resistivity, gamma, radiometric, and sonic logging from 2004, 2007, 2008, and several campaigns since 2011.

During 2008, a combined magnetotelluric, direct current resistivity and induced polarization survey (the Titan-24 survey), was a major source of target generation for much of the subsequent exploration during 2009 to 2013. In all, the survey generated 64 individual anomalies, with 25 classed as high priority. Additionally, the VTEM survey from 2009 also provided a number of targets over the same period. The key targets were followed up with base of till sampling and local ground-based EM surveys to further define targeting at more detailed resolution. Many of these were tested with diamond drilling, including the Satovaara, Lipatti, Saivel North, and Mustaselkä anomalies among others.

The base of till survey over the northern part of the Kevitsa mine area identified several Cu anomalies which were investigated and deemed sub-economic. This area now forms the 2A extension of the Kevitsa mines waste rock dump area. Additionally, the area that now hosts the Kevitsa tailings facility was subject to drilling in 2010. Some low-grade mineralisation was intersected but was considered to be uneconomic at the time.

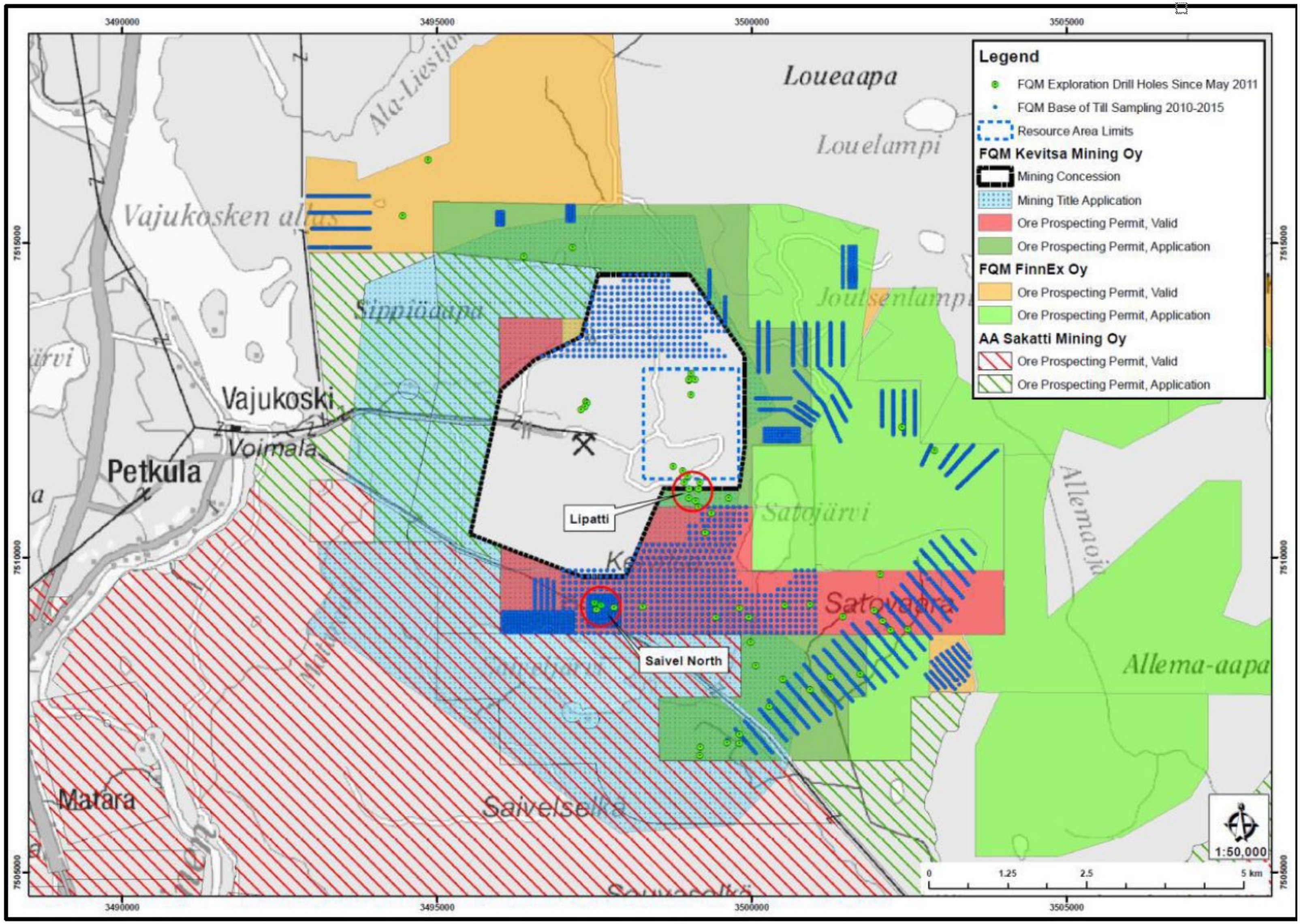


Figure 8-1: Base of till sampling conducted by First Quantum Minerals between 2010 and 2015 (Source: Booth, 2015)

8.3 Exploration since June 2016

Boliden conducts exploration work within Kevitsa Mining Concession and adjacent Exploration Permit areas through Boliden FinnEx OY, a separate entity from the mine operator. Since the release of the 2016 Kevitsa Mineral Resource estimate (Gray *et al.,* 2016), the exploration work has focused outside the Kevitsa mine area. A summary of exploration work carried out on behalf of Boliden within or adjacent to the Kevitsa Mine June 2016 to June 2018 is as follows:

- three diamond drillholes within Kevitsa Mining Concession;
- one near-mine exploration diamond drillhole outside Kevitsa Mining Concession;
- base of till sampling outside Kevitsa Mine at Hanhilehto, Marja, Liina and Hangaslaki targets;
- ground EM surveys outside Kevitsa Mine at Vaisko, Hanhilehto S, Hangaslaki and Mustaselkä targets;
- extended gravity survey outside Kevitsa Mining Concession;
- bought exploration permit areas adjacent to Kevitsa Mine from Anglo American Sakatti Mining, with associated data of exploration activities from 2010 to 2016 including geophysical surveys, base of till sampling and diamond drilling.

Previous exploration at Kevitsa includes extensive datasets of geophysical surveys and diamond drilling. This work was carried out by various entities, most importantly First Quantum Minerals and GTK. These activities are reported in detail in previous mineral resource estimate reports (Gray *et al.*, 2016; Lappalainen & White, 2010).

9 DRILLING

9.1 Drilling Summary

Mineral Resource definition, infill and exploration drilling at the Kevitsa property have been performed using diamond core drilling (”DD”). The location of the drillholes used for the 2011, 2016 and 2018 MRE are shown in Figure 9-1. A summary of the exploration drilling within the Kevitsa Mine area used for the 2018 MRE is listed in Table 9-1. Additional drilling has been conducted throughout 2019 and an updated resource model is in construction as of February 2020.

DD are spaced at 25 to 100 m along drill lines that are approximately 50 m apart; drillhole grid spacing increases with increasing depth below surface. The core drilling was completed at a range of core diameters, predominantly BQ-TK (40.7 mm core) and BRM (42 mm), but also NQ (48 mm) and WL-66 (50.5 mm).

RC grade control drillholes are drilled on an offset 15 m grid; however, they were not utilised as part of the 2018 MRE and are not described in detail herein. The location of RC drillhole collars is shown in Figure 9-2.

The 2016 MRE included assay data from 510 DD and the 2018 MRE includes data from 518 DD, which incorporated 8 infill holes from the 2017 drilling campaign. An additional 23 holes were drilled between 2017 and 2018 but the results were not received in time to use in the 2018 MRE.

Table 9-1: Summary of exploration drilling 1987 – 2018 used in the 2018 MRE

Company*	Period	DD Count	DD Meterage	RC Count	RC Meterage
GTK	1987-1994	244	32,720	-	-
SGL	2003-2008	68	25,873	-	-
FKMOY	2008-2016	205	91,799	3,195	139,091
BKMOY	2016-2018	22	8,789	1,636	84,586
BFXOY	2018	2	1,834	-	-
Total		**541**	**161,015**	**4,831**	**223,677**

*Notes: GTK = Geological Survey of Finland, SGL = Scandinavian Minerals Ltd, FKMOY = First Quantum Kevitsa Mining Oy, BKMOY = Boliden Kevitsa Mining Oy, BFXOY = Boliden FinnEx Oy.

9.2 Core Recovery

Core recovery is generally very good at Kevitsa (averaging >99% throughout the exploration history), as can be seen from the core recovery recorded per metre. The most significant core losses are suffered in the first few metres of overburden and strongly weathered bedrock. There are few shear zones with some core loss. In general, below 30 m of the original rock surface, core recovery does not affect grade estimation. In SRK's opinion, core recovery does not introduce bias into the analyses of the estimated elements so is not considered significant to the Mineral Resource estimate

9.3 Drillhole Surveying

All drillhole collar locations are referenced to Finnish National Grid Coordinate System Zone 3 coordinates. Collar surveying is conducted by the Mine Survey Department and the downhole surveying was completed by the drilling contractor Arctic Drilling Company ("ADC"). Down-hole surveying was carried out with a GyroSmart tool.

In the drilling campaigns prior to 2018, the collar positions have been surveyed by the Mine Survey department and independent contractor Rovamitta Oy. Rovamitta Oy was used for surveying from 2009 to 2012 for some of the drillholes (in total 59 holes). The down-hole surveys have been taken by the drilling companies, meaning that the tool has varied by the year and the contractor, but all with industry-standard methodology. There are drillholes with no downhole deviation survey of which are on average 47 m long, the majority of them being drilled by GTK; these holes were used in the 2018 MRE as the expected deviation was not considered to be material.

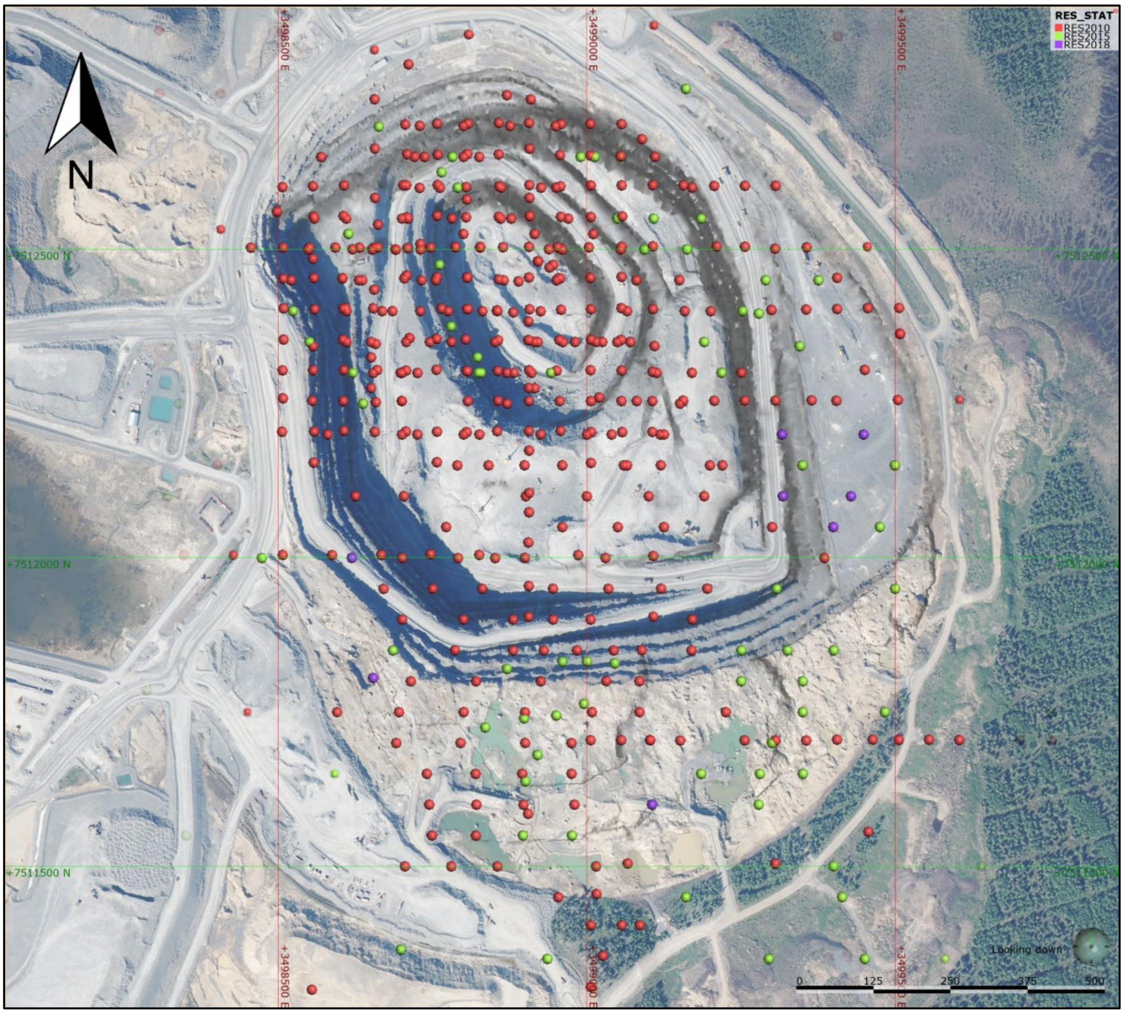


Figure 9-1: Diamond core drillholes completed at Kevitsa prior to MRE 2018 coloured MRE database (red = used for 2011, 2016, 2018 MRE; green = used for 2016, 2018 MRE only; purple = used for 2018 MRE only) and topographic surface with satellite imagery as of end-2019

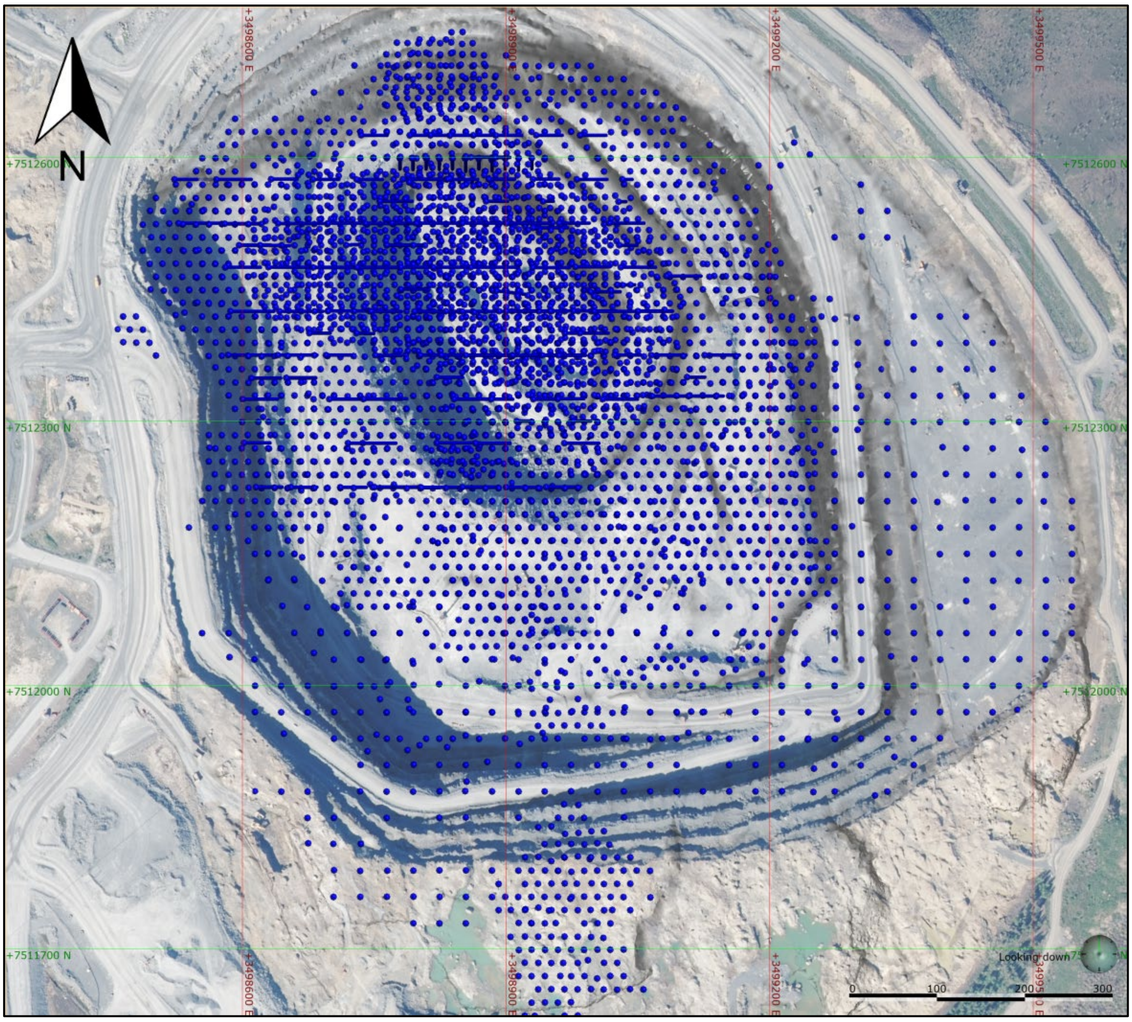


Figure 9-2: RC drillhole collars and topographic surface with satellite imagery as of end-2019

10 SAMPLE PREPARATION AND SECURITY

10.1 Introduction

The description of the sample preparation, analyses and security are summarised from Gray *et al*. (2016).

Sample preparation and analysis has good evidence of being managed in a secure manner at both on and off-site preparation and laboratory facilities. Drilling, logging and sampling data were collected from diamond core by reputable companies and suitably trained persons. Boliden has practised quality assurance and quality control (“QA/QC”) for the duration of their diamond drilling.

The Labtium Oy (“Labtium”) laboratory in Sodankylä was used for sample preparation and analysis, with results electronically uploaded into a secure database system. Samples were prepared and analysed in Finland. Labtium is a FINAS-accredited testing laboratory T025 meeting the requirements of international standard SFS-EN ISO/IEC 17025:2005. Regular laboratory visits and audits were completed by the geological team from Kevitsa mine since 2009. In previous campaigns, Labtium and GTK laboratories (local) and OMAC, ALS (international) laboratories were used for sample preparation and analysis.

All geological data held by Kevitsa mine is loaded to SQL database with DataShed software as the front-end. Regional exploration data, outside the remit of this report, is stored in a separate database maintained by Boliden FinnEX. There are links between the two databases to allow for collective viewing of both datasets at the same time.

10.2 Sample Preparation and Chain of Custody

10.2.1 Core

Core from all campaigns was logged and marked with sample intervals, sample numbers, and QC sample types, then photographed (as dry and wet core) before the core was split and divided into the pre-defined sample intervals. Both GTK and Scandinavian Minerals Ltd (“SGL”; previous explorer) applied systematic 2 m sampling downhole, where the sampling was not honouring the lithological contacts. FQM and Boliden sampled 2 m intervals but honoured lithological contacts; samples did not cross lithological boundaries.

Half core has been retained for reference purposes from all projects, unless a sample has an associated core duplicate (1/4 core remains) or samples have been taken for further study or testwork. Logging data from the original logging is held on site at Kevitsa and has been imported in the geological database.

Core, coarse and pulp rejects from SGL, FQM and Boliden drilling are stored on site. The Boliden sample preparation foreman maintains a map with the location of each drillhole and the corresponding coarse and pulp reject, stored. Most of the core drilled by GTK is at the GTK’s Finnish national core warehouse in Loppi.

Sample preparation for both the GTK and SGL drilling campaigns was completed by the GTK. The core was cut using a diamond saw and half cores are placed into bags, with the average sample weighing approximately 4 kg. A batch of samples consisted of 90 individual samples, inclusive of QC samples. QC samples included blanks, three commercial standards, and quarter core duplicates. Samples were crushed to 90% passing 2 mm and riffle split to 150 g.

This material was then milled to 90% passing 100 µm. Pulp material was sent to GTK laboratory in Rovaniemi for analysis.

Core drilled by FQM was cut by either employees or a subcontractor (GTK). The majority of samples drilled by FQM KMOY were prepared and analysed by Labtium in Rovaniemi. Labtium (Rovaniemi) was closed down in 2014, after which no drillhole samples were send by FQM for analysis. Samples were dried at 70°C in a forced air oven, then crushed using a robotised jaw crusher to >70% passing 2 mm. The samples were split down to 0.7 kg and then pulverised with LM2 pulverising mill to 90% passing 100 µm. A second laboratory, OMAC Laboratories Ltd (Alex Stewart Group Geochemical & Assays, now ALS Minerals), Ireland, was used briefly in 2009 for a limited number of primary assay results. OMAC laboratories has ISO/IEC 17015 accreditation. The sample preparation and analysis techniques were comparable with those used at Labtium. Check samples were sent to ALS Chemex Perth and ALS Otukumpu in Finland for independent umpire checks on the analytical precision at the primary laboratory. No sample preparation was required as part of this work.

Holes drilled by FQM FinnEx were logged on site by geologists. FQM FinnEx sample technicians cut the core on site, after which half core samples were samples were weighed, dried and crushed to product with 70% passing <2 mm and then split off to 250 g, pulverized and split to better than 85% passing 75 µm (laboratory code PREP-31). Each core sample batch included blank and standard samples inserted in the sequence by FQM FinnEx technicians. The blank samples were “silica gravel” (crushed quartzite) while the standards were OREAS commercial certified reference material (“CRM”) products OREAS 14P and OREAS 13b. These were inserted in the sample batches in random order so that each batch contained two to three blanks and at least one standard of both types. In addition, every batch had one to two of each of a core duplicate (1/4 core cut and inserted in the batch by FinnEx) and a coarse reject and pulverized reject. The latter two were produced by ALS Outokumpu laboratory.

Core drilled by Boliden was cut by sample technicians on site. Half of the core was placed into sample bags with sample tags and the remaining half was replaced in the original core box. A batch of samples consisted of approximately 100 individual samples, inclusive of QC samples. QC samples included two blanks, two CRM, and two quarter core duplicates. Sample lists are sent to the preparation laboratory included details of which samples should have a coarse duplicate prepared after crushing and duplicate after pulverizing. Once the sample batch was ready for analysis, samples were despatched to the sample preparation facilities at Labtium in Sodankylä. Chain of custody forms were sent with the samples and a copy retained on site for reference. Half core samples were then prepared by Labtium, the receiving laboratory. All samples drilled by Boliden Kevitsa Mine were sent to Labtium at Sodankylä. In June 2018 the ownership of Labtium Oy changed to Eurofins Scientific Group and formed new company Eurofins Labtium Oy. The laboratory and staff remained as with Labtium Oy. The samples were prepared by drying (method 10), crushing (method 31), splitting (method 35) and grinding (method 51). Samples are dried at 70°C, crushed to > 90% passing of < 2 mm and riffle split to 1 kg. This material was then pulverized to > 90% passing of < 0.1 mm particle size. 80 to 100 g was subsampled and returned to Boliden KMOY for archiving.

There is no documentation stating how the GTK density measurements were undertaken. Boliden use half core and very rarely 1/4 core, after cutting and without drying. The effect of moisture has been studied by comparing samples before and after drying; with insignificant differences noted due to the low porosity of the Kevitsa rocks.

10.2.2 Grade Control RC

The RC rig has an integrated 4-tier riffle splitter. The RC rig off-sider was responsible for labelling sampling bags with the hole identity and the sample interval. The rig operator communicates to his off-sider to change the sample bag at the end of each 3 m drilling interval. Samples were collected directly from the bottom tier of the riffle splitter. At the end of each shift, the drilling log and samples were delivered by ADC to the sample preparation facility on site at Kevitsa. Kevitsa Sampling Supervisors receive and check that all samples are present and that unique sample identities are allocated to each sample. Samples are grouped into batches of 80 to 90 samples. Samples were dried at 100°C and split using single tier splitter to a 2 kg sample. A duplicate was taken every 25 samples at the splitting stage to check sampling error associate with this process. Before sending to the laboratory, further QA/QC samples were inserted by the Sampling Supervisors. Two CRM and two blank samples were inserted per batch. Coarse duplicates (three per batch) were indicated on the sampling lists which go to the laboratory.

RC samples were sent to Labtium Sodankylä for final sample preparation and analysis. The 2 kg sample received from Kevitsa is dried and then crushed (method 31) to 70% passing 2 mm. Samples are then split down to 100 g (method 35) and pulverised to 90% passing 100 µm (method 40). Since August 2014, on completion of analysis, the remaining pulp sample was returned to the mine. This material was processed through the onsite XRD machine. Prior to this, pulp samples were discarded by the laboratory; this remains the case for coarse reject material.

10.3 Assay Analysis

All the DD pulp samples have used the same Aqua Regia digest method for total nickel (Ni) and copper (Cu), apart from drilling conducted by FQM FinnEx. The Boliden KMOY and FinnEx drilling programmes used an Agua Regia digest followed by ICP-OES analyses Labtium method 510P. Additional elements acquired are arsenic (As), silver (Ag), cadmium (Cd), cobalt (Co), chromium (Cr), iron (Fe), manganese (Mn), molybdenum (Mo), lead (Pb), antimony (Sb), and sulphur (S).

Nickel and copper sulphide results were available for a subset of the SGL drilling and all of the Boliden and FQM KMOY drilling. This method was introduced to analyse Ni in sulphides as opposed to Ni in silicates. Labtium method 240P is an ammonium citrate hydrogen peroxide leach with ICP-OES finish. Labtium and OMAC laboratories used this method of analysis of Ni, Cu and Co in sulphide.

Gold (Au), platinum (Pt), and palladium (Pd) have been assayed using lead collection fire assay techniques. Sample size has varied in the different campaigns. The GTK laboratory used a 25 g sample or 50 g sample with FAAS finish, whereas Labtium Rovaniemi used a 50 g charge weight with ICP-AES finish. Boliden KMOY has used Labtium Sodankylä (method 240P) which used a 25 g sample with ICP-OES finish.

FQM FinnEx samples were sent to ALS Loughrea in Ireland, and the analyses included near-total leach (four acid) multi-element ICP-MS method (laboratory code ME-MS61), as well as lead fire assay with ICP-AES finish (laboratory code PGM-ICP23) to obtain Pt, Pd, and Au. Some samples were selected for the L-ascorbic acid digest ICP-AES assays which yielded sulphide Ni (laboratory code ME-ICP09). All ALS Minerals laboratories and their above-mentioned assay methods are ISO 17025 accredited. ALS Loughrea is also an INAB accredited testing laboratory (Reg. No. 173T).

The primary laboratory used by each exploration/mining company at Kevitsa is provided in Table 10-1.

Table 10-1: Primary assaying laboratory used by Company

Campaign	Primary Lab	Aqua Regia (Total Ni, S etc)	Selective Leach (Sulphidic Ni, Cu, Co)	Multi-element	Fire Assay (Au, Pt, Pd)
GTK	GTK	X			X
SGL	GTK, Labtium	X	X		X
FQM KMOY	Labtium Rovaniemi	X	X		X
FQM FinnEX	ALS Loughrea			X	X
Boliden KMOY/FinnEx	Labtium Sodankyla	X	X		X

Notes:

1) Full set of elements analysed; Ag, As, Cd, Co, Cr, Cu, Fe, Mn, Mo, Ni, Pb, Sb, S.

2) The majority of samples were analysed using lead collection fire assay.

3) SGL switched from using GTK Rovaniemi to using Labtium Rovaniemi Laboratory in September 2007. Some of the drillholes were submitted for analysis by FQML after acquiring SGL in 2008.

10.4 XRD Analysis

Kevitsa has its own CubiX3 XRD machine on-site where grade control RC samples are analysed. The data is processed using HighScore software which uses a script to produce analysis for 26 minerals. The data are currently incorporated into the grade control block model used for short-term mine planning and provides additional geological information to the process plant to optimise the circuits for increased recovery.

An external company, Stenman Minerals AB ("Stenman"), also analysed the samples but using a Synchrotron device and processed through the Bruker Topas programme to produce analysis for 31 minerals. The samples analysed using Stenman are termed Stenman 1 or priority 0 ("P0") to differentiate them from Kevitsa data in the database.

The majority of the DD samples are analysed at Stenman, while the RC pulp analysis has remained on site. This has resulted in two different datasets which are also different spatially, with Stenman analysing wider spaced and more distal diamond drilling samples (global), predominantly relevant to the later stages of mine development and Kevitsa analysing close spaced (local) RC and some of the diamond drilling more relevant to the core of the deposit.

Due to the differences in the data and lack of samples analysed using both methods for comparison, it was considered inappropriate by FQM and Boliden to combine the datasets. Despite the Synchrotron analysis and the HighScore script being more suitable to the geology of Kevitsa and with a higher level of precision, the Kevitsa data remain the primary dataset given its ongoing use for all the grade control RC data and metallurgical data.

In order to better understand the differences in the two datasets and to increase the DD coverage in the Kevitsa dataset, two sections were selected and the pulps that had originally been analysed at Stenman were re-analysed at Kevitsa. The differences in the two original datasets are still seen in this smaller dataset (termed Stenman 2 or priority 1, "P1") where samples have been analysed by both companies. Kevitsa is working with Stenman to improve the script which would be used to reanalyse the raw Kevitsa data to better correlate with the

Stenman data. For the time being, the two datasets are considered separate and the results should be considered as qualitative, given the large differences observed.

A summary of the methodologies is provided in Figure 10-1.

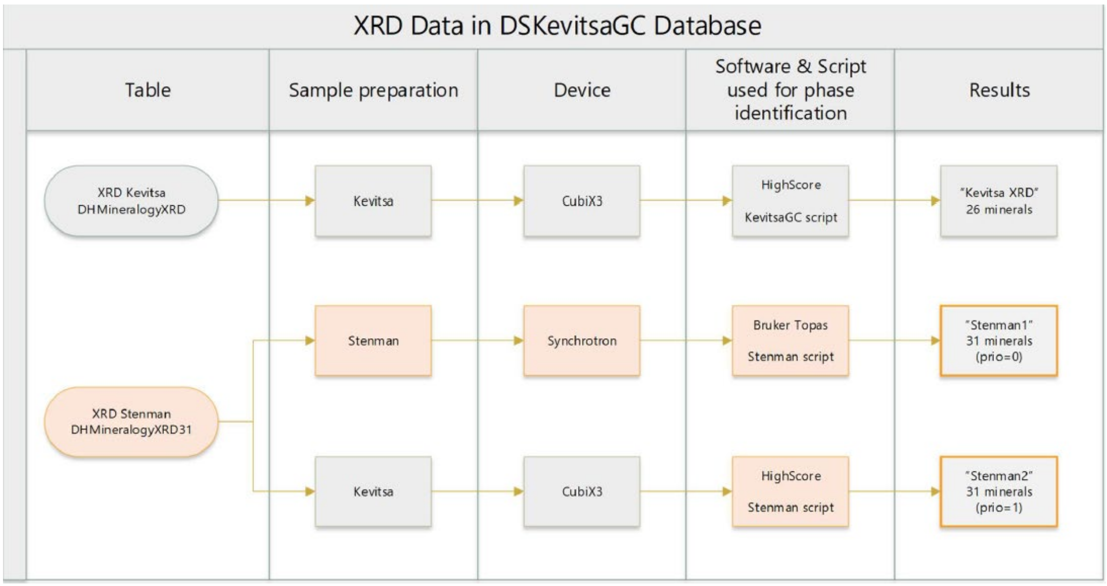


Figure 10-1: XRD datasets for Kevitsa

10.5 Analytical Quality Control Data

10.5.1 Introduction and pre-2018 QA/QC

QA/QC programs have been carried out over the lifetime of the Kevitsa Project. A description of these until the divestment of the Project to Boliden was provided in Gray *et al*. (2016).

FQM applied systematic QA/QC practices on all DD and grade control RC drillhole sampling. Historically, there was no documentation of QA/QC completed by GTK. SGL introduced QA/QC during their drilling; however, it was not available for the full dataset. In order to verify previous historical drilling campaigns, subsequent companies have run check programs. In 2001, at the request of SGL, SRK Consulting (UK) Ltd completed a review of the GTK drilling. Similarly, in 2008, FQM completed a series of umpire checks on the SGL drilling, comprising 1,230 samples being sent to a third-party laboratory, OMAC in Ireland. The program included quarter core, coarse samples, and pulp samples from across the deposit. Neither verification program raised any concerns pertaining to the quality of the data pre-2016; the conclusions below were made by Gray *et al* (2016):

The nickel and copper QA/QC results indicate that for the pre-2016 MRE data:

- assaying laboratories are reporting assays to acceptable levels of accuracy;
- standard failure rates are within acceptable levels;
- blank samples indicate that the sample preparation process is operating successfully and that contamination rates are low;
- field duplicate assays display low bias and good degree of precision;
- coarse crush duplicates display low bias and high degree of precision;
- umpire check samples display low bias and good degree of precision; and

- twinned drillholes display correlations between assays which are considered acceptable.

It was considered that the QA/QC results indicated that the drillhole assays were suitable for use in the 2016 MRE.

10.5.2 Post-2016 QA/QC

A review of the QA/QC procedures and results for DD drilled since Boliden's acquisition of the Kevitsa mine is reported by Murto (2018) and summarised in Degen *et al.* (2018), with relevant sections reproduced below.

The analysis herein refers to DD drilled on the Mine since the 2016 MRE, focusing on the drillholes included into the 2018 MRE. Altogether, 31 drillholes were drilled in 2017 - 2018, results for 8 were received between 01 January and 15 June 2018 and were used as part of the 2018 MRE.

In 2018, results from 11 batches of DD samples, altogether results from 1,140 samples, a mix of original and QC samples, were received; a summary is presented in Table 10-2. The following analysis and tables are reproduced from Murto (2018).

Table 10-2: Summary of QA/QC samples for the 2018 drilling campaign

Sample Type	Total	% Samples	% Insertion Rate
Normal samples	774	89%	-
Blanks	19	2%	2%
Standards/CRM	17	2%	2%
Field duplicates (FDP)	17	2%	2%
Coarse duplicates (CDP)	27	3%	3%
Pulp duplicates (PDP)	18	2%	2%
Total QC Samples	**98**	**11%**	13%
Total Samples	**872**	**100%**	**-**

Blanks

Blanks for all elements report within acceptable ranges. There is one sample with anomaly in Ni, NiS, Cu, CuS, Co, CoS, and Pt. This is likely due to a contamination or sample swap given the same sample stands out in several elements and methods. Results for Au and Pd are below detection limit.

Expected values for Co by aqua regia ("AR") and Co by ammonium citrate leach ("SSL") are based on average results recorded between 2014 and 2018.

Certified Reference Materials (CRM)

For CRM where there is suitable certified figure, failure is determined by samples ±3 standard deviations ("SD") from the expected value. For laboratory standards with no certified figure, failure is determined by the result falling outside the upper and lower limits as given by the laboratory. Limits are provided by Labtium based on its analysis over time.

In summary:

- AMIS0316 performed well.

- AMIS0318 reports lower than expected values in Ni, Cu and Co sulphides by selective leach method. This was expected, as the method is not ideal for AMIS0318.
- No certified expected values for selective leach analysis. Expected values are certified for AR and same values are assigned to SSL. Au, Pt, and Pd are all reporting within ±2 SD limit for both CRM.
- Ni in AMIS0192 show high variability and the general trend is showing lower than expected values. There are 13 outliers, most of them lower than expected values.
- Cu in AMIS0192 show adequate results for the first 18 samples after which the general trend is lower than expected values with 5 outliers.
- AMIS0354 for Ni show quite high variability but the calculated mean is close to the expected value.
- AMIS0354 for Cu are showing high variability and is showing higher than expected values. Three samples fail Cu in SSL-ICPES. The Cu content of AMIS0354 is much lower than the average mineralisation grade so the relevance of this to Kevitsa is limited.

Duplicates

The assessment of pass or fail for duplicates has been taken in the context of both grade control and resource estimation and considers the mining cut offs when looking at the significance of failures. For AR and SSL, the results are reviewed in grade control (0.1%) and resource estimation (0.01%) accuracy. It is important to be aware what is happening at lower values (especially for Resource estimation); however, for production purposes, precision of samples above 0.1% / 1000 ppm is more relevant.

In summary:

- Most of the duplicate failures are at lower values (results above 0.01%).
- Au, Pd, and Pt are performing well. Most of the duplicate failures are for Cu and Ni.
- Failures in FDP and CDP are all by AR and SSL methods, no outliers in PDP and LDP.
- All failures in CDP are of one sample pair, which fail for Cu and Ni both, by AR and SSL. This is most likely due to sample swapping.
- Two sample pairs in FDP fail for Cu or Ni by AR and SSL.
- scatterplots show that there is a fairly good coverage of grades for all elements.
- Quantitile-quantile (“Q-Q”) plots for all analysis methods show no bias between original and repeat values.
- Mean absolutely percentage deviation (“MAPD”) plots show that FDP / CDP have the lowest precision, as expected, and LDP the highest precision between pairs.
- Ni, NiS, and Co all report acceptable number of pairs passing with 10% MAPD.
- Field duplicates of Cu, CuS, and Au falls just under the 10% MAPD limit; 85% of FDP falling within 10% MAPD for Cu, CuS, and Au. Other duplicates are within acceptable limits.
- Pt shows the poorest precision; 80% of FDP and PDP falling within 10% MAPD and 75% of LDP within 10% MAPD.

Murto (2018) summary

Both standards AMIS0316 and AMIS0318 demonstrate acceptable accuracy for use in grade control and resource estimation. AMIS0318 reports lower than expected values in SSL method in Ni, Cu and Co, which is consistent with earlier results; the method is not ideal for AMIS0318.

Laboratory standards provides some measure of accuracy for NiS and CuS.

Duplicates have acceptable precision for use in grade control and resource estimation

Blanks suggested that results are suitable for use in grade control and resource estimation and contamination has been controlled.

LGC comments

The use of CRM that appears to be inadequate for the SSL for Ni, Cu and Co in sulphide minerals and expected grade ranges of the deposit impacts the confidence in the database quality. It is difficult to judge the quality of the new drillholes but given the project history there is good reason to accept the assay results and consider the database fit for purpose of an MRE.

SRK comments on analytical QA/QC

SRK agrees with the statements of both Degen *et al.* (2018) and Murto (2018) and overall the assay quality for DD hole assay data is considered to be high with on-going surveillance of the SSL method required. The RC samples were not used for the grade estimation and the QA/QC results have not been analysed by SRK.

10.6 Density Analysis

A total of 254 holes within the resource area were utilised to measure in situ (wet) bulk density data collected by a conventional gravimetric (Archimedes) method. The data was collected weighing the whole core in air and in water. Density was calculated by dividing the weight in air by the difference between weight in air and weight in water. The majority of sampling for density was completed on 10 cm intervals representing a 5 m down-hole length. The measurements are completed without drying due to the very low moisture content. No quality assurance procedures, such as duplicates or standards, are completed currently.

In addition to the Archimedes measurements, the down-hole petrophysics dataset includes readings for gamma, density and susceptibility and electrical properties; resistivity and IP effect. Readings were taken between 2004 and 2012 by survey contractors on a campaign basis and focused on drillholes which had not collapsed over time. Density was recorded every 2 cm or 5 cm down the hole and were composited up to 1 m intervals for the purposes of estimation.

Figure 10-2 shows the location and distribution of the drillholes containing density measurements.

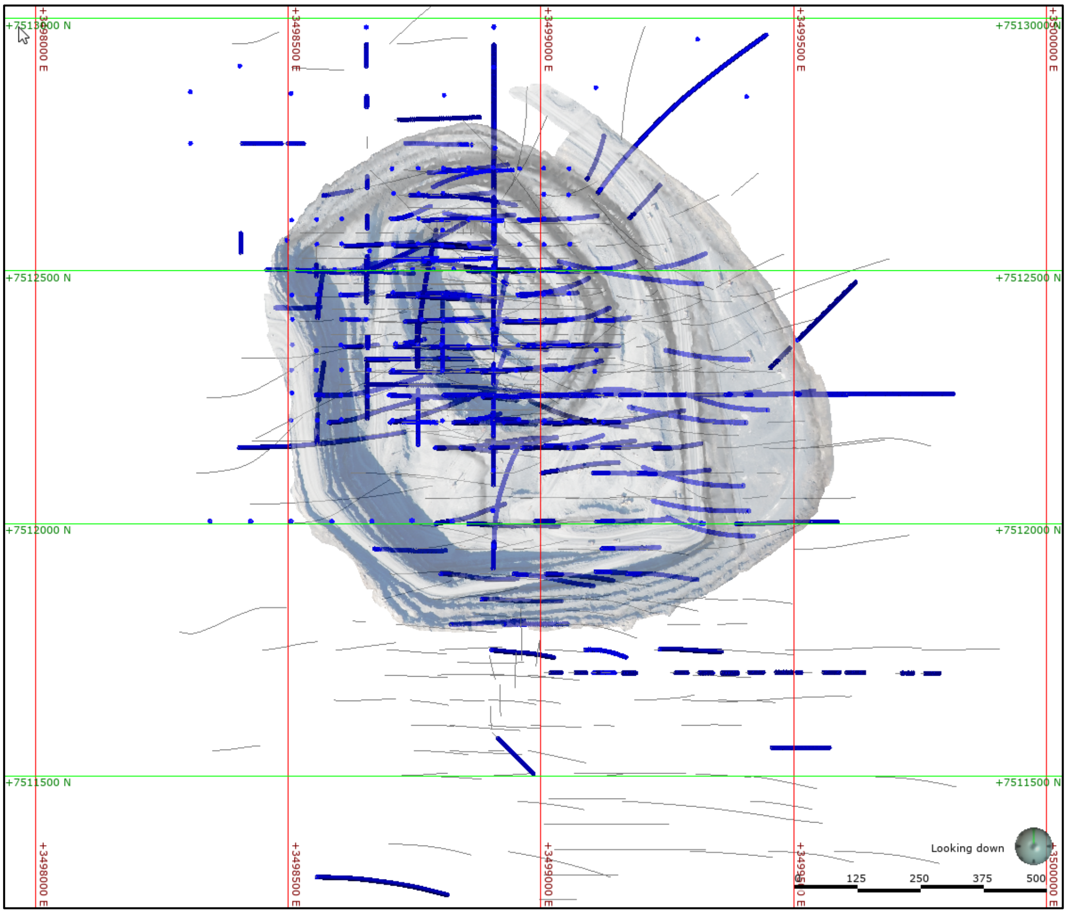


Figure 10-2: Plan view of Kevitsa pit showing holes containing density data (blue) (Source: SRK, 2019)

10.6.1 SRK comments on density

SRK considers the samples representative of the area covered by the Mineral Resource with adequate samples to provide a robust estimate. SRK notes that an update to the density database was made prior to the 2018 MRE, which in turn has impacted upon the tonnage estimates (average density of Measured and Indicated blocks of 3.18 g/cm^3 in the 2018 MRE compared to 3.16 g/cm^3 in the 2016 MRE). SRK notes that this change was a result of additional information, and as such, the differences in the estimates is based on sound analysis, and is therefore, appropriate.

