**FIRST QUANTUM**
MINERALS LTD.

# Kevitsa Nickel Copper Mine, Lapland, Finland

## NI 43-101 Technical Report

*30th March 2016*

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## TABLE OF CONTENTS

# TABLE OF FIGURES

# TABLE OF TABLES

# ITEM 1    **SUMMARY**

## 1.1    **Project Background**

This Technical Report on the Kevitsa Mine (the Mine) has been prepared by Mr Tony Cameron of Cameron Mining Consulting Ltd (CMC), and Messrs David Gray and Andrew Briggs of First Quantum Minerals Ltd (the issuer or FQM or the Company) as Qualified Persons (QP's) on behalf of FQM. The QP's have prepared this Technical Report to document updated Mineral Resource and Mineral Reserve estimates, economic evaluations and supporting ancillary documentation associated with FQM's Kevitsa Operation in Lapland, Finland.

This report and the resource/reserve estimates discussed therein represent the culmination of results of a series of reviews and updates undertaken by FQM (the issuer) since commissioning the project in August 2012.

This report supersedes the NI 43-101 Technical Report issued by FQM and dated 12 May 2011. Updates to this previous Technical Report (TR) are noted throughout.

## 1.2    **Property Location, Ownership and Approvals**

The Kevitsa Ni-Cu-PGE deposit is located in Finnish Lapland, 140 km north of the Arctic Circle.

Valid Mining Concession No. 7140 named Kevitsa covers the current mining, processing, tailings dam and waste dump areas. An application to extend Mining Concession 7140 to provide more space for the waste dumps was submitted to the authority in 2015.

In accordance with Finnish regulations, KMO owns the land within the mining concession. The land was previously under the control of the Finnish State Forestry Commission who are the principal landowner in the region surrounding the property.

The original 5 Mtpa environmental permit was replaced by the 10Mtpa Environmental Permit No. 79/2014/1 issued on 11th July 2014.

## 1.3    **Geology and Mineralisation**

The Kevitsa igneous complex is located within the Central Lapland Greenstone Belt (CLGB), which is a large domain mainly consisting of Paleoproterozoic, volcano-sedimentary, rocks in the Precambrian Fennoscandian Shield. The Kevitsa igneous complex is a layered intrusion comprising ultramafic to mafic igneous rocks with an estimated deposit age of approximately 2058 Ma. The intrusion has an arcuate shape at surface and extends to over 1.5 km in depth, where the shape becomes increasingly complex.

The known economic mineralisation is located in the centre of the main ultramafic unit of the Kevitsa layered intrusion. Olivine pyroxenite and its derivatives, host the economic mineralisation which predominantly occurs as disseminated Cu and Ni sulphides. The ore body consists of several irregular zones cut by faults and shear zones locally offsetting the mineralization. Two main ore types are currently distinguished; regular Ni-Cu ore which consists of a chalcopyrite-pentlandite assemblage and the Ni-platinum group element (PGE) ore with a predominantly pentlandite-millerite assemblage. PGE's occur mainly as Bi-tellurides and sperrylite.

## 1.4    Exploration Status

Exploration work has been active since the late 1970s. However, since the previous TR, apart from sampling associated with ongoing drilling (detailed in Item 10), there were no additional samples or exploration work undertaken by the issuer at Kevitsa or its immediate surrounds.

Exploration work, completed prior to June 2011, focused predominantly on geophysical methods including magnetic, radiometric, gravity, electromagnetic, electrical, downhole logging, seismic and magnetotelluric (Titan-24) surveys.

## 1.5    Operations Status

The Kevitsa operation has been mining and processing nickel-copper-PGE sulphide ore since achieving commercial production in August 2012. It is now in its fourth year of operation and Stage 2 (out of a total of 4 stages) of the open pit is currently being mined. Based on current estimates, mining and processing are due to be completed in 2033.

## 1.6    Mineral Resource Estimate

The Kevitsa Mineral Resource estimate (Table 1-1) was updated in January 2016 using the available updated drillhole database, which includes all diamond drill hole and reverse circulation sample results together with an updated interpretation of the geological model relevant to the spatial distribution of nickel, copper, PGE and gold mineralisation. Interpolation parameters were based upon the geology, styles of mineralisation, drill hole spacing and geostatistical analysis of the data. Mineral Resource estimates were classified according to geological continuity, QAQC, density data, drillhole grid spacing, grade continuity and confidence in the panel grade estimate and have been reported in accordance with the guidelines of the Australasian JORC Code (JORC, 2012), which in turn complies with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014).

**Table 1-1        January 2016 Kevitsa Mineral Resource statement depleted of mined material as at 31 December 2015 and using a 0.22% NiSEq cut-off grade.**

| Category | Density | Tonnes (Mt) | Ni (%) | NiS (%) | Cu (%) | Au (ppm) | Pt (ppm) | Pd (ppm) |
|---|---|---|---|---|---|---|---|---|
| Measured | 3.17 | 108.7 | 0.25 | 0.23 | 0.34 | 0.10 | 0.21 | 0.13 |
| Indicated | 3.16 | 167.0 | 0.26 | 0.24 | 0.34 | 0.09 | 0.16 | 0.11 |
| **Total measured and indicated** | **3.16** | **275.7** | **0.26** | **0.23** | **0.34** | **0.09** | **0.18** | **0.12** |
| Inferred | 3.16 | 57.1 | 0.22 | 0.20 | 0.31 | 0.06 | 0.12 | 0.07 |

## 1.7    Mineral Reserve Estimate

As at the end of December 2015, Kevitsa has 155 Mt of Mineral Reserve remaining. This estimate uses the categories of Mineral Reserve estimates permitted under the CIM Guidelines, 2014, within the designed final pits based on Measured and Indicated Mineral Resources.

**Table 1-2      Kevitsa Mineral Reserve Estimate depleted as at end December 2015 using a 0.22 % NiSEq cutoff grade.**

| Kevitsa Ore Reserve Estimate December 31 2015 | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | Tonnes (Mt) | Ni % | Ni(S) % | Cu % | Cu(S) % | Au g/t | Pd g/t | Pt g/t |
| Pit Only | | | | | | | | |
| Proven Reserve | 88.1 | 0.25 | 0.23 | 0.36 | 0.34 | 0.11 | 0.14 | 0.22 |
| Probable Reserve | 66.3 | 0.28 | 0.25 | 0.36 | 0.34 | 0.11 | 0.15 | 0.22 |
| Total | 154.4 | 0.27 | 0.24 | 0.36 | 0.34 | 0.11 | 0.15 | 0.22 |
| Stockpile | | | | | | | | |
| Proven Reserve | 0.1 | 0.23 | 0.19 | 0.25 | 0.24 | 0.09 | 0.16 | 0.21 |
| Probable Reserve | 0.0 | - | - | - | - | - | - | - |
| Total | 0.1 | 0.23 | 0.19 | 0.25 | 0.24 | 0.09 | 0.16 | 0.21 |
| Total Pit + Stockpile | | | | | | | | |
| Proven Reserve | 88.2 | 0.25 | 0.23 | 0.36 | 0.34 | 0.11 | 0.14 | 0.22 |
| Probable Reserve | 66.3 | 0.28 | 0.25 | 0.36 | 0.34 | 0.11 | 0.15 | 0.22 |
| Total | 154.5 | 0.27 | 0.24 | 0.36 | 0.34 | 0.11 | 0.15 | 0.22 |

## 1.8      Production Schedule

As at the end of December 2015, the Mine has 18 years of production remaining using current open pit reserves and stockpiles with a process feed rate of 7.6 Mt per year for 2016/17 and 9 Mt per year from 2018 onwards.

**Figure 1-2      Total Mining Volume (Mtpa) - Kevitsa Life of Mine Schedule depleted to end of December 2015.**
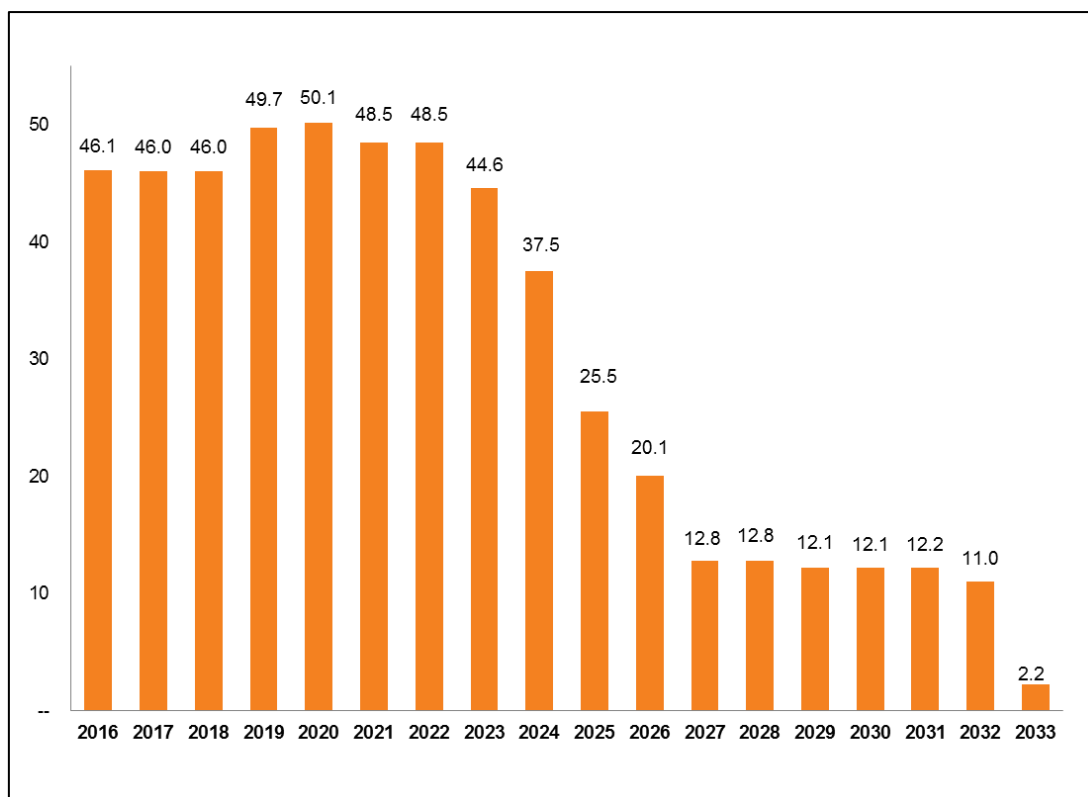
**Table 1-3        Kevitsa Life of Mine Schedule**