11 DATA VERIFICATION

11.1 Introduction

The Competent Person for Mineral Resource statement, Dr Lucy Roberts, visited the Kevitsa mine in November 2019. During the visit, the CP has gained confidence in the available DD and RC drillhole data, the geology models and understanding of the prevailing mineralisation. Dr Roberts believes the geological understanding and data available for this Kevitsa MRE update is of good quality and is representative of the prevailing mineralisation relevant to the deposit.

11.2 Site Visit Checks

SRK checked the following whilst on-site:

- in-pit observations served to verify the prevailing geology and its association with the different styles of mineralisation as per the logged data and 3D geology models;
- mining and run of mine stockpiling of mineralised material was verified through visual checks, grade control and reconciliation processes; and
- reconciliation process has been developed since mining start-up. Reconciliation results and final metal products have served to verify the accuracy of the Mineral Resource and Reserve estimation process. As an operating mine, reconciliation data supports results for the Mineral Resource, Mineral Reserve and grade control models.

11.3 Database Checks

SRK checked the following in the input database:

- DD and RC drillhole collar coordinates were verified through visual observation and digital checks against database data;
- sampling methods and data correspond to visual inspection of samples taken from stored core and samples and are correctly represented against the original sample sheet records and the stored database data;
- limited random selection of original laboratory assay results was verified against those in the database; and
- QA/QC data was investigated together with the process used for analysis and were verified as robust for assuring assay accuracy, precision and controlling contamination.

11.4 SRK Comments on Data Verification

Multiple phases of drilling have been conducted at Kevitsa. The drilling, sampling, logging, assaying, and analytical methodologies used are consistent with industry best practice. Additional diamond drilling will be available for the 2020 MRE. For the 2018 MRE, the geological modelling, and grade and tonnage estimation was based on diamond drilling alone. For the 2020 MRE, SRK and Boliden will investigate whether the grade control reverse circulation drillholes can also be used to support the grade interpolation and improve the quality of the tonnage and grade estimate.

12 MINERAL PROCESSING AND METALLURGICAL TESTING

As the Kevitsa mine is operational, the testwork completed prior to commissioning is not considered material to the Project currently and is not described in detail herein. More detailed information can be found in the last technical report (Gray *et al*, 2016).

Details of the operational processing and metallurgy are provided in the recovery methods Section 15.

13 MINERAL RESOURCE ESTIMATE

13.1 Introduction

The MRE process described herein was undertaken by LGC in November 2018 and described fully in the accompanying MRE technical report (Degen *et al*, 2018). SRK has reviewed this MRE and has summarised the key processes and decisions made by LGC below.

LGC constructed a block model of the distribution of the sulphide mineralisation in the Kevitsa Cu-Ni-PGE sulphide deposit and derived a block model. Seven grade elements (Cu, Ni(S), Co(S), Au, Pt, Pd, and S) and density were modelled and estimated.

Mineralisation domain modelling was conducted in Leapfrog Geo by Boliden. Grade shells for Cu and Ni(S) have been produced for initial orientation and modelling purposes. Statistical and geo-statistical analysis was carried out using Snowden Supervisor.

13.2 Available data and database integrity

The following data was used for the 2018 MRE:

- collar, survey and assay (including specific gravity) data for drillholes up to 15 June 2018;
- QA/QC results database;
- pit survey as of 30 September 2018;
- 5 m LIDAR topography; and
- bottom of till surface interpreted form drillhole intersects and pre-stripping.

No issues were identified with the input data files; however, LGC noted that the absence of consistent geological logging information prevented a lithological model being generated. The mineralisation is not lithologically-controlled and so this is not considered an issue. In addition, the lithological units are generally gradational phases of intrusions which would be very difficult to model effectively.

13.3 Coordinate system

Finland's national coordinate system KKJ is replaced by the pan-European coordinate system ETRS89, but remnants of KKJ are still in use.

KKJ is derived from the Finnish national adjustment (1966) of the ED50 (European Datum 1950) coordinate system by shifting and rotating ED50 plane coordinates so, that they optimally fit to KKJ's predecessor, the VVJ Helsinki System.

KKJ-coordinates can be presented in geographical (latitude, longitude) or in rectangular grid-coordinates (northing, easting). KKJ is 2D-coordinate system and does not contain any definition of the height system. If the height of a point is given in connection with the horizontal coordinates, it is usually the orthometric height in the national height system (N60, for example).

The KKJ-grid consists of six zones, each three degrees wide. Very often, only zones one through four are represented, because these cover Finland (nearly) entirely. Parameters for zone three are also used countrywide and is called 'Uniform Coordinate System', in Finnish 'Yhtenäiskoordinaatisto' or YKJ.

13.4 Structural Model

The Kevitsa structural model was finalised at the end of February 2018. Modelling concentrated on larger scale structures in the Kevitsa Mine area. The scope of the structural model was to increase the understanding of structural controls of the disseminated Cu-Ni mineralisation and geology of the Kevitsa ultramafic intrusion. Some of the modelled large-scale structures are informing the 2018 grade shells, which were used as definition for mineralisation grade shells of the Mineral Resource and grade control models. The structural model covers the final pit design area and the immediately adjacent area.

The creation of the structural model in Leapfrog Geo and interpretation of large structures in the Kevitsa mine area was described by Kokko (2018). The modelled faults are shown in Figure 13-1 and the main two faults used to offset mineralisation wireframes in Figure 13-2 and Figure 13-3.

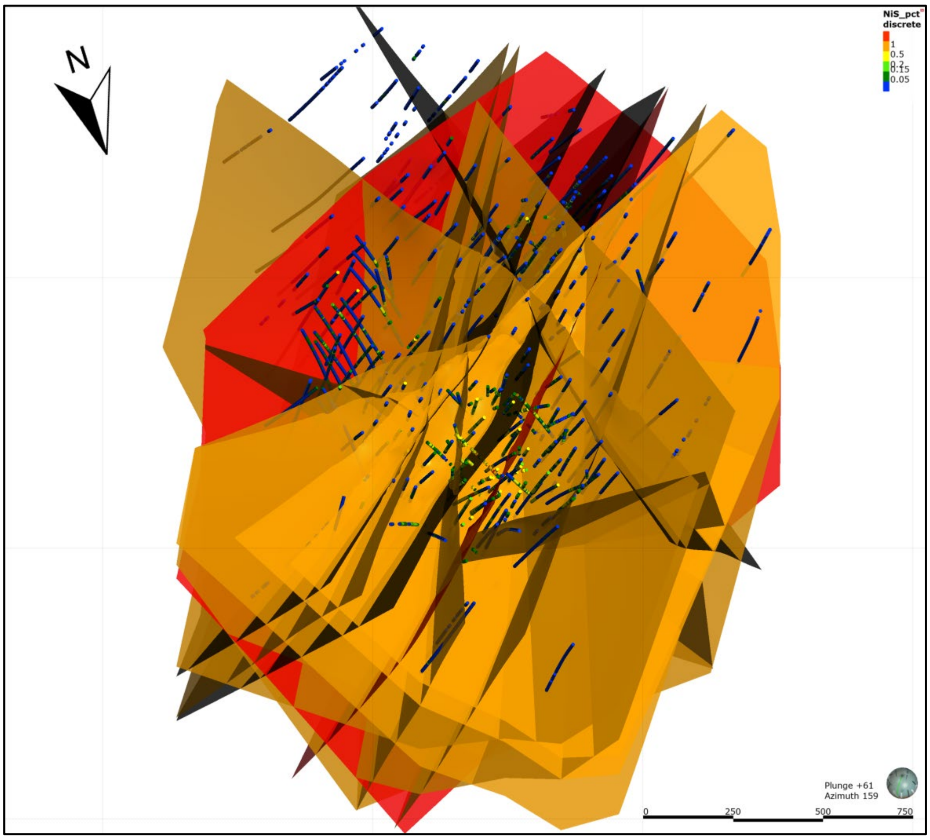


Figure 13-1: Oblique 3D view showing main faults (2 and 14 in red) and other modelled faults (orange) and drillholes coloured by Ni(S) domain

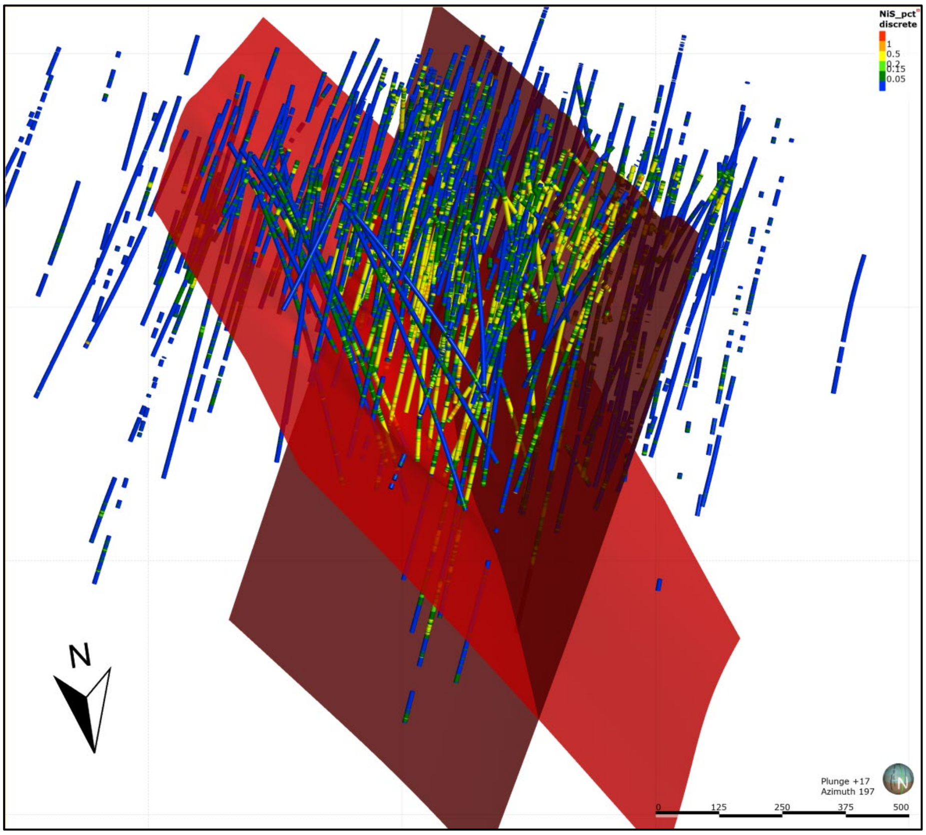


Figure 13-2: Oblique 3D view showing main faults (2 and 14 in red) and drillholes coloured by Ni(S) domain

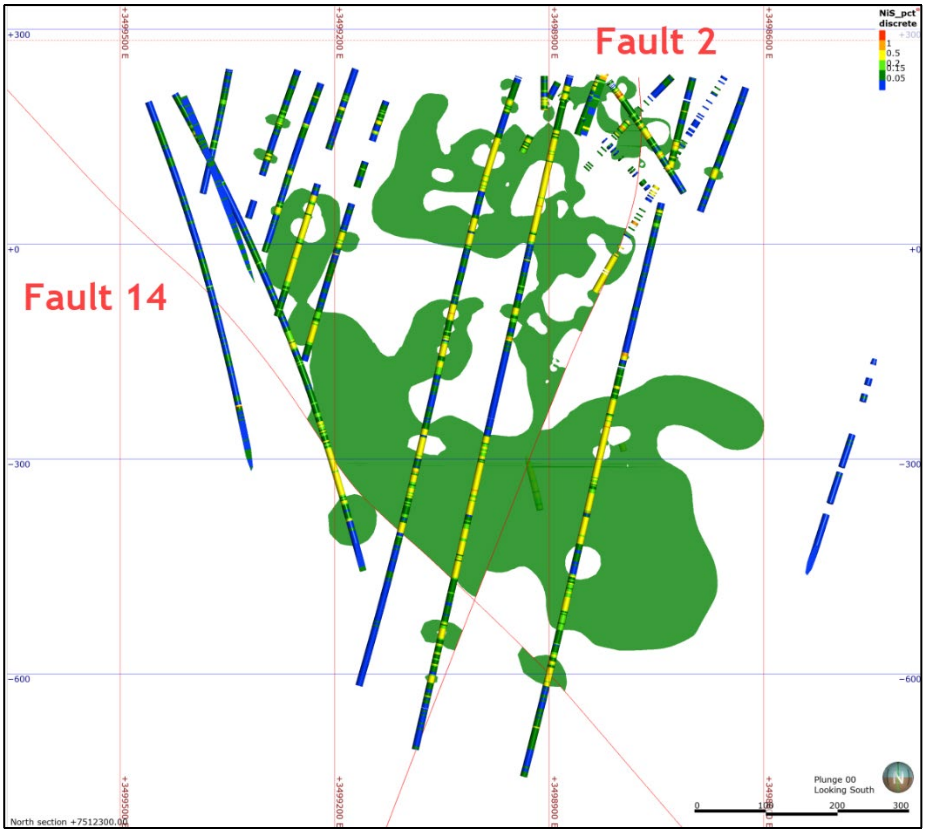


Figure 13-3: Cross-section (Y: 7512300) showing main faults (2 and 14) with low-grade Ni domain and drillholes coloured by Ni(S) domain

Prior to the 2018 model, there are four previous structural models of Kevitsa, which were completed by Jigsaw (2009), WSP Finland Ltd ("WSP"; 2014 and 2015) and Booth (2015).

The Jigsaw (2009) model was informed by reprocessed ground magnetic data, diamond drill core logging, the quarry and outcrop mapping. The model and the work is described in Standing *et al*. (2009). Their approach was to enhance the understanding in structural controls on mineralisation and geology.

The WSP scope was in rock mechanics, pit wall stability and mine planning. Orientated drill core structures from logging and downhole videos, Rock Quality Designation ("RQD") and 3D photogrammetry mapping were the main datasets for the WSP models. The WSP structural model was built containing 3D surfaces that highlight the most distinct and spatially continuous brittle structures. The WSP models are described in the reports from WSP (2014 and 2015).

The Booth (2015) model focuses only the faults that have the most apparent control on mineralisation and geology. Booth (2015) has used the data from 2D/3D seismic reflection data interpretations from Kaukolinna (2014) and from the top of bedrock surface provided by Kevitsa Mine as well some of the WSP structures.

There are other structural interpretations, which are more in focus of near-mine and vicinity of the Kevitsa mine from Koivisto *et al*. (2015).

13.4.1 LGC comments

Data density and distribution is sufficient for modelling the major structures within the designed pit area. The data density decreases going deeper from the already mined areas and away from the designed pit. The southern part of the designed pit stage 4 and stage 3 areas are not as well informed due the lack of mapping and topography data. The drilling density decreases as well in these areas.

13.4.2 SRK comments

The structural geology seems to be relatively well-understood; however, the importance of faulting on controlling/offsetting the mineralisation is not obvious with no sharp contacts observed in most places. The use of faulting to control the domains should therefore be reconsidered; however, it is not expected that this would significantly change the modelled volumes (globally). SRK suggests that this be further investigated during the production of the 2020 MRE.

13.5 Grade Domains

13.5.1 Data used

Use of both DD and RC drillholes was attempted, but assay conflicts between neighbouring RC holes made implicit modelling using RC information very difficult. The mineralisation is highly discontinuous (nugget effect) over short distances in parts of the deposit and often, sample intervals even less than 1 m apart may have very different grade characteristics. This leads to conflicting assignments of the respective intervals to waste and mineralised categories. Lack of RC sample input into the mineralisation model is not considered critical as most grade control data are within the central part of the deposit (Stage 2), which is also well defined by diamond drilling.

13.5.2 Compositing

The predominant sampling length is 3 m for RC holes and 2 m for DD holes. RC holes were excluded from the 2018 MRE, all assay data were composited to 2 m intervals (straight down from collar, assay table level). Compositing at the top-level dictates that any shell interpolation using these data be affected by any changes made to the compositing interval. Interval lengths less than 2 m were added to the previous interval.

13.5.3 Grade thresholds

Threshold grades were assigned based on Ni and Cu grade distribution, with histograms, probability plots and scatterplots shown in Figure 13-4 to Figure 13-9. The thresholds are not primarily a reflection of NSR factors. Currently, Ni(S) and Cu contribute 42% and 46%, respectively, to the economic yield.

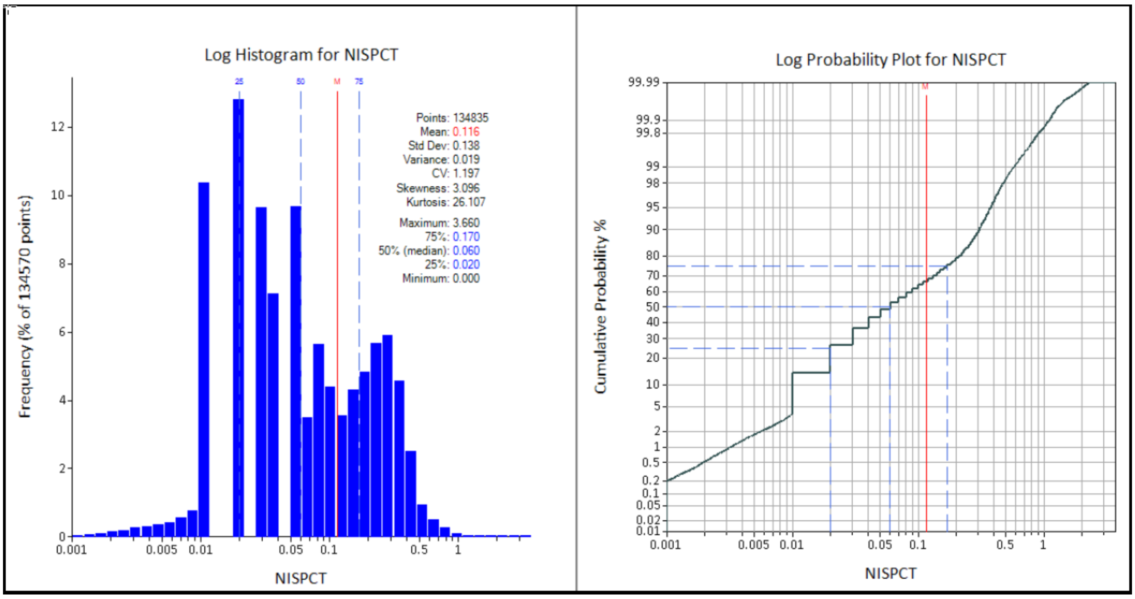


Figure 13-4: Ni (sulphidic) assay histogram and log probability plot

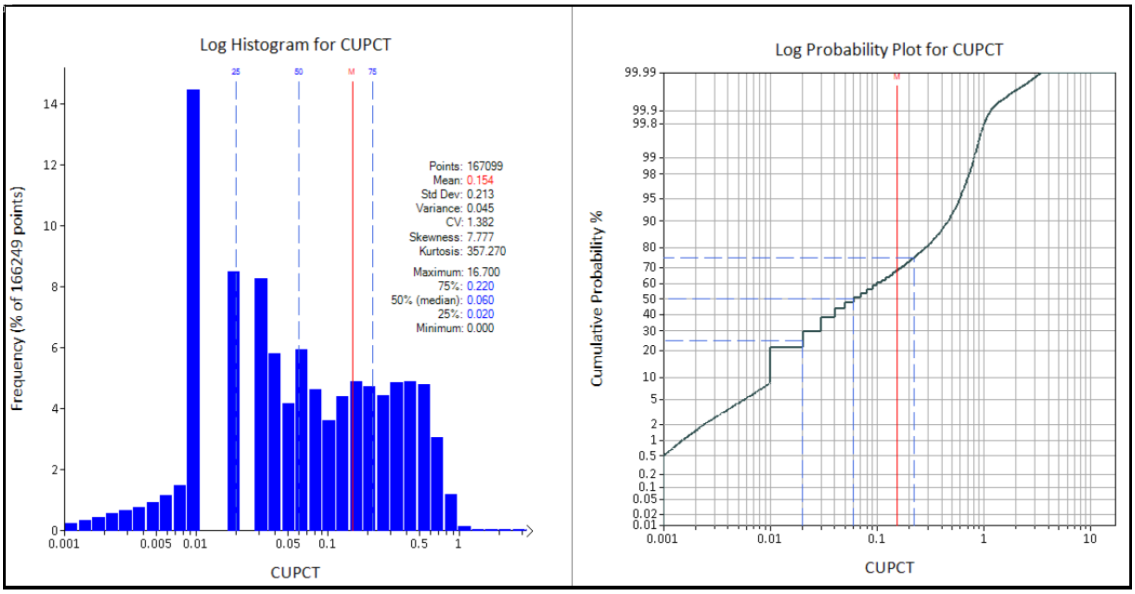


Figure 13-5: Cu (total) assay histogram and log probability plot

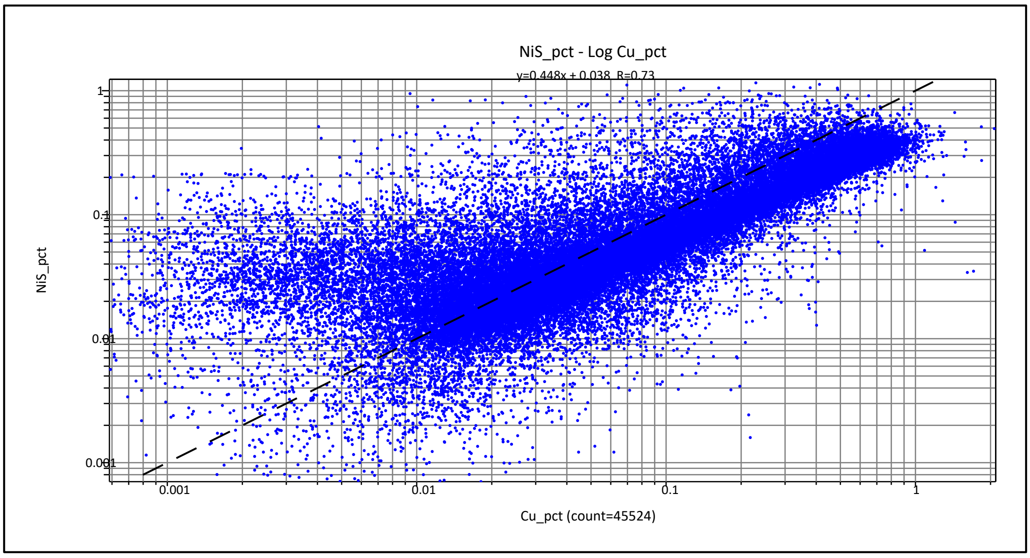


Figure 13-6: Scatterplot showing Cu % vs Ni(S) %

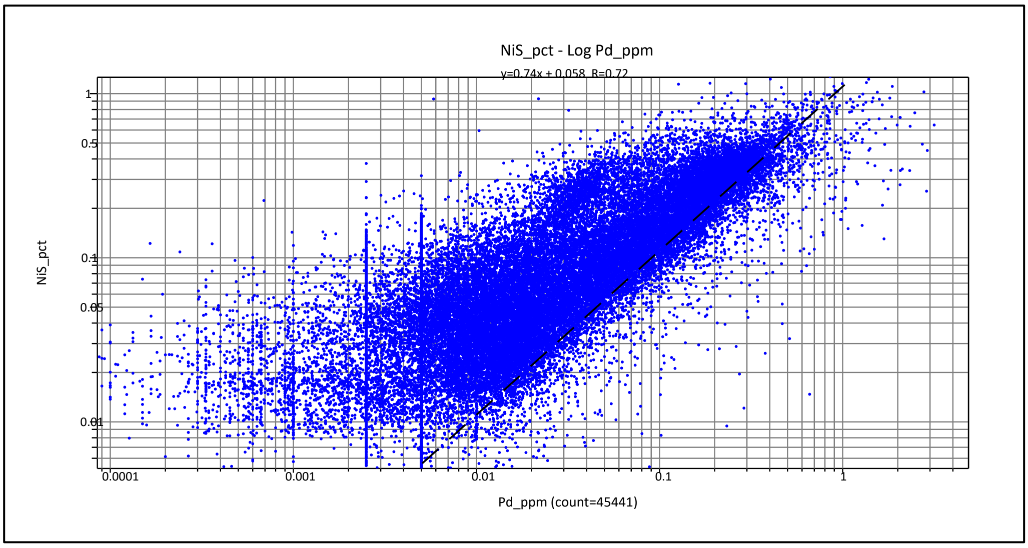


Figure 13-7: Scatterplot showing Pd (ppm) vs Ni(S) %

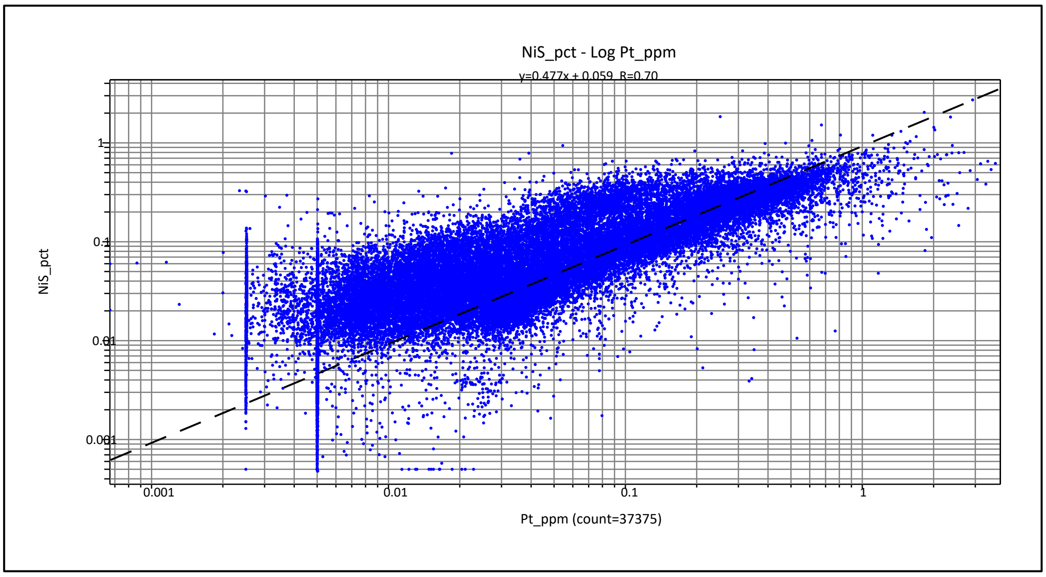


Figure 13-8: Scatterplot showing Pt (ppm) vs Ni(S) %

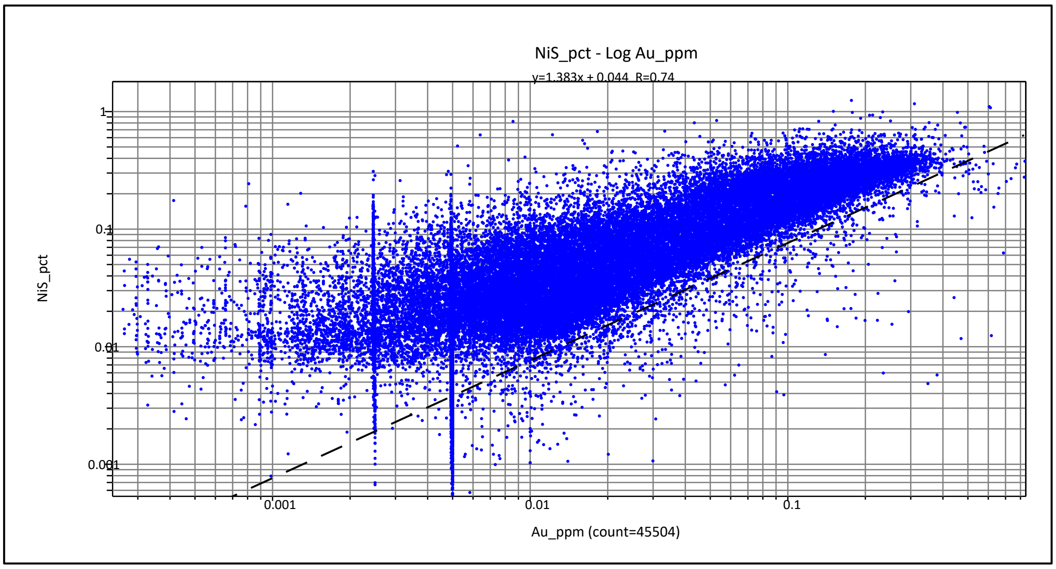


Figure 13-9: Scatterplot showing Au (ppm) vs Ni(S) %

Weak inflections (statistical breaks) in Ni(S) and Cu grade distribution data were used to determine the low grade, high grade and Ni-PGE shell thresholds:

- low-grade Ni(S) domain: >0.15% NiS,
- low-grade Cu domain: >0.15% Cu;
- high-grade Ni(S) domain: >0.30% NiS,
- high-grade Cu domain: >0.30% Cu;
- Ni-PGE domain: >0.40% NiS;
- very high-grade (0.40) Cu domain: >0.40% Cu; and
- very high-grade (0.50) Cu domain: >0.50% Cu.

13.5.4 Density

No apparent correlation exists between the metal grades and density, with scatterplots of S and other metals compared to density showing very little correlation, as demonstrated in Figure 13-10. This is presumably due to the generally disseminated nature of the sulphides and low overall proportion of the rock. The subtleties in the rock types is therefore likely to be the dominant control.

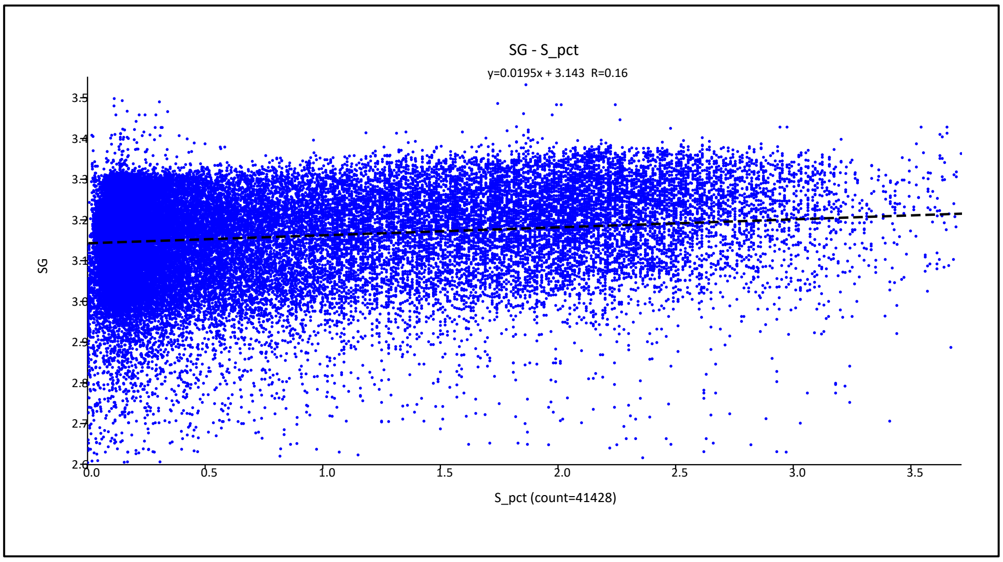


Figure 13-10: Scatterplot of sulphur (S) vs density (SG)

13.5.5 Resulting wireframes/meshes

The resulting domain wireframes/meshes were generated in Leapfrog Geo using an 'intrusion' method of modelling based on the various domaining criteria (above) and split into fault blocks using the structural model described above. The meshes are displayed in Figure 13-11 for copper and Figure 13-13 for nickel.

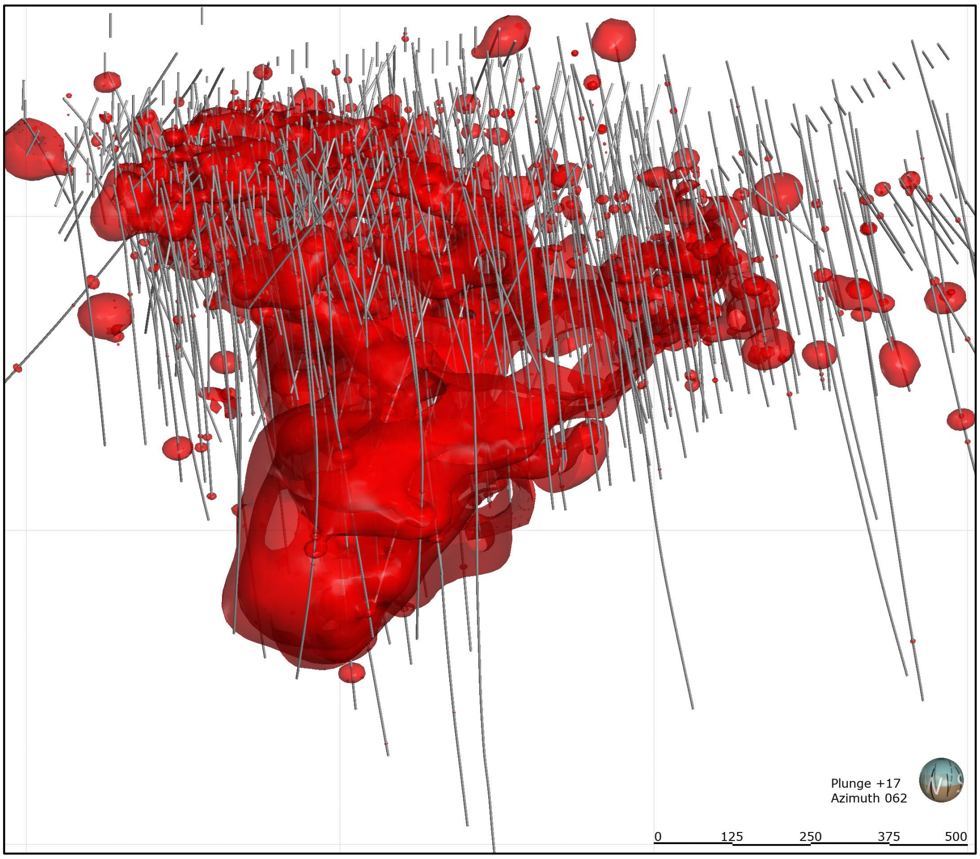


Figure 13-11: Copper low-grade domain meshes

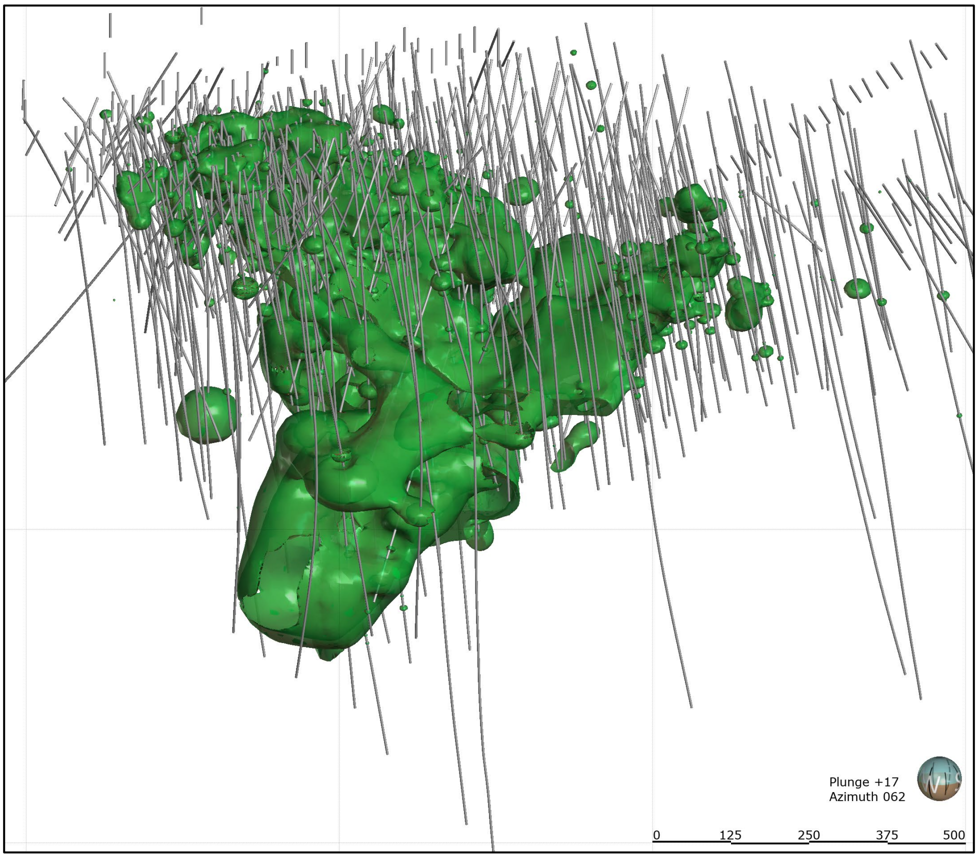


Figure 13-12: Copper high-grade domain meshes

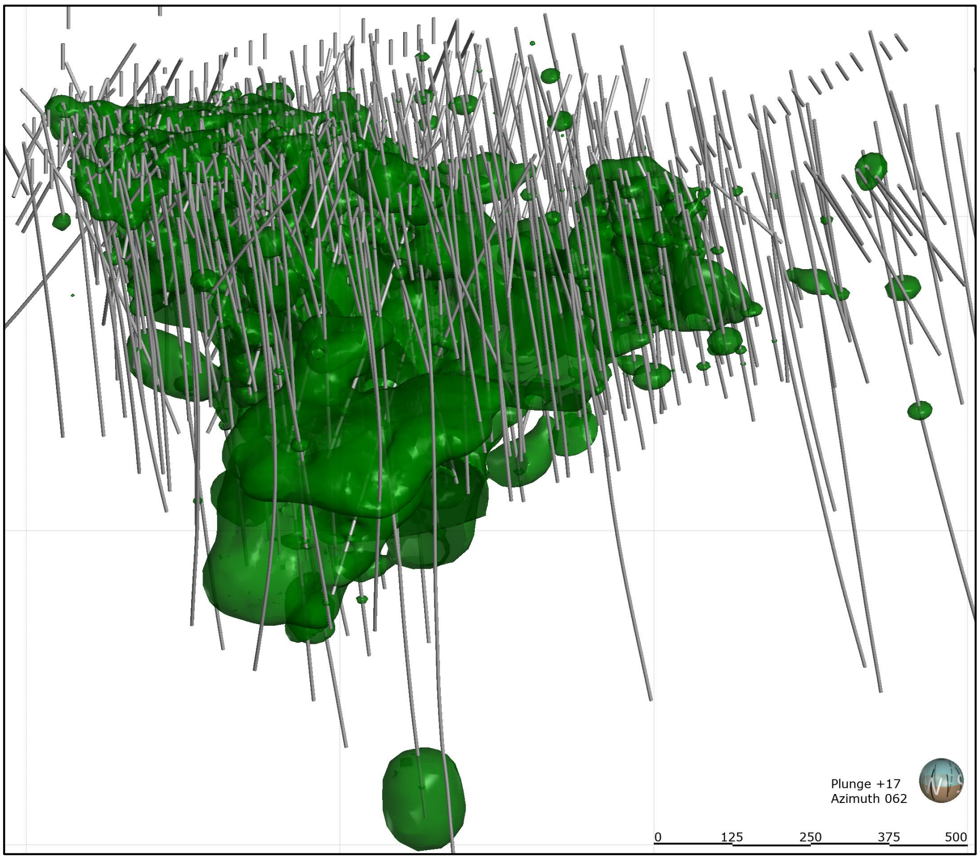


Figure 13-13: Nickel low-grade domain meshes

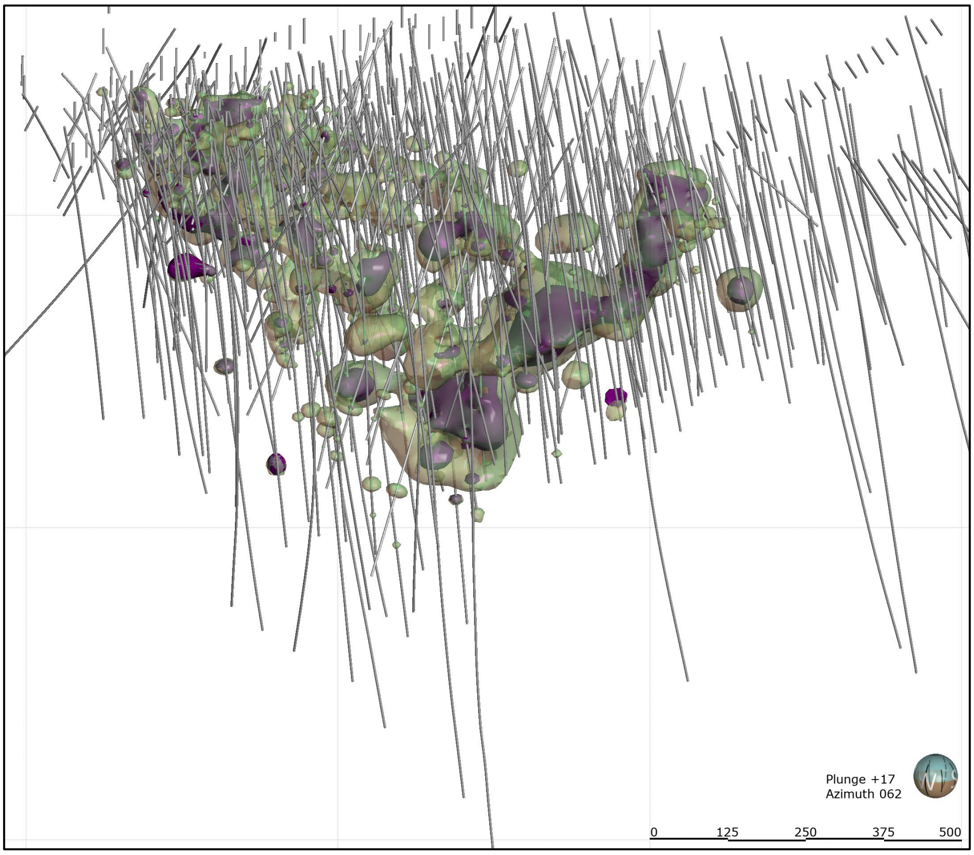


Figure 13-14: Nickel high-grade (green) and PGE (purple) domain meshes

13.5.6 SRK comments on domaining

In order to maintain consistency between support used for modelling and estimation and ensure contacts are adhered to, SRK suggests using the same composite length for geological modelling and grade interpolation in future. This is not considered to be a material issue by SRK.

Domaining is driven by modelling of grade shells, due to the nature of the mineralisation, and host lithology. The dominant control on the distribution of the mineralisation appears to be the disseminated nature of the mineralisation, in association with the complex fault morphology. Due to the nature of grade shell modelling, the resultant morphology is typically complex, but this is generally consistent with the data trends observed in the exploration data, and the closer spaced reverse circulation grade control drilling.

The focus of structural modelling has been on defining large scale faults which offset the mineralisation and have a certain degree of control on localising the mineralisation. Faulting also impacts on the amount of talc encountered, which is a key parameter for processing plant productivity (talc domaining is to be completed as part of the 2020 MRE). The structural modelling completed which supports the 2018 MRE and also the 2019 Mineral Resource statement herein, also includes a matrix which indicates the data used to support the modelling of each of the structures. SRK considers this approach to be reasonable, and notes that this can be further refined to illustrate the risk associated with each of the major structures, and how the risk may impact on other disciplines, such as geotechnical or metallurgical factors. There is no strong zonation seen within the model and so the use of separate Ni and Cu domains may not be required.

13.6 Statistical and geostatistical studies

13.6.1 Statistical analysis

Initial univariate statistical analysis is shown in Table 13-1 and Table 13-2 for the combined nickel and copper domains, respectively, and split by fault block ("FB"). Notably, high coefficients of variation ("CoV"; standard deviation/mean) are reported for precious metals and sub-domaining of the melonite-rich PGE mineralisation will be attempted as part of the 2020 MRE. The overall low CoV show that the grade clustering procedure has created relatively stationary statistical domains.

Table 13-1: Initial length weighted and domained statistics for Ni(S) grade shell assays (CoV >1.5 = orange)

FB	FIELD	NRECORDS	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	COV
	NISPCT	94171	93675	0.000182	3.83	0.05	0.06	1.21
	AUPPM	94171	94115	0	30.00	0.02	0.10	6.47
	PTPPM	94171	55489	0	6.16	0.04	0.10	2.30
	PDPPM	94171	94115	0	30.00	0.02	0.11	5.18
	COSPPM	94171	39362	0.54	2000.00	45.35	51.72	1.14
1	NISPCT	4668	4666	0.01	16.76	0.27	0.28	1.02
	AUPPM	4668	4666	0	1.97	0.09	0.10	1.07
	PTPPM	4668	3072	0.002	13.40	0.28	0.48	1.73
	PDPPM	4668	4666	0	3.18	0.18	0.26	1.43
	COSPPM	4668	1497	13.8	1850.00	126.33	79.12	0.63
2	NISPCT	19931	19922	0.00164	3.17	0.29	0.15	0.51
	AUPPM	19931	19851	0	4.70	0.13	0.12	0.95
	PTPPM	19931	15038	0	7.89	0.31	0.32	1.05
	PDPPM	19931	19737	0	8.69	0.19	0.23	1.17
	COSPPM	19931	7722	5	1820.00	122.47	65.79	0.54
3	NISPCT	78	77	0.05007	0.29	0.19	0.04	0.20
	AUPPM	78	77	0.002	0.11	0.01	0.02	2.51
	PTPPM	78	77	0.003	0.26	0.03	0.04	1.68
	PDPPM	78	77	0.003	0.15	0.02	0.02	1.19
	COSPPM	78	7	102	177.00	132.00	24.01	0.18
4	NISPCT	567	567	0.00741	2.70	0.30	0.27	0.90
	AUPPM	567	567	0.002	2.07	0.07	0.10	1.47
	PTPPM	567	306	0.005	0.68	0.12	0.11	0.91
	PDPPM	567	567	0.004	1.23	0.13	0.11	0.83
	COSPPM	567	297	5	1950.00	135.11	141.21	1.05
1;2;3;4	NISPCT	25244	25232	0.00164	16.76	0.28		
	AUPPM	25244	25161	0	4.70	0.12		
	PTPPM	25244	18493	0	13.40	0.30		
	PDPPM	25244	25047	0	8.69	0.19		
	COSPPM	25244	9523	5	1950.00	123.48		

Table 13-2: Initial length weighted and domained statistics for Cu grade shell assays (CoV > 1.5 = orange)

FB	FIELD	NRECORDS	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	COV
	CUPCT	89669	89660	0	16.70	0.05	0.13	2.60
	SPCT	89669	85977	-0.002	33.40	0.51	0.89	1.73
1	CUPCT	5264	5261	0.00408	2.95	0.35	0.21	0.59
	SPCT	5264	5081	0.02	27.80	1.79	1.12	0.63
2	CUPCT	23380	23372	0.00095	4.78	0.41	0.24	0.60
	SPCT	23380	23089	0.00593	24.80	1.74	0.91	0.52
3	CUPCT	20	20	0.07	0.45	0.24	0.11	0.45
	SPCT	20	20	0.71	2.31	1.19	0.48	0.41
4	CUPCT	856	856	0.00647	2.30	0.26	0.18	0.68
	SPCT	856	856	0.02	21.60	1.63	1.37	0.84
1;2;3;4;	CUPCT	29520	29509	0.00095	4.78	0.39		
	SPCT	29520	29046	0.00593	27.80	1.75		

13.6.2 Evaluation of outliers

Log-probability plots and histograms were used to identify the presence of extreme outlier grades for the samples of each element. Outlier samples were reviewed visually for their location in relation to the surrounding data in order to assess their potential impact upon block grade estimates. Capping of outlier/anomalous values was used to reduce the CoV and eliminate the impact of high-grade sample populations that have not been domained separately in order to minimise the risk of high-grade samples affecting poorly informed block estimates. The grade caps presented in Table 13-3 were applied.

SRK suggests that in future the grade caps are applied after compositing; however, it is not considered a material issue to the grade estimate overall.

Table 13-3: Summary of grade cap values applied

FIELD	capped at	No of caps	%caps	dMean	%mean diff
NiS	1	116	0.46	0.00	-0.94
Au	0.5	130	0.70	0.00	-2.21
Pt	2.5	81	0.32	-0.01	-1.77
Pd (FB1)	1.5	44	0.74	0.00	-1.26
Pd (FB 2-4)	2.5	26			
CoS	300	77	0.31	-2.51	-2.08
Cu	1	275	0.94	0.00	-0.85
S	uncapped				
Density	3.4	94	0.23	0.00	-0.01

13.6.3 Absent data

Unassayed sections of the drill core were treated as core loss; that is, samples with missing assay results have been set to absent. This means if these samples were mineralised but just not assayed, the grade interpolation with be negatively biased and if they were barren the grade would be positively biased. This is an issue for <0.1% of the data within the mineralisation domains and so SRK does not consider this a material issue.

13.6.4 Compositing

The predominant sampling length is 3 m for RC holes and 2 m for DD holes. RC holes were excluded from the 2018 MRE; however, all assay data were composited to 3 m intervals (straight down from collar, assay table level) to be consistent with the grade control block model estimate. Compositing from the collar-down (and not within domains) dictates that any shell interpolation using these data be affected by any changes made to the compositing interval. Interval lengths less than 3 m were added to the previous interval.

Capped composite statistics were compared to sample raw statistics; the mean and CoV were reduced insignificantly for all grades being interpolated.

13.6.5 Variogram analyses and parameters

Experimental variogram analyses were carried out for all metals, sulphur and density. Initially it was attempted to establish variogram models for all Ni(S) and Cu grade ranges (as given by the Leapfrog grade shells) within each of the four fault blocks. It became apparent that this approach means that too few composites are available to inform reliable variogram models. In addition, this approach would make little if any geological sense as the metal grades are continuously distributed across the established grade ranges and form a single continuous sample population. An important improvement to be made in the future will be the development of a geological model on which geological domaining can be based.

13.6.6 SRK comments on statistical analysis

SRK strongly recommends that all intervals without assays inside the mineralisation wireframes must be investigated and the reason explained. They could be un-sampled for a number of reasons (core loss, lack of mineralisation, lost sample, lost assay) and the grade interpolation around these samples will be biased either negatively or positively depending on why there are no assay results. If the intervals are not core loss, then a default value (SRK recommends half the detection limit to assume it is unmineralised) must be used rather than the sample being ignored (such as it was in the 2018 MRE); however, the number of affected intervals represents <0.1% of the core drilling database and therefore not considered to be a material issue.

SRK considers the grade caps used to be appropriate although notes that capping should be undertaken after compositing, which may reduce the number of samples requiring capping.

In SRK's opinion, the variograms produced are appropriate given the mineralisation style. The lack of continuity within individual domains is like to be due to a lack of data, rather than an inherent aspect of the grade distribution. As such, the variograms used to estimate the grades are considered suitable, but will need review during the 2020 MRE process, to reflect any changes in interpretation.

13.7 Block Model

13.7.1 Block model framework

The horizontal block size of 10 m (X) by 10 m (Y) is a reflection of the grade control drill spacing and the fact that the Grade control model and the Mineral Resource model are combined into a single model. A 12 m block height was chosen as it reflects the operation's 24 m bench height.

In well-informed areas of the Mineral Resource area (<50 m) model, a block size of 10 x 10 x 12 m was deemed acceptable by LGC. Poorly informed areas with sparser drilling information generally require a larger block size, but these areas are generally located at great depth or have sub-economic grade. Due to the low variability for the most important economic commodities (Cu-Ni), this is not considered a material issue by SRK.

13.7.2 Grade estimation parameters

To define the optimum search parameters for estimation, kriging neighbourhood analysis (“KNA”) was undertaken. KNA, as presented by Vann *et al*. (2003), is used to refine the search parameters in the interpolation process to help reduce conditional bias of block estimates. The criteria considered when evaluating a search area through KNA, in order of priority, are:

- slope of regression of the ‘true’ block grade on the ‘estimated’ block grade;
- distribution of Kriging weight;
- proportion of negative weights; and
- Kriging variance

KNA provides a useful tool to optimise a search area. It is a useful tool to help determining an optimum search area for any estimation or simulation exercise.

A single parent cell size block was created in areas, well (<50 m spacing), reasonably (50 - 100 m spacing) and poorly (>100 m spacing) supported by drillhole information and estimations of the block were carried with varying numbers of samples and the result for Kriging efficiency and slope of regression were plotted in a graph. This procedure was repeated for all estimated variables.

As a result of the analysis, LGC used a minimum of 25 and a maximum of 45 composites for the estimation of Ni(S), 30 and 70 for Cu and between 15-30 and 30-50 for the other estimated variables, as shown in Table 13-4.

Table 13-4: Estimation search parameters

SDESC	SREFNUM	SMETHOD	SDIST1	SDIST2	SDIST3	SANGLE1	SANGLE2	SANGLE3	SAXIS1	SAXIS2	SAXIS3	MINNUM1	MAXNUM1	SVOLFAC2	MINNUM2	MAXNUM2	SVOLFAC3	MINNUM3	MAXNUM3
NISPCT	10	2	40.5	27	42.3	80	100	135	3	1	3	25	45	3	25	45	5	5	45
CUPCT	20	2	72	21.6	48.6	60	170	170	3	1	3	30	70	3	15	70	5	5	15
AUPPM	30	2	30.6	16.2	28.8	80	120	150	3	1	3	15	30	2	5	30	3	1	30
PTPPM	40	2	122	27.9	99.9	70	150	0	3	1	3	30	50	2	30	50	3	15	50
PDPPM	50	2	134	16.2	72.9	70	150	0	3	1	3	30	50	2	30	50	3	30	50
COSPPM	60	2	90	63	63	80	120	10	3	1	3	25	45	3	5	45	5	5	45
SPCT	70	2	72.9	41.4	36	10	10	20	3	1	3	30	50	4	15	50	7	15	50
DEN	80	2	152	67.5	67.5	10	10	170	3	1	3	30	50	3	15	50	5	15	50

The estimation process is guided by the four fault blocks and the Ni(S) and Cu grade shells. Ni(S), Au, Pt, Pd, and CoS were estimated within each of the four fault blocks’ combined Ni(S) grade shells. This means that all Ni(S) grade shells (low grade, high grade and “PGE grade”) were combined into four fault block mineralisation shells. They were surrounded by a single non-mineralised or “waste grade” block estimates.

Cu, S, and density were estimated following the same principles, but into the Cu grade shells.

13.7.3 Model validation

In order to check that the estimation strategy applied is appropriate, the model was validated using different techniques:

- visual validation of block estimates against informing composites;
- statistical comparison of raw composite against block model estimates;
- validation plots to compare the block model estimates against informing composites along different slices through the deposits; and
- review of the KNA statistics.

Visual Validation

Visual checks have been carried out by viewing cross-sections and plans in different orientations, with different composite and block estimates displayed. Figure 13-15 provides an example of a plan view, in which the block model and corresponding informing composites are plotted for Ni(S). Figure 13-16 shows a cross-section through the block model.

LGC considered there to be a good correlation between the informing composite drillhole data and the block estimates.

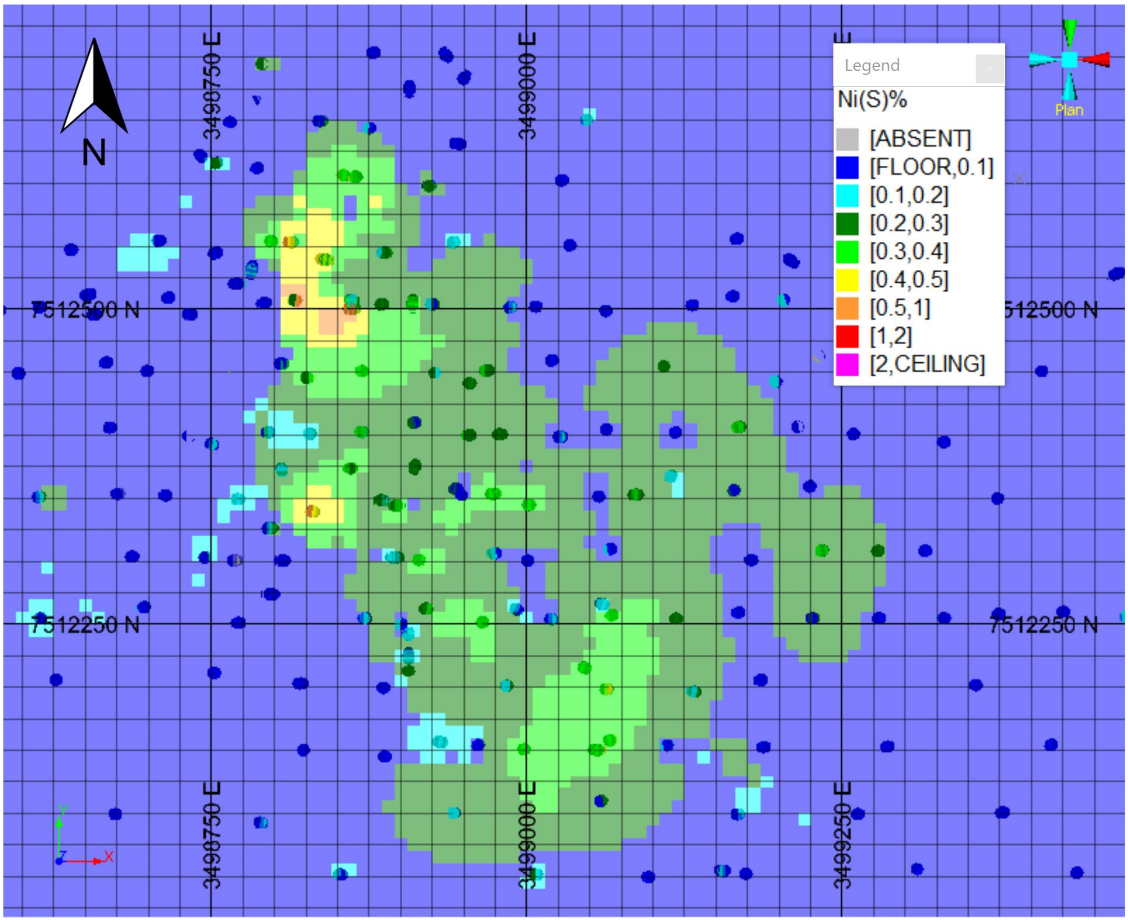


Figure 13-15: Plan view (-100 m Z) of Ni(S) estimate block model and sample composites (section thickness 10 m)

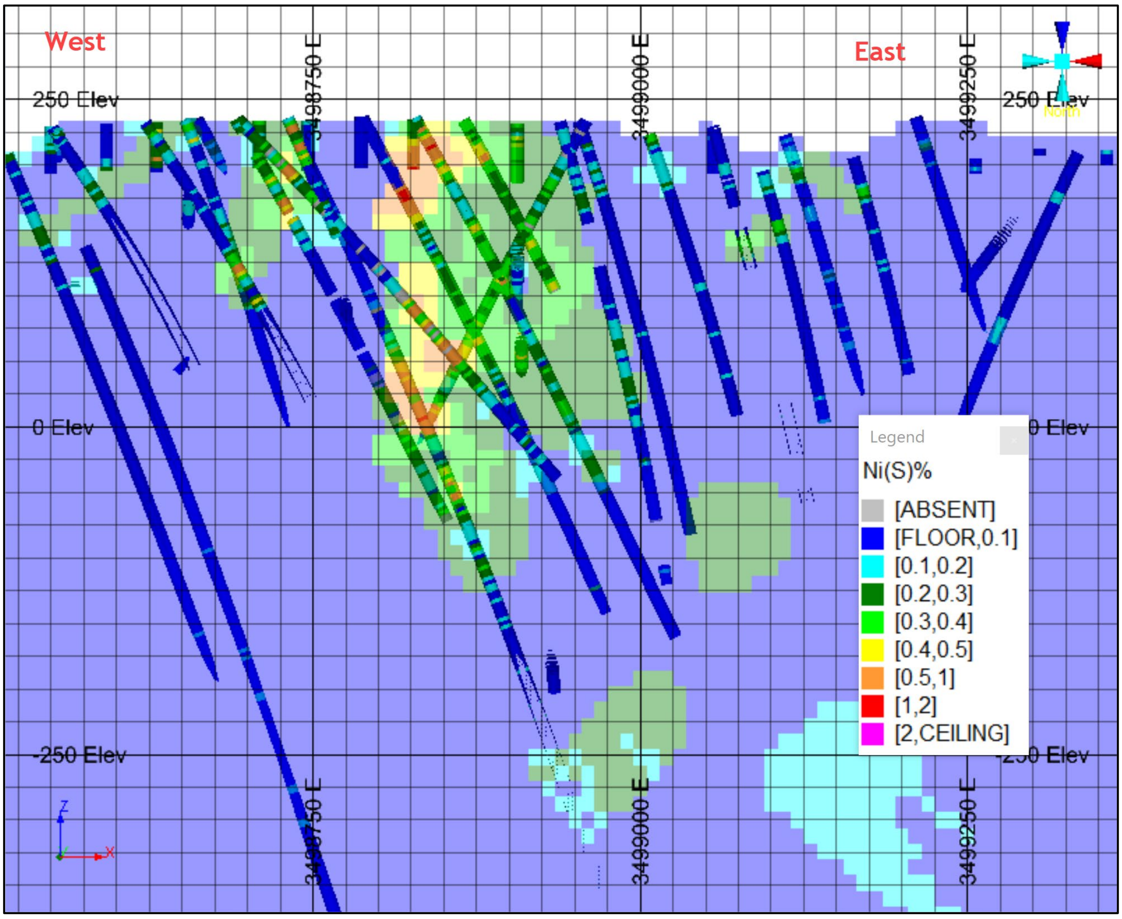


Figure 13-16: West-east cross section (Y: 7512500) of Ni(S) estimate block model and sample composites (section thickness 20 m)

Statistical Validation

In order to compare the model results to the input data, LGC has produced a comparison table of block model global mean grades and capped composite mean grades.

LGC noted that PGE grades appear to be significantly underestimated; this may be due to clustering effect and the relatively small numbers of composites per estimated fault block.

Fault block 2, holding the majority of available composites and having the largest volume of all fault blocks, shows a reasonable statistical comparison with the estimation input data.

In LGC's opinion, the fault block 2 estimates showed good correlation with the informing composites. The erroneous assignment of barren grades for S and Co(S) are insignificant in the context of the MRE.

KNA Statistics

For the review of KNA statistics block model, histograms of the slopes of regression ("SL") and Kriging efficiency ("KE") were produced for Cu and Ni(S). KE measures the difference between the true (unknown) block grades and the estimated grades. It varies between minus infinity and 100. KE values of greater than 50 are generally considered to be good.

The estimation yielded rather poor results for SL. The results for KE are on average just positive and therefore should be regarded as poor. In LGC's opinion, the estimation results should be therefore considered with caution.

Validation/Swath plots

As part of the validation process, the block model and input composites that fall within defined sectional criteria per estimation domain were compared and the results displayed graphically to check for visual discrepancies between grades on north-south and west-east sections as well as plan views.

Whilst this process does not truly replicate the samples used in the estimation of each block and does not account for anisotropies, the process of sectional validation quickly highlights areas of concern within the model and enables a more thorough and quantifiable check to be undertaken in specific areas of the model. Each graph also shows the number of samples available for the estimation. This provides information relating to the support of the blocks in the model. It is not unusual to see erratic graphs for the input data with large swings from minimum to maximum within a few sections, while the model graph averages these swings. The validation plots therefore give a qualitative indication to the amount of smoothing that has been introduced into the model.

Although LGC did not comment on the resulting plots, the images provided show that the block estimates are adequately smoothed when compared to the input composite data.

13.7.4 SRK comments on block modelling and estimation

The methodologies used to define and estimate the grade into the block model is consistent with industry best practice. SRK notes that there are some minor aspects which may be adjusted during the completion of the 2020 MRE, but these are generally minor, and are not considered to be material to the stated Mineral Resources. These aspects include:

- overall composite length (originally 2 m for mineralisation domain modelling compared to SRK's recommendation of 3 m), particularly if the reverse circulation data is included for grade estimation;
- review of capping strategy, to indicate whether the caps applied are relevant, or whether a high-grade restriction approach may be more appropriate;
- chosen block size, in comparison to drillhole spacing, particularly at depth;
- overall search strategy (such as minimum / maximum number of composites, rotations, number of samples per drillhole, etc); and
- use of hard / soft boundaries across faults and other controlling features.

SRK does not consider any of these aspects to be material; but considers that these aspects should be reviewed as part of the 2020 MRE. After review, it is likely that some of these aspects maybe unchanged, but SRK considers that review is warranted.

Density was estimated alongside the grade variables, although SRK notes that the amount of density sampling is relatively limited in comparison to the available grade data. Furthermore, SRK notes that no statistical comparisons are provided to indicate whether there is any relationship between grade and density. For the 2020 MRE, SRK recommends that this be undertaken, and should any trends be identified, ensure that these are reflected in the block model.

Validation of the block models used several industry standard techniques, including swath plots, statistical comparisons, and visual checks. The validation methodologies indicated that no significant biases had been introduced. SRK notes that the supplied swath plots indicate some areas of the block models whether the composite and block model grades are mildly divergent. SRK does not consider this to be material to the reported 2018 or 2019 Mineral Resource statements. SRK will review the block model validation for the 2020 MRE using a similar approach, and comment on any biases noted. SRK recommends a study of the level of smoothing, and optimisation of the estimate to match grade control sampling and reconciliation results.

13.8 Mineral Resource Classification

The block model was classified and reported using the guidelines of the PERC Standard (2017). Classification was primarily based upon confidence in the drillhole data, geological continuity, and the quality and confidence of the resulting kriged estimates. Geological confidence is supported by the available close spaced drill data and the mapping observations within the pit. Confidence in the kriged estimates was associated with drillhole grid spacing, QA/QC of sample data, KE and SL values.

Measured Mineral Resources were generally deemed appropriate in areas where the drill grid spacing was less than 25 m, KE was greater than 80%, and SL values were greater than 0.8. Indicated Mineral Resources were assigned to block estimates where the drill grid was between 25 and 75 m, with KE between 60 and 80%, and SL values greater than 0.6. In addition, blocks meeting the Measured Mineral Resource criteria above but with low-quality density estimates or low confidence in the geological model were assigned to Indicated Mineral Resources.

Block estimates that did not meet the Measured or Indicated Mineral Resource criteria and that were within 100 m of a single drillhole with geological continuity, were assigned to the Inferred Mineral Resource category. Typically, blocks designated as Inferred Mineral Resource had KE greater than 40% and SL values greater than 0.4.

The block model coloured by Mineral Resource classification category is provided in Figure 13-17.

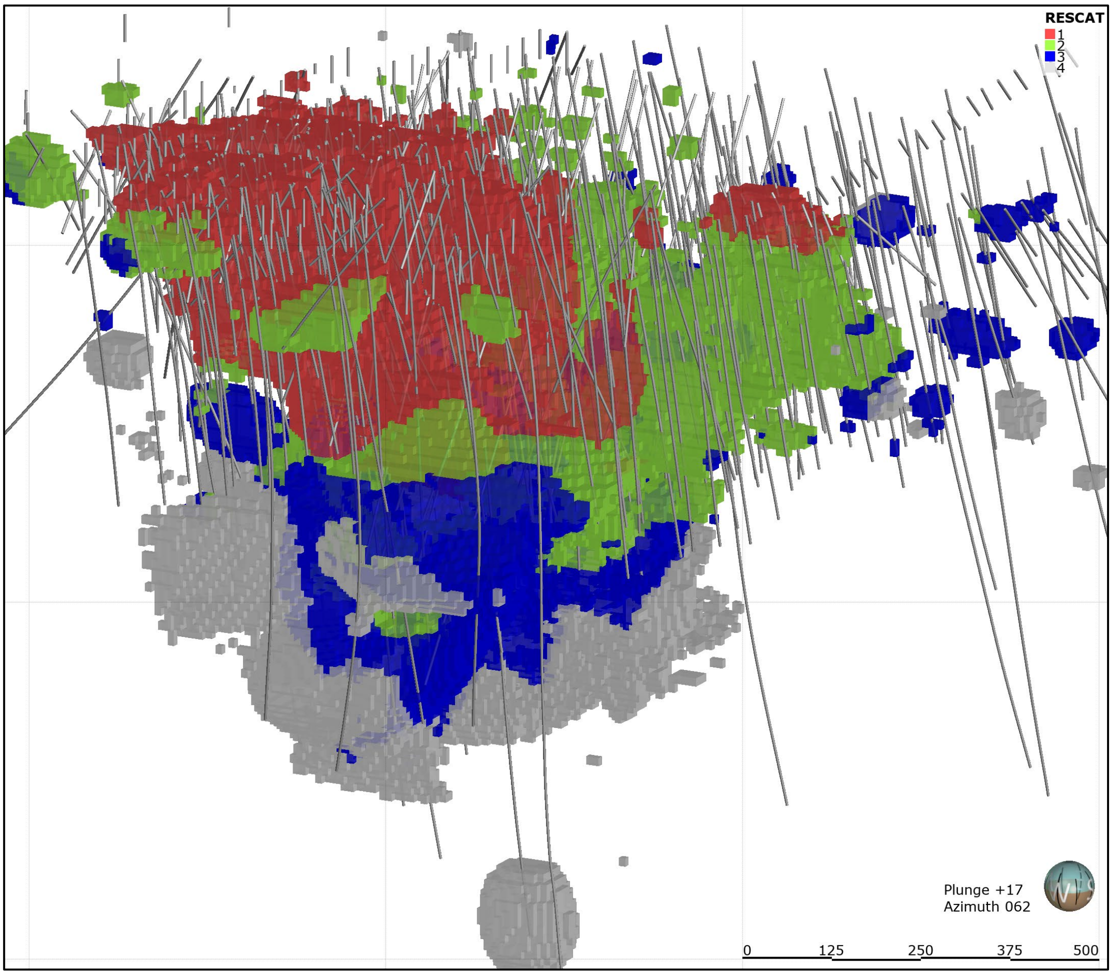


Figure 13-17: 3D view of the block model (mineralisation blocks only) coloured by Mineral Resource classification category (1 = Measured, 2 = Indicated, 3 = Inferred, 4 = Unclassified)

13.8.1 SRK comments on Mineral Resource classification

The classification approach used for the 2018 model is broadly consistent with that used for the 2016 model. SRK agrees with the approach taken, and as such, has made no changes for the 2019 Mineral Resource statement. As additional drilling will be available for the 2020 MRE, SRK will review the classification in conjunction with the new drilling, and make any changes, as considered appropriate. This will include comparison of using DD and RC together or separately.

A minor number of blocks designated as waste within the block model were classified as Inferred and Indicated Mineral Resources and reported. These blocks were estimated using an unconstrained grade estimate outside of the mineralisation wireframes and in SRK's opinion should not be classified and reported due to the lack of demonstrated geological continuity. SRK has subsequently re-classified these blocks as waste for the 2019 Mineral Resource statement.

13.9 Block Model Reconciliation

An internal report was provided by Boliden dated September 2019, recording the reconciliation of the 2016 and 2018 MRE block models to the grade control model and actual production data. The results comparing tonnage, Ni and Cu grade are displayed in Figure 13-18 to Figure 13-20, respectively. The results indicate that the MRE 2018 block model performed reasonably well on a monthly basis, which gives SRK further confidence in the quality of the estimated block model and confirms that the classification categories are satisfactory.

SRK notes that the grade reconciliation for Ni(S) for the full 12 months of 2019 indicates a slight under-performance (total 2019 results 0.22% in trucking data compared to 0.24% in the 2018 model for a 10% difference). This may be a result of several factors, and SRK considers this difference to be within the scale as to be expected for the level of confidence in the Mineral Resource model. The grade reconciliation, however, should be carefully monitored during 2020, and reviewed after a further 12 months of production, to ensure that the classification applied to the model is still warranted. This review should include comparisons with the 2016 MRE, the 2018 MRE, and the 2020 MRE.

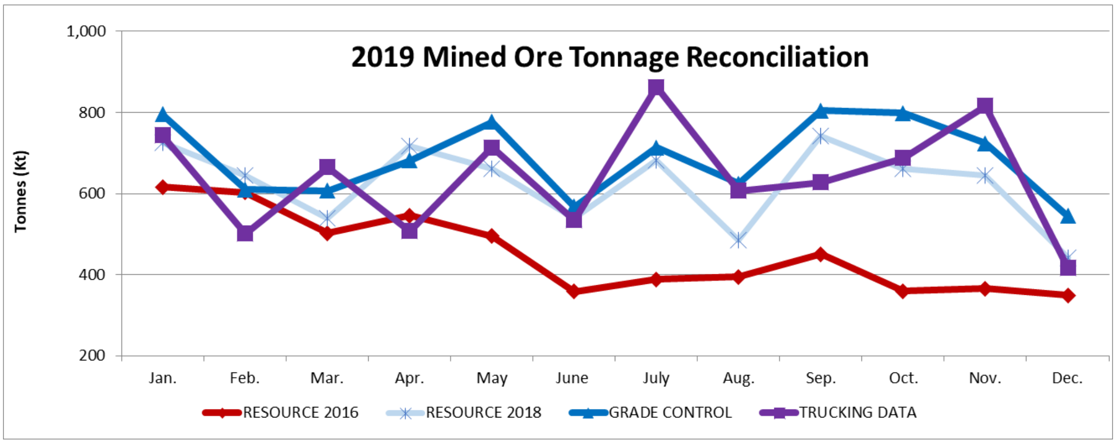


Figure 13-18: 2019 tonnage of ore mined from MRE 2016 block model, MRE 2018 block model, grade control block model and production (truck)

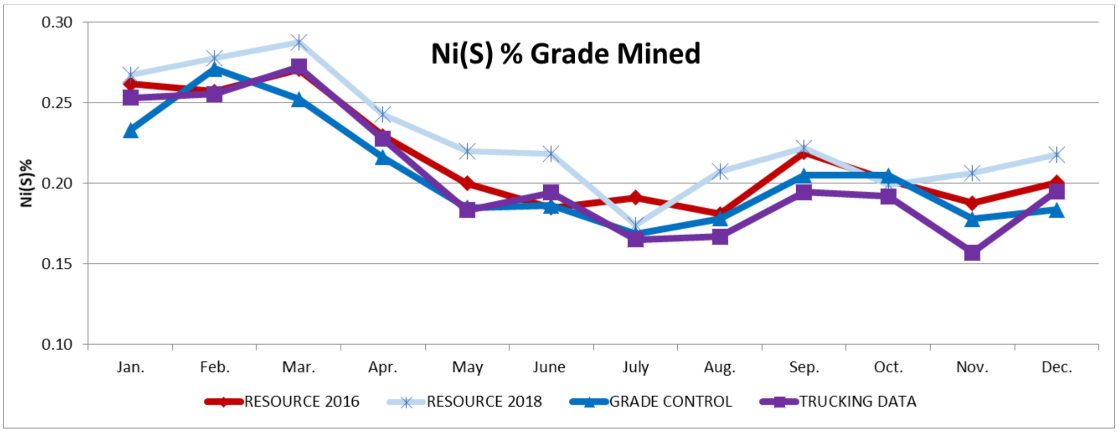


Figure 13-19: 2019 Ni(S)% grade of ore mined from MRE 2016 block model, MRE 2018 block model, grade control block model and production (truck) data

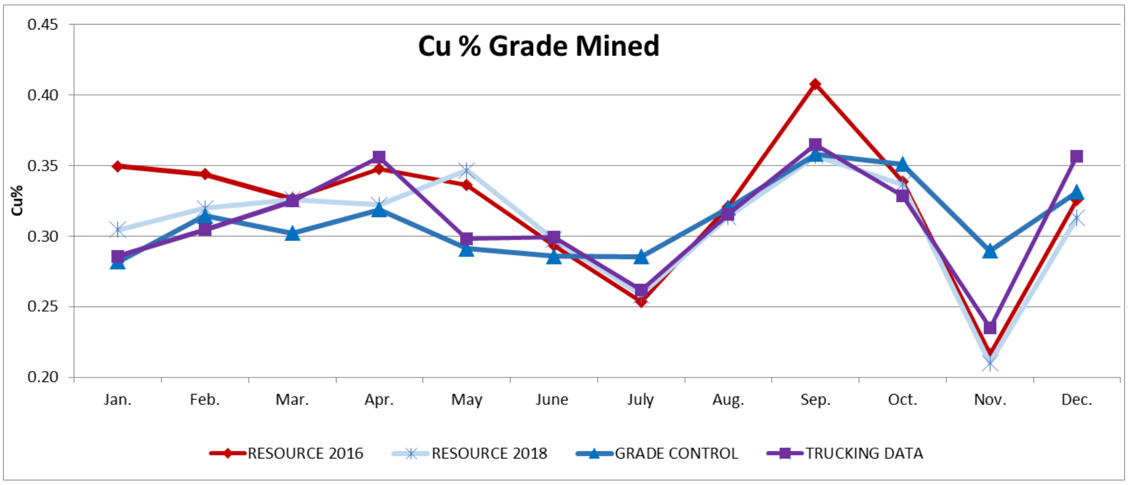


Figure 13-20: 2019 Cu% grade of ore mined from MRE 2016 block model, MRE 2018 block model, grade control block model and production (truck)

13.10 Assessment of Reasonable Prospects for Eventual Economic Extraction

13.10.1 Pit optimisation

The 'reasonable prospects for eventual economic extraction' ("RPEEE") requirement in the PERC Standard generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade considering reasonable extraction scenarios and processing recoveries. In order to meet this requirement, Boliden considers that the majority of the modelled Kevitsa mineralisation is amenable for open pit extraction.

In order to determine the quantities of mineralised material demonstrating RPEEE by an open pit, Boliden used a pit optimiser and reasonable mining and financial assumptions to evaluate "reasonably expected" to be mined from an open pit.

The optimisation completed in late 2018 (Ojanen, 2019) was carried out in Whittle 4.7.2, which uses a LERCHS-GROSSMAN graph-based algorithm to define nested pit shells from a mineralisation block model.

The optimisation parameters, described in detail in Section 14.3, were selected and simplified based on mining experience from Kevitsa but also to reflect a reasonably optimistic sense about the potential for the deposit to have future prospects for economic extraction. An undiscounted revenue factor (revenue factor = 1) optimised pit generated from the selected parameters was selected. This pit shell includes all cash positive blocks irrespective of the time value of money. Block selection for inclusion within the shell was carried out by cash-flow, so no cut-off was applied during the optimisation process. Boliden considers that the blocks located within the undiscounted revenue factor = 1 pit envelope demonstrate RPEEE and can be reported as a Mineral Resource.

The reader is cautioned that the results from the pit optimisation are used solely for the purpose of testing the RPEEE by an open pit and do not represent an attempt to estimate Mineral Reserves. The results are used to assist in the preparation of a Mineral Resource statement and to select an appropriate resource reporting cut-off grade.

The resulting pit shell used to report the 2018 and 2019 Mineral Resource statements is shown in Figure 13-21.

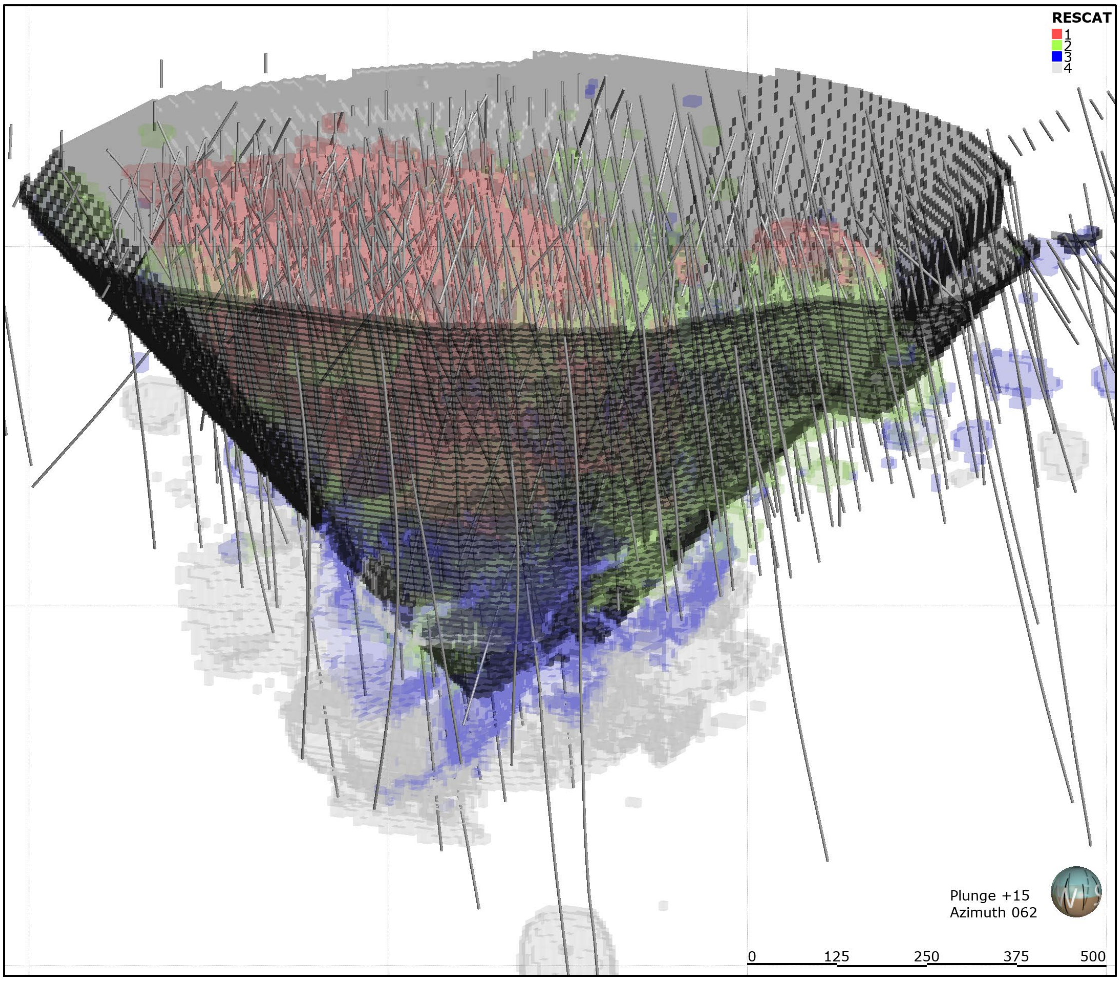


Figure 13-21: Whittle pit shell used for Mineral Resource reporting and block model coloured by Mineral Resource Classification

13.10.2 Metallurgical factors

The Kevitsa mineralisation is processed through the process plant which exists on site. The plant produces two concentrates with grades of approximately 9% Ni and 23% Cu as main payable elements. By-products include Au, Pt, Pd, and Co.

NSR values used in the Whittle optimisation process as selling parameters to enable block selection by cash flow, and thereby define cash-positive blocks.

The basis for these selling factors is derived from the NSR formula for Kevitsa, which in turn is based on process recovery figures from the process plant as well as general terms for payables and deleterious elements. Prices are set from Boliden's Long-Term Price ("LTP") outlook for 2019 onwards (more details on the NSR calculations and inputs are provided in Section 14.3). Figures used for the Kevitsa optimisation are a simplified, yet optimistic outlook on the possible price variations on the revenue generating elements. The NSR calculation (in EUR) is provided in Equation 13-1.

Equation 13-1: NSR 2018 cut-off calculation

$$NSR\ (2018) = (60 \times\ NiS) + (42 \times Cu) + (6 \times Pt) + (6 \times Pd) + (9 \times Au) + (50\ \times CoS)$$

13.10.3 Mineral Resource 2018 cut-off grade

An in situ cut-off value used for the 2018 Mineral Resource reporting by LGC was determined based upon the value of the material which would cover processing costs including a diluting fraction. On this basis, a cut-off value of EUR 10/t was chosen which corresponds to a 0.16% Ni(S) based on the NSR value strictly for Ni(S). This in turn was applied to blocks with Cu grades based on the following Ni(S) equivalent formula (Equation 13-2).

Equation 13-2: Nickel sulphide equivalent cut-off calculation

$$\mathrm{NiEq}\ (\%) = Ni(S)\ (\%) + 0.60\ Cu\ (\%)$$

The equivalency formula is based on a combination of in situ metal grades, process recoveries, and the relative value of Ni and Cu concentrates which are produced at Kevitsa. A final cut-off grade of 0.16% NiEq was subsequently used to report the 2018 Mineral Resource statement.

13.10.4 Mineral Resource 2019 cut-off grade

SRK considers the use of a NiEq to be unnecessary due to the more robust calculation provided by the NSR value and so has opted to use the NSR values. For the 2019 reporting, adjustments were made to the NSR calculation based on the predicted mining and processing parameters for 2020 onwards (Equation 13-3; described in detail in Section 14.3).

Equation 13-3: NSR for blocks planned to be mined from 2020 onwards

$$\mathrm{NSR\_LTP} = (64.47 \times \mathrm{NiS}) + (43.83 \times \mathrm{Cu}) + (6.80 \times \mathrm{Pt}) + (9.18 \times \mathrm{Pd}) + (8.97 \times \mathrm{Au}) + (68.32 \times \mathrm{CoS})$$

A cut-off of EUR 10 / t was used by SRK to align with the Mineral Reserve reporting procedure, which has been tested against the financial model produced by SRK to confirm the suitability for Mineral Resource reporting (refer to Section 18).

13.11 Mineral Resource Statement

The 31 December 2019 Mineral Resource statement for Kevitsa prepared by SRK is presented in Table 13-5 (inclusive of Mineral Reserves) and Table 13-6 (exclusive of Mineral Reserves) with notes explaining the reporting procedure provided underneath.