| | | UNITS | | TOTAL | 2016 | 2017 | 2018 | 2019 | 2020 | 2021 - 2025 | 2026-2030 | 2030-2033 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **MINING** | | | | | | | | | | | | |
| | Waste | Tonnes | Mt | **376.7** | 36.3 | 38.3 | 36.3 | 41.7 | 40.4 | 151.2 | 24.8 | 7.6 |
| | Ore | Tonnes | Mt | **154.37** | 9.8 | 7.7 | 9.7 | 8.1 | 9.7 | 46.5 | 45.1 | 17.8 |
| | Total Mined | Tonnes | Mt | **531.0** | 46.1 | 46.0 | 46.0 | 49.7 | 50.1 | 197.7 | 70.0 | 25.4 |
| | Strip ratio | | t:t | **2.4** | 3.7 | 5.0 | 3.7 | 5.2 | 4.2 | 3.2 | 0.5 | 0.4 |
| **PROCESSING SUMMARY** | | | | | | | | | | | | |
| | Feed to Plant | Tonnes | Mt | **154.5** | 7.6 | 7.6 | 9.0 | 9.0 | 9.0 | 45.0 | 45.0 | 22.30 |
| | | Ni | % | **0.24%** | 0.25% | 0.25% | 0.24% | 0.23% | 0.23% | 0.23% | 0.25% | 0.27% |
| | | Cu | % | **0.35%** | 0.32% | 0.34% | 0.34% | 0.31% | 0.33% | 0.32% | 0.38% | 0.38% |
| | | Pt | g/t | **0.22** | 0.31 | 0.30 | 0.30 | 0.24 | 0.23 | 0.24 | 0.18 | 0.14 |
| | | Pd | g/t | **0.15** | 0.21 | 0.19 | 0.19 | 0.15 | 0.14 | 0.17 | 0.12 | 0.09 |
| | | Au | g/t | **0.11** | 0.14 | 0.14 | 0.13 | 0.10 | 0.11 | 0.11 | 0.10 | 0.09 |
| | | | | | | | | | | | | |
| | Plant Recovery | Ni | % | **66.1%** | 66.4% | 66.4% | 66.3% | 65.9% | 65.9% | 65.9% | 66.3% | 66.0% |
| | | Cu | % | **89.9%** | 89.8% | 89.9% | 90.0% | 89.7% | 89.8% | 89.7% | 90.3% | 89.8% |
| | | Pt | % | **53.0%** | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% |
| | | Pd | % | **61.0%** | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% |
| | | Au | % | **52.0%** | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% |
| | Metal in Concentrate | | | | | | | | | | | | |
| | | Ni | Kt | **247.3** | 12.6 | 12.6 | 14.6 | 13.4 | 13.5 | 67.4 | 73.5 | 39.7 |
| | | Cu | Kt | **479.6** | 21.8 | 23.0 | 27.8 | 25.1 | 26.3 | 127.7 | 152.5 | 75.4 |
| | | Pt | koz | **574.1** | 40.1 | 38.5 | 45.6 | 36.6 | 34.6 | 185.4 | 140.5 | 52.8 |
| | | Pd | koz | **445.8** | 30.7 | 28.3 | 33.5 | 27.2 | 24.1 | 152.8 | 109.1 | 40.2 |
| | | Au | koz | **278.7** | 17.8 | 17.2 | 20.2 | 14.5 | 16.5 | 83.0 | 76.6 | 32.8 |

## 1.9        Processing

The Kevitsa Nickel Copper mine is mature and is capable of successfully operating at design capacity. KMO geological, mining, and processing personnel have developed a good understanding of the mineralogical and blending requirements for plant feed as well as how to operate the plant efficiently. All concentrate produced in 2015 was within contract specifications.

### 1.9.1   Process Description

The circuit is currently capable of treating 9 Mtpa of ore, equivalent to 1,125tph at 91.3% (8,000hrs per year) availability.  A block flowsheet summarizing the process is provided in Item 17.

ROM ore from the open pit is crushed in a gyratory crusher to 80% minus 150mm, followed by screening on a double deck vibrating screen for pebble generation.  The screen oversize and undersize is recombined and conveyed to the mill feed stockpile.

Mid-size material is used as grinding media in the pebble mill, and is conveyed to a storage bin. Excess mid-size material, overflowing the pebble bin, is cone crushed in secondary and tertiary crushers and returned to the stockpile feed conveyor, rejoining the coarse and fine material from the screening plant.

Material from the stockpile, comprising coarse lumps and fines is conveyed to two 7MW AG mills operating in parallel.  The mills operate partially in closed circuit with hydrocylones, and partially in open circuit with a portion of the discharge slurry being pumped from each mill to the pebble mill. The 14MW pebble milling circuit operates in closed circuit with a dedicated set of hydrocyclones. Pebbles are fed from the pebble bin to the pebble mill as grinding media.

Overflow from the cyclones on all three mills at a size of 80% passing 75 microns gravitates to the copper flotation circuit, and underflow is returned to the AG or pebble mills.

Cyclone overflow slurry is collected in a surge tank, and pumped to copper flotation. Copper flotation tailings are pumped to nickel flotation, and nickel flotation tailings to the sulphide flotation circuit.

Copper rougher and scavenger flotation concentrates are subject to regrind in a high intensity grinding (HIG) mill to assist in separation of copper and nickel minerals and cleaned in 4 stages of cleaners operating in counter current mode (concentrate moving from 1$^{st}$ to 4$^{th}$ cleaners, and tailings from each bank moving upstream. Fourth cleaner concentrate is the final concentrate, whilst first cleaner tails are pumped to nickel flotation.

The nickel cleaning circuit is similar, except that there are 5 cleaner stages, and the regrind mill is employed on the 2$^{nd}$ cleaner cons.

The sulphide flotation circuit comprises rougher flotation and a single cleaning stage. This circuit is employed to remove residual sulphides from the bulk of the flotation tailings so that they will be none acid generating when pumped to the tailings storage facility. Sulphide concentrates are stored in a lined storage pond.

Copper and nickel concentrates are thickened and filtered prior to transportation to off-site smelters.

Water supply to the circuit comprises raw water from the Vajukoski Pond, and process water, which is a combination of run-off water from the site, pit dewatering water, and decant water from the tailings pond.

### 1.9.2  Process Improvements

Since start up there have been several initiatives to improve plant efficiencies, and to increase throughput. These modifications are detailed in Item 13 and summarised as follows:

Geo-metallurgical program:

- Introduction of an XRD program for improved understanding of the mineralogy, which together with the Geological department has identified the main gangue minerals that control mill throughput and flotation efficiencies.
- Improved blasting techniques, leading to a reduction in coarse material produced in the mine.

Comminution:

Changes in the comminution circuit include the following:

- Addition of a second cone crusher to improve the crushing of pebbles.
- Installation of a screen above the pebble bin to generate pebbles of different sizes, so that one crusher could act as a tertiary crusher with a smaller closed side setting.
- Modification of the primary screen to enable a larger aperture to be used on the top deck – reducing the coarse oversize quantity to the stockpile (and AG mills) and increasing the amount of material for secondary crushing.

- Installation of a second 7MW drive in the pebble mill to enable it to be used as a ball mill at some future date.
- Installation of ball charging facilities on all mills.

Flotation:

- Installation of two column cells – one for copper duty to assist in the separation of copper and nickel sulphide minerals, and one on the nickel circuit to assist in the reduction of talk in the concentrates, and improve Ni concentrate grades.
- Installation of a HIG (high intensity grinding) mill to assist in the liberation of nickel sulphide minerals from copper concentrates.
- Addition of a 500m3 cell on the copper circuit – this was for testwork purposes, but now supports the higher processing rates.
- Improved reagent dosing capabilities – particularly for the nickel flotation circuit.

Concentrate Handling:

The nickel concentrate filter was increased in size to handle increased concentrate generation, or lower grade concentrates.

## 1.10    Project Infrastructure

All infrastructure required by the Mine is in place including sealed roads, power lines and substations, process plant, site offices, workshops, tailings dam, and waste storage facilities.

## 1.11    Environmental Status

The 10 Mtpa environmental permit was issued on 11th July 2014. However the permit appeals process is still ongoing due to various minor anticipated appeals and KMO expects this process to be successfully completed in 2017.

The Kevitsa mine is fully compliant with all final effluent discharge limits and air quality standards. There have been some challenges with internal compliance points with regards to nickel in drainage from the mine dumps prior to treatment in the neutralisation facility. The site water discharge pumping capability was upgraded in 2015 in line with the increased effluent discharge volumes approved in the new 10Mtpa permit. Work has started on enhancing mine drainage treatment capacity for the expanded mine. Kevitsa has applied for postponement of the deadline for increased treatment capacity from January 1st 2016 to April 30th 2017. Kevitsa's view is that postponing compliance of the stipulation does not present a significant risk of environment deterioration. The application was sent to the permitting authority at the end September 2015 and the application was published 4th February 2016. No decision has been made (likely earliest end of 2016).

Kevitsa's Asset Retirement Obligation was $21,239,062 on 31st December 2015. To-date KMOY has deposited Euros 17,068,848 with the Lapland Environmental Authority as a guarantee for restoration of the mine site. Environmental spend in 2015 was $1,757,915.

## 1.12    Capital and Operating Cost Estimates

The Kevitsa mine and process plant have been running for almost 4 years and were expanded to the current 9 Mtpa capacity in 2014/15. Capital expenditure required to be spent is for sustaining the

Mine, installation of trolley assist, additional mining equipment, waste stripping, and closure/reclamation. Operating costs are well known and the drivers understood. Even though the Mine is profitable in the current low commodity price conditions, FQM is currently reviewing all operations with an aim to reducing costs and improving profitability.

## 1.13    Economic Evaluation

The economic evaluation of the Kevitsa mine shows that it is profitable at the December 2015 consensus metal prices, which have been utilised to determine a cutoff grade of 0.22% NiSEq. The mining schedules run to date indicate that there is sufficient higher grade ore available to ensure the project cashflow remains positive.

## 1.14    Conclusions and Recommendations

### 1.14.1  Mineral Resource estimate

In respect of the Mineral Resource estimate, and the opinion of David Gray (QP), the classifications applied to the estimates of the Kevitsa Ni-Cu-PGE deposit accurately reflect the confidences in the available sample data, the geological model and the resulting grade estimates.

The understanding of close spaced geological and grade continuity has been improved by using the grade control drilled reverse circulation (RC) samples in this estimate. In turn this has added confidence to definition of different domains of mineralisation and the robustness of the resulting block estimates. The quality of this RC data was assured through a sound program of quality assurance and quality control (QAQC) together with twin hole analysis against the higher quality diamond drilled sample assay results. No deviations or biases between these two data sets were noted.

Inferred resource classification was applied to those estimates located in the deepest portions of the deposit as well as those areas having only one mineralised drillhole intersection within 100 m.

Compared to the previous estimate, the 2016 updated Mineral Resource estimate has increased total available nickel sulphide (NiS) metal by 6% and has increased total available copper metal by 4%.  Metal increases have resulted from expanded mineralised volumes which have been guided by:

- on-mine reconciliation data of the 24 Mt mined since the previous estimate,
- additional RC drilling, which has improved domain delineation and geology detail,
- in-pit mapping,
- updated 3D seismic structural interpretations,
- a comprehensive data set of XRD data supporting mineralisation, lithology and alteration definition, and
- alignment and improvement of estimation methods.

The Kevitsa geology and mining team have completed a due diligence review of the input data, estimation methods, block estimate results and classification.  Findings of this review support this updated estimate to be robust in methodology and representative of the input data and current mining performances.

Recommendations include:

- Consider extensional diamond drilling in order to improve confidence in deposit extents and deeper zones of mineralisation.
- Consider increasing confidence of Indicated Mineral Resources with a program of infill diamond drilling particularly covering the next 5 years of Mineral Reserves.
- Complete correlation of the developed neural network mineralisation domains with lithology and alteration from XRD data.
- Continue to improve the definition of the respective mineralisation domains mineralogy and the spatial delineation thereof. 3D structural geology models are likely to become important to improving the understanding of Kevitsa geology and impacts on mineralisation.
- Improve upon current estimation methods in both resource and grade control estimation routines. There is some opportunity for increasing domain detail which will in turn improve spatial analysis and allow for more accurate block estimates.

### 1.14.2  Metallurgical Testwork and Processing

The current metallurgical facilities at Kevitsa are capable of handling 9Mtpa of ore to produce on spec copper and nickel concentrates at design recoveries.

Recent work provides a solid framework for continued improvements in metallurgical processing at Kevitsa. Of vital importance is the understanding of the role of the gangue components in the ore on plant capacity, metal recovery and concentrate grades. Thus the geomet programs need to be progressed and expanded, and the blast optimization work needs to develop further. Further improvements in throughput in the milling circuit can be realised by limiting the quantity and size of the coarse lumps in the feed to the AG mills. Further optimization of the primary screen and secondary crusher circuit, combined with optimized blasting will help in this regard. Optimization work should also continue, with a view to a reduction in operating costs, and improved recoveries and concentrate grades. Ultimately a feasibility study will be required to evaluate the scope to increase throughput to 11Mtpa.

### 1.14.3  Geotechnical Engineering

Wall rock is competent and management systems are in place to monitor slope stability and review data. External geotechnical consultants (both from within Finland and overseas) are utilised to assist and advise the KMO team.

### 1.14.4  Mineral Reserve Estimate

The data is adequate to support the Mineral Reserve estimates using the categories of Mineral Reserve estimates permitted under NI 43-101 within the designed final pits and based upon Measured and Indicated Resources only.