Table 13-5: Mineral Resource Statement (inclusive of Mineral Reserves) effective of 31 December 2019*

Mineral Resource Category	Tonnes (Mt)	Sulphide Nickel (%)	Total Copper (%)	Gold (g/t)	Platinum (g/t)	Palladium (g/t)	Sulphide Cobalt (%)
Measured	88.2	0.24	0.35	0.10	0.20	0.13	0.01
Indicated	189.5	0.25	0.34	0.09	0.19	0.12	0.01
Meas+Ind	**277.7**	**0.25**	**0.34**	**0.10**	**0.19**	**0.12**	**0.01**
Inferred	19.2	0.22	0.33	0.06	0.13	0.09	0.01

**In reporting the Mineral Resource Statement, SRK notes the following:*

- *Mineral Resources have an effective date of 31 December 2019*
- *Competent Person for the declaration of Mineral Resources is Dr Lucy Roberts, an employee of SRK.*
- *Reported Mineral Resources are below the mined topography, dated 31 December 2019.*
- *Mineral Resources are reported inclusive of Mineral Reserves.*
- *Mineral Resources are reported as undiluted, with no mining recovery applied in the Statement. Assumptions for mining factors (mining and selling costs, mining recovery and dilution, pit slope angles) and processing factors (metal recovery, processing costs), during the optimisation process only.*
- *SRK considers there to be reasonable prospects for economic extraction by constraining within an optimised open pit shell constructed using long term market forecast commodity prices.*
- *Mineral Resources are reported above the optimised pit shell and above a Net Smelter Return ("NSR") marginal cut-off of EUR 10/t, which reflects the economic and technical parameters.*
- *Tonnages are reported in metric units, grades in percent or grams per tonne (g/t). Tonnages and grades are rounded appropriately.*
- *Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.*

Table 13-6: Mineral Resource Statement (exclusive of Mineral Reserves) effective of 31 December 2019*

Mineral Resource Category	Tonnes (Mt)	Sulphide Nickel (%)	Total Copper (%)	Gold (g/t)	Platinum (g/t)	Palladium (g/t)	Sulphide Cobalt (%)
Measured	26.5	0.23	0.33	0.08	0.16	0.10	0.01
Indicated	112.9	0.23	0.34	0.08	0.14	0.09	0.01
Meas+Ind	**139.4**	**0.23**	**0.34**	**0.08**	**0.15**	**0.09**	**0.01**
Inferred	17.8	0.22	0.33	0.06	0.13	0.08	0.01

**In reporting the Mineral Resource Statement, SRK notes the following:*

- *Mineral Resources have an effective date of 31 December 2019*
- *The Competent Person for the declaration of Mineral Resources is Dr Lucy Roberts, an employee of SRK.*
- *Reported Mineral Resources are below the mined topography, dated 31 December 2019.*
- *Mineral Resources are reported exclusive of Mineral Reserves.*
- *Mineral Resources are reported as undiluted, with no mining recovery applied in the Statement. Assumptions for mining factors (mining and selling costs, mining recovery and dilution, pit slope angles) and processing factors (metal recovery, processing costs), during the optimisation process only.*
- *SRK considers there to be reasonable prospects for economic extraction by constraining within an optimised open pit shell constructed using long term market forecast commodity prices.*
- *Mineral Resources are reported above the Whittle pit shell and above a Net Smelter Return ("NSR") marginal cut-off of EUR 10/t, which reflects the economic and technical parameters and below the mine design pit shell used to report the Mineral Reserve.*
- *Tonnages are reported in metric units, grades in percent or grams per tonne (g/t). Tonnages and grades are rounded appropriately.*
- *Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.*

13.12 Grade Sensitivity Analysis

The Mineral Resources of the Kevitsa Mine are sensitive to the selection of the cut-off grades. To illustrate this sensitivity, the block model quantities and grade estimates within the conceptual pit used to constrain the Mineral Resources are presented at different Ni(S) and Cu cut-off grades in Figure 13-22 and Figure 13-23, respectively. The reader is cautioned that the figures presented in this table should not be misconstrued with a Mineral Resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of a cut-off grade.

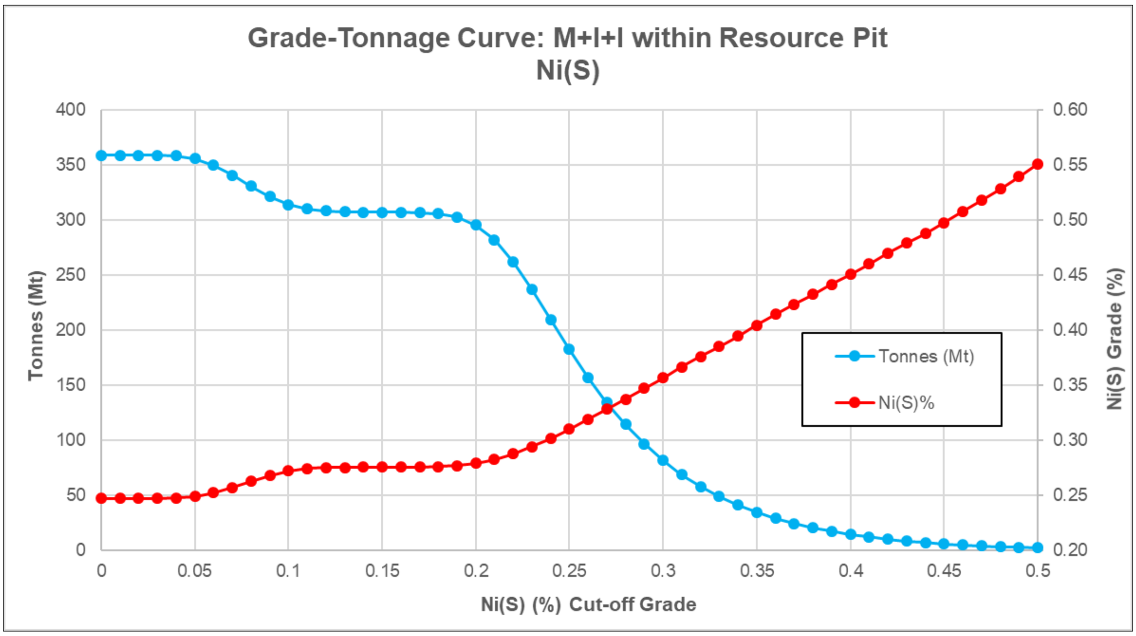


Figure 13-22: Grade-tonnage curve for sulphidic nickel in Measured+Indicated+Inferred blocks within the Mineral Resource pit shell

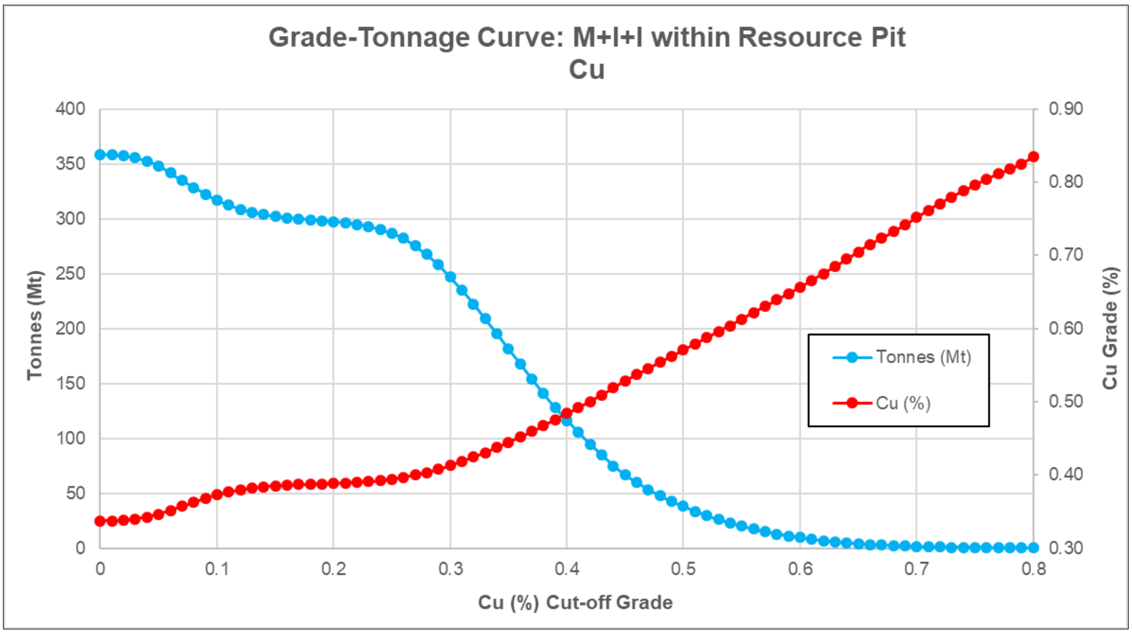


Figure 13-23: Grade-tonnage curve for total copper in Measured+Indicated+Inferred blocks within the Mineral Resource pit shell

13.13 Mineral Resource Reconciliation 2018-2019

The 2019 Mineral Resource statement was based on the MRE completed in 2018. Since the previous statement on 31 December 2018, four main changes have occurred, which are depicted in the waterfall charts in Figure 13-24 to Figure 13-26:

- **Depletion**: material has been extracted from the open pit. This has been depleted from the 2018 MRE block model. The total material depleted from the model using topographic surveys completed at end-2018 and at end-2019 equates to 8.3 Mt which is slightly higher than the 7.5 Mt reported by the mine production team. This may be due to a number of factors but mainly the difference between NSR cut-off for reporting ore in the Mineral Reserve (NSR = EUR 15 /t) compared to Mineral Resource (NSR = EUR 10 /t).
- **Methodology/refinement**: Material outside of modelled domains (considered as waste) but with estimated grades and a classification category of Measured, Indicated, or Inferred assigned were not reported as part of the 2019 Mineral Resource statement but were in 2018. This resulted in a loss of 7.9 Mt split across Measured, Indicated, and Inferred Mineral Resources.
- **Commodity**: changes to the approach used in determining the cut-off grade (from a NiEq cut-off grade to an NSR) has resulted in a slight decrease of some 0.5 Mt.
- **Stockpiles**: 153 kt of material was reported and included from the stockpiles.

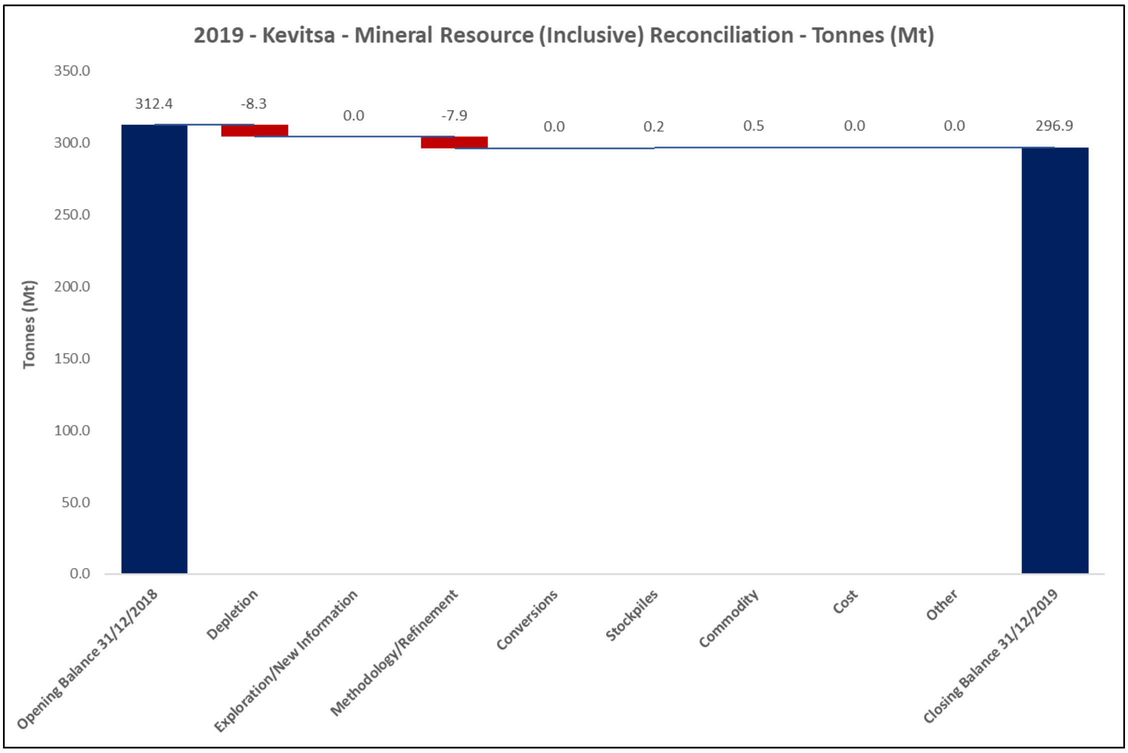


Figure 13-24: Mineral Resource (inclusive) 2018-2019 tonnage waterfall chart

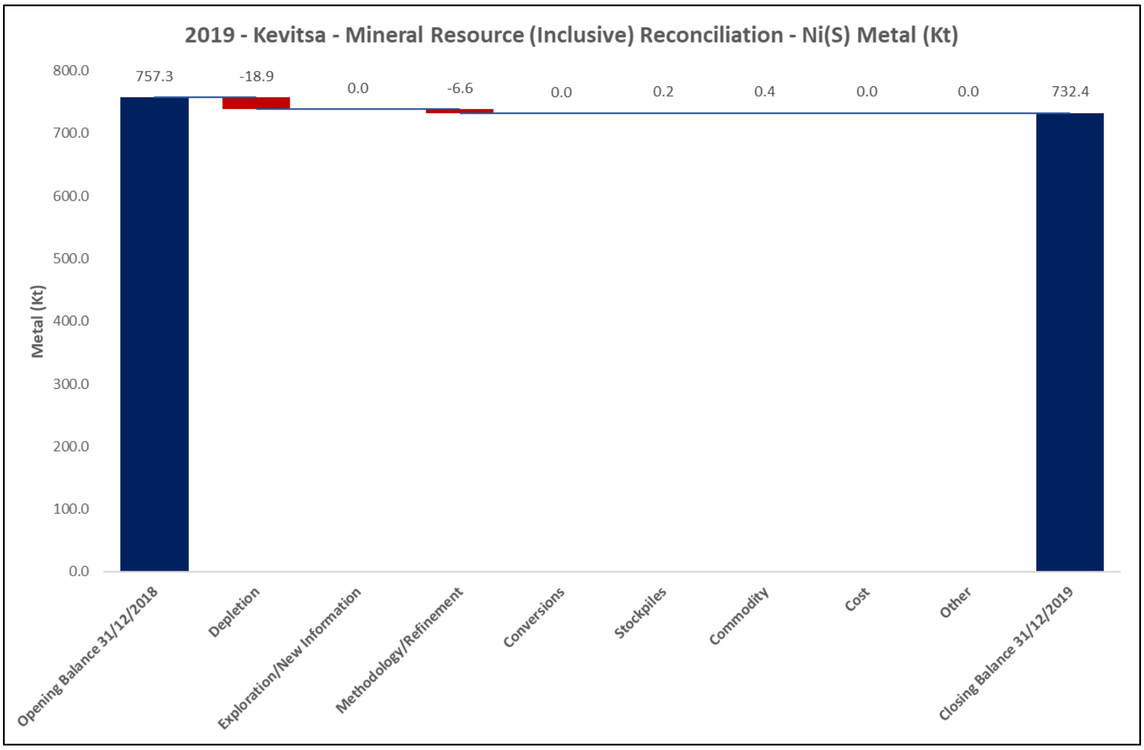


Figure 13-25: Mineral Resource (inclusive) sulphidic nickel metal waterfall chart

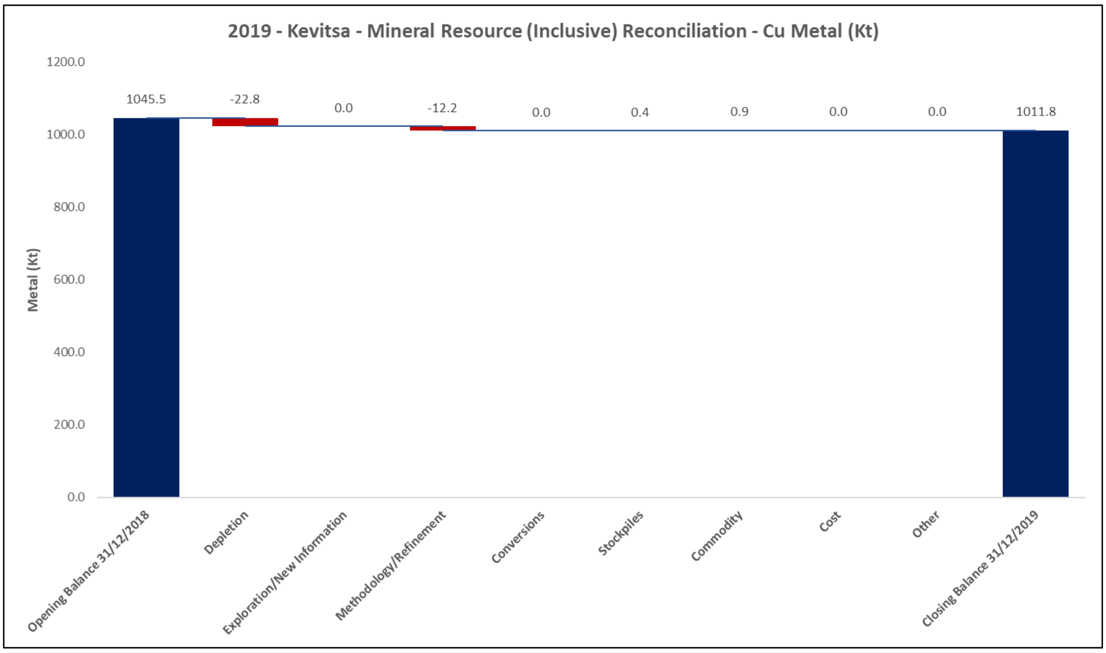


Figure 13-26: Mineral Resource (inclusive) copper metal waterfall chart

13.14 Exploration Potential

The Kevitsa disseminated mineralisation is known to extend beyond the planned Stage 4 design pit. Additional drilling is being conducted to evaluate the potential to expand the currently planned pit. Apart from the disseminated mineralisation, there is potential for higher grade massive and semi-massive ore, assumed to occur close to the basal contact of Kevitsa intrusion, beneath the known disseminated mineralisation. Drilling of these higher-grade targets has been sparse due to the depth (>800 m); however, more drilling is planned. Mining of such potential deep mineralisation would likely be in an underground scenario.

14 MINING METHODS

14.1 Introduction

Kevitsa is established as a large-scale conventional open pit mine, active in production since August 2012. The mine is largely owner operated, with bush-clearing and pre-stripping being completed by a local Finnish mining contractor.

The economic limit of the Mineral Resource was established through the Whittle 4D Pit optimisation process and tested in a strategic mine plan. From the pit optimisation, a final optimal pit shell was selected along with interim pushbacks shells.

Stage designs which incorporate ramps and bench geometry were designed according to the Geotechnical engineering criteria. The final designs and pushbacks were then scheduled in Deswik's interactive scheduler ("Deswik.IS") to produce a life of mine plan ("LoMp"). Based on the LoMp, the primary equipment fleet requirements were estimated from first principles, and the mining budget cost estimation was completed.

14.2 Operational Overview

14.2.1 Mining method

Mining at Kevitsa comprises of conventional open pit truck and shovel, preceded by drilling and blasting and followed by stockpiling and waste dumping.

Loading is completed by a large 36 m^3 electric hydraulic shovel, supported by smaller hydraulic shovels and wheel loaders. Production haulage is achieved by a fleet of 220 t diesel and 313 t diesel electric trucks.

Production drilling is completed by large 225 mm diameter down-the-hole ("DTH") rigs, supported by smaller 165 mm diameter rigs for smaller patterns and pre-splitting. Charging and blasting with emulsified explosives is completed by a blasting contractor.

Production is further supported by various auxiliary equipment which includes wheel loaders, road graders, bull dozers, water trucks, service trucks, lighting plants, and submersible pumps.

14.2.2 Historical mining

Historically, the mine has achieved 6.9 to 8.3 Mtpa ore mining and total mining tonnage of 28.1 to 42.5 Mtpa (Table 14-1). Mining has taken place since 2014 at average yearly stripping ratios of between 3.07 to 4.61.

Table 14-1: Historical mining production totals at Kevitsa 2014-2019

Parameter	2014	2015	2016	2017	2018	2019
Ore (Mt)	6.9	6.6	7.7	8.3	7.9	7.5
Waste (Mt)	21.2	30.4	31.9	34.2	33.5	33.5
Total (Mt)	**28.1**	**37.0**	**39.6**	**42.5**	**41.4**	**40.0**
SR (t:t)	3.07	4.61	4.14	4.12	4.24	4.47

14.3 Pit Optimisation

A pit optimisation study was completed by Boliden in February 2018 based on a 2016 resource model. In 2018, the MRE was updated which called for an update to the pit optimisation study which was completed early in early 2019. The 2019 pit optimisation study (Ojanen, 2019) determined that the pushback designs (Stage 2, 3 and 4 - based on the 2018 study) are still optimal, but that an additional cutback (Stage 5) can potentially be mined.

During 2019, various limitations were investigated that would prohibit the validity of an additional Stage 5 pushback. Although Stage 5 can be justified economically, it was identified that the current tailings and waste storage capacity are insufficient and further investigations into the feasibility of a Stage 5 will need to be completed. Whilst these investigations were underway, the mining of Stage 2, 3, and 4 commenced in 2019, and Boliden has taken the decision to issue Mineral Reserves based on the Stage 4 design as the final pit.

Since the Stage 4 design and the 2019 Mineral Reserves have been based on the 2018 pit optimisation study, in this section SRK has reviewed the economic input parameters and results. The pit optimisation input parameters, methodology and results were presented in the document - “BOL_MAIN-#1227092-v2-Kevitsa_Pit_Optimisation_February_2018.PDF”.

14.3.1 Mining model

The mining model which was used for the pit optimisation, “res_gc_con30012018.dm”, is a grade control model based on the 2016 Resource model. The mining model which includes Measured, Indicated, and Inferred Mineral Resource material was depleted by the end of year December 2017 topography for the pit optimisation. The model block dimensions and origin are shown in Table 14-2.

Table 14-2: Mining model dimension and origin

Description	Field
X-origin	3498285.0
Y-origin	7511250.0
Z-origin	-1014.0
X-dimension	10
Y-dimension	10
Z-dimension	12

14.3.2 Pit optimisation methodology

The calculation of a Whittle NPV, the usual criteria for selecting an optimal pit, is largely dependent on the discount rate and the high-level scheduling methodology applied in Whittle. Whittle produces nested pit shells with a discounted cashflow (“DCF”) for each nested pit. Three relative DCF are presented based on three different scheduling methodologies applied in Whittle:

Best*:* The best cash flow is achieved when each of the nested pit shells are mined in sequence. Such a sequence, although optimal for cash flow, is impractical since nested pit shells are often closely layered (like the layers of an onion) and would imply that thin pushbacks could be mined.

Specified*:* The specified cash flow is based on the mining engineer pre-selecting some of the nested pit shells which would represent more practical pushbacks to represent a mining sequence. A scheduling algorithm then determines optimal mining rates within the selected nested pushback shells.

Worst: The worst cash flow is achieved when the selected final pit shell is mined from top to bottom without any consideration for nested pit shells or pushbacks. This is undoubtedly would be practical but usually presents the lowest economic scheduling option.

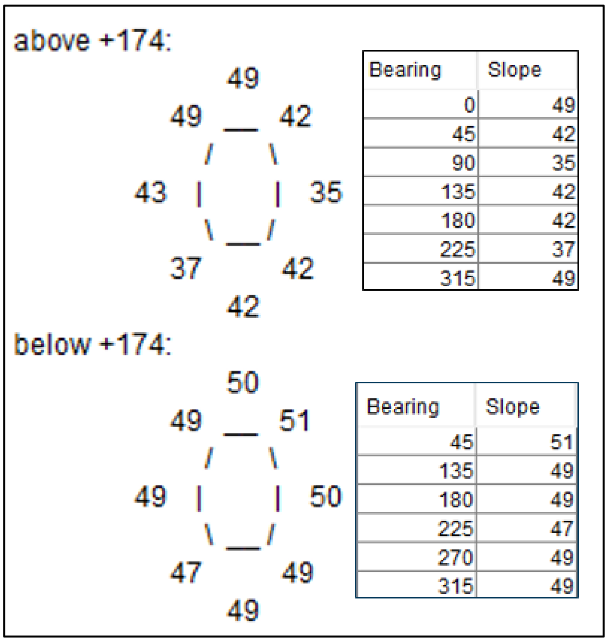
The selection criteria for the final pit at Kevitsa was guided by the pit which achieved the highest discounted cashflow (10%) in the specified case. SRK considers this as a good practice for selecting a robust pit shell to provide the best economics.

14.3.3 Pit optimisation input parameters

Geotechnical slope angles

Geotechnical slope angles were back calculated from the existing pit designs and used in Whittle as shown in Figure 14-1. Within the pit optimisation, distinction is being made between weathered and fresh material by elevation. Weathered material generally occurs above Z (elevation) >174 m, which requires lower Overall Slope Angles ("OSA"). The slope angles were applied as radial bearings in Whittle, which applies an overall slope angle in a radial position perpendicular to the slope face in a clockwise direction.

SRK considers the back calculation and application of existing slope angles from designed pits as good practice to ensure that the pit optimisation caters for the effect of ramps and geotechnical berms as far as possible.



Bearing	Slope
0	49
45	42
90	35
135	42
180	42
225	37
315	49

Bearing	Slope
45	51
135	49
180	49
225	47
270	49
315	49

Figure 14-1: Radial overall slope angles – geotechnical input parameters

Economic Input parameters

The economic input parameters used for the 2018 pit optimisation are shown in Table 14-3.

Mining costs were split into fixed and variable costs. Sustaining capital was included in the pit optimisation study for both mining and processing. The modifying factors are based on an historical reconciliation between actual mined and processed tonnages and grades and the resource model.

Processing recoveries and selling costs were incorporated into the NSR prices applied in Whittle. The 2017 long term metal prices applied for Cu and Ni (the main value driver metals) in the pit optimisation are comparable to the SRK sourced consensus market forecast prices in 2019 Q4 and are considered to be reasonable.

Mining costs incorporate a fixed cost which includes drilling, blasting, grade control, engineering, and mining related general and administrative costs. A variable component is included for vertical lift, incorporated in the mining model and increasing the mining cost with depth. The vertical lift estimate was based on a simplistic calculation using CAT 795 engine hour costs and a formula to calculate additional truck cycle times. The resultant variable cost is low in SRK's opinion (expected to be within a EUR 0.003-0.01/t/m range), and SRK recommends the revision thereof for future pit optimisations.

Pit shells were generated using ore selection by cashflow, in which Whittle evaluates a block based on the NSR values incorporated in each block in the mining model. The NSR formulas incorporated Payables and Deductibles as shown in Table 14-4 and processing recoveries in Table 14-5.

Table 14-3: Whittle economic input parameters February 2018

Input description	Input	Unit
Mining Costs		
Fixed: Drilling, Blasting, Loading, GC, Engineering, Admin etc.	1.51	EUR/t
Hauling fixed	0.36	EUR/t
Hauling (variable, vertical lift)	0.00116	EUR/t/m (vert)
Hauling Waste (fixed, additionally for waste/reject only)	0.127	EUR/t
Sustaining Capital (fixed, mining ton related costs)	0.25	EUR/t
Average Mining Cost (excl. additional for waste hauling)	**2.35**	**EUR/t (pit 10)**
Modifying factors		
Mining Dilution (Global Estimate)	7	%
Mining Recovery (Global Estimate)	93	%
Processing costs		
Milling	6.90	EUR/t
Overhead (environmental, general and administration (“G&A”))	1.60	EUR/t
Sustaining Capital (throughput related costs)	1.15	EUR/t
Total Processing Cost	**9.65**	**EUR/t**
Metal Prices		
Ni(S) - Metal Price	16,000	USD / t
Cu - Metal Price	6,200	USD / t
Au - Metal Price	1,200	USD / oz
Pt - Metal Price	1,150	USD / oz
Pd - Metal Price	750	USD / oz
Co(S) - Metal Price	14	USD / lb
NSR Prices		
Ni(S) - NSR Price	61.2	EUR / 10 kg
Cu - NSR Price	40.5	EUR / 10 kg
Au - NSR Price	8.9	EUR / g
Pt - NSR Price	7.54	EUR / g
Pd - NSR Price	4.7	EUR / g
Co(S) - NSR Price	27.0	EUR / 10 kg
Other		
Discount rate	10	%

Table 14-4: Payables and Deductibles incorporated in the NSR calculation

Parameter	Kevitsa Cu Concentrate		Kevitsa Ni Concentrate
Payables			
Cu (%)		Deduct 1 unit	Pay 80%
Ni (%)		-	Pay 90%
Co (%)		-	Pay 35%
Au (g)		Deduct 1 g	Pay 70% if content exceeds 1 g
Pd (g)	Pd <= 6.68	deduct 2 g	Pay 70% if content exceeds 1 g
	Pd > 6.68	pay 70%	
Pt (g)	Pt <= 6.68	deduct 2 gr	Pay 70% if content exceeds 1 g
	Pt > 6.68	pay 70%	
Deductibles			
Treatment Charges (“T/C”) (t)		USD 80.00	USD 190.00
Refining Charges (“R/C”) Cu (lb)		USD 0.08	USD 0.50
R/C Ni (lb)		-	USD 1.00
R/C Co (lb)		-	USD 3.00
R/C Au (oz)		USD 6.00	USD 35.00
R/C Pd (oz)		USD 15.00	USD 35.00
R/C Pt (oz)		USD 15.00	USD 35.00

Table 14-5: Processing Recoveries incorporated in the NSR calculation

Metal	Kevitsa Cu 23% Concentrate	Kevitsa Ni 9.2% Concentrate
Cu (%)	=27.078*Cu feed%+73.163	7.0
Ni (%)	-	=28.792*LN (NiS feed%)-496.1*NiS feed%^2+205.44*NiS feed%+92.682
Co (%)	-	60
Au (%)	40	12
Pd (%)	29	32
Pt (%)	26	27

14.3.4 Pit optimisation results

The pit optimisation results are shown in a nested pit shell graph as shown in Figure 14-2. The existing Stage 2 and Stage 3 designs were imported into Whittle, which formed Pit 1 and Pit 2 in the nested pit shell graph. Whittle was used to generate further nested pit shells beyond Pit 2 (Stage 3 and beyond).

The skin analysis for each pit shell in Figure 14-3 show the ore and waste quantities and specified NPV (at 10%) which was used in selecting the optimal pit. Pit 10 (the final selected pit) includes 137 Mt ore and 235 Mt waste.

The optimal pit was selected based on the maximum NPV for the specified case (EUR 935 M), which is Pit 10 in Figure 14-3. The optimal pit shell is occurring at the revenue factor 0.68 pit (calculated according to the Whittle “Specified” scheduling case; see 14.3.2). From the graph, the pit is selected at a point where a major increase in stripping would be required for mining to continue.

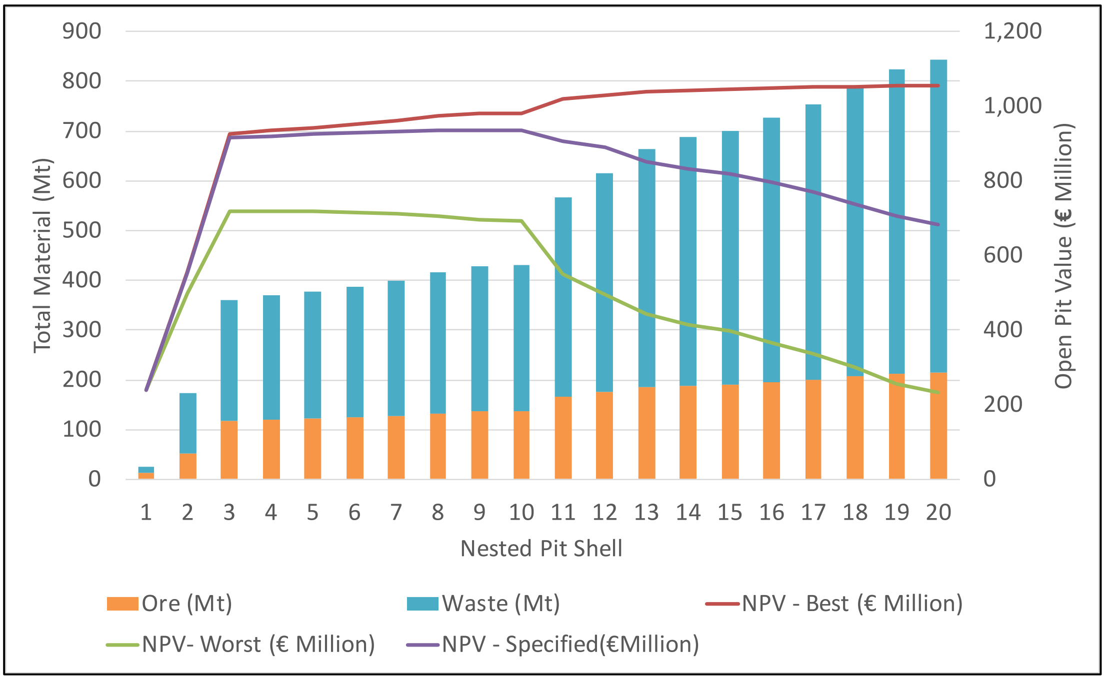
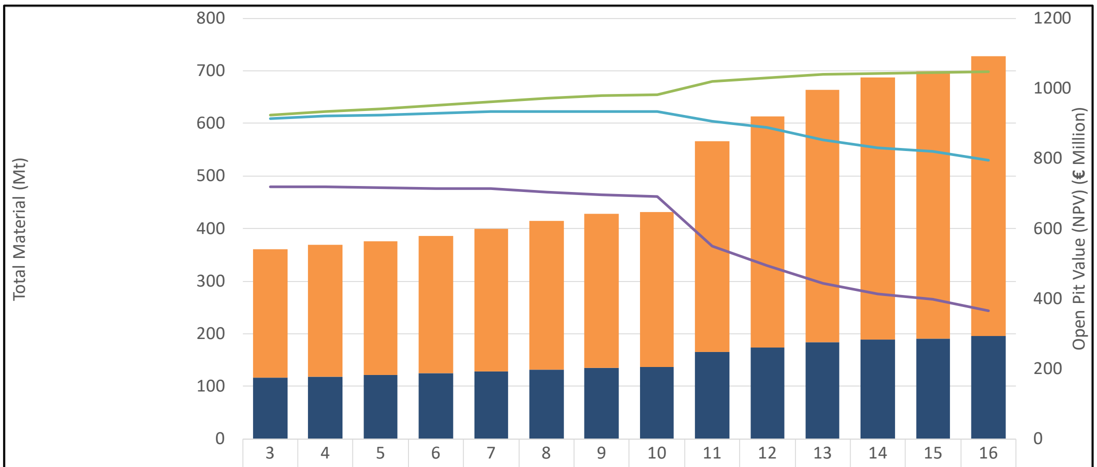


Figure 14-2: Nested pit shells – Pit optimisation results



	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Waste (Mt)	244	250	255	262	271	283	292	295	400	439	479	498	508	531
Ore (Mt)	117	119	122	124	128	133	136	137	165	175	185	189	191	196
RF	0.54	0.56	0.58	0.60	0.62	0.64	0.66	0.68	0.70	0.72	0.74	0.76	0.78	0.80
NPV - Best (€ Million)	925	934	942	951	962	973	980	982	1019	1030	1040	1043	1045	1048
NPV - Specified(€Million)	914	920	924	928	933	934	935	935	907	889	852	830	819	795
NPV- Worst (€ Million)	719	718	718	715	713	705	696	692	550	494	445	413	398	366

Figure 14-3: Skin analysis - Pit optimisation results

14.3.5 SRK comments on pit optimisation

In SRK's opinion, the pit shell selection point has provided for a robust economic limit, with enough upside potential to weather most technical and economic risks.

As mentioned in Section 14.3, the 2018 pit optimisation was based on an earlier (2016) MRE block model and input parameters from 2017 / 2018. The main output resulting from the pit optimisation was the pit shell which was used for the design of Stage 4 (the final pit). Since the pit shell was chosen at a low revenue factor (RF=0.68), it provided for robust economics with upside potential. In 2019, Boliden investigated the upside potential in an updated pit optimisation study which showed potential for a Stage 5 design, pending additional dumping and tailings storage requirements.

SRK considers that the methodology applied for the pit optimisation was comprehensive all be it producing a conservative final pit for the pit design with upside potential. SRK was made aware that the mine is currently engaged in a mineral resource update, to which an updated pit optimisation would aim to further investigate upside potential.

14.4 Pit Designs

Pit and stage designs with suitable bench geometry was completed in Geovia's Surpac mining software and was based on the strategic schedule results and final optimal pit shell (Figure 14-4). The pit and stage designs, LoMP and Mineral Reserves were evaluated using the 2018 Resource model titled, "bm_04_kev_mre2018.22.dm". The total inventory within the pit designs is shown in Table 14-6. The final pit design is comparable with the optimised pit shell to within a 7% margin for ore tonnes and metal content, which is considered acceptable by SRK.

The pit designs incorporated the minimum geotechnical parameters as discussed in section 14.10. Two interim stages (pushbacks), Stage 2 and Stage 3, are currently being mined, with Stage 4 (the final pit) having started waste stripping activities on the eastern pit face (Figure 14-5). The interim pushbacks provide for enough mining width (100 to 240 m) for the large equipment that is used on site.

To reduce the effects of bench overspill from Stage 4 waste mining onto the Stage 3 production faces, the Stage 3 design was altered to provide a 50 m catchment berm on the +78 m elevation. To achieve this, a temporary "triple bench" configuration (36 m bench height) was incorporated at this elevation (Figure 14-6). Given the competent nature of the "fresh" material at this elevation, SRK considers this an acceptable design alteration, with a marginal effect on the interim reserves.

Table 14-6: Mining Inventory (In situ) 31 December 2019

Mining (in situ)	Units	Total
Ore	Mt	141.1
Ni	%	0.26
Cu	%	0.34
Au	g/t	0.11
Pt	g/t	0.23
Pd	g/t	0.15
Co	%	0.01
Waste	Mt	187.95
Stage 4 Pre-stripping	Mt	2.61
Total	Mt	329.04
Stripping Ratio	t:t	1.33

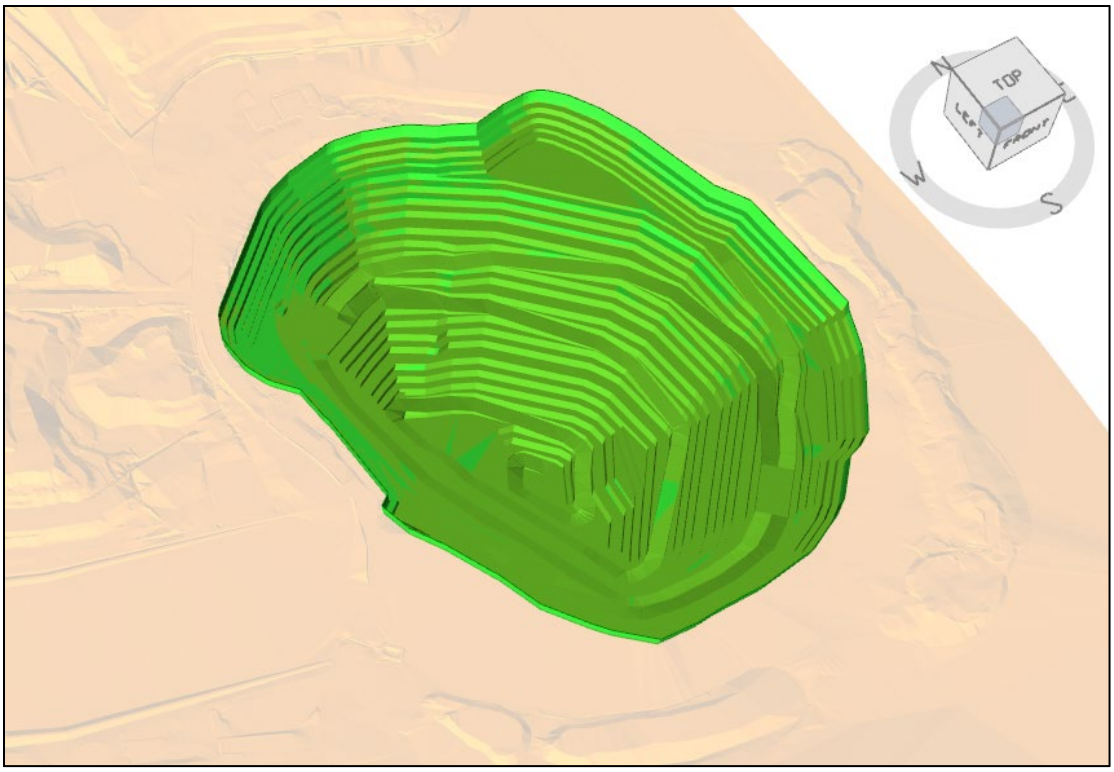


Figure 14-4: Stage 4 (Final) pit design

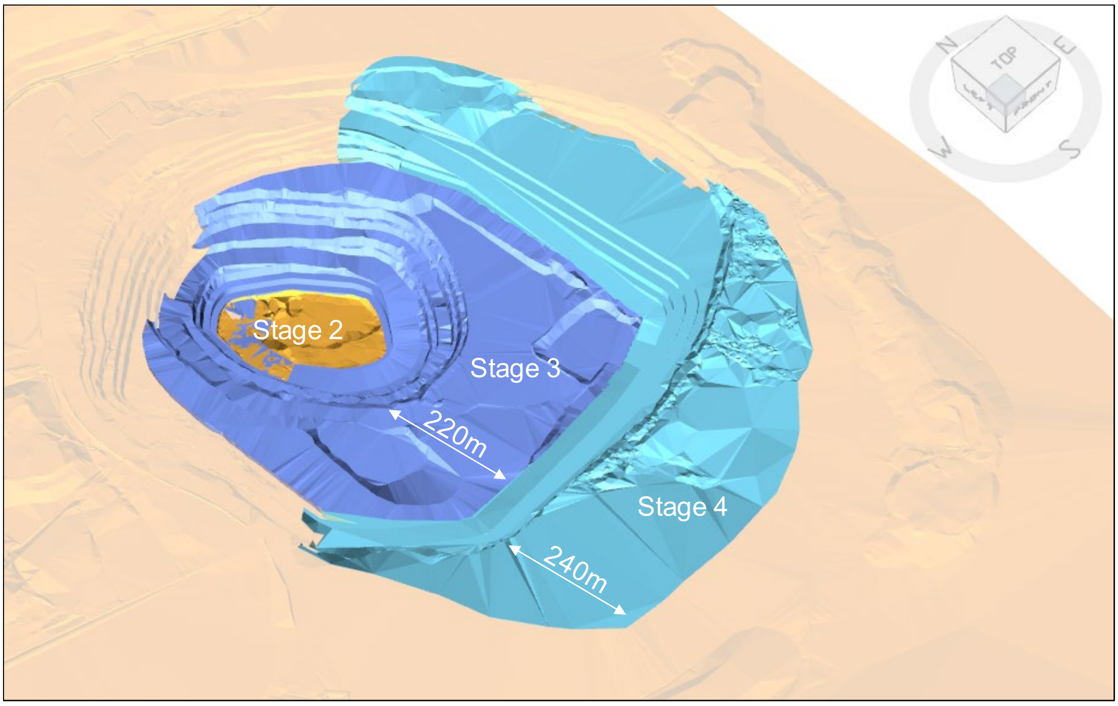


Figure 14-5: Kevitsa pushbacks solids (Stage 2, 3, & 4)

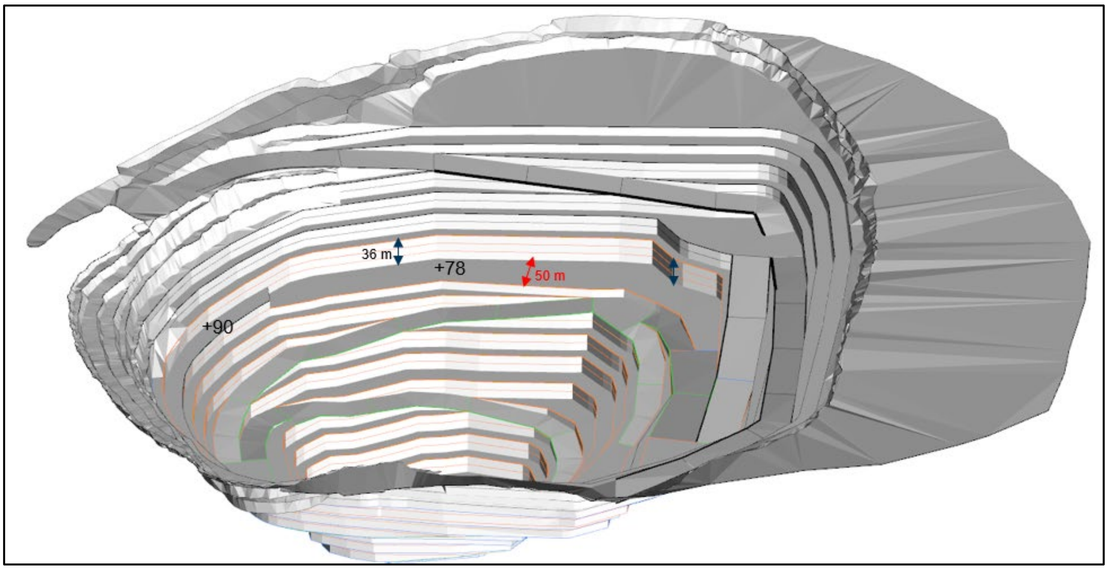


Figure 14-6: Kevitsa Stage 3 design alteration

14.5 Life of Mine Plan

A production schedule was completed for the LoMp using Deskwik's landform and haulage software package. The results from the production scheduling are shown in Figure 14-7. Stage 4 levelling (waste stripping) by the mining contractor will end in 2020. Waste stripping (35 Mtpa) is significantly reduced in 2023 to 15 Mtpa, with further decreases from 2027 onwards. Recent expansions in the concentrator plant will see a ramp-up in ore mining production in 2020 to achieve 10 Mtpa of ore from 2021 onwards.

The significant decrease in the production profile in 2023 raises the question whether the mine plan is optimal for an owner operated equipment fleet. SRK understands that excess capacity of the heavy mining equipment at Kevitsa could be transferred to Boliden's Aitik mine in Sweden after 2023. Equally, the mine is currently investigating the potential for an additional Stage 5 which could utilise the additional equipment capacity. Although not ideal, SRK accepts the production profile as achievable.

Ore in the LoMP and resulting Mineral Reserves has been defined by an NSR formula applied as an operational cut-off. An NSR >EUR15/t was used in the LoMP which relates to the unit costs for processing and mining. The NSR formula combines factors for processing recoveries, metal prices, payability, treatment and refinement charges. During the audit of the Mineral Resource and Mineral Reserves, various NSR formulae were provided by the mine without stating the assumptions upon which each formula was based. SRK back-calculated the numbers to understand and verify the significance of each. SRK recommends that, for transparency, all NSR formulae and supporting information should be fully documented.

For the budget plan (2020), an NSR formula using a 15% higher price Ni price was used to define Ore. Raising the prices meant that more marginal ore was included in the LoMp to fill the processing plant for 2020. SRK recommends that in such a case, rather than inflating the prices, the mine should lower the cut-off NSR price to a marginal cut-off (EUR 10 / t). SRK also recommends that all marginal ore be stockpiled separately.

Within the mining model, ore within the Stage 4 pit design is defined by a marginal NSR cut-off for 2020, based on the budget plan metal prices and processing recoveries for 2020.

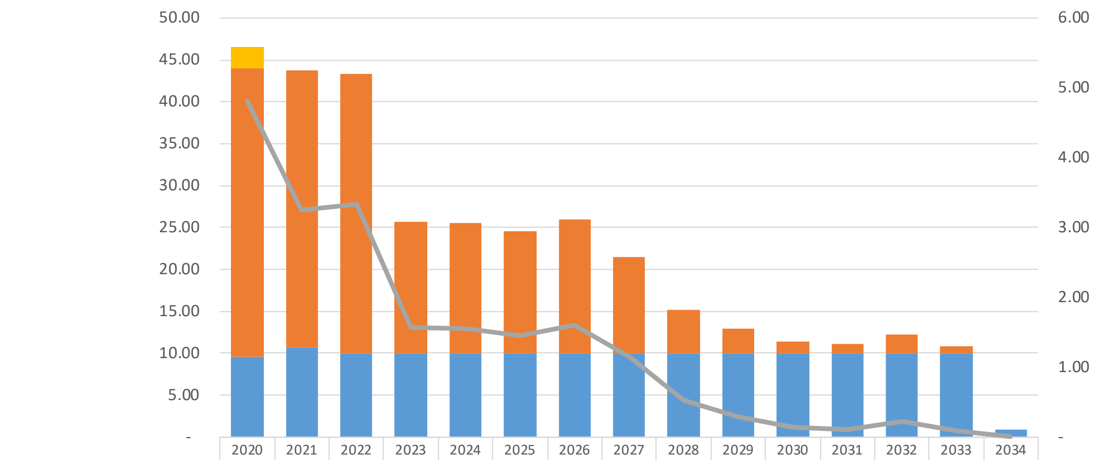
The NSR cut-off (EUR/t) was:

$$\mathrm{NSR\_BUD} = (72.11 \times \mathrm{NiS}) + (38.83 \times \mathrm{Cu}) + (7.96 \times \mathrm{Pt}) + (12.64 \times \mathrm{Pd}) + (12.51 \times \mathrm{Au}) + (44.93 \times \mathrm{CoS})$$

Beyond 2020 for the remainder of the LoMp, the NSR cut-off (EUR/t) based on long-term forecast metal prices was:

$$\mathrm{NSR\_LTP} = (64.47 \times \mathrm{NiS}) + (43.83 \times \mathrm{Cu}) + (6.80 \times \mathrm{Pt}) + (9.18 \times \mathrm{Pd}) + (8.97 \times \mathrm{Au}) + (68.32 \times \mathrm{CoS})$$

SRK was not provided with the basis for the abovementioned NSR formulae, and when compiling a NSR formula from the data provided for the financial model, a slightly different NSR formula resulted (see formula in 18.3). SRK believes this to be due to slight differences in metal recoveries at the time. The NSR terms associated with NiS were slightly more conservative for SRK but comparable for Cu.



	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Stage 4 levelling (Mt)	2.61														
Waste Tons (Mt)	34.38	33.08	33.28	15.67	15.60	14.57	16.03	11.51	5.21	2.93	1.39	1.10	2.27	0.93	-
Ore Tons (Mt)	9.58	10.72	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	9.94	0.85
Stripping Ratio (t:t)	4.81	3.25	3.33	1.57	1.56	1.46	1.60	1.15	0.52	0.29	0.14	0.11	0.23	0.09	-

Figure 14-7: LoMp production profile

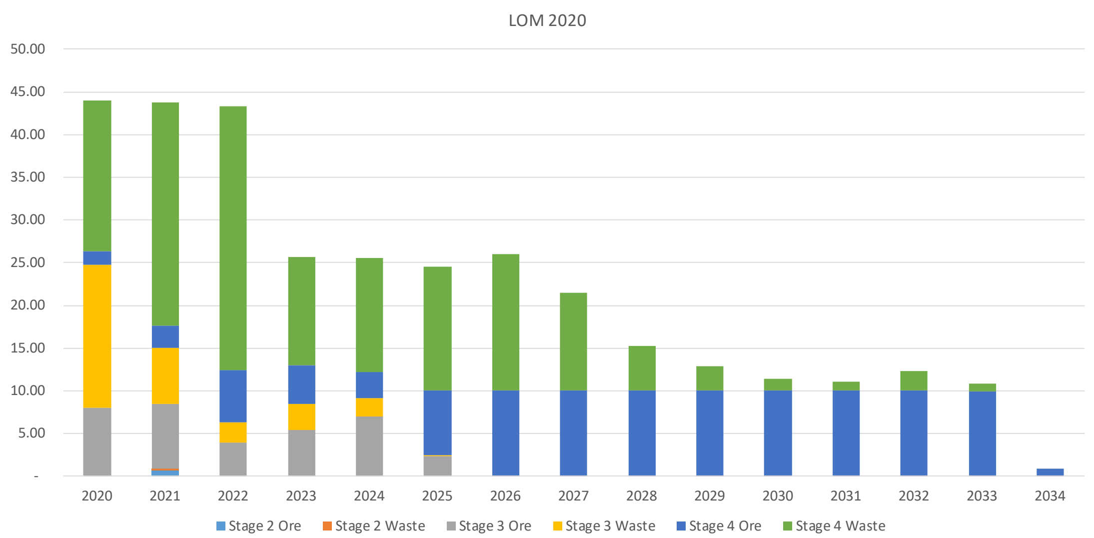


Figure 14-8: LoMp production profile per Stage

14.6 Waste Rock Dumps

Current waste dumping is taking place at Dumps 1 and 2 as shown in Figure 14-9. Waste rock dump ("WRD") preparation is currently underway at Dump 3. WRD preparation consists of either peat or bentonite sealing.

Peat sealing consists of laying down a layer of moraine, followed by a layer of peat (1 to 2 m). If peat is unavailable, bentonite sealing is used, which consists of +300 mm of morraine layered onto a bentonite carpet. The dumping of blasted rock commences once the morraine is frozen according to regulation, which also provides for improved underfoot conditions.

WRD preparation largely takes place in the summer months since morraine and peat stockpiles are largely frozen in the winter. The mine makes use of sub-contractors for WRD preparation. SRK was satisfied that sufficient interim dumping capacity (see interim designs in Figure 14-10) is available on Dumps 1 and 2 whilst Dump 3 is being prepared.

Waste types are classified based on the sulfur grades and nickel content and care is taken to "encapsulate" sulfur and nickel bearing rock in the waste dump according to a defined procedure.

The LoMp estimates that 64 million bank cubic metres ("BCM") of waste material will be mined at a net swell and compaction of 30% which equates to 84 million loose cubic metres ("LCM"). The final waste dumps will be 70 m high and the current design provides for remaining total dumping capacity in excess of 100 million LCM. Future WRD design iterations will incorporate an additional lift of 20 m as recently approved by the authorities. SRK is therefore satisfied that sufficient dumping capacity is available for the LoMp.

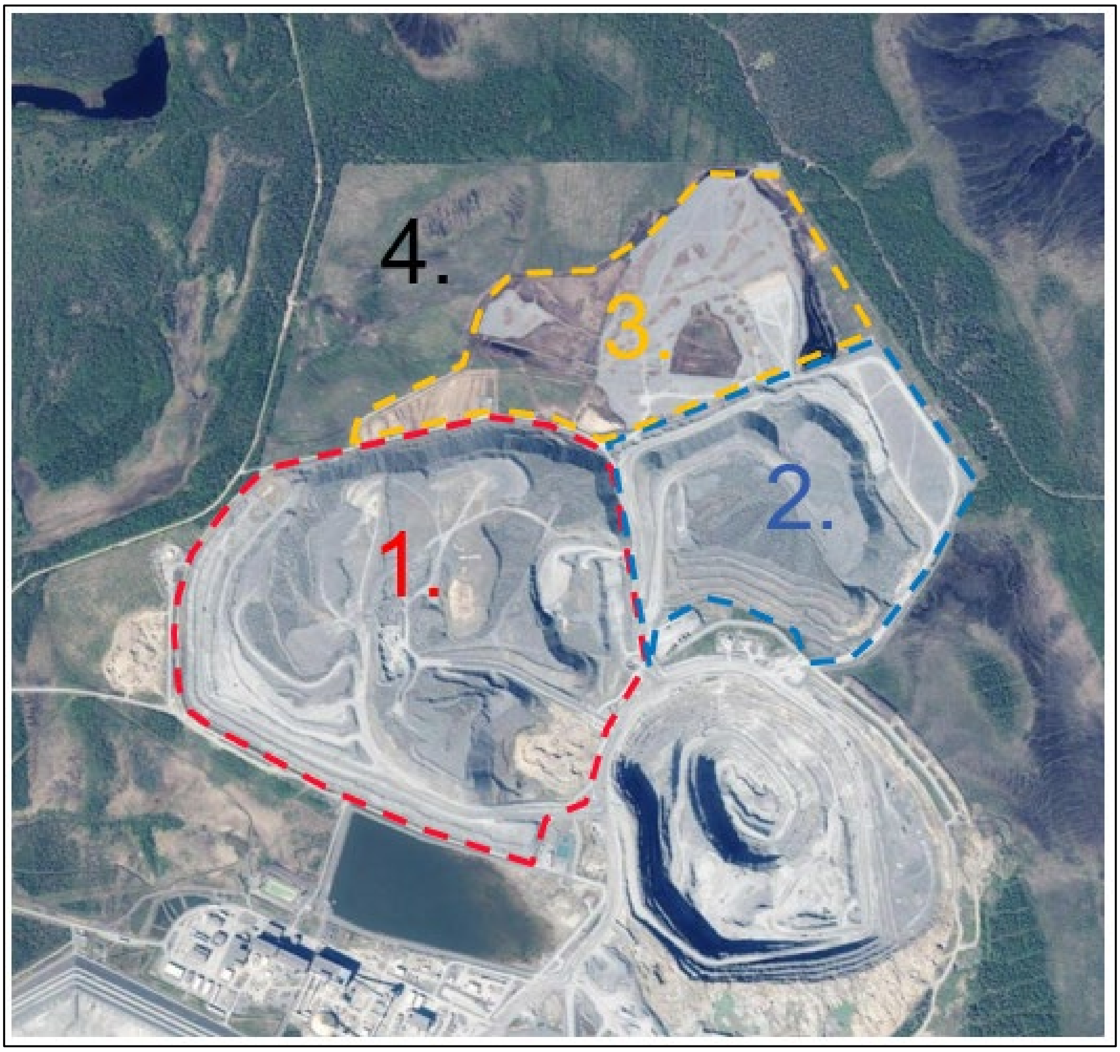


Figure 14-9: Waste dump locations for the Kevitsa mine

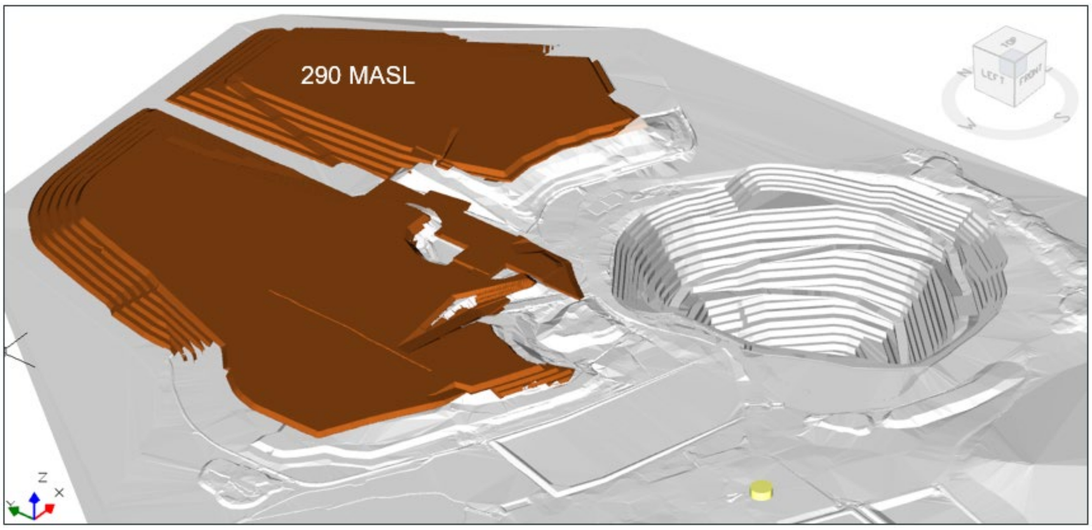


Figure 14-10: Interim waste dump designs for Kevitsa

14.7 Ore Stockpiling

All mined ore is sent to the run of mine ("RoM") Ore stockpile at Kevitsa and is therefore re-handled and there is no direct feeding into the primary crusher. The current stockpile "fingers" have a maximum storage capacity of 1.0 Mt. High grade and marginal ore are blended in the pit and stockpiled by blast blocks to achieve a uniform grade to be fed into the crusher. Ore blending is managed by grade control geologists on site.

The total ore available on the RoM stockpiles at on 31 December 2019 is shown in Table 14-7.

Table 14-7: RoM Stockpile totals at 31 December 2019

Blast	Tonnage	Ni%	Ni(S)%	Cu%
C1090X020	71,600	0.21	0.17	0.33
C1090X020	7,687	0.19	0.16	0.30
C1090X020	62,803	0.15	0.11	0.23
C1090X020	9,953	0.12	0.07	0.12
D1186X002	5,000	0.41	0.36	0.16
Total	**153,300**	**0.19**	**0.15**	**0.27**

14.8 Mining Equipment

The primary equipment fleet for Kevitsa is shown in Table 14-8. To ramp up ore mining to 9.5 Mtpa (7.5 Mtpa achieved in 2019), 34 Mtpa waste will need to be mined for 2020-2022. In total, 17 additional Komatsu 830-E were acquired in 2019 as well as one new CAT 6060 Face shovel in 2019; an additional CAT 6060 will be commissioned in 2020.

A trolley assist system will be implemented on the waste dump at Kevitsa for which construction will start in 2020 and the newly purchased Komatsu 830-E are trolley assist ready.

Loading and trucking requirements were calculated from first principles for Kevitsa using conservative productivity assumptions. On-site mining engineers are engaged in training to perform a Deswik haulage analyses to be included in future iterations of the LoMp. SRK considers that the current primary equipment totals to be sufficient for future production requirements.

Kevitsa mine has also commenced the installation of a fleet management / pit control system which will become operational in 2020. The system will provide fleet management and loading optimisation systems as well as remote drilling facilities.

All primary equipment is subject to Original Equipment Manufacturers ("OEM") maintenance and repair ("MARC") contracts at Kevitsa. OEM's share workshop facilities at Kevitsa, and the construction of a new truck workshop has recently been completed to accommodate the recent increased number of trucks operating at the mine.

Table 14-8: Primary Equipment fleet

Equipment type	Specifications	Qty
Drill rigs	Atlas Copco Pit Viper 271 Electric, 225 mm DTH, 20 m boom height, 6.3 kV	2 + 2 new units in 2019
	Atlas Copco SmartROC D65, 165 mm DTH, Diesel	6 units
Loaders	Komatsu PC8000 Electric hydraulic Face Shovel, 720 t, 36 m^3 bucket, 6.3 kV	1 unit
	CAT 6060 Diesel Face Shovel, 569 t, 34 m^3	1 new unit in 2019, 1 new in 2020
	Komatsu PC 5500 Diesel hydraulic Face shovel, 533 t, 29 m^3	1 unit
	Komatsu WA 1200 Diesel Wheel loader, 220 t, 19 35 m^3 bucket	1 unit
Haul trucks	Cat 795F, 313 t Diesel Electric trucks	4 units
	CAT 793F, 220 t Diesel trucks	11 units
	Komatsu 830E-5 Diesel Electric, 250 t	17 units in June 2019

14.9 SRK Comments on Mining Operations

SRK considers the Kevitsa operation as a well-established, world-class operation with minimal technical or economic risks to production. In summary, the mining operation at Kevitsa consists:

- Pit optimisation study and strategic schedule completed for Kevitsa was comprehensive, and ensured that a robust final pit design with practical pushbacks were identified for the mine.
- Final pit design adheres to geotechnical design parameters and closely mirrors the robust economic pit shell identified in the pit optimisation study;
  - pushback designs (Stage 2, 3 & 4) incorporate reasonable mining widths (100 to 240 m) as well as other considerations which allow for practical mining; and
  - total mining inventory (in situ) includes 187.95 Mt of waste, 141.1 Mt of ore, and 2.61 Mt of Stage 4 pre-stripping waste. The overall stripping ratio is 1.33.
- LoMp was simulated in a suitable mine scheduling software packages to achieve a maximum capacity in 2020 of 45 Mt total mining. An additional 2.61 Mt pre-stripping will be completed by a mining contractor in 2020.
- To define ore, an NSR >= EUR 15 / tonne was used in the LoMp which relates to the unit costs for processing + mining. The NSR formula combines factors for processing recoveries, metal prices, payability, Treatment, and Refinement charges. In the due process of auditing the Resources and Reserves, various NSR formulae were referenced by the mine without stating the assumptions upon which each formula was based. SRK had to back-calculate these to understand and verify the significance of each. SRK recommends that, for transparency, all NSR formulae and supporting information should be fully documented.
- For the2020 budget plan, an NSR formula using a 15% higher price Ni price was used to define ore. Raising the prices meant that more marginal ore was included in the LoMp to fill the processing plant for 2020. SRK recommends that rather than inflating the prices, the mine should lower the cut-off NSR price to a Marginal cut-off (EUR 10 / t). SRK also recommends that all marginal ore be stockpiled separately.

- Waste stripping is decreasing over time with high production tonnages in 2020 to 2022. Ore mining will ramp up in 2020 to achieve 10 Mtpa Ore from 2021 onwards.
- Waste dump preparation is carried out by a mining contractor and consists of either peat or bentonite sealing. Waste dump preparation predominantly takes place during the summer, is scheduled in advance and sufficient dumping capacity is available for the LoMp whilst ongoing dump preparation takes place.
- LoMp will require 84 million LCM dumping capacity (30% net swell and compaction), and sufficient capacity is available within the current approved dump designs.
- Kevitsa will become a fully owner operated mine when the mining contractor finishes the pre-stripping of Stage 4. The primary equipment fleet totals for the LoMp have been estimated from first principles and, in SRK's opinion, is based on conservative assumptions. Recent primary equipment purchases include two new Atlas Copco Pit Viper 271 Electric rigs, 2 x new Cat 6060 Diesel Face Shovels and 17 x Komatsu 830E-5 Diesel Electric 250 t trucks (trolley assist ready).

SRK recommends that future LoMp scenarios include a detailed haulage analysis to ensure that primary equipment totals are sufficient and are optimised.

14.10 Geotechnical Engineering

14.10.1 Introduction

SRK undertook an appraisal of the geotechnical engineering parameters applied for the 2018 optimisation, pit design and also operational management of pit slopes. SRK conducted a review on several key aspects that informed the Mineral Resource and Mineral Reserve reporting; including:

- data availability and collection for geology, structural, hydrogeological, and geotechnical information;
- data collection procedures and applicability to failure modes and stability influences;
- slope stability analysis performed and slope design parameters for Stage 3 to Stage 5;
- pit optimisation process and reconciliation to design;
- falls of ground location and investigation reports;
- constructed slope adherance to design parameters;
- pit hazard identification methods and management of controls to minimise slope stability risk including aspects such as monitoring, reports, seasonal performance, etc; and
- pit design analysis and adjustments to design to manage ground control issues.

The geotechnical appraisal was conducted utlising these information sources:

- pit optimisation: Boliden, February 2018 (Ojanen, 2018), performed by Boliden Technical Support;
- pit optimisation; Boliden, January 2019 (Ojanen, 2019), performed by Boliden Technical Support;
- slope design guidelines: 'Boliden Kevitsa' January 2018, 'Rock mechanical parameters for

Kevitsa Mine';

- general review of all geotechnical studies and slope design work from 2014 to 2017 conducted by external consultants:
    - WSP Consultants Finland, 2014 (Somervuori, 2014) and 2015 (Somervuori, 2015);
    - ITASCA Consultants Sweden, 2017 (Alvarez *et al.*, 2017); and
    - Turner Mine Geotechnical Pty Ltd ("TMG"), 2009 (Turner, 2009) and 2011 (Turner, 2011);
- comments from Senior Geotechnical Engineer (Pekka Bergström) regarding design application, overall current pit performance, and risk management.

14.10.2 Data availability and suitability

Rock Mass Data Collection

Core drilling is extensive with a reduced number of drillholes intersecting the east and west walls. Geotechnical core logging is performed and is to a reasonable quality for the slope design. There is a focus on the 'Q' system logging which was not originally configured for slope stability assessments. Boliden is not collecting inputs for modified rock mass rating ("MRMR") to be calculated explicitly. The MRMR is derived from Q using generic published equations but this is not yet verified as a suitable conversion for the Kevitsa rock mass. Numerical modelling uses geological strength index ("GSI") and this is determined directly from relationships with the Q values.

Structural data is collected using Sirovision. This is used to produce 3D photogrammetry models of established slopes and is being collected by the Geology department. This coverage is extensive and in high detail. There is a possibility to optimise the time spent on data collection versus requirement for geotechnical monitoring.

Strength testing is a combination of laboratory unconfined compression strength ("UCS"), Brazilian tensile strength tests, and several point load tests ("PLT"). The coverage of testing is suitable for the low variation in lithology in the deposit area.

Pit mapping procedures were not reviewed directly; however, pit geologists indicate that awareness is strong and data interpretation is generally good for geological needs, but it is lacking in collecting geotechnical data parameters. These data will inform both the localised design changes to manage risk as well as provide higher confidence into slope stability analysis.

Models

A reasonable lithology model and understanding of alteration for application into rock mass model has been generated. This model is suitable for geotechnical purposes to show interaction of ore limits with pit boundaries and limit of overburden and weathering zones on walls.

A model of large-scale structures has been developed. Major structures are identified by a combination of drillhole data and Sirovision mapping. The localised jointing/fracturing is not separated into structural domains for ongoing use in pit design changes. The structures were identified by drillhole core orientation with a point to point explicit modelling approach to join mapping with drillhole intersections.

Main rock mass model is defined by similar quality and strength. There is no significant variation in rock quality expected according to the lithological model. Localised alteration both increases and decreases the expected rock strength. The major structures delineate the boundaries of geotechnical domains.

Pit mapping procedures were not reviewed directly; however, conversation with pit geologists indicate that awareness is strong and data interpretation is generally good for geological needs but is lacking in geotechnical characterisation.

14.10.3 Slope stability analysis and slope design

Industry standard approaches have been applied to assess the slope stability from bench, inter-ramp, and overall slope angle scale. There are several stages in the slope design history determined from the documents supplied for review; the earliest being 2008, the latest being for the potential Stage 5 Design in 2017, and Boliden adjustments in early 2018. The phases of design work and geotechnical audits conducted by various companies is shown as a timeline in Figure 14-11.

Kinematic instability in fresh rock is the main failure mode expected at bench and multi-bench scale. The inter-ramp and overall slope scale stability is based on large structure forming planes and wedges and this is not well-understood.

Rock mass failure is not expected in slope toes, since the range of rock strength at Kevitsa are typically high (>200 MPa) and therefore intact rock failure is not expected in slope heights is less than 450 m. Itasca has modelled this in its 2017 analysis. Step path failure potential is possible and the likelihood of this is not quantified yet as the joint persistence data appears to not have been used in modelling.

Slope design has evolved over several years with the analysis being performed by external consultants. Thorough geotechnical design processes have been applied to an acceptable industry standard (Alvarez *et al.*, 2017). Boliden has adjusted locally the recommended design parameters as shown in Figure 14-12 and listed in Table 14-9.

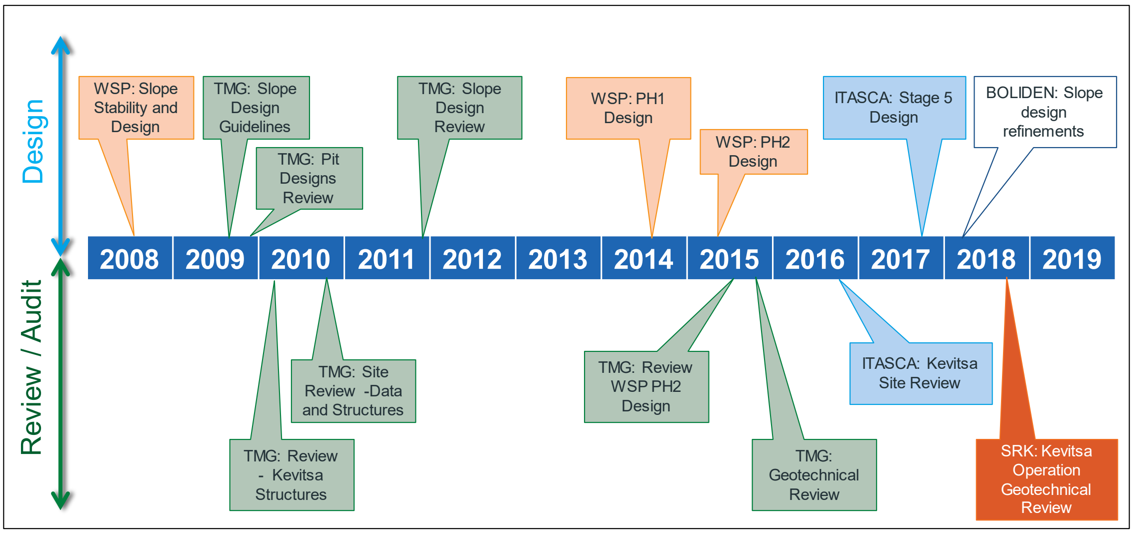
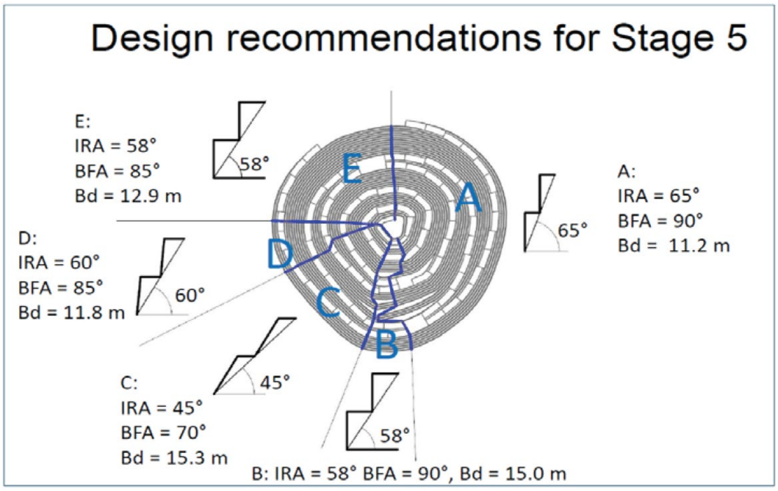
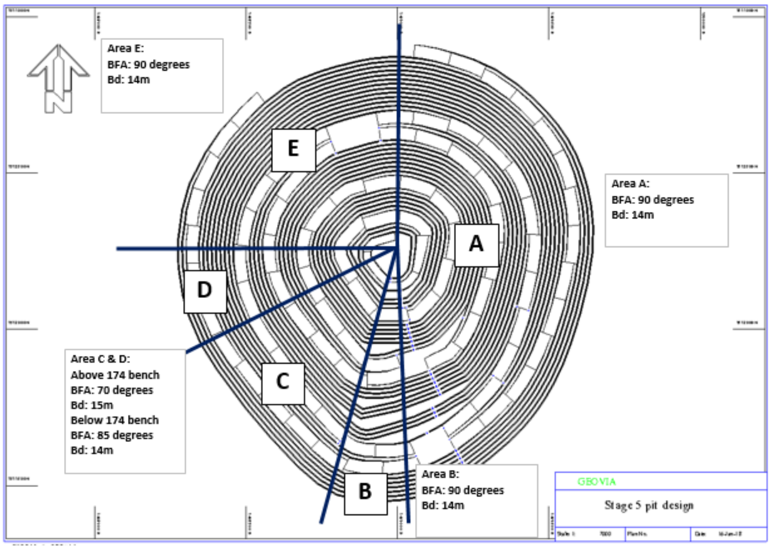


Figure 14-11: Summary timeline of slope design and geotechnical reviews by various consultants



ITASCA (2017) design recommendations for all slope sectors



Boliden Kevitsa: Adjustments to slope design recommendations for Area 'C'

Figure 14-12: Slope design parameters by design sector

Table 14-9: Recent Slope Parameter Design guidance revised by Boliden to Itasca (2017) recommendations

Domain	Batter Height BH (m)	Bench Face Angle BFA (°)	Catch Berm Width Bd (m)	Inter Ramp Angle IRA (°)	Maximum Inter Ramp Height (m)
A	24	90	14	73.7	192
B	24	90	14	73.7	192
C&D (above 174 RL)	24	70	15	55.9	192
C&D (below 174 RL)	24	85	14	69.2	192
E	24	90	14	73.7	192

The influence of a large structure parallel and behind the Stage 4 wall was raised during the review. This is not addressed in the current slope stability analysis. The potential influence of this structure not suitably analysed yet to understand future risk which is apparent in the immediate (1 to 2 years) development of the final Stage 4 east wall. Boliden has identified that a geotechnical assessment is required in 2020 to inform medium to long term mine planning of the east wall slope stability related to the NE-flt-2 structure influence.

Hydrogeological influence is addressed in WSP 2015 reporting (Somervuori, 2015), which has identified large structures influencing the pit slope with variable transmissivity properties. Future relevance of water into slope stability must be continually understood. The role of large structures to locate water inflow into the pit and influence slope stability is to be addressed in a 12-month planning cycle at a minimum. More detail on water management is provided in Section 14.11.

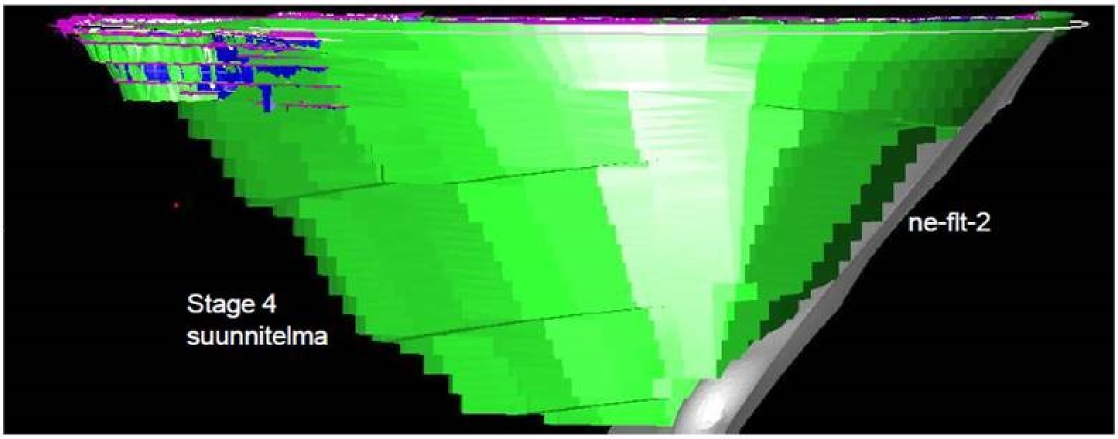


Figure 14-13: Stage 4 pit design with NE-FLT-2 Structure located near east wall

14.10.4 Pit optimisation inputs

The pit optimisation process considers individual slope components and not restricted to overall slope angle only. This approach is considered a high standard for deep pit design (>480 m) and more representative of the extraction ratio that will be realised in a pit design after optimisation. The process is acceptable as overall slope angles are derived by:

- all slope components utilised including changes with depth; and
- count of ramps and geotechnical berm widths.

The resulting overall slope angles utilised in the Whittle optimisation are not overly aggressive (37 to 45° above +174 m level, 46 to 52° below +174 m level). This incorporates the realised angles from the as-built slopes including ramps and geotechnical berms and therefore is ‘calibrated’ to what can be achieved.

In terms of the pit optimisation process, it is understood that the Geotechnical Site Engineer is not intrinsically involved in pit optimisation process which is completed in Boliden head office in Sweden. SRK believes there is an opportunity to improve the usage of an overall slope angle (“OSA”) by better collaboration within Boliden disciplines in calibrating the geotechnical inputs according to previous design.

SRK has not identified any ‘Fatal Flaws’ in the current pit optimisation process; therefore, it is deemed there is a minimal economic risk to the reserve based on the optimisation process.

14.10.5 Pit design and mine planning inputs

Pit design modifications are made by site experience and slope designs are adjusted on site for the short term mine planning process. Several aspects have evolved:

- ramp width changes - wider;

- geotechnical berms widened to 50 m to allow for access to remove of spill material, instead of pushing off and down the slope (based on Stage 2 learnings); and
- local adjustment for 36 m high bench face to accommodate 50 m wide geotechnical berm.

Reconciliation of actual constructed slopes to design is not routinely conducted to optimise the pit design process. This is not a critical gap, but SRK recognises an improvement opportunity in this regard.

14.10.6 Slope management

Bench-scale wedge failures (example in Figure 14-14) are common and managed during scaling and blast removal. These failures are expected and managed individually. There is clear wedge fallout at bench scale where the larger structures intersect. These indicate a level of predictability in terms of preparation for crest loss (example in Figure 14-15) and rockfall protection.

No inter-ramp scale failures yet or expected in the Stage 3 design. The location of large structures are known; however, the analysis conducted so far does not highlight the risk of multi-bench scale failure.

The main risk SRK has identified is stage interaction, which can be mining up to 4 stages concurrently which poses rockfall risk to operations. Wider clean-up berm is now designed to allow truck and excavator access to load and move material versus dumping down the slope. This risk is recognised and being managed in the short term and has influenced design changes to provide wider catch berms

Figure 14-14: Example of typical bench scale wedge or planar failures

Figure 14-15: Example of frequent bench scale crest loss to structure

14.10.7 Pit hazard management

The current slope management approach is summarised:

- pre-splitting performance is very good, and applied to the final walls in Stage 2 and 3. Temporary interim walls are pre-split where required;
- scaling practice is good, with reasonable equipment availability using long reach excavators; and
- final wall approval system in development in order to allow for authorisation to drill next pattern below the slope. This indicates areas requiring scaling and time required to complete this. It is also a record of bench conditions in domains to form history.

Current monitoring is slope monitoring radar only and observational monitoring. Radar monitoring was established in 2016 using a single IDS mobile unit. The positioning is suited to mid- to long-term monitoring. This unit is mainly located on the upper SW Crest which allows good coverage from this position. There is an option to move the unit, but this is not very practical for long term monitoring. There is a feature to use 270º scanning range; however, practical experience suggests that the wide angle peripheries have a larger error range due to the radar vector not aligned to the probable wall movement vector.

From discussions with the responsible Geotechnical Engineer on site during the site visit, it was suggested that that a second radar is deemed a requirement but is not yet budgeted for. SRK feels that this requires addressing for risk management and control to achieving the LoMp. Monitoring of circular pits with radar can introduce error and low confidence in the radar scans at the peripheries; therefore, a second radar will service the opposite half of the pit which includes ramp access as well. Additionally, a second radar unit will provide an opportunity to move and monitor a specific high-risk zone for a required period and not reduce the overall pit monitoring by moving a single unit.

‘Slope Hazard Awareness’ training material is prepared and updated by the responsible Geotechnical Engineer. This is a presentation format that documents the hazards, risk identification and risk mitigation controls applied at Kevitsa. This material is high quality and applicable in SRK’s opinion. Routine hazard maps are constructed and issued to all operators and staff entering the operation. These maps detail and rank the areas of slope hazard with a map and descriptive photos of the locations (example in Figure 14-16).

Geotechnical reviews are internal and external and applied every 1 to 2 years. There have been continuous reviews conducted which is well-documented. The reviews accessed by SRK at the time of writing (end-2019) are shown in Figure 14-11.

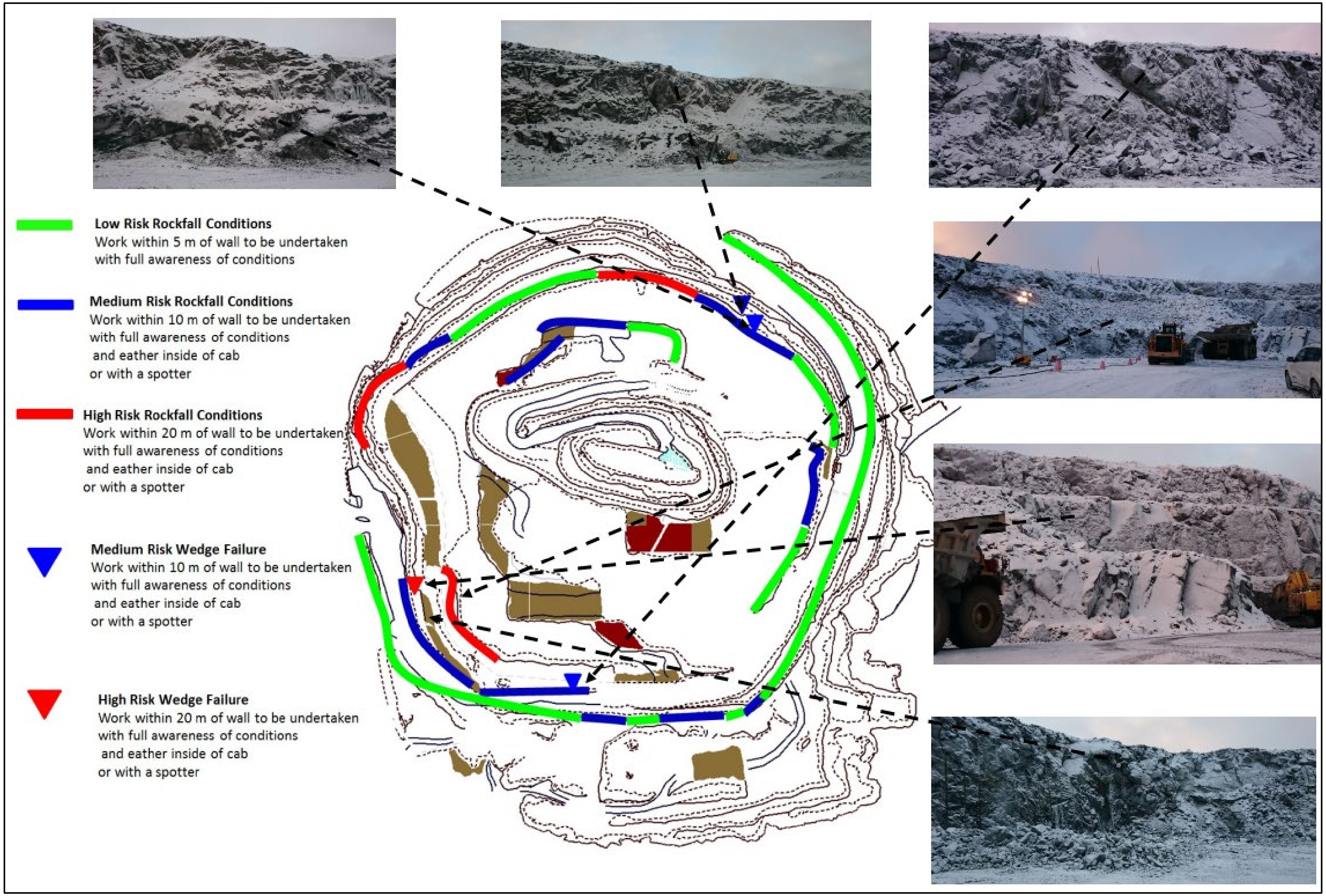


Figure 14-16: Example Hazard Map for Stage 2and 3 mining at Kevitsa

14.10.8 SRK comments on geotechnical engineering

The various geotechnical engineering aspects that contribute to the slope design, pit design and optimisation have been reviewed by SRK to contribute to the Mineral Resource and Mineral Reserve reviews to understand any fatal flaws.