Recommendations include:

- Continue to refine mine designs based on the results from ongoing grade control drilling, reconciliation, and Resource model updates.
- Monitor ore reconciliation to obtain data on the new model in relation to unplanned dilution and loss. Update estimates if required.

- Review the optimisation results, pit designs, and process plant operational parameters if and when metal prices improve. At long term consensus metal prices, the optimisation indicated that Stage 5 of the pit could be developed thus increasing the life of mine.
- Continue monitoring of pit slopes and using data obtained to review design parameters. It is noted that it may be possible to steepen some slopes and reduce waste stripping costs however this needs to be evaluated carefully and weighed up against the risk of wall failure.

### 1.14.5 Mining

The mine footprint has expanded over time such that the KMO owner operated mining fleet has now got sufficient room to operate efficiently (which is reflected in the production numbers increasing quarterly through 2015). The mining contractor has been working with KMO since mining commenced and is generally responsible for mining areas deemed too small for efficient operation of the KMO fleet. Production targets are being met and KMO and the contractor have sufficient equipment on site to meet the future targets. The only issue in relation to mining is slope stability. Whilst this may have the potential to close the pit, the wall rock is competent and management systems are in place and have worked effectively to date.

### 1.14.6 Environmental Compliance

It was noted during the site visits that the close working relationship between KMO personnel and personnel employed by the regulatory authorities means that the Mine is being constantly monitored and reviewed. It is recommended that KMO continue to work with the authorities to show the commitment to meeting environmental targets.

### 1.14.7 Social Compliance

The site has a good relationship with local communities, Sodankylä municipality, and reindeer herders, and that also applies to the relationships with the supervising and permitting authorities. At the time of writing this report, the issuer was not aware of any material social or community non-compliance or risks at Kevitsa.

### 1.14.8 Closure Plan

The mine closure and reclamation phase, which will follow once the extraction and mineral processing phase has been completed, will involve dismantling buildings and constructions, decommissioning the mine, and landscaping. The reclamation and landscaping of waste disposal sites will begin in stages as soon as the final dump heights have been reached. Monitoring of the site will need to continue for several years after mining has been completed.

# ITEM 2    **INTRODUCTION**

## 2.1      **Purpose of the report**

This Technical Report on the Kevitsa Mine (the Mine) has been prepared by Qualified Persons,  Tony Cameron of Cameron Mining Consulting Ltd (CMC), David Gray of First Quantum Minerals Ltd (FQM), and Andrew Briggs of First Quantum Minerals Ltd (FQM) for FQM.

The purpose of this Technical Report is to document updated Mineral Resource and Mineral Reserve estimates for the Mine, and to provide an updated commentary on the operational status of the Kevitsa mine.

## 2.2      **Terms of Reference**

The Company has prepared this Technical Report to document updated Mineral Resource and Mineral Reserve estimates, economic evaluations and supporting ancillary documentation associated with the Kevitsa Mine in Lapland, Finland. The Technical Report covers all mineralisation at the Mine and has been written to comply with the reporting requirements of the National Instrument 43-101:  'Standards of Disclosure for Mineral Projects' of the Canadian Securities Administrators (the Instrument) and with the 'guidelines of the Australasian JORC Code (JORC, 2012), which in turn complies with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014).

This report and the resource and reserve estimates discussed therein represent the results of a culmination of reviews and updates undertaken by FQM since the previous technical report dated 12th January 2010.

This report supersedes the NI 43-101 Technical Report issued by FQM for Kevitsa and dated January 2010.  The effective date for this Mineral Resource and Mineral Reserve estimate update is 30th March, 2016.

## 2.3      **Qualified Persons and Authors**

The Mineral Resource estimates were prepared under the supervision of Qualified Person (QP), David Gray.  Mr Gray of FQM, the issuer, meets the requirements of a Qualified Person according to his Certificate of Qualified Person attached in Item 28.

The Mineral Reserve estimates were prepared under the direction and supervision of Tony Cameron (QP). Mr Cameron of Cameron Mining Consulting Ltd. meets the requirements of a Qualified Person according to his Certificate of Qualified Person attached in Item 28.

Metallurgical testing, mineral processing and process recovery aspects of this report were prepared under the direction of Andrew Briggs (QP). Mr Briggs of FQM meets the requirements of a Qualified Person according to his Certificate of Qualified Person attached in Item 28.

The following table identifies which items of the Technical Report have been the responsibilities of each QP.

| Name | Position | NI 43-101 Responsibility |
|------|----------|--------------------------|
| David Gray <br><br> BSc (Geology), MAusIMM, PrSciNat(SACNASP) | Group Mine and Resource Geologist, <br><br> FQM (Australia) Pty Ltd | Author and Qualified Person <br><br> Items 7 – 12, 14 |
| Tony Cameron <br><br><br> BEng Hons (Mining), MEngSc, FAusIMM | Consultant Mining Engineer, <br><br> Cameron Mining Consulting Ltd. | Author and Qualified Person <br><br> Items 1-6, 15 and 16, 18 to 28 |
| Andrew Briggs <br><br> BSc(Eng), ARSM, FSAIMM, PEng(NAPEG) | Group Consultant Metallurgist <br><br> FQM (Australia) Pty Ltd | Author and Qualified Person <br><br> Items 13 and 17 |

## 2.4 Sources of Information

Geology and Mineral Resource sources of information are drilling logged and sample data, grade control reverse circulation sample data, in-pit geology mapping, as well as the relevant (current) information from the previous Technical Report on the property.

Mining and Mineral Reserve sources of information are the Mineral Resource, actual production and monitoring data (since 2012), and budget projections, as well as the relevant (current) information from the previous Technical Reports on the property.

Metallurgy, Processing, and Economic sources of information are the actual operating data acquired since copper production commenced in 2012, operating budget estimates, as well as the relevant (current) information from the previous Technical Reports on the property.

All other relevant information has been gathered from previous Technical Reports on the property and where necessary updated with information and/or reports provided and translated by senior site personnel.

## 2.5 Personal Inspections

The QP's have visited the Kevitsa Operation as follows: -

- David Gray visited the project in August 2014, March 2015, November 2015 and January 2016. In the first visits Mr Gray inspected drill core and drilling sites, reviewed geological data collection and sample preparation procedures, and carried out independent data verification. The most recent visit was for the purpose of finalising the latest geology model and enhancements to the grade control process.
- Tony Cameron most recently visited the property in January 2016 (4th to 20th). Mr Cameron inspected the open pit operations including observing and reviewing a number of production

and trim blasts. He also inspected the tailings and waste storage facilities, as well as the maintenance workshops. Previous visits to site were in December 2012, July 2014, and May 2015.

- Andrew Briggs has visited Kevitsa fourteen (14) times since FQM acquired the project in 2008. The eight (8) visits prior to August 2012 were involved with the plant design and commissioning. Following the plant commissioning, Mr Briggs' visits were involved in plant optimisation, installation of a second secondary crusher, regrind milling, column-flotation, re-work associated with the primary screen, and dealing with the problems of self-heating concentrates. The most recent site visit was in October 2014.

# ITEM 3    **RELIANCE ON OTHER EXPERTS**

The authors of this Technical Report do not disclaim any responsibility for the content contained herein.

# ITEM 4    PROPERTY LOCATION, DESCRIPTION AND TENURE

## 4.1    Location

The Kevitsa Mine 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. The town of Sodankylä is located approximately 40 km south by road and the nearest village Petkula is located 8 km west of the property.  A location map is presented in Figure 4-1.

Access to the mine site is via excellent, well-maintained all-weather sealed roads.  Port facilities are available at Kemi Harbour which is approximately 290 km from the property by road.

**Figure 4-1        Location of Kevitsa Operation, Finland**



## 4.2    Description

Kevitsa is situated in Finnish Lapland.  The area has a gently undulating terrain and is a plateau at level 220 m to 240 m with local hills rising to 350 metres. The Kevitsa deposit is sited at the watershed between the stream 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.

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

On the project area Kevitsa Hill is the highest point with an elevation 310 metres above sea level, the lowest areas being ~212 metres, and on western boundary of applied mining concession average elevation being 230 metres above sea level in the main resource area.

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

## 4.3      Tenure

The site operating entity is Kevitsa Mining Oy (KMO), of which, FQM holds a 100% interest. Mining Concession No. 7140 was granted to KMO by the Regional Ministry for Employment and Technological Development of the Province of Andalusia on August 6, 2003 and expires on August 6, 2033.  Figure 4-2 shows the project location relative to the main towns in the area.

**Figure 4-2      Location of the Kevitsa Project**
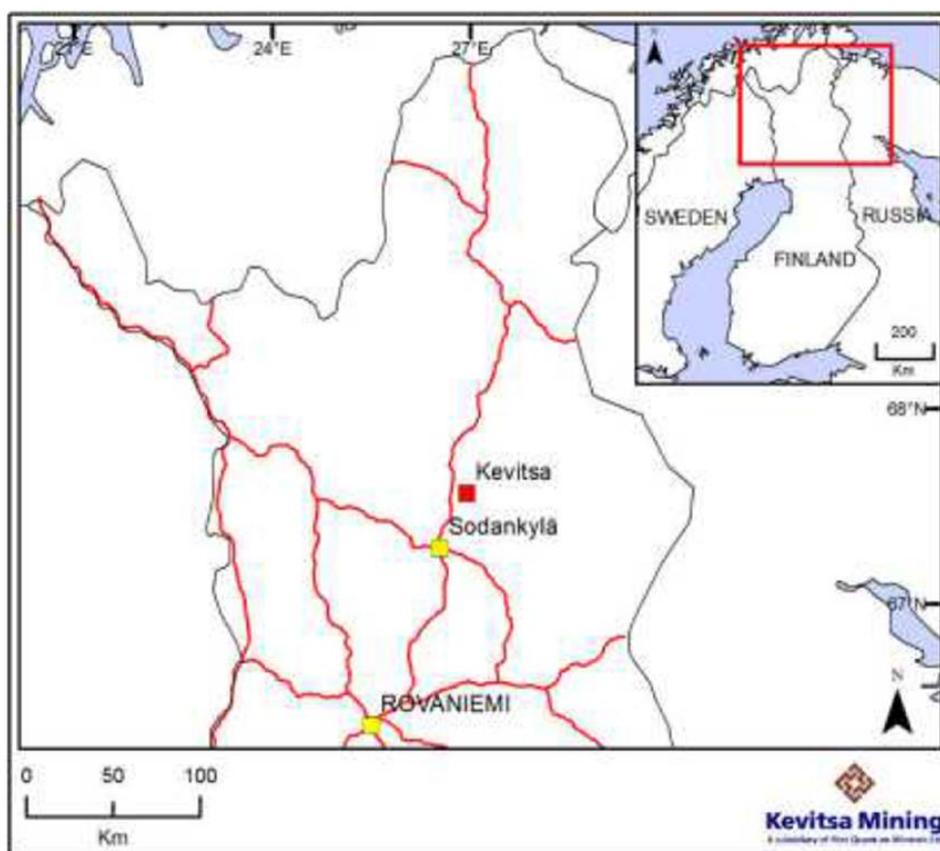


## 4.4      Rights and Surface Land Ownership

In accordance with Finnish regulations, KMO owns the land within the mining concession. The land was previously under the control of the Forestry authority who also control most of the surrounding land.

## 4.5      Royalties, Payments, Agreements

There are no mineral royalties in Finland.

## 4.6      **Environmental Liabilities and Permitting**

The Kevitsa is located in a forestry area well away from populated areas.  As the project has developed, KMO has incurred reclamation liabilities that are clearly identified in the permitting and budgets.

Permitting for the Kevitsa Mine included environmental approvals for operation of the mine, processing plant, waste disposal and tailings disposal, as well as water use permits and approvals.

## 4.7      **Archaeological**

All archaeological sites identified prior to the commencement of the Mine have been investigated by a registered archaeological company (Mikroliitti Oy).  For the archaeological sites encountered during operations that were not initially identified, once an indication of their presence exists, an archaeological team would be retained and brought in by KMO and the findings are catalogued before advancement of the Mine is allowed in the area that would affect the site.

## 4.8      **Potential Access and Exploitation Risks**

At this point in time, there are no other significant factors or risks that may affect access, title, or the right or ability of KMO to continue operations.

# ITEM 5   ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

## 5.1    Accessibility

Access to the Kevitsa project site is good. The main road from Rovaniemi is E75 (E4) that connects to the Village of Petkula by a surfaced local road until the Vajukoski hydropower station and dam are reached. A new road and bridge were constructed to access the site. This included the construction of two bridges, firstly over the Kitinen River and secondly over the Mataraoja Stream.

## 5.2    Climate

The climate at Kevitsa and surrounding area is subarctic, with average temperatures of 0.8 ºC and average precipitation of 507 mm. October to April has negative temperatures, with January being the coldest period averaging at -14.1 ºC, with half of the precipitation in the form of snow

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

Seasonal temperature variation is strong, as well as length of daylight. Winter usually starts in late October with the first snowfall and most frequently snow cover thaws in early May. In springtime, in April snow cover on the area can reach up to 100 cm in thickness, however, the long term average at the end of March is 79 cm. The area does not have permafrost; however, during winter the ground freezes thereby demanding expertise to avoid failures in road and building construction.

## 5.3    Physiography

Topography of the surrounding area is that of gently rolling hills. The original (before mining) property elevations range from 15 to 45 meters above sea level.

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.

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

On the project area Kevitsa Hill is the highest point with an elevation 310 metres above sea level, the lowest areas being ~212 metres, and on western boundary of applied mining concession average elevation being 230 metres above sea level in the main resource area.

## 5.4    Vegetation

Prior to acquisition of KMO, the property was used for forestry and reindeer herding. The closure plan states that, where possible, the land will be returned to its former use.  Areas not returned to agricultural use will be revegetated with native plant species.

5.5        **Local Resources**

Lapland is relatively sparsely populated however KMO aims to employ local personnel and contractors where possible.

As of December 2015, the relative proportions of workers are as follows:

- Local (Sodankylä)                      52%
- Regional (Lapland)                    36%
- National (Finland other)                6%
- International                             <u>6%</u>

            Total                              100%

5.6        **Infrastructure**

All infrastructure required by the Mine is in place including sealed roads, power lines and substations, process plant, site offices, workshops, tailings dam, and waste storage facilities. More information on infrastructure can be found in Item 18 of this report.

# ITEM 6 **HISTORY**

## 6.1 **Previous Mineral Resource and Mineral Reserve Estimates**

### 6.1.1 **Mineral Resources**

The previous Mineral Resource estimates were detailed in the Technical Report (dated May 2011) and completed by Qualified Persons, Mr Galen White of CSA Global (UK) Limited and Mr Markku Lappalainen of FQM's Kevitsa Mining Oy on behalf of First Quantum Minerals Limited. At the time of reporting these Mineral Resource estimates, mining had not yet started at Kevitsa. These Mineral Resource estimates were written to comply with the reporting requirements of the National Instrument 43-101: 'Standards of Disclosure for Mineral Projects' of the Canadian Securities Administrators (the Instrument) and in turn complies with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2005).

**Table 6-1     The previous Mineral Resource statement dated May 2011 and using a total nickel cut-off grade of 0.1% (before mining of 23 Mt of ore).**

| Category | Density | Tonnes (Mt) | Ni (%) | NiS (%) | Cu (%) | Au ppm | Pt ppm | Pd ppm |
|---|---|---|---|---|---|---|---|---|
| Measured | 3.18 | 89.3 | 0.29 | 0.26 | 0.40 | 0.12 | 0.23 | 0.17 |
| Indicated | 3.20 | 150.8 | 0.32 | 0.29 | 0.42 | 0.11 | 0.19 | 0.14 |
| **Measured and Indicated** | **3.20** | **240.1** | **0.30** | **0.28** | **0.41** | **0.11** | **0.21** | **0.15** |
| Inferred | 3.20 | 34.7 | 0.29 | 0.27 | 0.36 | 0.09 | 0.14 | 0.10 |

### 6.1.2 **Reserves**

The previous Ore Reserve estimates were detailed in the Technical Report (dated May 2011) and completed by Qualified Person, Mr Nick Journet of Dump Solver Pty Ltd on behalf of First Quantum Minerals Limited. At the time of reporting these Ore Reserve estimates, mining had not yet started at Kevitsa. These Ore Reserve estimates were written to comply with the reporting requirements of the National Instrument 43-101: 'Standards of Disclosure for Mineral Projects' of the Canadian Securities Administrators (the Instrument) and in turn complies with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2005).

**Table 6-2**          **The Previous Ore Reserve Statement Dated May 2011 (before mining of 23Mt of ore).**

| Category | Mt | Ni % | Sul Ni % | Cu % | Au ppm | Pt ppm | Pd ppm | Co % |
|---|---|---|---|---|---|---|---|---|
| Proved Reserve | 84.7 | 0.276 | 0.252 | 0.388 | 0.120 | 0.232 | 0.168 | 0.014 |
| Probable Reserve | 75.9 | 0.316 | 0.286 | 0.412 | 0.120 | 0.244 | 0.176 | 0.014 |
| *Total Mineral Reserve* | *160.6* | *0.295* | *0.268* | *0.399* | *0.120* | *0.233* | *0.172* | *0.014* |
| Waste | 469.8 | | | | | | | |
| Pit Size | 630.4 | | | | | | | |
| Strip Ratio | 3:1 | | | | | | | |

*Source: N. Journet,  Dump Solver Pty Ltd – May 2011*

The previous Ore Reserve was estimated inclusive of 3% mining losses and 3% mining dilution at zero grade.

## 6.2          Production

### 6.2.1    Mining Operations

Mining movement has increased each year since commencement of operations in 2012. Total 2015 ore and waste production was 37 Mt.

**Table 6-3**          **Kevitsa Mining Production Summary 2012 to 2015.**

| Production | | 2012 | 2013 | 2014 | 2015 | Total to Date |
|---|---|---|---|---|---|---|
| Ore | Mt | 3.14 | 6.68 | 6.72 | 6.42 | 22.96 |
| Waste/Overburden | Mt | 8.97 | 17.29 | 21.93 | 30.47 | 78.65 |
| Total | Mt | 12.11 | 23.96 | 28.65 | 36.89 | 101.61 |

The average strip ratio to date is 3.4 to 1 which is higher than the original life of mine average of 3 to 1 due to the initial pre-stripping activities.

### 6.2.2    Processing Operations

Table 6-2 below summarises ore processed and metal produced since the plant commenced operations in 2012.

**Table 6-2**          **Kevitsa Process Plant Production Summary 2012 to 2015.**

| Production | | 2012 | 2013 | 2014 | 2015 | Total to Date |
|---|---|---|---|---|---|---|
| Ore Processed | kt | 3,138 | 6,314 | 6,711 | 6,665 | 22,828 |
| Nickel Produced | t | 1,870 | 8,963 | 9,433 | 8,805 | 29,071 |
| Copper Produced | t | 3,448 | 14,775 | 17,535 | 17,204 | 52,962 |
| Platinum Produced | oz | 6,123 | 30,403 | 34,090 | 31,899 | 102,515 |
| Palladium Produced | oz | 5,419 | 24,639 | 25,990 | 25,196 | 81,244 |
| Gold Produced | oz | 2,172 | 11,723 | 12,844 | 12,847 | 39,586 |

# ITEM 7    GEOLOGICAL SETTING AND MINERALISATION

## 7.1    Regional Geology

The Kevitsa deposit is located in the centre of the Kevitsa igneous complex; in the ultramafic-mafic intrusive rocks dated at 2058 ± 4 Ma (Mutanen and Huhma, 2001). The Kevitsa igneous complex is enveloped by Paleoproterozoic supracrustal rocks of Central Lapland Greenstone Belt (CLGB) in the Precambrian Fennoscandian Shield. The CLGB is comprised (Figure 7-1) of several volcano-sedimentary stratigraphic groups (Räsänen, Hanski, Juopperi, Kortelainen, Lanne, Lehtonen, Manninen, Rastaa and Väänänen, 1996) with ultramafic intrusives (Hanski, Huhma, Rastas and Kamenetsky, 2001; Hanski and Huhma 2005).

The CLGB is divided to seven stratigraphic groups (Räsänen et al. 1996) from oldest to youngest; Salla, Onkamo, Sodankylä, Savukoski, Kittilä, Lainio and Kumpu. These volcano-sedimentary rocks of the CLGB have undergone multiple episodes of folding and thrusting resulting in overturning and structural repetition of the stratigraphic sequences.

The Salla and Onkamo Group were formed in an intracratonic rift environment at 2.5 – 2.4 Ga. The Salla Group consists of felsic metavolcanic rocks and it is overlain by the Onkamo Group siliceous, high-Mg basalts and mafic metavolcanics. After rifting ceased it was followed by extensive deposition of overlying Sodankylä Group clastic, metasedimentary rocks. Savukoski Group (Figure 7-2), which hosts the Kevitsa intrusive complex, represents a major marine transgression dominated by supracrustal rocks; black schists, phyllites, tuffites, mafic metavolcanics and the uppermost unit of ultramafic metavolcanics. The minimum age of 2060 Ma (Mutanen and Huhma, 2001) for Savukoski Group pelitic metasediments has been determined from the crosscutting ultramafic intrusive bodies. A tectonic contact separates the Savukoski group from overlying mafic volcanic rocks of the Kittilä Group. These overthrust faults with ophiolites have been interpreted as allochthonous. The Kittilä Group is a single large terrain of mafic volcanic rocks called the Kittilä Greenstone Complex. Both Savukoski and Kittilä Group rocks are overlain unconformably by the quartzites and conglomerates of the Lainio and Kumpu Groups. The Lainio and Kumpu Groups have a maximum age of 1.88 Ga. (Lehtonen, Airo, Eilu, Hanski, Kortelainen, Lanne, Manninen, Rastas, Räsänen and Virransalo, 1998; Hanski and Huhma, 2005).

Three major ductile deformational events (D1-D3), simultaneous and later shear zones are related to regional structures of the CLGB (Hölttä et al. 2007). The deformation event D1 comprises the oldest tectono-metamorphic feature S1 which is a bedding-parallel foliation. S1 can be seen perpendicular to the S2 foliation, in the hinges of the F2 folds. S2 foliation is the most prominent structural feature of the CLGB. In most cases it is sub-parallel to bedding and when associated to F2 folds S2 is axial planar to foliation. The tight or isoclinic, recumbent or reclined, F2 folds indicate the Northward thrust deformation event (D2). The aforementioned structures are deformed by one or multiple sets of later F3 folds of the D3 deformation event. F3 folds are dominated by E-W and N-S orientation with varying dips of axial surfaces from horizontal to vertical, and shear zones of varying orientations; in the Sodankylä area F3 folds are E-W oriented and their axial surfaces range from vertical to moderately dipping. Tectonic movements of D3 are complex and adjacent shear zones involve rotations.