Suitable data collection is in place with geotechnical drill core logging given the general high competency of the lithology types. The structural mapping could be improved in order to provide localised expectation of jointing that will lead to crest loss and bench scale instability. There is a suitable model of the major geological structures that influence the pit slopes up to Stage 3.

Slope stability analysis has been performed by various external consultants using industry standard practice. The most recent being ITASCA's work conducted in 2017. The role of minor structure is included in the analysis to derive bench scale parameters and required catch width. SRK could not determine if the stability analysis has included the influence of major structures in the Stage 5 design. As well as this, the potential for step-path failure at inter-ramp scale has not been suitably addressed. In deep pits with high intact rock strength the role of jointing controls stability and this structural fabric is to be suitably modelled in terms of persistence length as well as orientation. SRK recommends that the existing slopes are mapped for this data to enhance the 2017 slope stability analysis.

Boliden has adopted the recommended design parameters mainly and has made local adjustments based on operational knowledge and updated geotechnical data collected since 2017. The bench configuration results in realistic inter-ramp angles while providing catch bench width for steep (85 to 90°) bench face angles.

The Stage 4 design east wall stability will likely be compromised by the NE-FLT-2 structure and Boliden has recognised that a suitable technical study is to be conducted on this slope in the near term in order to inform the medium term mine plan.

The pit optimisation process is clear, and the slope angle inputs follow the geotechnical guidance parameters that are at a high resolution to provide height based inter-ramp angle inputs into the Whittle optimisation. These are corrected with the actual as-built slope angles, ramps and geotechnical catch benches to the developed depths. SRK considers this practice to be of a high standard and the resulting pit shells are realistic to accommodate a suitable pit design.

Slope hazard management is suitable, with failure modes well-understood and the pit, at this stage of development, has been constructed well with the minor-moderate bench scale instability risk handled suitably. Structurally controlled wedge and planar failure is common with resulting minor crest loss. The operation has a well-established process of scaling benches with a long reach excavator to remove unstable blocks. Pre-splitting is being conducted in the final walls and is producing clean bench faces.

The single slope stability radar is the only real-time monitoring instrument active at this stage. The responsible geotechnical engineer on site has recommended to the mine that a second radar to be implemented and SRK supports this due to the shape and size of the Stage 4 pit in development. This will provide higher confidence monitoring of all walls in a circular pit as well as providing further redundancy. As well as this, a second unit will allow targeted area monitoring when required and will not compromise continuous full pit monitoring. Additional systems such as prism monitoring is recommended to complement the radar monitoring. This is common industry practice in large pits and the Stage 4 development is early, allowing prisms to be installed.

Overall, SRK deems that there is low risk to achieving the final mine pit limits in the Stage 3 design. The risk to achieve the Stage 4 and Stage 5 designs (for future investigations) requires a higher level of analysis to quantify the probability of inter-ramp failure. This is to include a better understanding of jointing patterns and quantification of the hydrogeological influence.

14.11 Water Management

14.11.1 Introduction and sources of data

This section reviews the water management aspects of the Kevitsa mine. In broad terms, these comprise surface water and groundwater control in relation to the pit and surface infrastructure, the effects of both on the pit shell and slope design, the mine water balance, water supply, treatment and disposal of water to the environment.

The main documents reviewed to support the water management appraisal are listed below:

- WSP (2014). Kevitsa slope stability study, Phase I determination of geotechnical domains: geological structures and rock quality (Somervuori, 2014).
- Geosto Ltd (2015). Groundwater monitoring for FQM Kevitsa Mine open pit (Saksa, 2015).
- Golder Associates AB ("Golder") (2016). Conceptual Site Model and inflow analysis (La Touche and Yungwirth, 2016).
- Golder Associates (2017). Hydrogeological testing factual report (La Touche and Yungwirth, 2017).
- Golder Associates (2018). Kevitsa TSF monitoring wells and piezometer installation report (Kaczynski and Girard, 2018).
- Golder Associates (2018). Kevitsa site-wide groundwater model (Lindmark and Lelliot, 2018).
- Golder Associates (2018). Kevitsa site-wide water balance modelling using GoldSim (Garrick, H. 2018).

The rock mechanics study produced by ITASCA in 2017 (Alvarez *et al.*, 2017) has also been consulted for its coverage on the water aspects of pit slope design, but other parts of the report have not been reviewed (more detail is provided in Section 14.10).

SRK is aware that a separate report on pit water management was due to be produced following completion of Golder groundwater modelling report (2018), which was not available prior to SRK completing its review. As such, the present study is unable to comment in detail about the water management infrastructure proposed for the later stages of the mine life.

14.11.2 General hydrological setting

The area around the Kevitsa mine is characterised by quite gentle relief, with large expanses of flat ground to the west and NW in the direction of the River Kitinen, where ground elevation ranges between 200 and 210 masl and with rounded hills immediately east of the mine that rise to some 300 masl.

Meteorological data sourced from the climate station in Sodankyla indicates that the average monthly temperature in the Kevitsa area ranges between -13.4°C in January and 15°C in July and that precipitation and evaporation, respectively average 544 and 247 mm/year. Since precipitation during winter months is locked-up as snow, the ‘effective’ precipitation that occurs during the thaw in April, snow melt plus rainfall, is significant, leading to peak run-off and groundwater recharge occurring in the spring and early summer.

Run-off from the mine site is generally to the west in the direction of the River Kitinen. Local water features include: (a) three water courses: the River Kitinen to the west, the Ala-Liesijoki to the north and the River Luiro to the east; and (b) three lakes, one in the south (Saiveljarvi), a second to the east (Satojarvi), and a third to the north (Vaiskonlampi). The Satojarvi is a European Union Natura 2000 designated site and is therefore the subject of stringent environmental protections. The River Kitinen is an important source of energy for the Finnish grid due to the presence of several hydro-electric schemes along its course. The Vajukoski Dam and power plant are located just north of the mine approach road and some 5 km to the west of the facility.

The geology of the area broadly comprises a thin veneer of peat (mean thickness = 0.9 m) and recent unconsolidated glacial moraine (mean thickness = 3.3 m) overlying a bedrock complex consisting of a mafic-ultramafic layered intrusion and a medium-high grade metamorphosed, greenstone belt style assemblage of sediments and volcaniclastics. The bedrock in the area of the pit is cross-cut by faults, with the main fault set striking NW-SE, and possesses a weathered top surface that is some 20 m thick. These material characteristics define the four principal hydrostratigraphic units, namely:

- peat;
- shallow moraine;
- weathered and fractured bedrock; and
- fresh bedrock (faulted and fractured to varying degrees).

The peat hydraulic conductivity (K) ranges between 1E-04 and 1E-08 m/s, which is consistent with similar sites in Finland and with literature sources. In-situ hydraulic tests of the moraine yield Ks ranging between 1E-06 and 3E-05 m/s, which is typical of a silt or fine sand. Much lower Ks for the moraine were derived from laboratory samples, but these are less likely to be representative since ex-situ tests frequently either destroy (through handling) or miss in selection preferential pathways that exist naturally in the ground. Testing of the bedrock by WSP and Golder shows that K decreases with increasing depth, which is entirely consistent with patterns observed elsewhere in bedrock. The top weathered zone has an average K of 3E-05 m/s with the underlying, fresh bedrock decreasing from 1.5E-07 m/s in the top 20 to 30 m to 3.2E-09 m/s between 380 and 550 m below ground level (“mbgl”).

Regionally, groundwater flow direction is expected to be westwards towards the River Kitinen; however, water level monitoring in shallow hydrogeological units at the site show that dewatering of the pit has created a cone of depression with groundwater in surrounding formations flowing towards the pit. The deep bedrock pressure records also support this pattern, although it is worth remarking that observations by Golder point to some higher K faults as exerting a locally strong NNW-SSE preferential orientation of flow.

Discharge from the pit and other key locations including the waste rock dump (“WRD”), the storm drain, the TSF, the modular effluent treatment plant (“METP”) and the discharge to the

River Kitinen are monitored for water quality. Golder has noted elevated concentrations of nickel and manganese in the discharge from the WRD and that the wetlands, to which this discharge reports, did not seem to have the desired effect of polishing and improving the water. The assumption was that there is an element of groundwater contributing to the make-up of the wetlands and that this is elevated in metals, thereby reducing water quality rather than improving it. SRK is also aware that there have been at least two cases of leachate leakage from the TSF, on the WNW side of the facility and the other on the SW side. These seep locations, which are now monitored using uPVC Casagrande wells have picked up nickel and chloride contamination. It is also important to note at this juncture that SRK has not seen or is not aware of what remedial works are planned to mitigate these leaks.

Water from the pit and the waste rock reports to a water reservoir and is either forwarded to the process plant, as part of the make-up, or to the METP and from thence to the River Kitinen. Water from these sources is occasionally sent to the nearby wetland during Summer months for polishing, but this step is now normally by-passed. Only if the nickel concentration is more than 5 mg/l in the WRD discharge is it sent directly to treatment. Process make-up also comes from the River Kitinen and from water recycled from the TSF, either directly or by way of the decant and water reservoir. Surplus water from the process also reports back to the reservoir where it is stored either for later re-use or for treatment and, to a limited extent discharge into the wetland during the summer months.

14.11.3 Site characterisation studies

SRK has not received documentation on the designs and supporting studies for surface water management and is therefore not able to comment on this aspect of the site characterisation. However, extensive reporting is available on the groundwater regime, which is discussed below.

The earliest records of testing are discussed in WSP's Phase 1 slope stability study and relate to Poyry's 2011 field study. The report shows that the mine area was subjected to extensive testing in the northern, eastern and southern quadrants of the pit, using a combination of flow logging, packer and pumping tests; however, the intrusion contact zone and the western side of the pit did not receive the same level of attention, for which no reason could be found. These tests were useful in revealing the local properties of the bedrock with the groundwater regime (a) clearly dominated by fracture flow, and (b) the top, more weathered part of the formation having the most elevated K.

Following a data review and gap analysis by Golder in 2016, some supplementary field testing was undertaken in 2017. This comprised 23 packer tests in 7 drillholes (DBH1-7), five of which were subsequently installed with vibrating wire piezometers ("VWP"). These tests complemented and extended coverage of the site whilst broadly corroborating the hydrogeological characteristics of the bedrock in the area of the pit. SRK notes Golder's comments about the packer test results skewing the K upwards on the grounds that these tests were targeted at geological structures. To get a more balanced representation, future testing programmes might consider continuous testing down the hole irrespective of whether structures exist. This is feasible, but would require an adjustment to the procedure, especially if such work is performed on the back of an infill drilling or geotechnical drilling campaign.

Shallow formation groundwater levels at the site are measured in a series of standpipe piezometers as part of the environmental monitoring programme. The original network, comprising KEVG-series holes appears to have focussed around the perimeter of the TSF and to the east and south of the present pit (based on what is presented in the Geosto report). This was upgraded in 2018 with the installation of additional SBP- and SB-series holes to extend coverage across the whole site and to significantly improve resolution. SRK notes Golder's comments in their modelling study regarding the sparse nature of the data from the shallow wells, in particular the reliance on periodic manual water level measurements. Golder also noted that a number of the shallow wells near the pit were dry but observing the water level contour plan in its report it seems there are others around the TSF and at the western extremity of the site that are also dry.

In terms of the future operation, the value of the shallow wells could be much improved by installing transducer-logger units in a selection of the wells to get continuous water level and temperature records that can be used to align changes in the hydrograph more closely to climatic and other effects on site. In this regard, SRK notes that climatic effects, for example the spring thaw, are evident in the vibrating wire transducer responses in the bedrock monitoring holes (for example, DBH7), although the strength of the signal varies between holes and with monitoring interval depths, probably indicating the variability in K and hydraulic connectivity of the fractured rock mass.

14.11.4 Groundwater modelling

A detailed groundwater model of the site was developed by Golder in 2018 to predict flows into the future pit and to assist in the selection of dewatering design for managing pit water.

The modelling was undertaken using MODFLOW - Unstructured Grid ("USG") with the limits of the model domain defined by the River Kitinen in the west, the River Luiro in the east, the River Ala-Liesijoki in the north and the catchment boundary south of Saiveljarvi marking the southern border. The model grid was formed of 100 x 100 m cells that were reduced down to 12.5 x 12.5 m in the immediate vicinity of the mine. The unstructured grid capability of the MODFLOW model was used to increase cell density in areas of faulting. The model was vertically-discretised into 17 layers to represent the different lithologies, states of weathering and changes in K and storage (S). The latter parameters were derived from a combination of field test results and literature sources. Golder also built some anisotropy in to the model to reflect the preferential flow direction imposed by the presence of faulting across the pit footprint. Recharge was obtained from the HBV model (Swedish Meteorological and Hydrological Institute) produced to support the site-wide water balance.

Initial model calibration was accomplished in steady state to match existing conditions and the model then run in transient mode, at monthly time steps for the remaining life of mine to predict the pattern and quantity of inflow to the pit through various stages of deepening and pushback.

Golder tested three dewatering scenarios to gauge the suitability of different approaches. The first scenario assumed natural gravity drainage to the pit, whilst the second and third used different arrangements of advance dewatering wells around the periphery of the pit to actively induce drawdown. The modelling results indicated that advance dewatering wells, whether spaced equally around the pit or specifically targeted at faults, did not make a large enough difference to pit inflows to confer an advantage, although it was noted the dewatering would likely increase drawdown in the nearby wetland.

SRK believes the choice of model, the model construction and calibration to be appropriate for the Kevitsa mine; however, the model simulations seem to be focussed on managing pit inflows from a purely operational point of view and it is not immediately clear why the groundwater study has not been linked more closely with pit slope depressurisation and optimisation. Whilst

ITASCA has considered groundwater in its geotechnical study, it has been considered by employing various phreatic surfaces behind the slope and with no specific regard to local hydrogeological characteristics. Geosto in its review (Saksa, 2015) discusses whether special measures for depressurisation might be required, identifying two rock quality domains near the base of the future pit, B and C, that are both fractured and of low strength. Gesto flags the fact that the low K of these domains means there is a possibility that pore pressures will be slow to dissipate. SRK concurs with this line of thought and considers that the project will likely benefit from a closer scrutiny of material strengths and the ability of slopes to drain and depressurise.

14.11.5 Mine water balance

An initial spreadsheet-based water balance model for the site was produced by FQM. This was evaluated and updated by Golder in 2016 and then significantly improved in 2017 with the transposition of the existing model from Excel into a Monte-Carlo simulation software programme called GoldSim. The main aims of the 2017 study were to improve the simulation of the freeze-thaw cycle, to incorporate a stochastic climate model, to update surface water and groundwater flows based on the HBV model and undertake a more detailed assessment of discharge performance from site due to storm events. Additionally, the model had to meet certain objectives in respect of managing the water inventory, namely that volumes in the water reservoir should be maintained at optimum capacity (70% of total volume), the overall volume of water on site (in all facilities) should range between 1 Mm^3 and 3 Mm^3 and that water quality and chemical loading (particularly Ni and Cu) should be regulated to ensure they achieve compliance with the site discharge permit.

SRK notes that the calibrated water balance model matched existing data reasonably well, although it was acknowledged that both the effluent treatment rate and off-site discharge were slightly lower in the GoldSim model. The longer-term predictions through to end of mine life confirm that off-site discharge and treatment requirements will steadily increase as a result of increasing pit inflow and sublimation losses. The mean inventory, whilst mostly remaining within the 1 Mm^3 to 3 Mm^3 bounds, is predicted to exceed the upper bound during the summer in four individual years. A sensitivity study undertaken to assess treatment requirements established that discharges could be managed if the ETP capacity was increased from 500 to 750 m^3/year; however, if capacity is kept at 500 m^3/year, then it is impossible to keep the site inventory below 3 Mm^3 and, hence there is a significant risk of uncontrolled release to the wetland.

SRK considers the water balance to be robust and the recommendations made at the end of the study to be sensible, particularly in respect of the need to establish a more accurate understanding of the sublimation rate. The report also recognises at the end that climate change is a factor that will likely 'shift the probability distribution' although it is not modelled. SRK agrees based on experience of similar projects in Finnish Lapland; climate change predictions point to increased temperatures and rainfall for the region, which may lead to an increase in the open water season during the summer and reduction in snowpack depth during winter. This effect will be combined with an increase in mean annual run-off, which may increase the water available over the annual period. SRK therefore considers that it would be advantageous for the operator to factor climate change into future water balance predictions so that appropriate contingencies are incorporated in the mine infrastructure designs.

14.11.6 SRK comments on water management

The various water management aspects that contribute to the slope design, pit design and optimisation have been reviewed by SRK to contribute to the Mineral Resource and Mineral Reserve reviews to understand any fatal flaws.

No water management flaws have been highlighted in the review affecting the reporting of Mineral Resource and Mineral Reserve statements.

15 RECOVERY METHODS

15.1 Metallurgy

The predominant copper mineral is chalcopyrite. Cubanite is also present in lesser amounts. Pentlandite is the predominant nickel mineral.

If the copper mineralisation is identified as high in cubanite, the plant targets a lower Cu grade in Cu concentrate.

Small, high grade PGM zones are present in the orebodies. These zones are identified and the high grade PGM ore is blended with the bulk feed.

Talc is present in some areas of the deposit. This is identified by geology and the ore is handled separately.

15.2 Processing

15.2.1 RoM stockpile

The RoM stockpiles are controlled by geology department. Large stockpiles, nominally 1 Mt, are formed to try to maintain a reasonably consistent feed in terms of metal grades and ore hardness. These stockpiles represent 6 to 7 weeks plant feed.

Ore is blended on these stockpiles with reference to Cu and Ni grades, chalcopyrite:cubanite ratio and pyrrhotite:pentlandite ratio, and ore hardness.

High talc ores are stockpiled separately and processed in batch mode since this material has a significant detrimental effect on Cu and Ni recoveries. High talc ores can contain up to 30% talc.

15.2.2 Flowsheet

The process flowsheet can be considered conventional. Comminution and flotation flowsheets are shown in Figure 15-1 and Figure 15-2.

RoM ore, top size nominally 1200 mm, is crushed to -150 mm by a large gyratory crusher rated at 2000 tph. Primary crushed ore is screened to remove -130+25 mm material for use as grinding media in the pebble mills. The +130 mm material and the -25mm material is stockpiled and fed to the two autogenous grinding mills ("AG"). Excess -130+25 mm material and AG mill pebbles are crushed in two Metso MP800 cone crushers each rated at 560 tph and added to the AG mill stockpile. The crushing and screening plant incorporates dust removal equipment.

Crushed ore is ground to 76 to 78% -75 µm in two AG mills operating in parallel and a single pebble mill. All three mills are 8.5 m diameter and 8.5 m long, the AG mills incorporate 7 MW drives and the pebble mill 14 MW of power, installed dual pinion 7 MW drives. One AG mill is variable speed.

Flotation of Cu and NiS concentrates is achieved by sequential flotation. Ni flotation tailings are further treated by flotation to produce a high-sulphur concentrate which is stored in a separate tailings impoundment from the low-sulphur tailings.

Cu flotation is achieved in rougher and rougher-scavengers with four stages of Cu concentrate cleaning, the latter stage a single flotation column. The Cu rougher and rougher-scavenger are all 160 m^3 cylindrical cells and the residence time is nominally 30 minutes. The original flotation circuit has been upgraded by the addition on a single 500 m^3 cell ahead of the copper rougher bank. An Outotec HIG mill is used for Cu concentrate regrind.

Ni flotation is achieved in two lines of rougher and rougher-scavengers with four stages of Ni concentrate cleaning. The Ni rougher and rougher-scavenger are all 300 m^3 cylindrical cells and the residence time is nominally 60 minutes. A small ball mill is used for Ni concentrate regrind.

As the Ni tailings still contain significant sulphide material, typically 1 to 2.5% sulphide sulphur, the tailings are further processed through additional flotation cells to separate a high-grade sulphur concentrate, typically 30% S content, from the bulk of the tailings mass. This is achieved in rougher cells with a single stage of cleaning. The high-sulphur concentrate and the low-sulphur tailings stream are stored in separate tailings ponds. The pond for the high-sulphur concentrate is lined. Based on the original design criteria and the 2017 to 2019 historical operating data the typical mass yield of high sulphur concentrate is around 1.4%.

Talc flotation is performed in the single 500 m^3 cell when treating high talc ores.

The Cu and Ni concentrates are dewatered separately by thickening followed by automated pressure filtration. Concentrate moistures are typically 9% by weight.

The plant incorporates a high level of instrumentation and process control equipment. This includes integrated AG/pebble mill controls, flotation feed particle size analysis ("PSA"), froth cameras in Cu and Ni rougher/rougher scavengers/cleaners, and Courier on-stream analysers incorporating four six stream multiplexers.

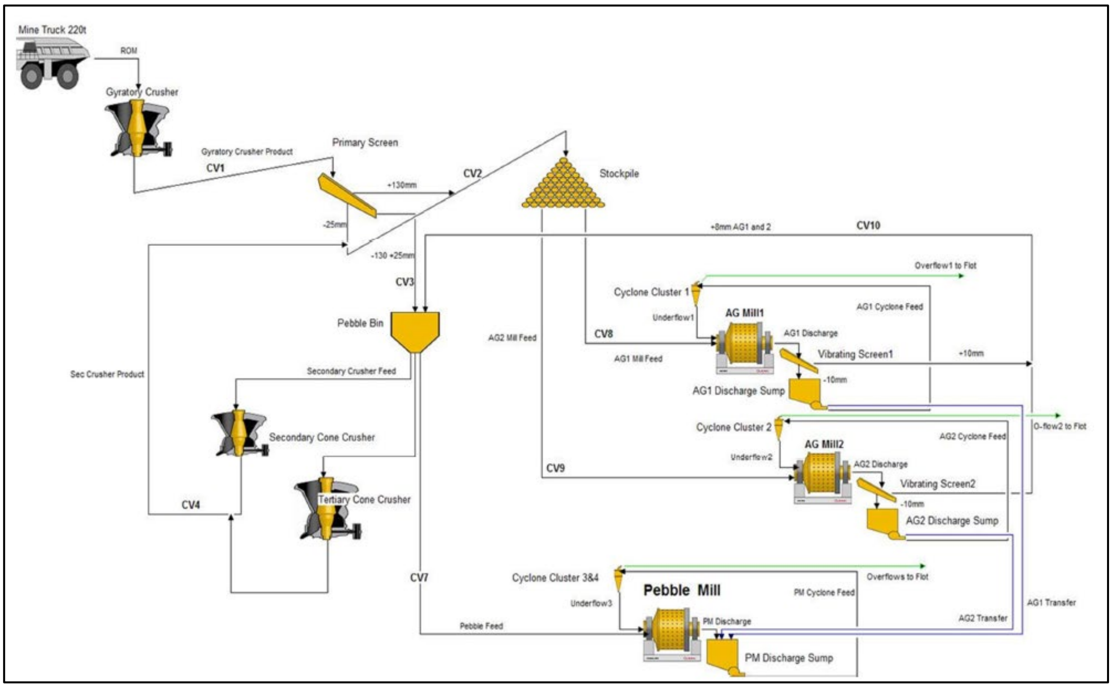


Figure 15-1: Kevitsa Crushing and Grinding Circuits (Source: Boliden, 2019)

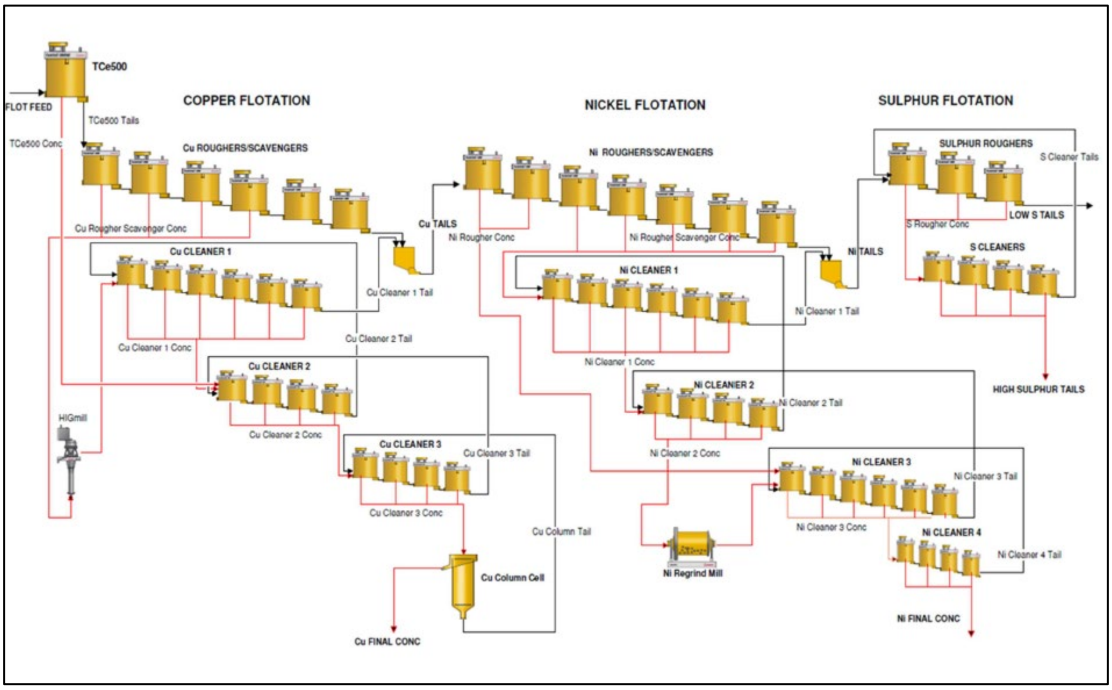


Figure 15-2: Kevitsa Cu and Ni Flotation Circuits (Source: Boliden, 2019)

15.3 Process Equipment

All equipment installed on the plant can be considered standard for the industry and is from high quality suppliers; there are no reported issues.

15.3.1 Plant performance

Plant performance for 2017 to 2019 are given in Table 15-1.

15.3.2 Plant throughput and feed grades

The concentrator has demonstrated a throughput of 7.6 to 7.9 Mtpa over the period 2017 to 2018. Cu and Ni feed grades are lower in 2019 than previous years.

SRK noted that the tonnages of Cu and Ni in feed were significantly lower in 2019 than previous years.

Cu and Ni total tailings grades are similar, resulting in lower metal recoveries in 2019.

The expansion project is designed to increase the concentrator capacity to 10 Mtpa. Based on the study, the new mill will process up to 3.6 Mtpa fresh feed.

SRK does not have any reservations regarding the design and this capacity should be achievable.

Table 15-1: Kevitsa concentrator historical performance (2017-2019 Oct)

Item	2017					2018					2019				
	Q1	Q2	Q3	Q4	2017	Q1	Q2	Q3	Q4	2018	Q1	Q2	Q3	Q4 to Oct	2019 to Oct
Dry tonnes	1,958,633	1,916,716	2,026,211	2,009,600	**7,911,160**	1,885,958	1,880,872	1,899,960	1,915,305	**7,582,095**	1,782,029	1,745,930	2,049,806	740,431	**6,318,196**
%Cu	0.403	0.395	0.434	0.440	**0.418**	0.412	0.416	0.388	0.361	**0.394**	0.299	0.305	0.292	0.324	**0.301**
%NiS	0.245	0.262	0.253	0.248	**0.252**	0.254	0.280	0.280	0.246	**0.265**	0.234	0.186	0.166	0.165	**0.191**
ppm Au	0.149	0.152	0.159	0.169	**0.157**	0.154	0.162	0.149	0.136	**0.150**	0.121	0.104	0.096	0.109	**0.107**
ppm Pt	0.292	0.333	0.321	0.325	**0.318**	0.357	0.399	0.377	0.323	**0.364**	0.325	0.214	0.186	0.206	**0.235**
ppm Pd	0.192	0.221	0.201	0.202	**0.204**	0.228	0.252	0.231	0.189	**0.225**	0.188	0.115	0.095	0.108	**0.128**
Cu conc-dry tonnes	25,917	25,961	29,431	30,406	**111,715**	27,705	29,569	27,119	25,360	**109,753**	19,842	19,370	21,563	8,287	**69,062**
% Cu	23.4	23.6	24.2	23.9	**23.8**	23.0	22.7	23.4	23.2	**23.1**	22.7	22.5	20.8	21.3	**21.9**
% NiS	1.0	0.9	0.8	0.8	**0.9**	0.9	0.9	0.8	0.9	**0.9**	0.9	0.9	0.8	0.9	**0.8**
ppm Au	4.65	4.87	3.72	4.86	**4.51**	4.66	4.77	5.17	4.66	**4.81**	4.92	3.94	3.59	3.87	**4.11**
ppm Pt	8.18	7.88	6.49	7.30	**7.42**	8.64	9.46	7.30	7.43	**8.25**	7.84	6.20	4.92	6.42	**6.30**
ppm Pd	4.94	5.55	4.94	5.05	**5.11**	5.80	6.64	6.66	6.45	**6.39**	7.45	4.90	3.70	4.33	**5.19**
% Cu rec to Cu conc	76.9	80.8	80.9	82.1	**80.3**	82.0	85.8	86.0	85.0	**84.7**	84.5	82.0	75.0	73.5	**79.4**
% Ni rec to Cu conc	5.6	4.7	4.7	5.1	**5.0**	5.0	5.0	4.0	4.8	**4.7**	4.2	5.1	5.0	6.0	**4.9**
% Au rec to Cu conc	41.4	43.5	34.1	43.4	**40.5**	44.5	46.4	49.5	45.4	**46.5**	45.2	42.1	39.3	39.9	**42.0**
% Pt rec to Cu conc	37.1	32.0	29.3	34.0	**33.0**	35.6	37.3	27.6	30.5	**32.8**	26.9	32.1	27.8	35.0	**29.2**
% Pd rec to Cu conc	34.1	34.1	35.7	37.8	**35.4**	37.4	41.5	41.2	45.2	**41.2**	44.2	47.2	41.0	44.8	**44.2**
Ni conc-dry tonnes	33,083	33,283	37,003	35,925	**139,293**	34,451	36,892	37,943	35,956	**145,242**	31,507	24,776	24,664	9,023	**89,969**
% Cu	2.9	2.3	2.3	2.3	**2.4**	2.0	1.3	1.3	1.5	**1.5**	1.2	1.9	3.2	2.8	**2.1**
% NiS	10.1	10.6	9.4	9.6	**9.9**	10.0	10.0	9.2	9.3	**9.6**	9.3	8.6	7.5	8.4	**8.5**
ppm Au	0.80	0.81	1.73	0.75	**1.04**	0.76	0.68	0.71	0.65	**0.70**	0.66	0.65	0.81	0.66	**0.70**
ppm Pt	3.45	5.01	4.72	3.70	**4.22**	4.08	4.41	5.73	4.17	**4.62**	4.96	3.25	4.04	3.03	**4.04**
ppm Pd	3.21	3.97	3.10	2.70	**3.23**	3.36	3.39	3.26	2.54	**3.14**	2.92	2.05	2.32	2.00	**2.42**
% Cu rec to Ni conc	12.0	10.1	9.8	9.3	**10.2**	8.8	6.2	6.8	7.6	**7.4**	6.9	8.7	13.2	10.7	**9.9**
% Ni rec to Ni conc	69.7	70.1	67.8	69.2	**69.2**	71.7	70.2	65.4	70.9	**69.4**	70.1	66.1	53.9	62.4	**63.6**
% Au rec to Ni conc	2.6	2.5	7.4	2.6	**11.6**	2.5	2.3	2.1	1.9	**9.0**	1.6	1.7	2.2	1.8	**9.3**
% Pt rec to Ni conc	12.3	21.2	23.3	16.5	**23.4**	16.9	18.6	29.6	18.1	**24.3**	20.5	11.2	15.3	9.7	**24.5**
% Pd rec to Ni conc	12.5	15.8	12.6	10.8	**27.9**	13.2	12.8	11.3	7.4	**26.8**	7.4	4.8	5.9	5.0	**26.9**
Float tails - dry tonne	1,899,633	1,857,472	1,959,778	1,943,269	**7,660,152**	1,823,802	1,814,411	1,834,898	1,853,989	**7,327,101**	1,730,680	1,701,784	2,003,579	723,122	**6,159,165**
% Cu	0.046	0.037	0.042	0.039	**0.041**	0.039	0.034	0.029	0.028	**0.033**	0.026	0.029	0.035	0.053	**0.033**
% NiS	0.062	0.068	0.072	0.066	**0.067**	0.061	0.072	0.089	0.062	**0.071**	0.062	0.055	0.070	0.053	**0.062**
ppm Au	0.076	0.074	0.075	0.085	**0.078**	0.074	0.076	0.063	0.064	**0.069**	0.056	0.052	0.050	0.059	**0.053**
ppm Pt	0.129	0.144	0.145	0.154	**0.143**	0.160	0.169	0.164	0.151	**0.161**	0.154	0.102	0.088	0.099	**0.112**
ppm Pd	0.075	0.079	0.075	0.080	**0.077**	0.084	0.084	0.073	0.058	**0.075**	0.055	0.033	0.029	0.036	**0.038**

15.3.3 Metal recovery

The historical metallurgical performance is given in file reference: “Millstats 2017-2019.xlsx”.

The budgeted metal recoveries to the Cu and Ni concentrates and the Cu and Ni concentrate tonnages are given in file reference: “Recovery data SRK.xlsx”.

The mill feed metal and mineral contents are given in two files: “Budgetti 2020.xlsx” & “Budgetti 2021-2034.xlsx”.

The relationship between head grade, metal recovery and concentrate grades has not been received. Site personnel noted that copper recovery is influenced by the Cu mineralisation (chalcopyrite and cubanite). No information has been provided to SRK describing this relationship.

Treatment of high talc ores results in significant copper losses. The site team advised SRK that up to 4% loss of Cu recovery from Cu concentrate occurs (when pre-flotation of talc is used during summer time with low residual reagents in process water). As the high talc ores are stockpiled and fed through in batches, the overall effect on annual Cu recovery is relatively minor, less than 0.5% overall. SRK’s check calculations of the LoM recoveries indicate that this has been included in the LoM figures. The talc concentrate is sent to low sulphur tailings disposal.

Copper concentrate

The following provides detail on the copper concentrate recoveries and grades in the forward-looking LoMp:

- Cu grade of Cu concentrate is 23% Cu. Historically, between 2017 and 2019 (October), Cu grades achieved in Cu concentrate were 23.8%, 23.1% and 21.9%, respectively.
- average Cu recovery to the Cu concentrate is 82.8%. Historically Cu recovery achieved for 2107, 2018 and 2019 (to October) was 80.4%, 84.7% and 79.6%, respectively.
- Ni content of the Cu concentrate is 0.80% Ni. Historically around 5% Ni has been recovered to the Cu concentrate with annual grades of 0.89%, 0.86% and 0.85% Ni for years 2017 and 2019 (October). In the financial model, Ni is not considered recoverable/payable in the Cu concentrate.
- Au recovery to Cu concentrate is 40%. Historically Au recovery achieved is 42 to 46% between 2017 and 2019 (October).
- Pt recovery to Cu concentrate is 26%. Historically Pt recovery achieved was 29 to 34% between 2017 and 2019 (October).
- Pd recovery to Cu concentrate is 29%. Historically Pd recovery achieved was 36 to 44% between 2017 and 2019 (October).

In SRK’s opinion, the metal recoveries in the LoMp are reasonable, and no further adjustment of Cu recovery to Cu concentrate due to high talc ores is required.

Nickel concentrate

The following provides detail on the nickel concentrate recoveries and grades in the LoM plan:

- Ni grade of Ni concentrate is 9.2% Ni. Historically, between 2017 and 2019 (October), the Ni grades achieved in Ni concentrate were 9.92, 9.58 and 8.42, respectively (drop in 2019 also reflected in resource block model grade decrease).
- Ni recovery to the Ni concentrate is 70.8%. Historically Ni recovery achieved for 2107, 2018 and 2019 (to October) was 69.5, 69.5 and 63.0%, respectively.
- Cu content of the Ni concentrate is 1.2% Cu. Historically around 9% Cu has been recovered to the Ni concentrate with annual grades of 2.4, 1.5 and 2.1% Cu for years 2017 and 2019 (October).
- Co content of the Ni concentrate is 0.40% Co. The Co recovery is calculated using the Co content in concentrate, the Co feed grade and the Ni concentrate tonnage. The average LoM Co recovery to Ni concentrate is 66% and the range 60 to 79%.
- Au recovery to Ni concentrate of around 9% has been historically achieved between 2017 and 2019 (October).
- Pt recovery to nickel concentrate is 27%. Historically Pt recovery achieved was 23.1, 24.0 and 24.3% between 2017 and 2019 (October). Historically, Pt grade in concentrate has been typically 2.4 g/t Pt.
- Pd recovery to nickel concentrate is 32%. Historically Pd recovery achieved was 27.8, 26.3 and 27.1% between 2017 and 2019 (October). Historically, Pd grade in concentrate has been typically 3.4 g/t Pd.

Tailings

The following provides detail on the tailings grades in the LoM plan:

- average LoM copper in tailings is ~0.035% Cu.
- average LoM nickel in tailings is ~0.066% Ni.

Inputs to the Financial Model

Based on the review of the metallurgy, the following metallurgical factors were suggested for the financial model developed by SRK (Section 18) and used in Mineral Reserve estimation:

<u>Copper concentrate</u>

- LoM Cu concentrate grades are acceptable at 23% Cu.
- LoM Cu recovery to Cu concentrate are acceptable. A check of the effect of high talc on Cu recovery is evident in the LoM figures and is acceptable.
- LoM planned gold recovery to Cu concentrate of 40% appears to be low and a figure of 44% should be applied.

<u>Nickel concentrate</u>

- LoM Ni concentrate grade of 9.2% Ni is acceptable.
- Ni recoveries to Ni concentrate are acceptable.
- LoM Cu recovery to Ni concentrate of 7% appears to be too low and a figure of 9% Cu should be used.

- LoM % Co in Ni concentrate of up to 0.4% is acceptable.
- LoM plan Au recovery to Ni concentrate of 12% appears to be high and a figure of 10% should be applied.
- LoM plan Pt recovery to Ni concentrate of 27% appears to be high and a figure of 24% should be applied.
- LoM plan Pd recovery to Ni concentrate of 32% appears to be high and a figure of 27% should be applied.

It is noted that historically Cu, Ni and Co in tailings average 0.033% Cu, 0.062% Ni and 0.004% Co.

15.3.4 Concentrate quality

The Cu and Ni concentrates quality is good, Cu concentrate 20 to 26% Cu and Ni concentrate 8 to 11% Ni. The typical analyses are given in Table 15-2.

Table 15-2: Cu and Ni concentrate quality

Conc	Cu	Pb	Zn	Ag	Au	Fe	Ni	S	Co	MgO	SiO_2
	%	%	%	g/t	g/t	%	%	%	%	%	%
Copper conc	20-26	<0.1	< 0.1	30 - 60	5-10	25-30	0.6-1.0	25-35	<0.05	3-9	5-15
Nickel conc	0.5-3	<0.1	< 0.01		0.5-1	30-40	8-11	25-35	0.3-0.6	3-9	5-20

Impurities, Hg, Cd, Cr, Bi, Sb, As, Se, Sn, Mn, Mo, Al_2O_3, BaO, CaO, and TiO_2 are all at acceptable levels.

From the historical concentrator figures (Table 15-1), the Cu concentrate Cu grade is normally 22 to 24% Cu and Ni concentrate Ni grade is normally 8.5 to 10% Ni, which fell in 2019. Ni in Cu concentrate is typically 0.9% Ni. Cu in Ni concentrate is typically 2% Cu.

15.4 Expansion Project

The expansion project is detailed in a study document "BOL_MAIN-#1197188-v1-Kevitsa 9 Mt Feasibility study Technical Report Feb....pdf". At the time of the site visit the expansion project was nearing completion and commissioning was due to begin in January/February 2020.

The study mentions a range of different design tonnages 9.5/9.9/10 Mtpa. Based on the LoM plan the concentrator expansion project will increase the annual throughput from the current 7.6 Mtpa up to 9.9 Mtpa; the permit specifies 10 Mtpa.

The expansion project includes some major equipment additions and a number of smaller circuit modifications (pumps, piping, electrical drives) to allow for the increase in solid/pulp flows throughout the circuit.

The main circuit modifications are:

- replacement of primary screening in crushing;
- addition of an additional 130 m^3 copper rougher;
- addition of a third SAG mill (8.75 m diameter x 8.75 m length, 14 MW) and associated circuit including feeders, feed conveying, mills discharge screen, mill sump and pumps, hydrocyclone pack and pebble recycle conveyors;

- one additional nickel concentrate pressure filter; and
- extension to the concentrator building.

SRK does not have any issues with the design and projected throughput of the expanded plant.

A detailed schedule for the expansion project has been received. The project started in 2018 and final hot commissioning is scheduled for completion end-February 2020. The target completion date was verbally confirmed during the site visit in November 2019.

Key dates for the project are given in Table 15-3.