**Figure 7-1      Regional geology map highlighting the position of Kevitsa igneous complex in relation to the Central Lapland Greenstone Belt (CLGB) geology (Hölttä, Väisänen, Väänänen and Mnninen, 2007).**
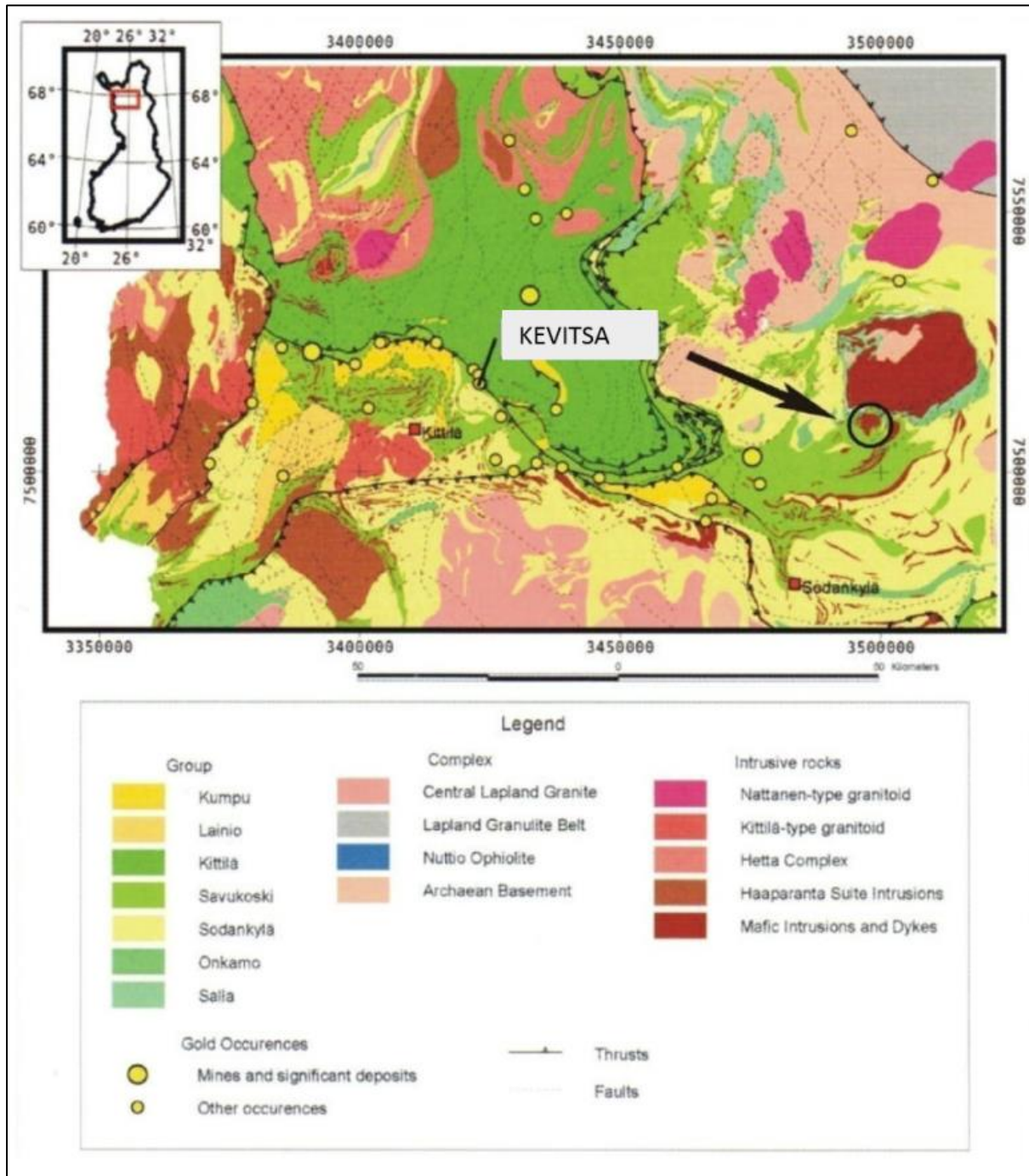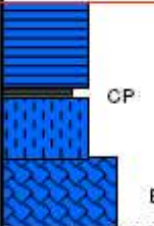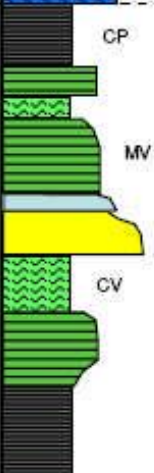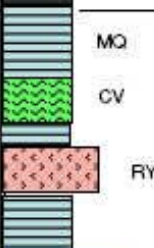
**Figure 7-2      Kevitsa regional stratigraphy. Modified by FQML, 2009; after GTK Report of Investigation 140, (Lehtonen, M. et al 1998). *Group ages based on U/Pb/Zr dating **U-Pb zircon age of 2057Ma is controversial due to possible contamination by metamict zircons.**

## 7.2      **Local geology**

The Kevitsa intrusion is a layered, mafic-ultramafic complex. The host rocks of the deposit are ultramafics, mainly olivine websterite and olivine pyroxenite (Figure 7-3) located in N-E part of the intrusion. The intrusion extends to at least 1.5 km in depth. Gabbro occurs at the top of the intrusion located to South-West side of the ultramafics. A dunite body, discordant to the intrusive layering, is present in the middle of the intrusion and another dunite body is located at the bottom of the intrusion. Xenoliths of variable size are common in the ultramafics, and within the ore body.  The xenolith composition is generally sedimentary, mafic or ultramafic. The Kevitsa intrusion is surrounded by supracrustal rocks such as mafic volcanics, phyllites and carbonaceous schists.

**Figure 7-3      Bedrock geology of the Kevitsa intrusion and surrounding country rocks. The red circle is a planned final pit position.**



The Kevitsa area has undergone several tectonic and metamorphic events which are evident in the intrusion and in the country rocks (Hölttä et al. 2007). The NNE-SSW trending Satovaara fault, and other structures which are associated with it, are a structurally significant feature of the area. The Satovaara fault has deformed the eastern margin of the Kevitsa intrusion.

Metamorphism has modified mineralogy; amphibole alteration of ferromagnesian minerals such as olivine, orthopyroxene, and clinopyroxene is very common and overprints the majority of the Kevitsa mineralisation. Other common alteration minerals include chlorite, serpentine and carbonate. Talc concentration is usually low but can be locally elevated, especially near late fractures and veins. Epidote alteration is also observed in association with faults and shear zones. Magnetite is present as a primary mineral crystallized from the intruded magma but also exists as a hydration product of pyroxenes together with amphibole.

## 7.3      **Mineralisation**

The main Kevitsa Ni-Cu-PGE mineralisation is located at the centre of the main ultramafic intrusive, which is composed predominantly of olivine websterite and olivine pyroxenite. The sulphide mineralisation is disseminated in style and the overall mineralisation volume is irregular in shape. The mineralised zones are cut by several steep faults and shear zones, thought to locally offset mineralisation.

Two economically important mineralisation styles can be distinguished; regular Ni-Cu mineralisation and Ni-PGE mineralisation. The predominant mineralisation type is Ni-Cu and comprises approximately 95 % of the deposit. Ni and Cu grade variability is relatively low but there are discrete zones of Cu and Ni-rich mineralisation. The mineralisation is characterized by a Ni tenor of between 0.1 to 0.7% Ni. The Ni-PGE zones appear to be structurally controlled and more discreet compared to the regular Ni-Cu zones and is distinguished by a higher Ni tenor.

The main sulphide minerals are chalcopyrite and pentlandite occurring together with pyrrhotite and magnetite. The sulphide grain size varies from fine to medium and occurs predominantly between the cumulate minerals displaying primary magmatic textures. Locally sulphides occur as semi-massive veinlets and pyrrhotite-rich net-textured zones. Cu-rich veinlets, typically a pyrrhotite-chalcopyrite-magnetite assemblage, are associated with carbonate veining and are likely related to local hydrothermal remobilization. The Ni-PGE mineralisation has a higher Ni content in additional mineral phases such as millerite. The range of PGE minerals is considerable and the dominant minerals are Pt-Pd bismuth tellurides and sperrylite (Gervilla and Kojonen, 2002). PGE minerals occur mainly as inclusions in amphibolite which is most likely due to local alteration. Origin of the PGE mineralisation remains debatable (Mutanen, 1997; Hanski et al., 1997; Yang, Maier, Hanski, Lappalainen, Santaguida, and Määttä, 2012; Vaillant, Barnes, Fiorintini, Santaguida, and Törmänen, 2015).

# ITEM 8    DEPOSIT TYPE

Kevitsa is a magmatic, layered-intrusive, Ni-Cu-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 Ni-Cu-PGE deposit located 20 km to the South-West 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. This can be seen in the Kevitsa ultramafic rocks (Figure 8-1). Simultaneous with mineral crystallization the chemical composition of the rock changes from MgO-rich to Al2O3-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.

**Figure 8-1**    **Mineralogical and geochemical trend in Kevitsa ultramafic rock differentiation.**
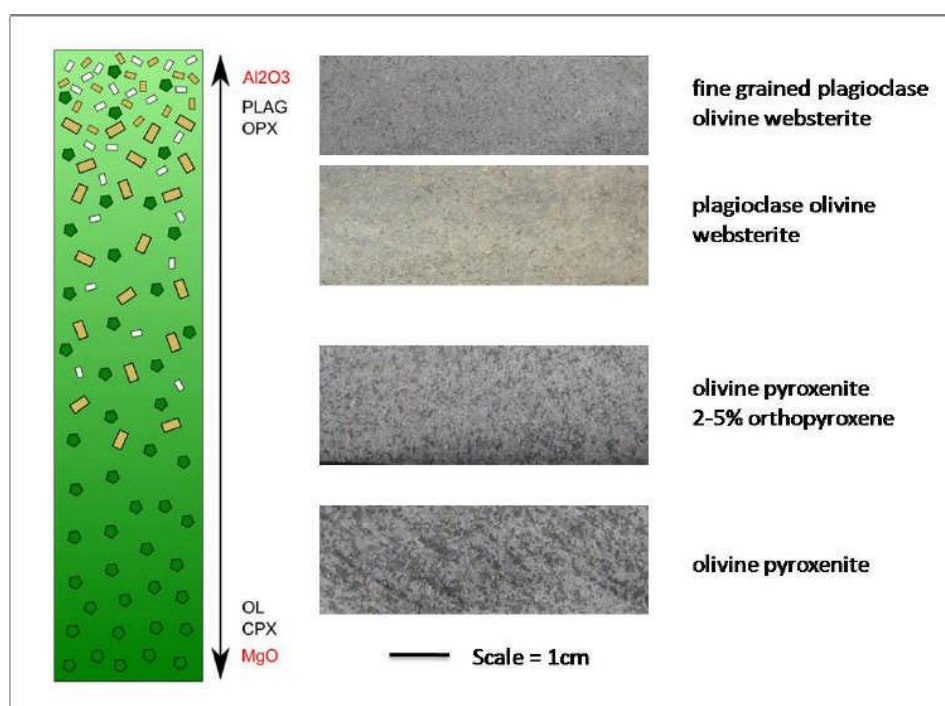


Accordingly modelling of the domains of mineralisation has taken cognisance of the presence of igneous layering and structural deformation together with the significant alteration and remobilisation of mineralisation. Definitive 3D modelling of the respective lithologies together with their relationship to the different styles of mineralisation is problematic. As a result, neural network analysis and probabilistic estimation methods have been used to guide defining the spatial position of mineralised volumes.

# ITEM 9   **EXPLORATION**

Apart from sampling associated with ongoing drilling (detailed in Item 10), there were no additional samples or exploration work undertaken by the issuer at the Kevitsa deposit.

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 FQML and previous owners, mainly the Geological Survey of Finland (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 surveys from 1978, 1982, and 1984 on a 100 x 20 m grid on various orientations
- Electromagnetic
    - airborne single-frequency electromagnetic 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 200m spacing, and reduced to 100 m spacing over the Kevitsa-Satovaara Igneous Complex
    - horizontal loop, frequency ground electromagnetic (Slingram 1984 and Maxmin 1987) and VLF at different frequencies from 1993 to 1995
    - local ground based EM 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
    - The 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 12.5m interval spacing and maximum receiver offset of 2502 m
    - 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 2009-2013. 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 Satovaara, Lipatti, Saivel North, and Mustaselkä among others. Between 2010 and 2015 base of till surveys were conducted in the following areas in and around the Kevitsa Mine lease:

- the northern part of Kevitsa Mine area 2010
- Sato-oja 2011
- Satovaara 2011
- Satovaaranjänkä 2012
- Lipattikuusikko 2011
- Saivel North 2011-2013
- Haapaselkä 2013
- Satojärvi North 2013
- Satovaara North 2013-2014
- Pikku Vaiskonselkä 2014
- Satovaarankuusikko 2014
- Saiveljärvi 2014
- Vaju 2015

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 mineralization was intersected but was considered to be uneconomic. Table 9-1 outlines the exploration tenement areas as seen on the figure above. This is split into current valid permits and those for which applications have been submitted.

**Figure 9-1        Base of Till sampling conducted by First Quantum Minerals between 2010 and 2015.**
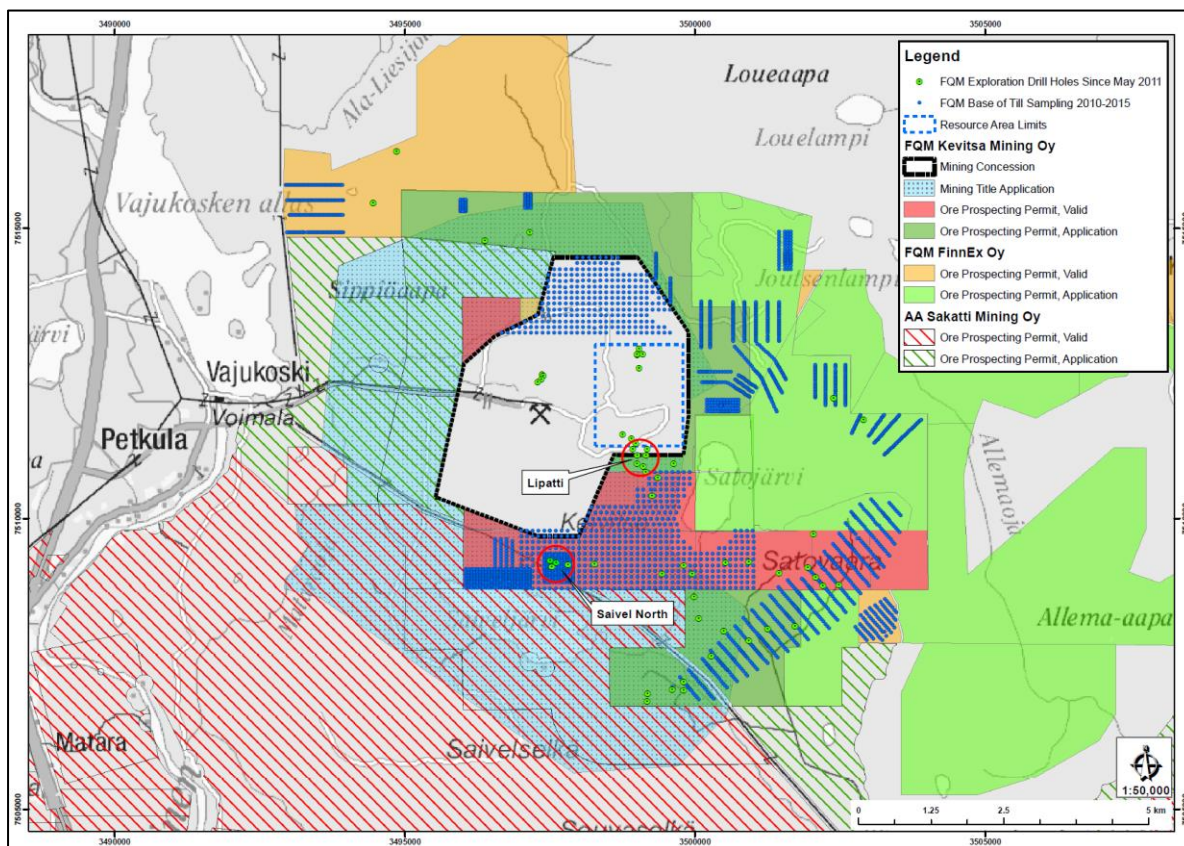


**Table 9-1        Exploration tenements as per figure 9-1**

| Type | Owner | Area (km²) | No. of Blocks | Permit ID | Notes |
|------|-------|-----------|---------------|-----------|-------|
| Valid Mining Concession | FQM KMOY | 14.1 | 1 | 7140 | |
| Valid Ore Prospecting Permits | FQM KMOY | 10.6 | 13 | 8890/1 to 8890/13 | Valid until Mining Concession extension is granted. |
| Valid Ore Prospecting Permits | FQM FinnEx Oy | 19.4 | 5 | ML2013:0078, ML2013:0079 | Belong to larger permits beyond the Near Mine area |
| Applied - Mining Concession, extension | FQM KMOY | 54.5 | 1 | 7140 | Applied Mining Title extension for the required auxiliary area |
| Applied - Ore Prospecting Permits | FQM KMOY | 13.2 | 5 | ML2014:0111, ML2014:0112 | |

| Applied      -      Ore Prospecting Permits | FQM FinnEx Oy | 45.2 | 15 | 9093/1, 9093/2, ML2013:0080 , ML2014:0106 , ML2014:0113 , ML2014:0114 , ML2015:0064 | |
|---|---|---|---|---|---|

# ITEM 10  **DRILLING**

## 10.1  **Diamond core drilling**

Diamond drilling has been used for resource definition, infill and exploration across the Kevitsa deposit. The deposit, has been drilled (Figure 10-1 and Figure 10-2) under the ownership of several companies since the 1980s; the Geological Survey of Finland (GTK), Outokumpu (OKU), Scandinavian Minerals Ltd (SGL) and First Quantum Minerals Limited(FQM/Kevitsa Mining Oy (KMOY)) as well as Finnex (FXOY).

**Figure 10-1      A plan view of the diamond drillhole collars in relation to pit location and mine infrastructure. Grey points are collar positions of diamond holes outside the resource area.**
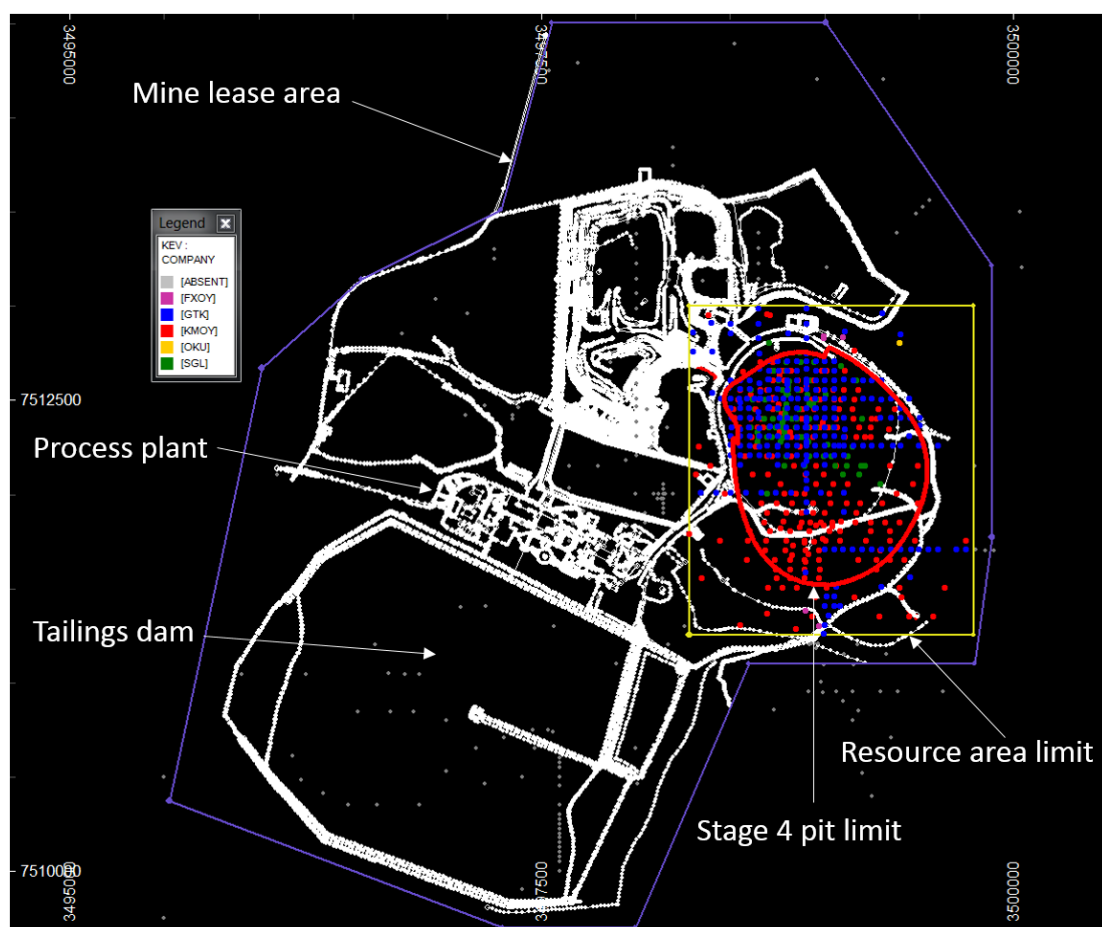
**Figure 10-2**      **A plan view of drilling within the resource area and coloured by company. Holes drilled since 2011 (additional since the previous resource estimate), are highlighted by a circle surrounding the collar position.**
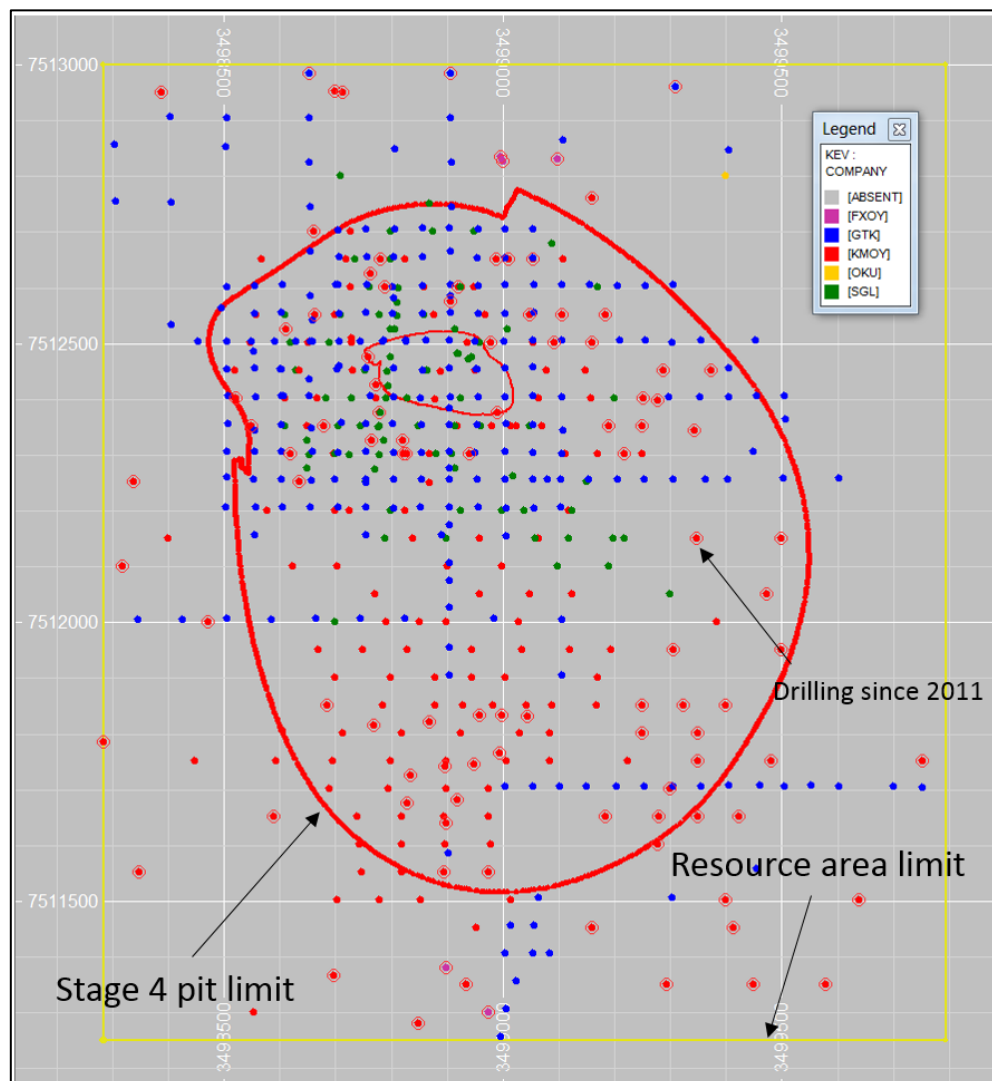


Table 10-1 and Table 10-2 summarise diamond drilling completed by respective owners as per the Kevitsa Mine Geology database. Of the 745 drilled, logged, sampled and assayed diamond holes, 546 were selected within an area deemed relevant to this mineral resource estimate (Figure 10-2). The holes included in Table 10-1 have assay results. Table 10-2 includes completed drilling on the wider mine lease and surrounds but does not include regional exploration. There were 164 drilled holes with absent sample results which were either not sampled or are still waiting on return of results (11 from KMOY/FXOY drilling and 128 from older GTK drilling).