Table 15-3: Expansion schedule

Activity	Start date	End date	Status
Overall schedule	December 2017	January 2020	
Process Engineering	March 2018	December 2018	Complete
Engineering	December 2017	May 2019	Minor items outstanding
Procurement	December 2017	September 2019	Minor items outstanding
Grinding mill	January 2018	April 2019	Delivery complete
Filter	March 2018	December 2018	Delivery complete
Electrics	February 2018	April 2019	Delivery complete
Civil works	January 2018	April 2019	Complete
Construction	May 2018	December 2019	
Installation works	August 2018 *	September 2019	All major items complete
Commissioning non mill **	May 2019	September 2019	
Commissioning (mill) ***	December 2019	End January 2020	Planned
Mill Ramp-up	February 2020	February 2020	One month allowed

* major items, excludes early preparation works
** includes testing/cold commissioning/hot commissioning filter, flotation, pumps etc.
*** includes testing/cold commissioning/hot commissioning of stockpile and mill circuit

15.5 Processing Plant Operating Costs

The detailed build-up of the processing operating costs using physicals and unit costs were not provided to SRK prior to the review. The financial cost files give the total annual cost in Euro ("EUR").

The process operating costs per tonne of ore for 2020, 2021 and 2022 are: EUR 8.42, 7.73 and 7.64/t, respectively; the costs fall thereafter to between EUR 6 and 5/t. SRK has not been provided with further details at this point.

From the expansion study document the summarised operating costs are given as follows (base case date February 2018): EUR 8.79/t (current annual production 7.6 Mtpa) to EUR 8.31/t (predicted annual production 9.9 Mtpa). These costs are inclusive of concentrate transport (refer to Section 18.4) and tailings pond costs.

Other tables in the study identify that the expansion reduces the plant operating cost from 7.12 to EUR 6.58/t of milled ore (concentrate transport excluded).

The total production cost for Kevitsa is reduced by approximately EUR 1/t of ore from the baseline cost of EUR 13.2/t to EUR 12.1/t for the expansion case.

For commentary on applied operating costs in the Economic Assessment, refer to Section 18.

15.6 Process Plant Capital Costs

Final expansion project expenditure in 2020 is EUR 2 M. Capital expenditure for the plant related items for 2020, 2021, 2022 (as captured in the overall site capital programme as presented in Section 18.5) is given in Table 15-4.

Table 15-4: Plant capital expenditure

Investment project name	Investment Specification	Reason/Explanation	2020 kEUR	2021 kEUR	2022 kEUR
Tailings Pumping System Update	Mine sustaining	To meet 9,5Mt throughput flowrates	4,400		
Kevitsa Control Room Office Area Modification	Non specified investments	New office/control room building to meet space demands	2,500	2,500	
9.5 Mt expansion project	**Expansion investments**	**Finalize 9.5 Mt project**	**2,000**		
Decanter Pump Update	De-bottleneck investments	Floating pumping station; to ensure process water quality and ensure sufficient capacity	780		
Primary screen building isolation and heating	Mine sustaining		500		
Primary crusher dump pocket fix	Mine sustaining		500		
Sulphuric Acid Storage Tank	Mine sustaining	More capacity; makes transportation easier.	300		
Mill 1 and 2 new feed chutes	Mine sustaining		300		
Fixed Vacuum Cleaning Lines in the Process	EHS investments	Makes cleaning easier/faster for contractor	300		
Surge Tank Agitator Update	De-bottleneck investments	Better mixing capacity	295		
TSF B overflow directly to the METP	De-bottleneck investments	Limit production losses due to poor quality process water	250		
Reagents preparation system - Update	De-bottleneck investments	To increase reagent mixing capacity	220		
Mill 1 and 2 new screening o/s chutes	Mine sustaining		200		
Flotation HVAC Update	Mine sustaining	Current HVAC insufficient.	200	1,750	
Conveyor belt reeling and replacing machine (2PC)	Mine sustaining		150		
CVR 9 new feed chutes	Mine sustaining		100	100	
CVR 5 feed chute	Mine sustaining		100		
Pebble bin gates	Reinvestments		96	100	
Conveyor tunnels isolated and heated	Mine sustaining		80	200	150
Concentrate Storage Capacity Expansion	Reinvestments	Makes transportation planning more flexible.	60	500	
Automatic lubrication systems for flotation cells	Mine sustaining		50	40	50
Expansion of primary crusher service area	Mine sustaining		50	300	
Flash Flotation (for the #4 mill circuit)	Expansion investments	Test flash flotation suitability to AG4 circuit. Aim is to increase recovery.	30	350	
Sulphur Flotation Feed Sampler	Mine sustaining			610	
CVR 6 feed chute	Mine sustaining			100	
Secondary crusher building isolation and heating	Mine sustaining			500	
Expansion of secondary crushers service area	Mine sustaining			50	400
Prim. Mill 1 VSD refurbishment	Reinvestments			150	
Ni Regrind Update	Reinvestments	New type of mill (HIG, SMD or ISA). Aim is to increase recovery.		500	
Rock Breaker for Primary Crusher	De-bottleneck investments			584	
Intelligent Instrument in Milling	De-bottleneck investments	To optimize mills throughput.		550	
Mill Feed Control and Management	De-bottleneck investments	To optimize mills throughput.		210	
HIMU Optimization	Expansion investments	To increase Secondary crushing capacity		2,500	2,500
BDMS, Big Data Management System	Non specified investments	Single point data access		180	

15.7 SRK Comments on Processing and Metallurgy

The following comments and issues are provided relating to the processing/metallurgy impacts on modifying factors for Mineral Reserve reporting:

- key issue identified is that the lower Cu feed (head) grades are currently resulting in lower Cu recoveries to Cu concentrate, which may continue to impact the quality of the product in the LoMp;
- equipment used is considered industry standard and high quality;
- existing plant proven up to 8 Mtpa;
- expansion project almost complete to increase plant capacity to 10 Mtpa (design), 9.5 Mtpa planned;
- expansion project (new mill) due to be commissioned in February 2020 and should be online end Q1 2020 (subject to ore availability);
- approximately EUR 2 M budgeted spend outstanding (included in the budget);
- historical operating figures show a fall in Cu and Ni feed grades and expansion is required to maintain copper and Ni concentrate production levels;
- lower feed grades may impact Ni recovery, but historical performance indicates that the impact can be managed by adjustments to the processing regime;
- Au and PGM levels in concentrates are low but payable, further information may be required to inform the financial model;
- no operating information on Ag levels in feed or concentrates; and
- Cu and Ni concentrates are clean (no contaminants).

15.8 Tailing Storage Facility

15.8.1 Introduction

SRK has completed a desktop review of the design and monitoring documentation related to the Kevitsa Tailings Storage Facility (“TSF”), which is operated by Boliden in Finland. SRK has not performed an inspection of the facilities discussed in the following report section and therefore cannot comment specifically with regard to the current condition of the TSF from a geotechnical or dam safety perspective.

Tailings are deposited as a slurry at 30% solids w/w by sub-aerial discharge in a full perimeter paddock style impoundment. The tailings storage area is divided in two individual cells, referred to as TSF A (flotation tailings storage which represent 98% of total tailings mass) and TSF B (high sulphur tailings storage, which represents 2% of the total mass). The current tailings production rate is 7-8Mtpa, increasing to 9.5 Mtpa from 2022. The remining mine life (related to Stage 4 mining activities) is forecast to extend until 2034.

SRK has undertaken the design review using International Best Practice Guidelines (Canadian Dam Association. Guide to the Management of Tailings Facilities. Version 3.1, 2017). Where the TSF designs do not conform to these guidelines, SRK has made recommendations for future work which should be considered to ensure that each TSF meets compliance across all areas.

15.8.2 Documents reviewed

The following key documents were reviewed during preparation of this section:

- Kevitsa TSFA: Tailings Characterisation, Seepage and Stability Analysis. Golder Associates. Report No. 1780041/B.0. November 2017 (Romain, 2017).
- Kevitsa TSFA: Deposition Modelling over the Life of the Facility. Golder Associates. Report No. 1779683.RPT01.B0. March 2018 (Girard, 2018).
- independent review of dam safety at the Kevitsa Mine Finland. Independent Technical Review Board (ITRB). Document no. 190118LR_Kevitsa_ITRB-Boliden Comments.docx. M10160A02.730 (Rönnblom Pärson, 2019).
- Kevitsa Mine Tailings Storage Facility Monitoring Report: Annual Report, 2018. Golder Associates, March 2019 (Girard, 2019).
- Deposition Memo Fall 2019, Boliden. Doc Reference Deposition Memo Fall Rev2.docx (Golder (unnamed), 2019).
- Memorandum: Status update Kevitsa Closure Plan and Current Best Estimate for Kevitsa Closure Costs. Boliden, July 2019 (Boliden (unnamed), 2019).

15.8.3 Key technical issues

The following technical issues were observed by SRK:

- The current embankment crest of the facility is at elevation 247 m RL (referred to as' Stage 5 Raise' by Boliden). The current maximum tailings elevation is approximately 245 m RL and the pond is 240 m RL. This indicates that the minimum freeboard of 1.5 m between the pond and embankment crest is currently being maintained.
- During 2018, Golder completed volumetric modelling to confirm the remaining storage capacity in the TSF A Cell. Approximately 34.7 Mt of tailings had been deposited in the TSF as of September 2017. A range of in situ dry densities and upstream slope geometries were assumed during volumetric analysis. The outcomes of this assessment under each scenario are summarised in Table 15-5. At this time, the remaining storage requirement was some 126.6 Mt until 2031. Golder estimated that the TSF will reach design capacity (Stage 13: 270 m crest elevation) during early 2030. SRK notes that the Stage 4 mining plan has since been extended to 2034 (140 Mt storage requirement from January 2020). An additional 39 Mt of tailings will be generated over years 2030 to 2034 (inclusive), which equates to approximately 26% of the total tonnage storage capacity estimated by Golder. This material will have to be stored in an alternative location.

Table 15-5: TSF A Remaining Storage Capacity (Golder 2019)

Criteria			Tailings (Million tonnes)	% of Tailings Produced
Tailings Produced			161,4	-
Available Storage in TSF A	6,6 H:1V Average Slope	at 1.6 t/m³	142,5	88
		at 1.7 t/m³	149,3	93
		at 1.8 t/m³	156,0	97
	5,3 H:1V Average Slope	at 1.6 t/m³	149,0	92
		at 1.7 t/m³	156,1	97

- SRK understands that Boliden has initiated early stage concept studies to investigate potential storage options for tailings post 2034 (Stage 5 mining phase); however, there is no plan in place to address the 25% deficit in storage capacity for storage of tailings generated during Phase 4. Whilst there is sufficient storage capacity for another 10 years of production, SRK notes that permitting of new TSF facilities in Finland can take up to 10 years from concept to commissioning Phase (particularly at sensitive sites in proximity to Natura 2000 areas) and hence engineering studies and permitting activities should be considered during 2020 to mitigate the risk of scheduling delays towards the end of the TSF life.
- SRK notes that due to the design of embankment raises (upstream raise method), the available surface area for storage of tailings is reduced significantly as the TSF moves towards terminal elevation. Golder has estimated that the facility will reach a maximum rate of rise of 3.2 m/annum at the permitted final crest elevation in 2030. The potential to raise the facility above this elevation (if permitted in the future) is limited, as the rate of rise may exceed safe operating criteria.
- Current water balance modelling indicates that approximately 2.5 Mm^3 of storage capacity is required in the TSF (Golder, 2018). Volumetric modelling by Golder indicates that during the latter stages of construction, the maximum pond volume (i.e. below the 1.5 m minimum freeboard marker) will be limited to 1.5 Mm^3. This could further impact storage capacity during the latter stages of the operational life of the facility.
- Increased seepage flow rates through the basal layers of the TSF (where there is no basal liner and compacted peat only) have the potential to impact groundwater quality in proximity to the TSF. Updates of the water balance and predictive modelling have been recommended (by Golder) to address this risk. SRK understands that these studies are ongoing. Remediation measures may have to be designed if predictive models indicate that environmental discharge limits are breached in the future (perhaps as a result of increased production rates); however, this is not currently the case. Boliden reports that pilot testing of a new active groundwater extraction/treatment system was undertaken during 2019. The design of the water treatment system has yet to be finalised.
- Recent piezometer cone penetrometer test (“CPTu”) investigations have been carried out within the tailings material by Golder (2017). This work was intended to characterise the liquefaction susceptibility of the tailings stored within the facility. The investigations identified a loose, potentially contractive layer in the NW corner which extended from the starter embankment towards the centre of the facility. Limit equilibrium stability analysis (drained and undrained analysis) was carried out (Golder, 2017) to assess the impact on global slope stability. The required factor of safety against slope failure was obtained under all scenarios. Golder recommended that additional CPTu investigations were completed around all embankment sections, to ensure that the presence of potentially liquefiable material is accurately mapped around each sector and incorporated in subsequent stability analysis.
- Detailed deposition planning (including 3D modelling) is currently undertaken by Boliden, to ensure that tailings deposition is occurring in the correct sector and that construction of embankment raises can scheduled within the short 7-month summer construction season each year. SRK considers this approach to be in line with international best practice and allows the operator to effectively plan over the short term.
- SRK considers the operating management system (“OMS”) documents to be systematic

and detailed. The documentation meets the requirements set out in the Mining Association of Canada ("MAC") Guidelines. It is noted that the latest OMS document is out of date (latest version issued during 2018); however, this information is readily available to all Boliden employees on the internal OMS (alongside all relevant documentation to the facility, such as audit templates, monitoring plans, and environment, health and safety legislation).

- The latest independent technical review board ("ITRB") audit for the facility was carried out during 2018. The following key comments were made with regards to the TSF operation and management of the facility:
  - Design flood for both TSF Cells is based upon a 5000-year return period and does not consider climate change. This is not considered to be sufficient for this consequence category facility ('High Consequence Category'). As defined in the Canadian Dam Association Dam Safety Guidelines ("CDA", 2013), the target for flood storage should be 1/3 between 1:1000 and 'Probable Maximum Flood', which will exceed this value. Should a higher storm storage be considered at the TSF, this could impact the LoM storage of the facility.
  - Additional CPTu test work was recommended by the ITRB, to ensure that the presence of loose, contractive material has been accurately quantified around each embankment section. SRK notes that Golder has subsequently completed CPTu testing and analysis to partially address this concern.
  - ITRB recommended than an Engineer of Record ("EOR") was appointed to ensure that the TSF is operated and maintained as per the design intent.
  - ITRB concluded that the current dam break inundation study required an update, with the results being used to update the Emergency Preparedness Plan ("EPP").
  - Additional work was recommended to develop the EPP for the project. It was acknowledged that Boliden had a comprehensive side wide rescue plan in place; however, additional information was required to ensure that the EPP meet the requirements set out in MAC Guidelines
- An annual monitoring report is produced by Golder which summarises collated data and any deviations recorded. All collected data are checked against 'trigger levels', which have been defined by Golder through stability analysis. No significant exceedances were recorded during 2018 and hence the facility was operated within anticipated parameters. SRK considers the number and location of instruments to be suitable for a facility of this size. Additional monitoring provisions should be considered when Stage 6 embankment construction is complete.

15.8.4 Suggested edits to financial model

Based upon review of the data provided, SRK recommends the following key edits to the financial model:

- Allowance for shortfall in Stage 4 tailings storage for period 2030-2034. This is estimated as EUR 10 M for construction (2028 to 2029) and EUR 4 M per annum for subsequent lifts thereafter.
- Contingency for closure engineering. Should additional imported borrow material be required for cover design, Boliden estimates that a surcharge of EUR 7.9/m^2 could be

required for closure. Based upon the TSF A area only, this could increase the current closure provision by some EUR 20 M.

15.8.5 SRK comments on TSF

In SRK's opinion, the following areas required further work to improve the TSF management:

- Design work should be progressed for the preferred new TSF location, such that an optimised solution for tailings storage post 2030 can be realised. The permitting status and timeline for the new TSF locations need be checked in line with the project implementation schedule to ensure that no delays will be incurred to project as a result of government approvals.
- Given the size and complexity of the TSF and based on SRK experience with other international mining operations, at least one dedicated full-time intermediate to senior geotechnical engineer would generally be assigned to this facility. In addition, international practice generally has an EOR associated with a TSF (not required under Finnish regulations).
- Some key documentation, notably the EPP, related to the TSF should be updated to meet the requirements set out in international guidelines such as MAC (2019). This should be updated with the results of an updated dam break study.
- Tailings planning: TSF height will increase by 25 m over the next 10 years. This will require diligent tailings management, planning, design, and implementation to achieve this with the current summer and winter operational constraints.
- Closure design: additional closure engineering is required to reduce the uncertainties associated with borrow material sources for the engineered cover system. This should be completed well in advance of decommissioning of the facility, to minimise the risk of unforeseen expenditure at closure.

16 PROJECT INFRASTRUCTURE

Construction at Kevitsa began in 2012 with production commencing in 2014. The following infrastructure has been completed on-site:

- mining machinery workshop;
- primary crushing / screening;
- interim crushed ore stockpile storage area;
- fine crushing plant;
- milling hall;
- flotation plant;
- concentrate processing plant;
- final concentrate storage area;
- chemical storage area;
- main warehouse;
- small machine repair shop;
- office building; and
- sample processing facilities.

The construction of a new Heavy Mining Machinery workshop building was completed in 2019 to support the additional mining trucks acquired by Kevitsa.

16.1 Power Supply

Power supply is sourced direct from the grid which is mostly from hydropower and nuclear reactors. The Vajukoski hydropower station and dam are located close to the mine.

16.2 Water Supply and Water Systems

Water supply and water management systems are in place. A description can be found in Section 14.11.

16.3 Fire Protection System

Fire detection and alarms are provided for all substation buildings. The main plant has been provided with fire detection where appropriate and all plant and workshop areas have fire hydrants strategically located for manual fire protection in addition to hand-held extinguishers.

16.4 Plant Buildings

The process plant is provided with a range of non-process related buildings to support the operations. These include, but are not limited to control rooms, laboratories, workshops, warehouses, security and mine access infrastructure, and ablution facilities. The entire facility is enclosed to allow year-round operation and access.

16.5 Mine services

The mining fleet services facility is located near the Process plant. The workshop is fully enclosed and heated to allow year-round activities. The mining and ancillary contractors have their own workshops and maintenance facilities in areas allocated to them

16.6 Roads and site access

Access to the mine site is via excellent, well-maintained all-weather sealed roads. The town of Sodankylä is located approximately 40 km south by road and the nearest village Petkula is 8 km west of the property.

17 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

17.1 Introduction

This section covers SRK's review of the environment, social and governance ("ESG") aspects of the Kevitsa operation. The review included a site visit in November 2019 and a review of various documents provided by the Kevitsa management team during the site visit. The majority of the documentation provided were in Finnish. This is a constraint in terms of reviewing details of some aspects such as monitoring programmes. The key documents reviewed are summarised Table 17-1.

17.2 Setting

17.2.1 Administrative

The area is governed by the regional land use plan of Northern Lapland, in which it has been designated as a mining site, and by a local master plan for the Lokka-Koitelainen-Kevitsa area (2001), in which it has been reserved for mining and quarrying.

17.2.2 Environmental

There are several Natura 2000 sites near the Kevitsa mine: The nearest two are Koitelaiskaira Wilderness (FI1301716, 43,938 hectares) immediately to the east of the mining site and Pomokaira Wilderness (FI1301712, 92,358 hectares) approximately 5 km to the west (Figure 17-1).

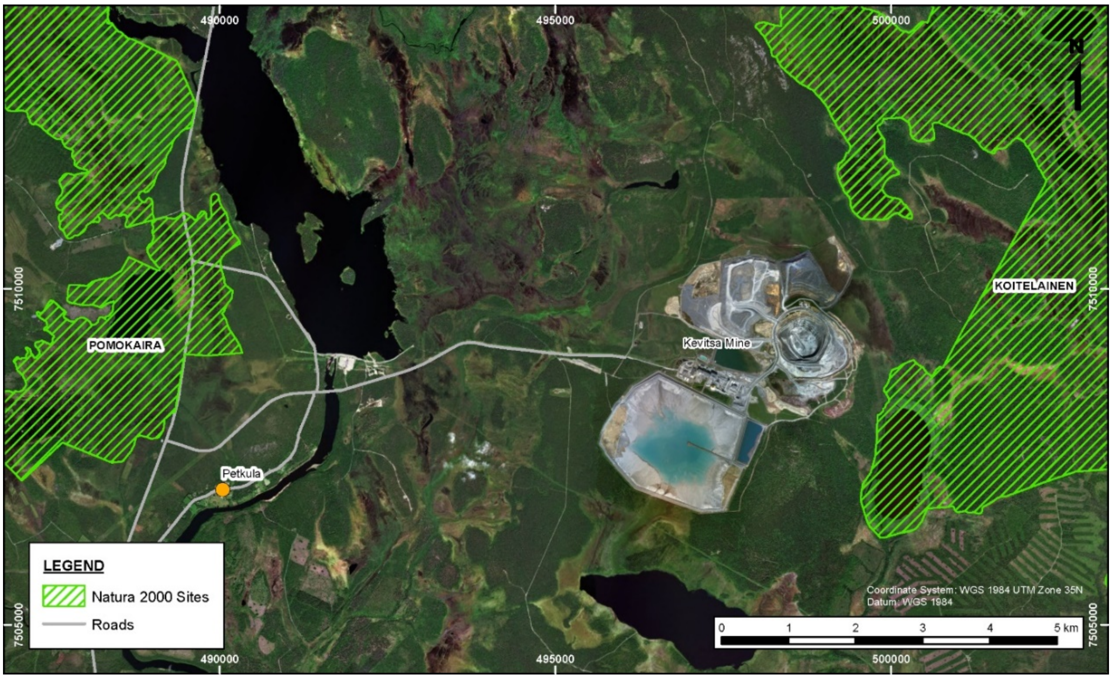


Figure 17-1: Kevitsa location and adjacent Natura 200 sites

Table 17-1: Documents reviewed by SRK

Document Ref	Document Name	Summary
BOL_MAIN-1564976 -v2-GAP-analysis+Kevitsa+ ISO+ 45001+ Finding+answers(1)	GAP-analysis Kevitsa ISO 45001 Desk review onsite - 2019	External audit findings with Boliden assessment of root cause, corrective action required and verification of effectiveness including target date for completion.
BOL_MAIN-# 1566875-v1-BOL- MAIN-_1535630-v1 -Kevitsa_Internal_Audit _Closing_Meeting-PPT	Preliminary Report – Internal Audit, Kevitsa 11 September 2019	Internal audit findings of positive observations, improvement areas and deviations. The report states that once all deviations and improvement areas will be inserted into the issues management system and corrective action implemented within 3 months.
Kevitsan sulkemissuunnitelma 20191031	Kevitsan Kaivoksen sulkemissuunnitelma 31. 10.2019	Mine closure Plan 2019 (document in Finnish) which discusses the closure vision, risk assessment and closure options for the pit lake (requires bioremediation), WRD and TSF.
Kevitsa – Closure Provision Memo Q3 2019	Memorandum:Status update Kevitsa closure plan and current best estimate for Kevitsa closure costs 2019	Discussion on updating closure costs based on changing the base case design (a concave cover on the TMF) to the modified design (convex cover on the TMF) to achieve compliance with water quality standards at closure.
Kevitsa EIA 2010	Scaling up of the Kevitsa mine environmental impact assessment 2011	Review of the EIA **abstract** to scale up annual production from 5 Mtpa to 10 Mtpa by expanding the open pit.
1655025.505_Kevitsa TSFA Task 1 Report_A.1_Oct17 final	Kevitsa TSFA groundwater remediation scheme, Task 1: data appraisal, gap analysis and conceptual model (October, 2017)	Report to develop a groundwater management scheme to address elevated concentrations of Cl, Co and Ni in groundwater in the vicinity of TSF A following a review of monitoring results by ELY. ELY required the company to present a justified and timetabled plan to prevent further deterioration of groundwater quality.
1.0_Boliden Kevitsa_ Ympäristötarkkailun vuosiyhteenveto_päivitetty	Boliden Kevitsa Mining Oy Kevitsan Kaivoksen Ympäristötarkkailun Vuosiyhteenveto 2018	Annual summary of 2018 environmental monitoring (document in Finnish). Monitoring was performed by Eurofins and includes monitoring of emissions, surface water, groundwater, freshwater ecology, terrestrial ecology, air quality and noise to applicable permits and obligations.
BOL_MAIN-#1566875-v1-BOL_MAIN-_1535630-v1-Kevitsa_Internal_Audit_ Closing_Meeting	Preliminary Report (internal audit 11.09.2019)	PowerPoint presentation of the internal audit closing meeting highlighting the positive observations, areas of improvement and any deviations.
BOL_MAIN-#1562830-v1-Kyläillan_24_10_2019_ pöytäkirja_AS	Kevitsan kaivoksen kyläilta	Stakeholder meeting (24.10. 2019) minutes, which also notes general concerns raised by the community members.
INST-21013-v.1.0 Environmental Deviation Reporting Instruction	Environmental Deviations Reporting Instruction	Reporting mechanism for complaints or comments from external stakeholder relating to the environment.
Boliden Annual & Sustainability Report 2018	Metals for a Sustainable Society - 2018 Annual & Sustainability Report	Group level annual report from 2018.
GDLN-22203-v.1.0 Guidelines for recruitment and selection within Boliden Mines' operations	Guidelines for recruitment and selection within Boliden Mines' operations	Group level recruitment guidelines for HR.
1 Työntekijän yleiset vastuut	The Employee's General Responsibility	General obligations of the employee.
BOL_MAIN-#1537522-v1-Eettinen_ohjeisto_Boliden_Kevitsav.fi.en	Code of Conduct	Ethical guidelines and code of conduct in the workplace (following UN Universal Declaration of Human Rights and the International Labour Organisation's core conventions).
BOL_MAIN-902674-v4-Procurement+Policy	Procurement policy	Corporate level procurement policy and guidelines.

The water bodies in the immediate vicinity of the Kevitsa mine are located within the drainage basin of the Kemijoki River and Kitinen River. The majority of the water bodies in the area are streams, and the lakes in the area are relatively small with the exception of Lake Vajunen. The nearest lakes are Lake Satojärvi and Lake Saiveljärvi. The nearest stream to the site is Mataraoja Brook, which has its source within the boundaries of the current site. The Kitinen River that runs approximately 5 km to the west of the site has been dammed to create. Vajunen Reservoir. The reservoir services the Vajukoski Hydropower Station. Kevitsa draws fresh water from the hydro power dam and also discharges directly to the same dam via a return pipeline (Figure 17-2).

There are no classified aquifers within the mining site or in its immediate vicinity. The nearest classified aquifer is located approximately 8 km to the south of the mining site boundary.

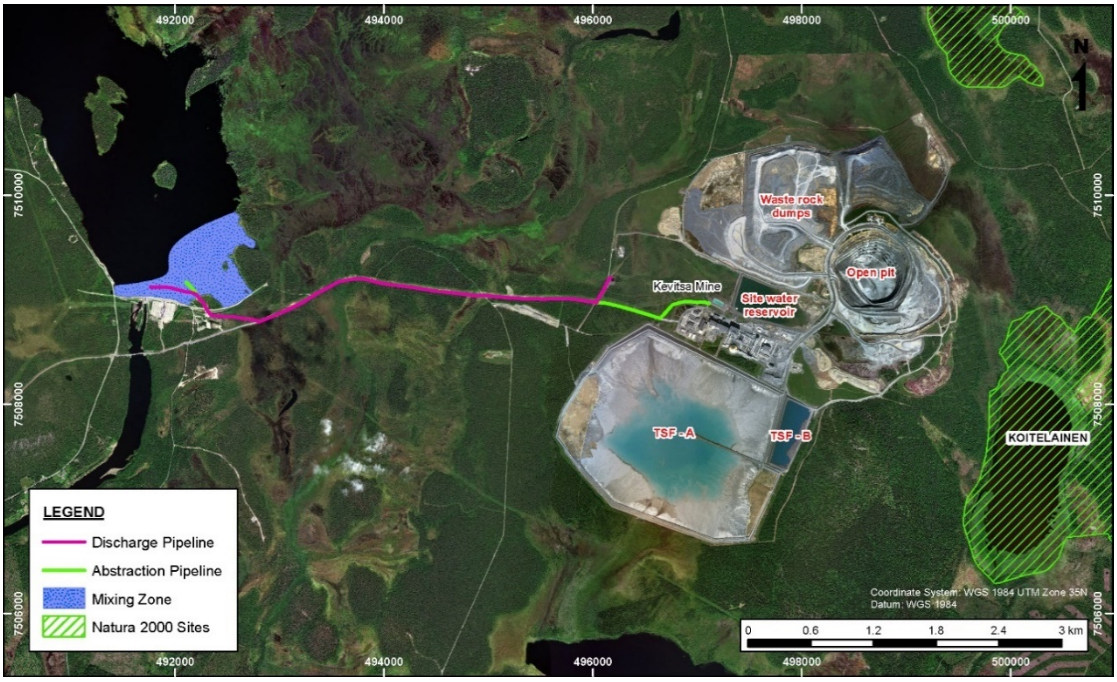


Figure 17-2: Kevitsa water abstraction and discharge routes

17.2.3 Social

There are no nationally, regionally, or locally recognised cultural landscapes in the vicinity of the mining site. The most important economies in the area have traditionally been forestry and reindeer husbandry. The area is also used for various recreational purposes, most importantly for hunting, fishing, and berry picking. There are no residential properties or other buildings within the mining site or in its immediate vicinity. The nearest holiday homes are located to the northwest of the mining site and on the southern banks of Lake Saiveljärvi. Lake Saiveljärvi is located immediately south of TSF-A. The nearest permanent homes are located in the village of Petkula, approximately 4 to 5 km from the site boundary (Figure 17-1).

17.3 Status of Environmental Approvals

17.3.1 Requirements

For the development of a mine an environmental impact assessment (“EIA”) is required, as stipulated in the Act on Environmental Impact Assessment Procedure (252/2017) and associated Decree on Environmental Impact Assessment Procedure (277/2017). The EIA process has a steering group made up of the company, local authorities, community representatives and non-governmental organisations (“NGO”). The report is made available to stakeholders for comment by the Lapland Regional Environmental Centre (“ELY”) who is the local authority responsible for evaluating the EIA and monitoring compliance with regulatory requirements during operation.

The final approval for a project is split into two separate processes in Finland, firstly the preparation and approval of an EIA; secondly applying for an environmental and water permit (combined process that includes the requirements of the EU Extractive Industries Waste Directive).

Once the EIA is approved, an application is made for the environmental and water permit/s in terms of the Environmental Protection Act (EPA 86/00), Environmental Protection Decree (169/00) and the Finnish Water Act (264/1961) (all as amended).

17.3.2 Status

The first Environmental and Water permit for the Kevitsa operation was granted in 2010 (No. 46/09/1; No. PSY-2007-Y 101 / Authorization 66) following the approval of the EIA (LAPELY /94/07.00/2010/07.00.11.04). Construction of the mine began in spring 2010 and production began in the summer of 2012. A monitoring program, prepared by Pöyry Finland Oy as part of the EIA, was approved by the Lapland ELY Centre on 20 April 2012.

In 2014, an Environmental and Water permit was granted for the expansion of production, which received the Supreme Administrative Court's decision No 522/1/16 on February 15, 2017.

The 2014 Environmental and Water permit contains 82 conditions covering all aspects of the operation including discharge water quality, waste facility closure bonds, monitoring requirement and closure requirements.

The Finnish permitting process is relatively complex and multiple permits are required to be in place at any one time. Kevitsa stated it currently has over 50 permits for the operation. To keep track of its legal obligations Kevitsa has subscribed to an on-line software programme provided by Ramboll (‘LAWLY’). This is a live system allowing Kevitsa to keep up to date with the relevant regulatory requirements for the operation. The software lists applicable legislation, licences and permits required and allows Kevitsa to record, track and assign responsibility for compliance. This is one of the more comprehensive compliance recording and tracking registers observed by SRK. At the time of the site visit the population of the system was still in progress.

Kevitsa has also developed a written procedure to formalise and record how documentation is delivered to, or received from, the regulatory authorities.

17.4 Overview of Environmental and Social Management at the Operations

17.4.1 Environmental Management Systems

Resources

Kevitsa has a designated health, safety, environment and quality (“HSEQ”) department with an overall manager and discipline area for each section. The Environment Section Head is supported by three technical environmental staff who are responsible for internal monitoring and maintenance of the management system elements of the operation. Environmental monitoring required from a regulatory perspective is outsourced to ensure independence and transparency.

The safety department consists of a Section Head and six support staff. There is also a quality section and health section responsible for the on-site clinic.

Management systems

Kevitsa is part of the New Boliden group of companies and in line with the corporate requirements has set an objective of becoming ISO 14 001 certified in 2020. The operation draws on the Boliden corporate HSEC policies and already has in place risk registers, operating procedures, monitoring programmes, audit programmes and emergency response plans.

There are a series of internal and external audits and inspections carried out through the course of the year. As noted in some of the internal reports, the scheduling of the audits and coordination between Kevitsa site management and Boliden can be improved.

Kevitsa has a comprehensive bio-physical and biological monitoring programme with compliance monitoring carried out by third parties.

The monitoring package is divided into the following sections:

1. Usage monitoring;

2. Emission monitoring;

3. Surface water quality;

4. Groundwater quality;

5. Biological monitoring in surface waters;

6. Biological monitoring of land;

7. Air quality; and

8. Noise.

Bio-physical monitoring is carried out monthly with internal checks done daily and weekly for specific internal controls. Biological monitoring is carried out every three years.

17.4.2 Social Management Systems

Kevitsa does not have a formal social management system in place and there was no stakeholder engagement plan provided for review. Minutes from a stakeholder engagement meeting dated October 2019 indicate that meetings that do take place are recorded. Main questions and concerns raised by the stakeholders appear to be related to traffic and noise pollution, compensation for affected communities from nearby villages and financial/corporate social responsibility support for the villages. Responses by the Company are also minuted.

The community meetings obviously provide a mechanism for the local residents to raise any issues or complaints; however, there does not appear to be a formal grievance mechanism in place and a grievance log was not provided for review. It is not clear if all issues are systematically logged and responses provided.

Kevitsa has a written procedure for the recording of community complaints or grievances when they are raised. This is part of the environmental incident reporting system (Centuri) and it is reasonable to expect that the majority of community grievances will be in relation to the environmental performance of the operation (INST-21013-v.1.0 Environmental Deviation Reporting Instruction).

17.5 Key Environmental and Social Issues

Groundwater monitoring has identified contamination of groundwater in the vicinity of the TSF. This suggests TSF water seepage through the peat/bentonite layer at this facility. A corrective action plan is being developed with the assistance of Boliden's external consultants, Golder Associates; however, the issue is currently unresolved. SRK understands that an additional TSF will be required to meet the waste storage requirements for the full LoM. The groundwater contamination will likely make the permitting of an additional TSF a challenge with additional scrutiny of site selection, design and closure plans. This could prove to be on the critical path for the operation to achieve production out to the current plan of 2034 and subsequently beyond this date.

The occurrence of a rare moss species (*Dichelyma capillaceum*) in the area of the planned WRD extension is putting the approved location and design of this facility at risk. It was originally discovered in 2017 and at the time was the most northern site known for the species in Finland. Boliden has subsequently conducted several surveys during the course of 2018 and 2019 and have identified a further 5 and 6 sites respectively where the moss occurs.

A rare frog species (*Rana arvalis*) has also been identified at the site as part of the baseline studies in 2012 for the mine expansion. As with the moss, Boliden has sponsored a number of additional surveys which have identified multiple additional sites outside the proposed WRD footprint where the species occurs. SRK understands that this provides options for the potential relocation of both moss and frog populations if required. The additional habitat areas also provide options should groundwater drawdown associated with the pit extensions impact on wetland areas that form part of the frog habitats.

Boliden has stated that it sees no constraints in term of land availability to relocate and expand the WRD should this be required. This may imply a longer haul distance. Should Boliden decide to continue with the current planned WRD expansion, there is a risk of delays associated with permitting the disturbance of the moss and frog species, which are included on the Annex II and Annex IV lists respectively under the EU Habitats Directive.

The 2018 environmental monitoring report concluded that elevated heavy metals were observed in some bioindicators (such as soil, humus and moss) collected from around the open pit and TSF areas. The report stated that the levels observed were higher than 2015 and probably due to dust deposition from blasting, traffic and tailings deposition. Higher dust concentrations were also observed further away from the mine and this was attributed to 2018 being generally a drier, warmer year contributing to the wider dispersion of dust. All other parameters (water discharges, surface water, air quality and noise) remained in line with previous year's results. SRK is not aware of any actions required are a result of the findings of the 2018 monitoring. Boliden will conduct further surveys in 2021 to determine whether the elevated levels continue to trend higher.

17.6 Closure Planning and Cost Estimate

Kevitsa has produced a very detailed closure plan for the operation. This includes detailed hydrogeology and geochemistry assessments. At the time of the visit the company had recently submitted an update to the authorities to address the changes to the operation including the planned increase in production. They anticipated being asked for additional information before the plan is accepted. Kevitsa has provided SRK with a closure provision memo dated Q3 2019. The memo states that trade off studies on various closure designs will be completed in Q4 2020. The memo includes cost estimates for the main closure costs components (Table 17-2). Given how difficult closure costs are to estimate SRK has applied a 25% contingency to the current estimate. This assumes that a concave TSF will be acceptable as a final landform. An additional approximate EUR 22 M will be required to transport fill material to create a convex post closure shape on the TSF. SRK has also included a 'place-holder' cost for staff retrenchment costs.

These costs are based on the following assumptions:

- 500 employees paid 3 months' salary (estimate EUR 2,000 per month) for retrenchment;
- concave final landform for TSF A with no requirements for additional fill material (add another ~EUR 22 M for a convex landform); and
- cost for rehabilitation of roads, pipelines, and other smaller infrastructure not included.

Table 17-2: Kevitsa Closure Cost Estimate

Infrastructure	Closure action	Area (ha)	EUR/m^2	Total EUR (000s)
Waste rock	-	307	17.9	54,953
TSF A	Assumes a concave final shape with water management	280	17.9	50,120
Additional material	*Assumes a convex shape is required for water and post closure seepage management*	*227*	*9.60*	*21,792 (Currently excluded from the total)*
TSF B	-	13	13.89	1,806
Water treatment	Ongoing treatment and plant	-	-	12,000
Pit recontouring	Recontour	-	-	1,000
Monitoring	Ongoing environmental monitoring	-	-	2,250
Ni rich moraine area	-	-	-	50
Processing plant and admin buildings	Demolish, removal of waste and rehabilitate ground	36	-	10,000
Other (SRK estimate)	Retrenchment (500 employees)	-	-	3,000
Sub-total				**135,179**
Contingency (SRK assumption)	25%	-	-	33,795
Total				**168,974**

17.7 SRK Comments on ESG

The various ESG aspects that contribute to mine plan and financial have been reviewed by SRK to contribute to the Mineral Resource and Mineral Reserve reviews to understand any fatal flaws.

SRK has provided an estimated closure cost to provide input into the financial model testing the economic viability of the project.

SRK considers that the ESG procedures in place are fit for purpose with no fatal flaws identified preventing the reporting of Mineral Resource and Mineral Reserve statements.

18 ECONOMIC ASSESSMENT

18.1 Introduction

SRK has prepared a technical economic model to test the economic viability of the Mineral Reserves and assess the suitability of the currently applied NSR cut-offs. SRK notes that no pre-existing financial model for Kevitsa was available for review, and hence SRK had to set one up based on information provided by the Company. Information in particular with regards to operating costs from 2021 onwards was very limited. The model and the results herein are presented in Euros or million Euros ("EUR M"), with an exchange rate of 1.17 USD:EUR applied to those components stated in USD. The model is presented in real money terms.

18.2 Technical Drivers

The basis of the economic assessment is informed by the mine and mill feed production plans. The Mining and mill feed LoMp production totals are inclusive of modifying factors.

The processing parameters as applied follow from SRK's review of the operations and are presented in Table 18-1. The recoveries for Cu to Cu concentrate and Ni to Ni concentrate were based on LoM average numbers, with year on year variations based on their respective head grades and mineralogical composition of the mill feed.

Table 18-1: SRK Adjusted Metallurgical Parameters

Processing Plant Input Parameters	Units	Cu Concentrate	Ni Concentrate
Metallurgical Recoveries			
Ni	(%)	-	70.8%
Cu	(%)	82.8%	9.0%
Au	(%)	44.0%	10.0%
Pt	(%)	26.0%	24.0%
Pd	(%)	29.0%	27.0%
Co	(%)	-	60.0%
Concentrate Grade			
	(%)	23.0% Cu	9.2% Ni

18.3 Commodity Prices and Smelter Terms

Commodity prices as applied in the technical economic model are as presented by the Company. SRK has compared these with latest consensus market forecasts ("CMF") available to SRK and is comfortable with the numbers used by the Company (Table 18-2). SRK notes that in the short-term different prices are typically applicable. Similarly, long-term prices are considered adequate, but SRK notes that short-term variability to smelter terms will be applicable.

Table 18-2: Commodity Prices

Commodity	Units	Company	CMF
Ni	(USD/t)	16,000	15,400
Cu	(USD/t)	6,600	6,500
Au	(USD/oz)	1,200	1,300
Pt	(USD/oz)	1,000	970
Pd	(USD/oz)	1,000	1,065
Co	(USD/lb)	20	18

Smelter terms have been provided by the Company and have been incorporated as such. Payable metal is calculated as follows:

- Cu concentrate:
  - Cu: deduct 1% unit: with a concentrate grade of 23%, this results in a ((23-1)/23=) 95.6% payability;
  - Au: deduct 1 g/t, remainder payable;
  - Pd: if grade in concentrate <= 6.68 g/t, deduct 2 g/t, otherwise 70% payable;
  - Pt: as per Pd; and
- Ni concentrate:
  - Ni: 90% payable;
  - Cu: 80% payable;
  - Au: if grade in concentrate > 1 g/t, 70% payable;
  - Pd: as per Au;
  - Pt: as per Au; and
  - Co: 35% payable.

Applicable treatment charges ("TC") and refining charges ("RC") are presented in Table 18-3. For comparison with other sections in this report, the above mentioned (SRK-adjusted) recoveries, commodity prices and smelter terms, result in the NSR formula as follows:

$$NSR\ (EUR) = (62.66 \times \mathrm{NiS}) + (44.40 \times \mathrm{Cu}) + (8.57 \times \mathrm{Pt}) + (8.48 \times \mathrm{Pd}) + (10.84 \times \mathrm{Au}) + (67.27 \times \mathrm{Co})$$

Table 18-3: Treatment and Refining Charges

Treatment and Refining Charges	Units	Cu Concentrate	Ni Concentrate
Treatment Charges			
	(USD/t)	80	190
Refining Charges			
Ni	(USD/lb)	-	1
Cu	(USD/lb)	0.08	0.5
Au	(USD/oz)	6	35
Pt	(USD/oz)	15	35
Pd	(USD/oz)	15	35
Co	(USD/lb)	-	3

18.4 Operating Costs

Operating costs for 2017-2019 and budget 2020 have been provided on a department by department basis and have for the purposes of this report been summarised in three main categories: mine, plant and General & Administration ("G&A"). Concentrate freight costs are captured by the Company under the processing department but are presented separately herein for 2019 and 2020.