A range of drilling inclinations was used during the various campaigns. 25% of holes were drilled vertically by GTK and were short, circa 40 m holes. The remaining holes were inclined between 45 – 80 degrees. KMOY diamond drilling was inclined between 70 – 80 degrees. Figure 10-4 and Figure 10-3 below shows the inclination of the drill holes from the three main drilling campaigns; GTK, SGL and KMOY. 97% of holes were drilled in an east-west direction. Drill direction and inclination was chosen to maximise the angle of intersection with the zones of mineralisation.

101 of the KMOY holes (Figure 10-2) have been added since the previous TR Mineral resource estimate.

**Table 10-1        Diamond drilling per campaign from 1984 to 2015 (Kevitsa Resource area).**

| Company | Year | Number of holes | Total length (m) | Ave. hole length (m) | Total length sampled (m) | Number of samples |
|---------|------|----------------|------------------|----------------------|--------------------------|-------------------|
| GTK | 1984-1995 | 250 | 33,520 | 134 | 29,834 | 15,620 |
| OKU | 1996-1998 | 1 | 256 | 256 | 16 | 13 |
| SGL | 2003-2008 | 82 | 27,429 | 335 | 26,986 | 13,611 |
| KMOY | 2008-2015 | 211 | 92,291 | 437 | 79,431 | 45,021 |
| FXOY | 2012-2015 | 5 | 2,063 | 413 | 998 | 553 |
| Total | | 549 | 155,559 | | 137,265 | 74,818 |

**Table 10-2        Diamond drilling per campaign from 1984 to 2015 external to Kevitsa Resource.**

| Company | Year | Number of holes | Total length (m) | Ave. hole length (m) |
|---------|------|----------------|------------------|----------------------|
| GTK | 1984-1995 | 99 | 7,214 | 73 |
| OKU | 1996-1998 | 10 | 1,467 | 147 |
| SGL | 2003-2008 | 3 | 336 | 112 |
| KMOY | 2008-2015 | 64 | 22,207 | 347 |
| FXOY | 2012-2015 | 20 | 7,908 | 395 |
| Total | | 196 | 39,132 | |

**Figure 10-3        Histogram showing the variation in inclination of diamond drilling for the three main drilling campaigns**
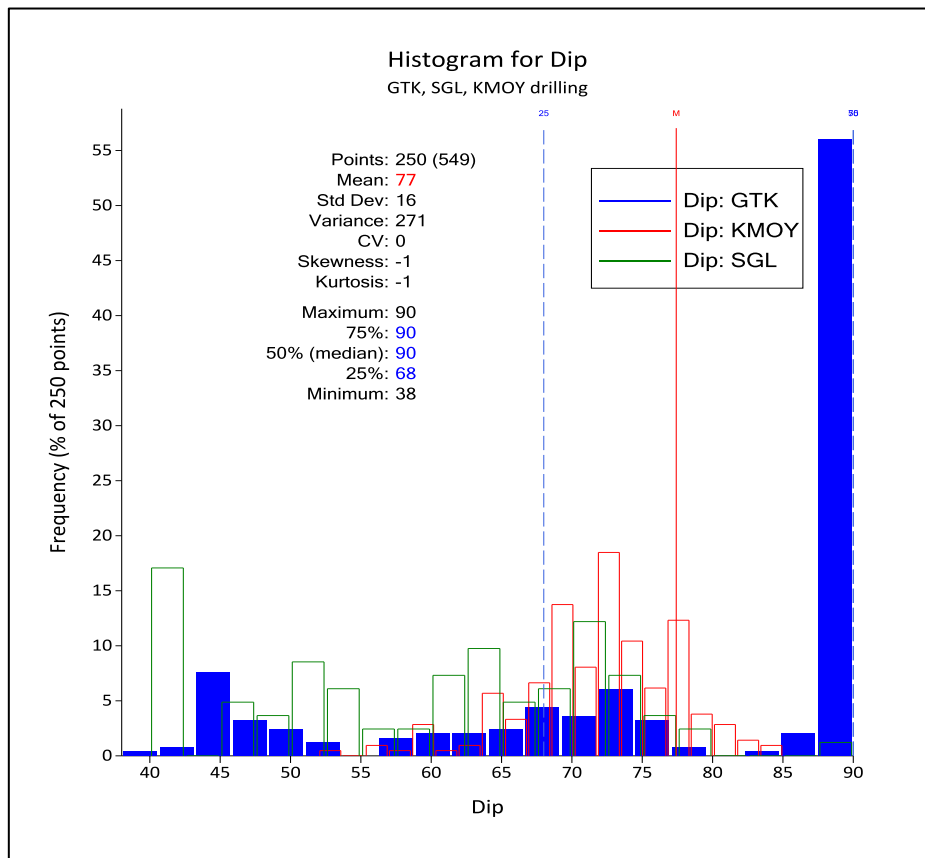
**Figure 10-4        Orientation of the inclined drilling within the resource area, coloured by company**
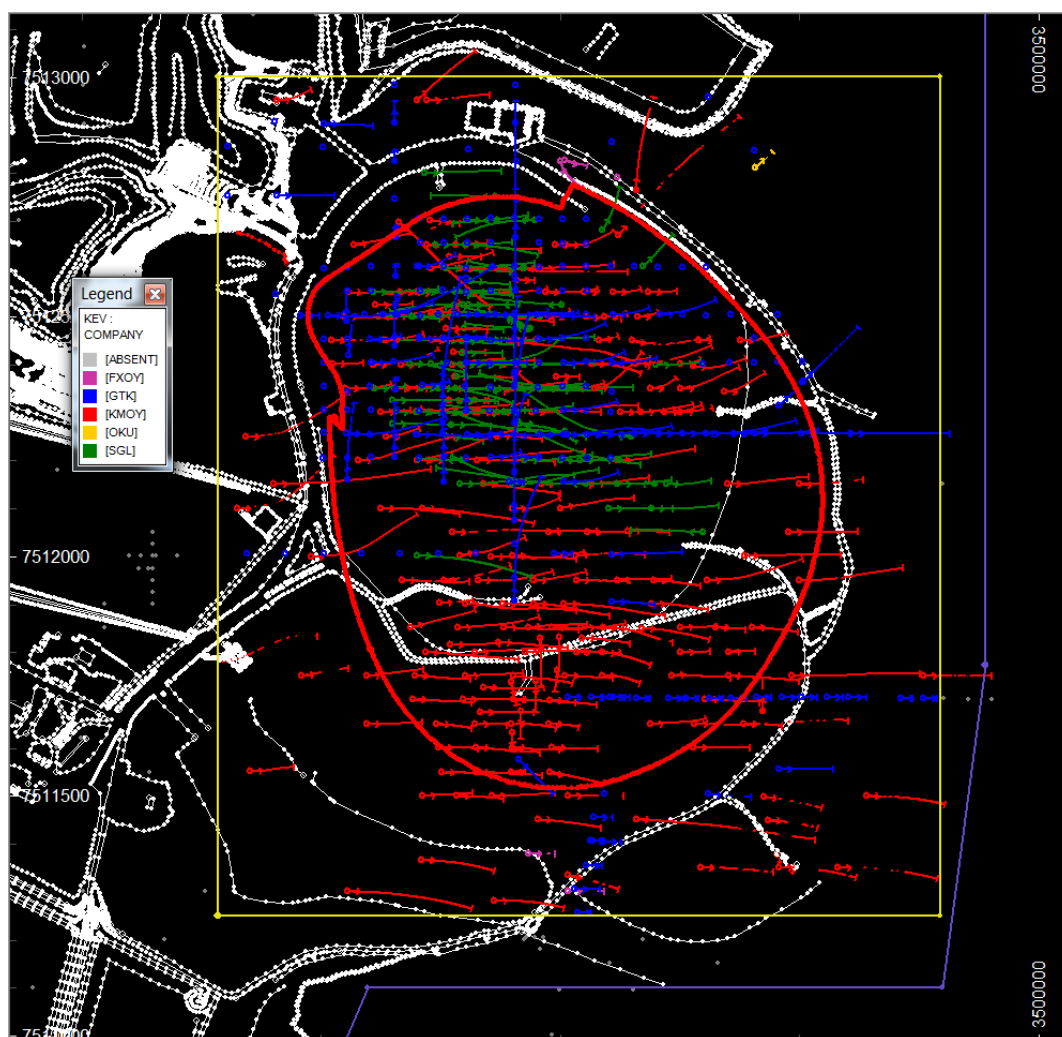


## 10.1.1  Diamond drilling campaigns

Diamond drilling until the end of 2010 has been detailed in the previous TR. This information is summarised here and supplemented with information on drilling completed by First Quantum Minerals since the last 2011 Mineral Resource estimate as per the previous TR.

The Geological Survey (GTK) of Finland drilled 480 diamond holes from 1984 – 1995. 250 diamond holes were within the resource area and 99 were in the surrounding mine lease area. The remaining GTK holes were not sampled. The initial GTK drilling in the resource area was short (circa. 40 m) holes with 40 holes in drilled in excess of 300 m deep.

Between 1996 and 1998 Outokumpu (OKU) drilled 12 holes, 1 of which falls within the resource area and has partial sampling. The remainder were located outside of the resource area or were not sampled or used in this estimate.

Scandinavian Minerals Ltd. (SGL) started drilling in 2003 and continued until First Quantum Minerals acquired SGL in 2008. The campaign totalled 90 drill holes, 85 of which were sampled and 82 of which locate within the resource area.

FQM completed a comprehensive diamond drilling program of 330 holes drilled between 2008 and early 2015. 216 of these holes fall within the resource area, 84 were located in the surrounding area and the remaining were either not sampled or abandoned during drilling. This program of drilling was focused on upgrading the previous TR Mineral Resource classification and to better delineate mineralisation, especially to the South and within the stage 4 pit limits.

Drillholes that were not sampled included those drilled for geotechnical purposes. These were used by the mining team, with guidance from WSP consultants, to assist in development of a structural model for studying pit wall stability.

Three drilling contractors have been used by First Quantum Minerals since 2008; KATI Oy, Suomen Malmi Oy (SMOY) and Arctic Drilling Company (ADC). The equipment used by each drilling company and the corresponding core diameter for completed holes within the resource area is shown in the table below.

**Table 10-3     Equipment and core diameter for drilling contractors used by First Quantum Minerals drilling within the resource area**

| Contractor | Equipment | Core diameter (mm) | Number of holes | Recovery (%) |
|---|---|---|---|---|
| ADC | BQ-TK | 40.7 | 117 | 98.9 |
| ADC | NQ | 48.0 | 5 | 99.9 |
| KATI | WL-66 | 50.5 | 17 | 99.8 |
| KATI | BQ-TK | 40.7 | 2 | 98.9 |
| KATI | NQ | 48.0 | 3 | 99.8 |
| SMOY | BGM | 42.0 | 72 | 99.4 |

Collar positions of all drill holes were survey either by an independent contractor (Rovamitta Oy) or the Mine Survey Department and referenced in Finnish National Grid Coordinate System Zone 3 coordinates. Down hole surveys were taken using Deviflex, Maxibor 2 or Gyrosmart deviation tools, depending on the drill contractor and year.