SRK notes that whilst a decrease in processing unit costs for 2020 is realistic considering the higher throughput, the sharp decrease in mining unit cost appears optimistic. All waste stripping costs are capitalised.

In addition, a high-level itemised budget for 2020-2034 was provided by the Company which wasn't split per department. Along with the capitalised waste stripping cost as projected in the capital expenditure forecast, this was then split into mining and other. For SRK's assessment, the G&A and concentrate freight costs were based on unit costs and remained flat since the Milled tonnes per annum were constant over the LoMp (10 Mtpa). This resulted in a variable unit cost for processing over the LoM (Table 18-5).

The mining unit cost for the mines' 2020 budget was lower than the historical costs achieved between 2017-2019. SRK was not provided with any details or feedback from the mine as to the reasoning for the reduction in unit costs. SRK suspects it might be due to a new equipment fleet, cessation of the mining contractor in the first quarter in 2020, and the commencement of a trolley assist system and the introduction of a slightly larger new fleet. In light of the lack of detail surrounding the reduction of the mining cost over time, SRK ran a sensitivity to mining costs to test the Mineral Reserve's sensitivity to an increased mining cost.

Table 18-4: Historical and Budget Unit Operating Costs

Parameter	Units	2017	2018	2019	2020 Budget
Technical Drivers					
Ore Mined	(kt)	8.4	7.9	7.7	9.6
Waste Mined	(kt)	34.0	33.4	32.2	37.0
Total Mined	(kt)	42.4	41.3	39.9	46.6
Mill Feed	(kt)	7.9	7.6	7.5	9.1
Cu con	(kt)	111.7	109.8	80.2	96.4
Ni con	(kt)	139.3	145.2	104.8	124.7
Unit Costs					
Mining (ore & waste)	(EUR/t mined)	2.78	2.98	3.14	2.47
Processing	(EUR/t milled)	6.55[1)]	7.22[1)]	6.73	6.35
G&A	(EURk)	15,639	14,515	22,591	22,634
Freight					
Cu con	(EUR/t Cu con)	Incl above	Incl above	34.71	36.74
Ni con	(EUR/t Ni con)	Incl above	Incl above	40.66	45.74
Total	**(EUR/t milled)**	**23.40**	**25.35**	**27.28**	**22.53**
1) Includes concentrate freight					

Table 18-5: Forecast LoM Unit Operating Costs (assuming SRK Adjusted Recoveries)

Parameter	Units	LoMp	2020	2021	2022	2023	2024	2025-2033	2034
Technical Drivers									
Ore	(kt)	141,089	9,579	10,720	10,000	10,000	10,000	89,940	850
Waste	(kt)	190,560	36,993	33,081	33,283	15,674	15,602	55,928	-
Total Mined	(kt)	331,650	46,572	43,801	43,283	25,674	25,602	145,868	850
Mill Feed	(kt)	140,243	9,072	10,002	10,000	10,000	10,000	90,000	1.169
Cu con	(kt)	1,592	96.3	95.9	87.6	96.9	108.1	1,093.2	14.4
Ni con	(kt)	2,576	124.7	150.3	183.2	169.8	150.5	1,769.6	28.1
Unit Costs									
Mining	(EUR/t $_{mined}$)	2.40	2.45	2.44	2.51	2.36	2.34	2.35	2.35
Processing	(EUR/t $_{milled}$)	6.68	6.11	6.90	6.16	7.00	8.19	6.58	5.56
G&A	(EUR k)	319,522	22,634	22,634	22,634	22,634	22,634	203,706	2,645
Freight									
Cu con	(EUR/t Cu $_{con}$)	36.74	36.74	36.74	36.74	36.74	36.74	36.74	36.74
Ni con	(EUR/t Ni $_{con}$)	45.74	45.74	45.74	45.74	45.74	45.74	45.74	45.74
Total	**(EUR/t $_{milled}$)**	**15.9**	**22.2**	**20.9**	**20.4**	**16.5**	**17.5**	**14.0**	**11.1**

18.5 Capital Expenditure

Capital expenditure as projected by the Company for the operation is shown as Table 18-6. Two adjustments have been made by SRK:

- inclusion of a capital allowance for construction (EUR 10 M) and annual lifts (EUR 4 M per annum) of a new TSF, which should cover the shortfall of the current facility (refer to Section 15.8.4); and
- inclusion of a closure cost provision (refer to Section 17.6).

All waste stripping costs are capitalised in full.

Table 18-6: LoM Capital Expenditure

Parameter	Units	LoMp	2020	2021	2022	2023	2024	2025-2033	2034
Capitalised Waste Stripping	(EUR M)	459.8	90.6	80.6	83.5	37.0	36.6	129.3	-
Other Sustaining	(EUR M)	23.9	17.3	3.8	0.7	2.1	-	-	-
Reinvestments	(EUR M)	10.8	6.5	3.1	0.3	0.2	0.2	0.7	-
De-bottleneck	(EUR M)	12.4	2.9	1.9	7.5	-	-	-	-
Expansion	(EUR M)	47.4	27.7	10.3	5.2	4.2	-	-	-
EHS	(EUR M)	55.7	6.2	8.7	5.4	4.9	4.1	26.4	-
Non-specified	(EUR M)	9.9	6.0	3.8	0.1	-	-	-	-
Additional TSF	(EUR M)	26.0	-	-	-	-	-	22.0	-
Closure	(EUR M)	169.0	-	-	-	-	-	-	169.0
Total Capital Expenditure	**(EUR M)**	**814.7**	**157.1**	**112.2**	**102.7**	**48.3**	**40.8**	**178.3**	**169.0**

18.6 Other Assumptions

For the purposes of the Mineral Reserve economic viability test, SRK has excluded the following (as deemed immaterial):

- any tax loss opening balances;
- working capital movements; and
- VAT movements.

Corporate income tax has been incorporated at a rate of 20%, with depreciation as supplied by the Company. Depreciation has not been adjusted for the additional capital expenditure as proposed by SRK, as due to the timing of this expenditure, its impact on tax payable is negligible.

18.7 Results

Results from the discounted cashflow model are presented in Table 18-7. SRK notes that total operating costs equal EUR 15.8/t, which compares well with the currently applied NSR operational cut-off of EUR 15/t for Mineral Reserves. The marginal NSR cut-off applied to the Mineral Resource of EUR 10.0/t matches the marginal operating cost (which excludes the cost of mining) of EUR 10.1/t.

The Net Present Values ("NPV") for the life of mine are presented at a range of discount rates in Table 18-8.

Table 18-7: LoM Summary

Parameter	Units	LoM
Moved Material		
Ore Mined	(kt)	141,089
Waste Mined	(kt)	190,560
Strip Ratio	(kt)	1.35
Mill Feed	(kt)	140,243
Grades		
Ni(S)	(%)	0.24
Cu	(%)	0.32
Au	(g/t)	0.10
Pt	(g/t)	0.21
Pd	(g/t)	0.14
Co(S)	(%)	0.01
Cu con	(kt)	1,592
Ni con	(kt)	2,576
Payable Metal		
Ni	(Mlb)	470.3
Cu	(Mlb)	842.6
Au	(koz)	154.5
Pt	(koz)	308.9
Pd	(koz)	200.3
Co	(Mlb)	6.8
TC/RC		
Cu con	(EUR M)	165
Ni con	(EUR M)	876
Total	(EUR M)	1,042
Net Revenue		
Cu con	(EUR M)	2,164
Ni con	(EUR M)	2,577
Total	**(EUR M)**	**4,741**
Operating Costs		
Mining	(EUR M)	795.0
Processing	(EUR M)	939.1
G&A	(EUR M)	319.5
Freight	(EUR M)	176.3
Capitalised Stripping	(EUR M)	(460)
Total	**(EUR M)**	**1,770**
EBITDA	(EUR M)	2,971
Corporate Income Tax	(EUR M)	329
Cashflow from Operations	**(EUR M)**	**2,643**
Total Capital Expenditure	(EUR M)	814.7
Net Free Cashflow	**(EUR M)**	**1,828**

Table 18-8: NPV at Range of Discount Rates

Discount Rate	NPV (EUR M)
0%	1,828
2%	1,544
4%	1,312
6%	1,123
8%	967
10%	**838**
12%	731
14%	640
16%	564

18.8 Sensitivity Analysis

SRK has tested the outcome of the economic assessment for some potential changes in inputs, especially with regards to changes in mining costs and potentially lower than planned nickel head grades.

During 2019, a 15% lower than expected nickel grade was processed. From the sensitivity analysis, the impact on NPV if the mill feed contained consistently 15% less NiS is presented in Table 18-9.

An NPV sensitivity to changes in mine operating costs, overall operating costs, commodity prices and capital expenditure, is presented in Figure 18-1, which confirms the LoMp to be robust.

Table 18-9: NPV Sensitivity to lower Ni head grade

Discount Rate	Base Case NPV (EUR M) (LoM Average 0.24% Ni)	Reduced Ni head grade NPV (EUR M) (LoM Average 0.20% Ni)
0%	1,828	1,590
2%	1,544	1,341
4%	1,312	1,137
6%	1,123	971
8%	967	834
10%	**838**	**720**
12%	731	625
14%	640	546
16%	564	479

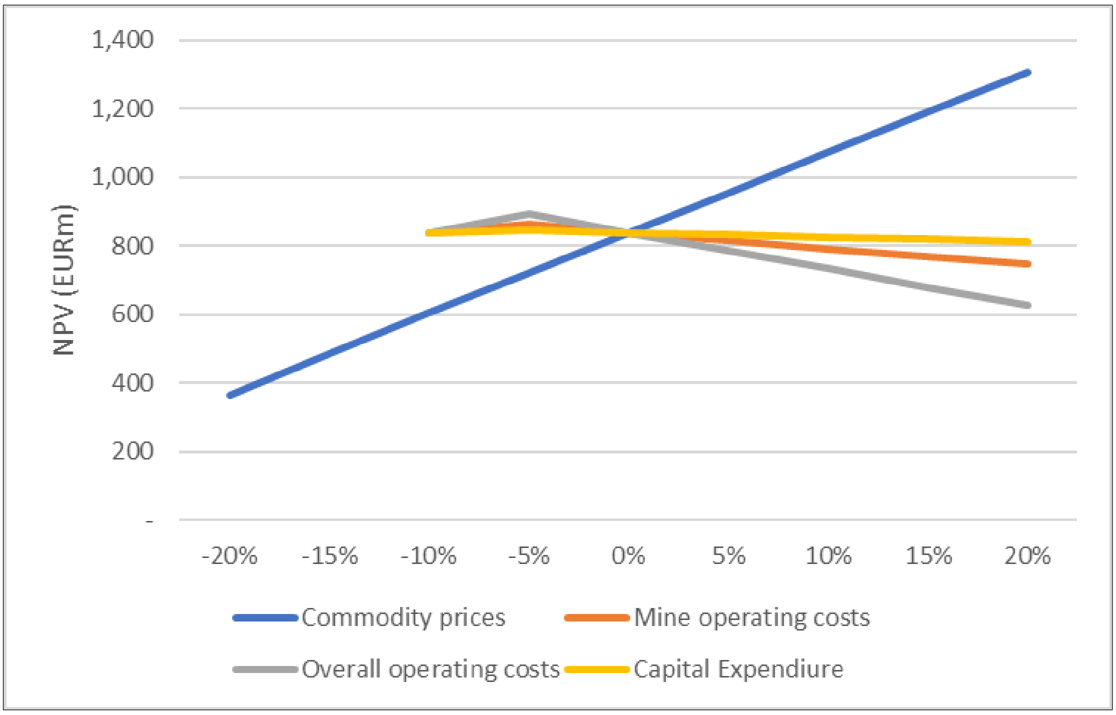


Figure 18-1: NPV Sensitivity to Changes in Inputs

18.9 SRK Comments on Economic Assessment

The Kevitsa Mineral Reserve LoMp returns a positive NPV, with sufficient margin to cover higher mining costs (more in line with those historically achieved) if these are deemed more appropriate. SRK recommends that the Company re-assesses its methodology to forecast longer term operating costs, using appropriate cost drivers. If mining costs are higher than those currently estimated by the Company (and possibly more in line with historically achieved unit costs, although SRK acknowledges the recent purchase of modern more efficient equipment and other initiatives to reduce mining cost), the Company may need to re-assess its currently applied NSR cut-off of EUR 15/t for Mineral Reserves. The marginal cut-off NSR of EUR 10/t as applied to the Mineral Resource seems appropriate.

19 MINERAL RESERVE ESTIMATE

19.1 Introduction

SRK has used the 2018 MRE block model and 2019 LoMp in addition to the actual mined topographic surface in order to report Mineral Reserves as of end-December 2019. More details on the mining operation providing input to the Mineral Reserve statement are provided in Section 14.

19.2 Mining factors

The mining factors used for the Mineral Reserve statement are based on the historical reconciliation between the in situ MRE block models (used in the LoMp) and the sampled mill feed. The historical comparison of the tonnage and metal content (tonnes) variance for the two main revenue drivers (Cu and Ni(S)) is summarised in Table 19-1.

19.2.1 Mining recovery

Historical Mining Recovery ("MREC") was calculated by comparing what is reported from the in-situ resource model to what arrives at the mill (mill feed / resource model). This was calculated for Cu and for NiS. The historical MREC is between 94 to 98% for the life of the operation. SRK noted that the most recent MREC for NiS in was 85% in 2019.

Given the yearly changes in metal content, Boliden has selected a 93% mining recovery which was applied as a global mining factor for the reserves. SRK considers this as an acceptable mining recovery but recommends that the poor reconciliation for NiS be addressed in the 2020 MRE and mining study that will follow. SRK ran a sensitivity on a 15% drop in NiS grade in the LoMp and is satisfied that the mine remains profitable all be it at a lower NPV.

19.2.2 Dilution

Historical Mining Dilution ("MDIL") was calculated by comparing ore tonnes reported from the in-situ resource model (x MREC) to what arrives at the mill. Two sets of MDIL is calculated based on the respective MREC from the two main revenue driving commodities. The historical MDIL is between 1.04 and 1.08, with the most recent 2019 value of 1.02.

Based on the historic reconciliation, Boliden has selected a MDIL of 1.07 which was applied as a global mining factor for reserves. SRK considers this as an appropriate global modifying factor.

Table 19-1: Historical comparison between resource model in-situ tonnes and mill feed*

Year	Mineral Resource - In Situ			Mill feed			Mining Factors **			
	Tonnes (kt)	Cu Content (t)	NiS Content (t)	Tonnes (kt)	Cu Content (t)	NiS Content (t)	MREC based on Cu Content (t:t)	MREC based on NiS Content (t:t)	MDIL based on Cu MREC (t:t)	MDIL based on NiS MREC (t:t)
2012	2,599	8,767	6,107	3,138	9,778	7,029	1.12	1.15	1.08	1.05
2013	6,092	16,800	13,653	6,301	17,792	14,088	1.06	1.03	0.98	1.00
2014	6,546	21,952	16,161	6,711	20,128	15,403	0.92	0.95	1.12	1.08
2015	6,641	20,015	14,695	6,665	19,181	13,196	0.96	0.90	1.05	1.12
2016	6,914	23,261	16,929	7,392	23,117	16,096	0.99	0.95	1.08	1.12
2017	8,079	34,947	21,711	7,911	33,100	19,872	0.95	0.92	1.03	1.07
2018	8,063	31,735	21,252	7,582	29,882	20,085	0.94	0.95	1.00	0.99
2019	7,481	21,792	16,441	**7,536**	22,190	14,043	1.02	0.85	0.99	1.18
Total	**52,416**	**179,268**	**126,949**	**53,237**	**175,169**	**119,812**	**0.98**	**0.94**	**1.04**	**1.08**

**Note: 2010 MRE model used for 2012-2015, 2016 MRE model used for 2016-2018 and 2018 MRE model used for 2019 reconciliation.*

*** MREC = (Mill content (t) / resource content (t)); and MDIL = (mill tonnes / (resource tonnes * MREC))*

19.3 Mineral Reserve Statement

The Mineral Reserve statement produced by SRK on behalf of Boliden in presented in Table 19-2 with notes explaining the reporting procedure.

Table 19-2: Mineral Reserve Statement effective of 31 December 2019*

Mineral Reserve Category	Tonnage (Mt)	Sulphide Nickel (%)	Total Copper (%)	Gold (g/t)	Platinum (g/t)	Palladium (g/t)	Sulphide Cobalt (%)
Proved	62	0.25	0.33	0.10	0.19	0.12	0.01
Probable	78	0.23	0.31	0.11	0.24	0.16	0.01
Prov+Prob	**140**	**0.24**	**0.32**	**0.10**	**0.21**	**0.14**	**0.01**

**In reporting the Mineral Reserve Statement, SRK notes the following:*

- *Mineral Reserve statement has an effective date of 31 December 2019.*
- *Competent Person for the declaration of Mineral Reserves is Mr Hanno Buys, an employee of SRK and professional member of The Institute of Materials, Minerals and Mining ("IOMMM") in the United Kingdom and registered as a Professional Mining Engineer ("Pr.Eng") with the Engineering Council of South-Africa.*
- *Reported Mineral Reserves are below the actual mined topography, dated 31 December 2019 and above the final stage 4 pit design "kev_stage4_28052019.dtm" (based on recommended pit slope angles), and are all contained within the pit shell used for the Mineral Resource Statement.*
- *Mineral Reserves are reported inclusive of mining modifying factors which are based historical reconciliation results, a 7% dilution and a 93% mining recovery are applied in the statement.*
- *Mineral Reserves are inclusive of a 0.153 Mt of ROM stockpile at 31 December 2019.*
- *A life of mine plan production schedule along with mining factors (mining recovery and dilution), processing factors (Recovery and Processing costs) and revenue factors (metal prices, selling costs) were incorporated in a financial model and economic analysis by which SRK determined the Mineral Reserves to be currently economic.*
- *Mineral Reserves are reported within the pit design at a Net Smelter Return ("NSR") operational cut-off of EUR 15/tonne ore.*
- *Tonnages are reported in metric units, grades in percent (%) or grams per tonne (g/t). Tonnages and grades are rounded appropriately.*
- *Mineral Reserves include 40 Mt of Ore to be mined at the last four years of the LoM (years 2030-2034) for which current TSF capacity is insufficient. These Mineral Reserves are dependent on Kevitsa identifying a suitable location, designing and obtaining relevant permits for additional TSF capacity within the next 10 years - prior to the tailings deposition.*
- *Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content*

19.4 Mineral Reserve Reconciliation 2018-2019

Except for depletion through mining and minor quantities reported in stockpiles, the only other change between 2018 and 2019 Mineral Reserve statements is with the geological model utilised. The previous Mineral Reserve statement of 31 December 2018 was reported using the 2016 MRE model, whereas the updated Mineral Reserve statement of 31 December 2019 was reported using the 2018 MRE model. The main change between the two geological models was the domaining and estimation methodology which resulted in higher density values (and thereby higher tonnage) and higher Ni(S) grades. This combination has resulted in an increase in material above the NSR cut-off which has remained the same. The Mineral Reserve reconciliation waterfall chart in RoM tonnes is presented in Figure 19-1, in nickel metal in Figure 19-2 and in copper metal in Figure 19-3.

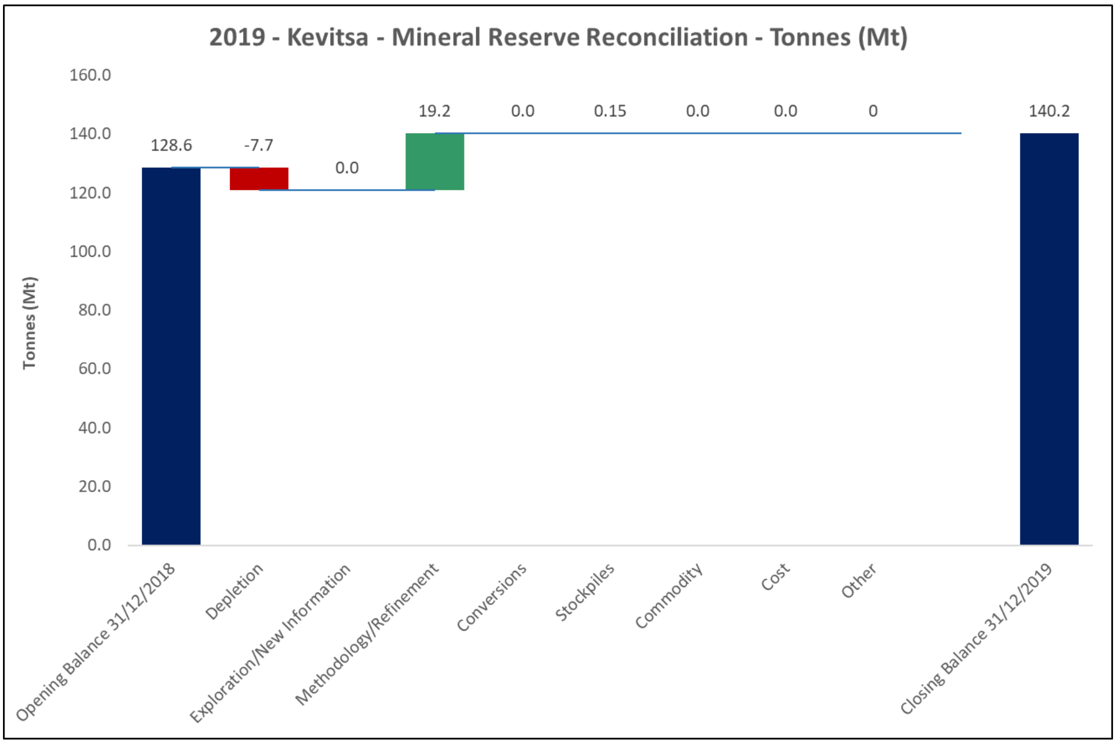


Figure 19-1: Mineral Reserve 2018-2019 tonnage waterfall chart

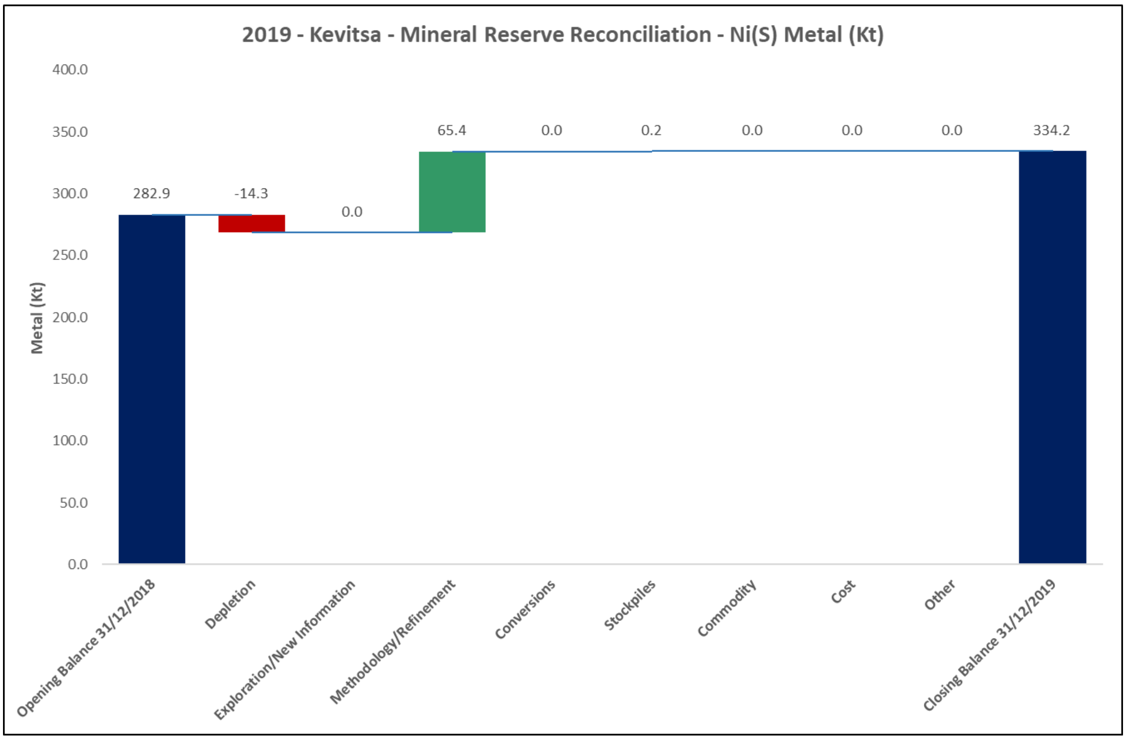


Figure 19-2: Mineral Reserve 2018-2019 sulphidic nickel metal waterfall chart

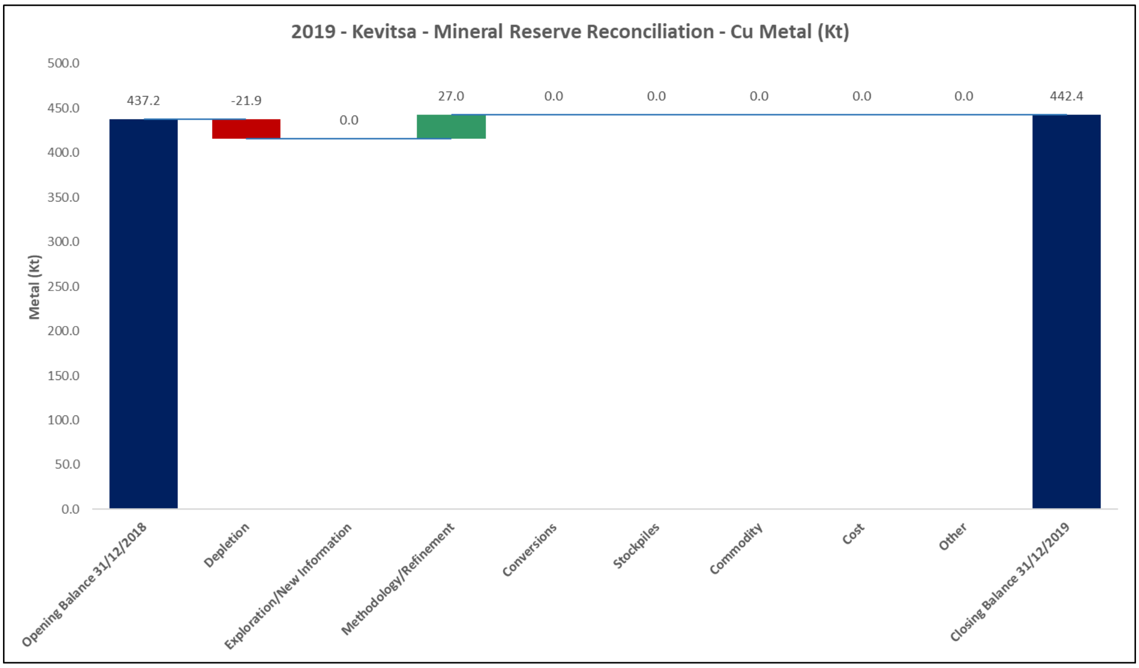


Figure 19-3: Mineral Reserve 2018-2019 copper metal waterfall chart

20 ADJACENT PROPERTIES

As well as Boliden's exploration claims, there are claims and claim reservations owned by Anglo American Plc adjacent to the Kevitsa Mining concession. This claim covers their nickel-copper-cobalt-PGE-gold Sakatti exploration project, which reported 44 Mt of Indicated and 4 Mt of Inferred Mineral Resources in 2016. Exploration and other studies continue on the claim in 2020.

As at the end of December 2019, there were no mining concessions granted adjacent to the Kevitsa mine.

21 CONCLUSIONS, RISKS AND RECOMMENDATIONS

This section summarises the potential risks as it relates to the Mineral Resources and Mineral Reserves and provide recommendations based on the risks.

21.1 Geology, Exploration and Mineral Resources

The geology is very well-understood with a well-documented history of exploration in the region. SRK's review of the 2018 MRE which was used to report the 2019 Mineral Resource statement highlighted no fatal flaws or significant issues requiring adjustments prior to reporting.

A number of minor suggestions are made by SRK for future Mineral Resource estimates:

- review composite length used for grade estimation;
- review capping strategy;
- review block model block size;
- review overall estimation search strategy;
- review use of hard / soft boundaries across faults and other controlling features; and
- review NSR calculation is appropriate using up-to-date costs and prices.

21.2 Mining

The following key conclusions and recommendations are provided relating to the mining impacts on modifying factors for Mineral Resource and Mineral Reserve reporting:

- **Marginal Ore in LoMp:** for the 2020 budget plan, an NSR formula relating to a 15% higher price Ni price was used to define ore. Raising the prices meant that more marginal ore was included in the LoMp to fill up the processing plant for 2020.
  - SRK recommends that in such a case, rather than inflating the prices, the mine should lower the cut-off NSR price to a marginal cut-off (EUR 10 / t).
  - SRK also recommends that all marginal ore be stockpiled seperately, to be used strategically in instances where ore shortfalls may occur.
- **Equipment optimisation:** waste stripping requires high production tonnages in 2020 – 2022 (28.1 to 42.5 Mtpa). Kevitsa has achieved 42 Mtpa in 2017 (with the aid of a mining contractor doing pre-stripping), new equipment has been acquired, and a fleet management system is currently being implemented. Kevitsa will become a fully owner operated when the mining contractor finishes the pre-stripping of stage 4 in 2020. Recent primary equipment purchases include 2 x new Atlas Copco Pit Viper 271 Electric rigs, 2 x new Cat 6060 Diesel Face Shovels and 17 x Komatsu 830E-5 Diesel Electric 250t trucks (trolley assist ready).
  - SRK recommends that future LoMp scenarios include a detailed haulage analysis (instead of first principle calculations) to ensure that primary equipment totals are sufficient and optimised.
- **Production profile:** significant decrease in the production profile in 2023 raises the question whether the LoMp plan is optimal for an owner operated equipment fleet. SRK understands that excess capacity heavy mining equipment at Kevitsa could be transferred to Boliden's Aitik mine in Sweden post 2023. Equally, the mine is currently investigating

the potential for an additional Stage 5 which could utilise the additional equipment capacity. Although not optimal for production, SRK accepts the production profile as achievable.

21.3 Geotechnical Engineering

The various geotechnical engineering aspects that contribute to the slope design, pit design and optimisation have been reviewed by SRK to contribute to the Mineral Resource and Mineral Reserve reviews to understand any fatal flaws.

- **Data collection:** in place and suitable; however, structural mapping could be improved in order to provide localised expectation of jointing that will lead to crest loss and bench scale instability. There is a suitable model of the major geological structures that influence the pit slopes up to Stage 4.
- **Slope stability analysis:** has been performed by various external consultants using industry standard practice. SRK could not determine if the stability analysis has included the influence of major structures in the Stage 5 design. Furthermore, the potential for step-path failure at inter-ramp scale has not been suitably addressed. In deep pits with high intact rock strength the role of jointing controls stability and this structural fabric is to be suitably modelled in terms of persistence length as well as orientation. SRK recommends that the existing slopes are mapped for additional data to enhance the 2017 slope stability analysis.
- **Wall stability:** the hydrogeological influence to slope stability is not quantified and suitably included in analysis. Stage 4 design east wall stability will likely be compromised by the NE-FLT-2 structure and Boliden has recognised that a suitable technical study is to be conducted on this slope in the near term in order to inform the medium term mine plan.
- **Pit optimisation:** process is clear, and the slope angle inputs follow the geotechnical guidance parameters that are at a high resolution to provide height based inter-ramp angle inputs into the Whittle optimisation.
- **Slope hazard management:** is suitable, with failure modes well-understood and the pit, at this stage of development, has been constructed well with the minor-moderate bench scale instability risk handled suitably.
- **Real-time monitoring:** single slope stability radar is the only instrument active at this stage; additional equipment is to be budgeted for to improve risk management controls.

Overall, SRK deems that there is low risk to achieving the final mine pit limits in the Stage 3 design. The risk to achieve the Stage 4 and Stage 5 designs (for future investigations) requires a higher level of analysis to quantify the probability of inter-ramp failure. This is to include a better understanding of jointing patterns and quantify the hydrogeological influence.

21.4 Water Management

The various water management aspects that contribute to the slope design, pit design, and optimisation have been reviewed by SRK to contribute to the Mineral Resource and Mineral Reserve reviews to understand any fatal flaws.

No water management flaws have been highlighted in the review affecting the reporting of Mineral Resource and Mineral Reserve statements.

21.5 Processing

The following key conclusions and recommendations are provided relating to the processing/metallurgy impacts on modifying factors for Mineral Resource and Mineral Reserve reporting:

- **Cu feed grade:** key issue identified is that the lower Cu feed (head) grades are currently resulting in lower Cu recoveries to Cu concentrate, which may continue to impact the quality of the product in the LoMp.
- **Plant equipment:** considered industry standard and high quality.
- **Plant throughput:** existing plant is proven up to 8 Mtpa.
- **Historical operating figures:** show a fall in Cu and Ni feed grades and expansion is required to maintain copper and Ni concentrate production levels.
- **Cu and Ni concentrates:** are clean (no contaminants).

Overall, no fatal flaws in reporting of Mineral Resources and Mineral Reserves were identified in relation to processing and metallurgy.

21.6 Tailings

The following key conclusions and recommendations are provided relating to the tailings storage facility impacts on modifying factors for Mineral Resource and Mineral Reserve reporting:

- **Storage capacity:** there is a significant shortfall in storage capacity for tailings generated as part of the Stage 4 mining plan. Should lower densities or steeper beach angles be realised, this could reduce the forecast storage capacity yet further. Optimising the deposition planning for the current TSF may partially alleviate this issue; however, it is likely that a new TSF will be required to store the shortfall identified between 2030 and 2034.
- **Rate of rise:** owing to the increased tailings production rate (9.5 Mpta) following plant expansion, the rate of rise of the facility will increase. Additional analysis is required to ensure that each embankment raise can be installed as designed, on potential contractive tailings materials, which are prone to static liquefaction. There is a risk that the scheduling of embankment raises will be impacted, as there is currently a very short period for construction each year (approximately 7 months).
- **Reduced pond capacity:** current water balance modelling indicates that approximately 2.5 Mm^3 of storage capacity is required in the TSF (Golder, 2018). Volumetric modelling by Golder indicates that during the latter stages of construction, the maximum pond volume (i.e. below the 1.5 m minimum freeboard marker) will be limited to 1.5 Mm^3. This could further impact storage capacity during the latter stages of the operational life of the facility.
- **Seepage through base of the TSF:** increased seepage flow rates through the basal layers of the TSF (where there is no basal liner and compacted peat only) have the potential to impact groundwater quality in proximity to the TSF. Updates of the water balance and predictive modelling have been recommended (by Golder) to address this risk. SRK understands that these studies are ongoing. Remediation measures may have to be designed if predictive models indicate that environmental discharge limits are

breached in the future (perhaps as a result of increased production rates); however, this is not currently the case. Boliden reports that pilot testing of a new active groundwater extraction/treatment system was undertaken during 2019. The design of the water treatment system has yet to be finalised.

- **Closure:** Boliden has prepared a conceptual level closure cost estimate, which is based upon typical unit rates per area (Boliden, 2019). Boliden has identified a significant lack of borrow materials (till for engineered cover system) during preparation of the closure plan. An additional 2.0 Mm^3 of material may have to be imported from distal sources, which is likely to significantly increase the closure provision. Alternative closure designs may need to be scoped out to partially mitigate this risk.
- **Permitting:** The risks associated with permitting delays associated with a new TSF are considered to be significant. SRK has not received designs covering future TSF and based on experience in the region, note the long lead times and potential project delays associated with permitting of new facilities.

The following opportunities for improvement have been highlighted by SRK:

- There is opportunity to minimise the rate of rise of the TSF through detailed deposition modelling and construction scheduling for future raises. Targeted deposition will reduce the risks associated with: 1) ice formation in the perimeter beach above water zones; 2) formation of low-density layers due to sub-aqueous deposition; and 3) formation of loose, potentially liquefiable layers beneath embankment raises.
- An update of the water balance would allow opportunities to reduce water storage on the TSF to be identified, thus increasing storage capacity in the facility. Thickening of the tailings for instance may reduce the amount of supernatant to be stored on the facility, which will be limited during the final stages of operations.
- Additional volumetric modelling could be considered during updates to the closure plan, to identify opportunities to minimise the large volumes of imported fill required to create a convex shedding upper surface, prior to installation of the engineered cover system.

21.7 Environment, Social and Governance

The various ESG aspects that contribute to mine plan and financial have been reviewed by SRK to contribute to the Mineral Resource and Mineral Reserve reviews.

SRK considers that the ESG procedures in place are fit for purpose with no fatal flaws identified preventing the reporting of Mineral Resource and Mineral Reserve statements.

21.8 Economics

SRK recommends that the Company re-assesses its methodology to forecast longer term operating costs, using appropriate cost drivers. If mining costs are higher than those currently estimated by the Company (and possibly more in line with historically achieved unit costs, although SRK acknowledges the recent purchase of modern more efficient equipment and other initiatives to reduce mining cost), the Company may need to re-assess its currently applied NSR cut-off of EUR 15/t for Mineral Reserves.

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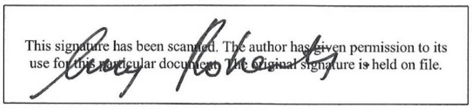
For and on behalf of SRK Consulting (Finland) Oy



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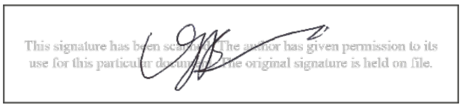

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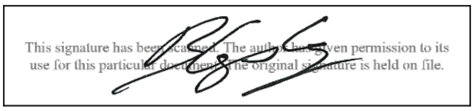

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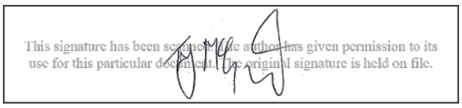

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Glossary

PERC Standard	Pan European Reserves and Resources Reporting Committee (PERC) Standard for Reporting of Exploration Results, Mineral Resources and Mineral Reserves (latest edition 2017)

Abbreviations

Au	Gold
Co(S)	Cobalt (sulphidic)
Cu	Copper (total)
Cu(S)	Copper (sulphidic)
Fe	Iron
Ni	Nickel (total)
Ni(S)	Nickel (sulphidic)
Pd	Palladium
Pt	Platinum
S	Sulphur

Units

Mt	Million metric tonnes (based on a dry in situ bulk density unless specified)
Kt	Thousand metric tonnes (based on a dry in situ bulk density unless specified)
t	Metric tonnes (based on a dry in situ bulk density unless specified)
Mtpa	Million metric tonnes per year/annum
ktpa	Thousand metric tonnes per year/annum
tpa	Metric tonnes per year/annum
tpd	Metric tonnes per day
%	Percentage
m	Metres
cm	Centimetres
µm	Micrometres
km	Kilometres
km^2	Kilometres squared
Mm^3	Million cubic metres
mbgl	Metres below ground level
kg	Kilograms
g	Grams
µg/l	Micro-grams per litre
mg/L	Miligrammes per litre
m/s	Metres per second
EUR	Euros (EUR)
dB	Decibels
Mpa	Mega pascals
GWh	Gigawatt hours
ha	Hectares
masl	Metres above sea level
mbgl	Metres below ground level

APPENDIX

A COMPETENT PERSONS' CONSENT STATEMENTS

Competent Person's Consent Statement – Mineral Resources

Pursuant to the requirements of paragraph 3.2 of the PERC Standard

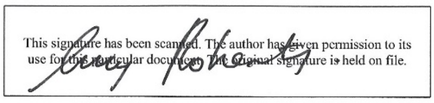
Technical Report for the Kevitsa Cu-Ni-PGE Mine, Finland for Boliden Kevitsa Mining Oy ("Boliden"). Effective Date of 31 December 2019.

I, Dr Lucy Roberts, confirm that:

- I have read and understood the requirements of the PERC Standard for Reporting of Exploration Results, Mineral Resources and Mineral Reserves ("PERC Standard").
- I am a Competent Person for Mineral Resources as defined by the PERC Standard, having at least five years' relevant experience in relation to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a professional Member or otherwise registered professional, with the Australasian Institute of Mining and Metallurgy ("AusIMM"), being an institution which is included in the current list of recognised professional organisations or a member institution of the European Federation of Geologists, or an organisation elsewhere included in the RPO list in Appendix 5 of the PERC Standard or as subsequently updated
- I have reviewed the Report to which this Consent Statement applies.
- I am a full-time employee of SRK Consulting (UK) Ltd ("SRK") and have been engaged by Boliden to prepare the Report for Kevitsa for the period ended 31 December 2019.
- SRK an independent consulting company and there is no other direct financial relationship between myself and the Company; however, SRK is being paid a pre-agreed fee (not linked with the outcome of the report) by Boliden to complete the report**.**

I verify that the Report is based on, and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Mineral Resources.

I consent to the release of the Report and this Consent Statement by the directors of Boliden.

Signature of Competent Person:	Date: 07.04.2020
	
Professional Membership: Member and Chartered Professional with the Australasian Institute of Mining and Metallurgy (MAusIMM(CP))	Membership Number: 211381

Additional Deposits covered by the Report for which the Competent Person signing this form is accepting responsibility: **None**

Additional Reports related to the deposit for which the Competent Person signing this form is accepting responsibility: **None**

Competent Person’s Consent Statement – Mineral Reserves

Pursuant to the requirements of paragraph 3.2 of the PERC Standard

Technical Report for the Kevitsa Cu-Ni-PGE Mine, Finland for Boliden Kevitsa Mining Oy (“Boliden”). Effective Date of 31 December 2019.

I, Mr Hanno Buys, confirm that:

- I have read and understood the requirements of the PERC Standard for Reporting of Exploration Results, Mineral Resources and Mineral Reserves (“PERC Standard”).
- I am a Competent Person for Mineral Reserves as defined by the PERC Standard, having at least five years’ relevant experience in relation to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a professional Member or otherwise registered professional, with the Institute of Materials, Mining & Metallurgy (“IMMM”), being an institution which is included in the current list of recognised professional organisations or a member institution of the European Federation of Geologists, or an organisation elsewhere included in the RPO list in Appendix 5 of the PERC Standard or as subsequently updated
- I have reviewed the Report to which this Consent Statement applies.
- I am a full-time employee of SRK Consulting (UK) Ltd (“SRK”) and have been engaged by Boliden to prepare the Report for Kevitsa for the period ended 31 December 2019.
- SRK an independent consulting company and there is no other direct financial relationship between myself and the Company; however, SRK is being paid a pre-agreed fee (not linked with the outcome of the report) by Boliden to complete the report**.**

I verify that the Report is based on, and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Mineral Reserves.

I consent to the release of the Report and this Consent Statement by the directors of Boliden**.**

Signature of Competent Person:	Date: 07.04.2020
	
Professional Membership: Professional Member of the Institute of Materials, Mining & Metallurgy (MIMMM)	Membership Number: 483399

Additional Deposits covered by the Report for which the Competent Person signing this form is accepting responsibility: **None**

Additional Reports related to the deposit for which the Competent Person signing this form is accepting responsibility: **None**