Core recovery was very good for drilling which took place under FQML ownership and is in line with previous drilling campaigns. Where core loss has occurred it was generally within the first 20 m from surface. There is no risk that core recovery would have any material impact on grade estimation.

## 10.2    Reverse Circulation (RC) and Blast Hole (BH) drilling

Seventeen reverse circulation (RC) holes (115 mm diameter) were drilled in 2010 and early 2011 as infill between the wider spaced diamond holes. The RC holes were inclined between 55 – 85 degrees and down hole surveys were taken using the Maxibor down hole survey tool. Two of these holes were not sampled and three of the locations were later redrilled with diamond drilling.

In February 2011 Grade Control (GC) drilling commenced at Kevitsa mining operations using both blast hole (BH) and RC drilling. The early infill RC and initial GC RC drilling was done by was done by Mäcklin Oy drilling contractors. KMOY operated their own RC drill rig for the period February 2012 to February 2013. During this time, in May 2012, the external drilling contractor changed to Arctic Drilling Corporation (ADC). Since February 2013 ADC has been the sole RC drilling operator on site.

During initial mining, blast hole sampling was used for grade control purposes but was slowly phased out with time. All blast sampling was stopped as of April 2015. Blast hole sampling was of inadequate quality and BH sample results were not available in a timely manner to allow for good mine planning. Blast hole sampling data was not included in the dataset for the resource estimation.

RC drilling was completed on a 15m x 15m staggered grid in mineralised zones. The drill grid was expanded to 30m x 30m in known waste zones. Drill holes were most commonly between 36 m and 72 m in length, with a mean length across the total dataset of 42 m; at these depths downhole deviation is controlled and within acceptable limits.

Since 2014 the RC sample interval has been 3m, giving 4 samples per 12 m mining bench; prior to this a 1m sample interval was used. For the unmined data, below the 31 December 2015 pit survey, the dominant sample length is 3m (Figure 10-5).

**Figure 10-5        RC sample length for samples below the 31 December 2015 pit surface.**
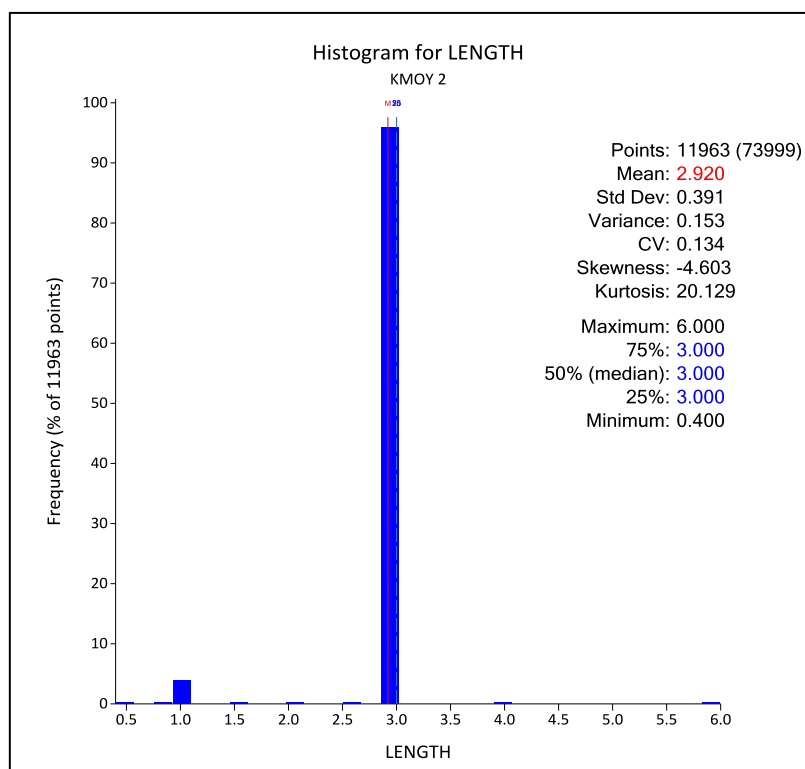


85% of the RC grade control drilling is vertical; of the inclined holes the dominant inclination is 60 degrees. Currently, holes are only inclined if drill coverage is needed in tricky locations such as close to the pit wall or on the pit edge, where vertical holes are not possible.

## 10.3     Exploration target drilling

In 2011 the Geology team was split into Mine Geology and Exploration; FinnEx Oy was created as a separate entity responsible for near mine and regional exploration. The map below shows the diamond drilling carried out by FinnEx since 2011 within the mine lease and the surrounding area (Figure 10-6).

**Figure 10-6**      **Map of the Kevitsa mine lease with surrounding exploration permits under the control of First Quantum Minerals and Anglo American (AA).   Exploration drilling is shown as green dots.**
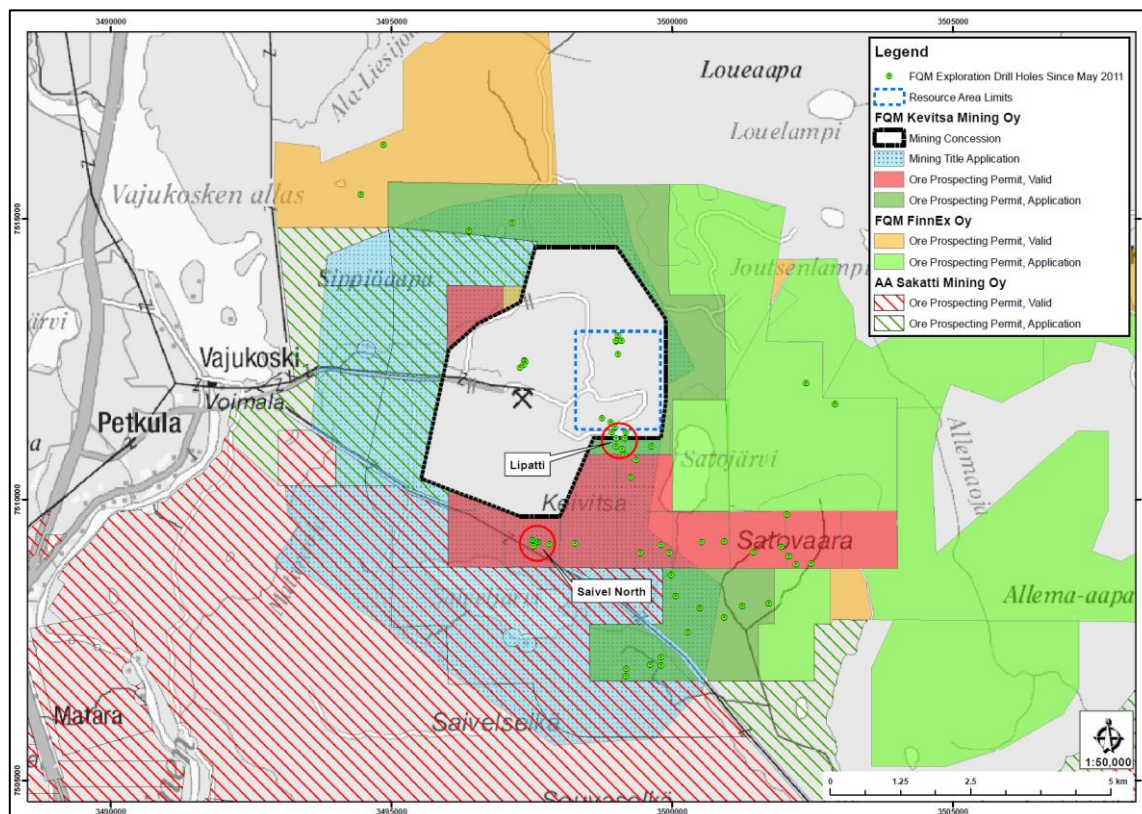


Diamond drilling of EM and VTEM anomalies initially showed promising results with disseminations of both Ni and Cu sulfide minerals, most notably at Lipatti and Saivel North. However intersections have tended to be small or at depth and therefore have so far been considered uneconomic.  Other targets were barren with anomalies often attributable to sulfidic black shales.

By 2013 most of the near mine shallow targets had been tested with only some deep targets remaining. Drilling of deep targets encountered deviation difficulties causing drill holes to miss their targets at depth. Follow up drilling has been conducted during the 2014-2015 on some remaining targets containing sulfides, most notably at Lipatti, and on VTEM anomalies to the north of the Kevitsa mine area. During 2015 exploration activities around Kevitsa were significantly curtailed due to FQM reductions in exploration budgets and personnel.

## 10.4      Summary of drilling used in the 2016 Mineral Resource estimate

549 diamond drill holes and 2,806 RC drill holes make up the dataset used for this Mineral Resource estimate; KMOY drilling makes ups 77% of the total drilling meters. In a variation to the previous TR estimate, this estimate has included results from the grade control RC sampling; the primary purpose of this drilling is to inform the grade control block model which guides mine production and feed to the plant. Including RC sample data has improved detailed delineation of zones of mineralisation, helped with the categorisation of sulphide domains and has supported grade estimates relevant to the scale of mining.  RC sample data has also provided valuable short range grade data for spatial analysis and derivation of variogram models used during ordinary kriging of

grades into the block model. RC drilling will have the most influence on the initial three to four benches within the resource estimate block model, thereafter the diamond drill data dominates estimate results (Figure 10-8).

The graph below (Figure 10-7) shows the percentage of the drilling per campaign below the 31 December 2015 pit surface.

**Figure 10-7**        **Drilling per campaign below the 31 December 2015 pit surface.**
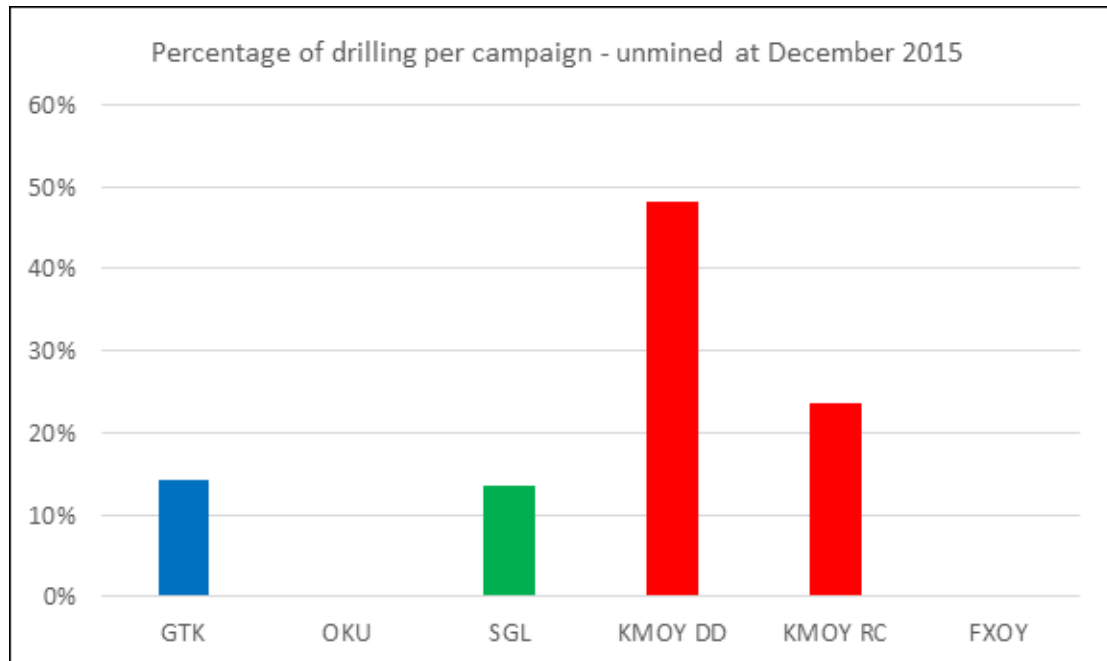
**Figure 10-8        RC drill coverage within in the pit in plan and section view. Showing the difference in the extents of diamond and RC drilling, both spatially and at depth**
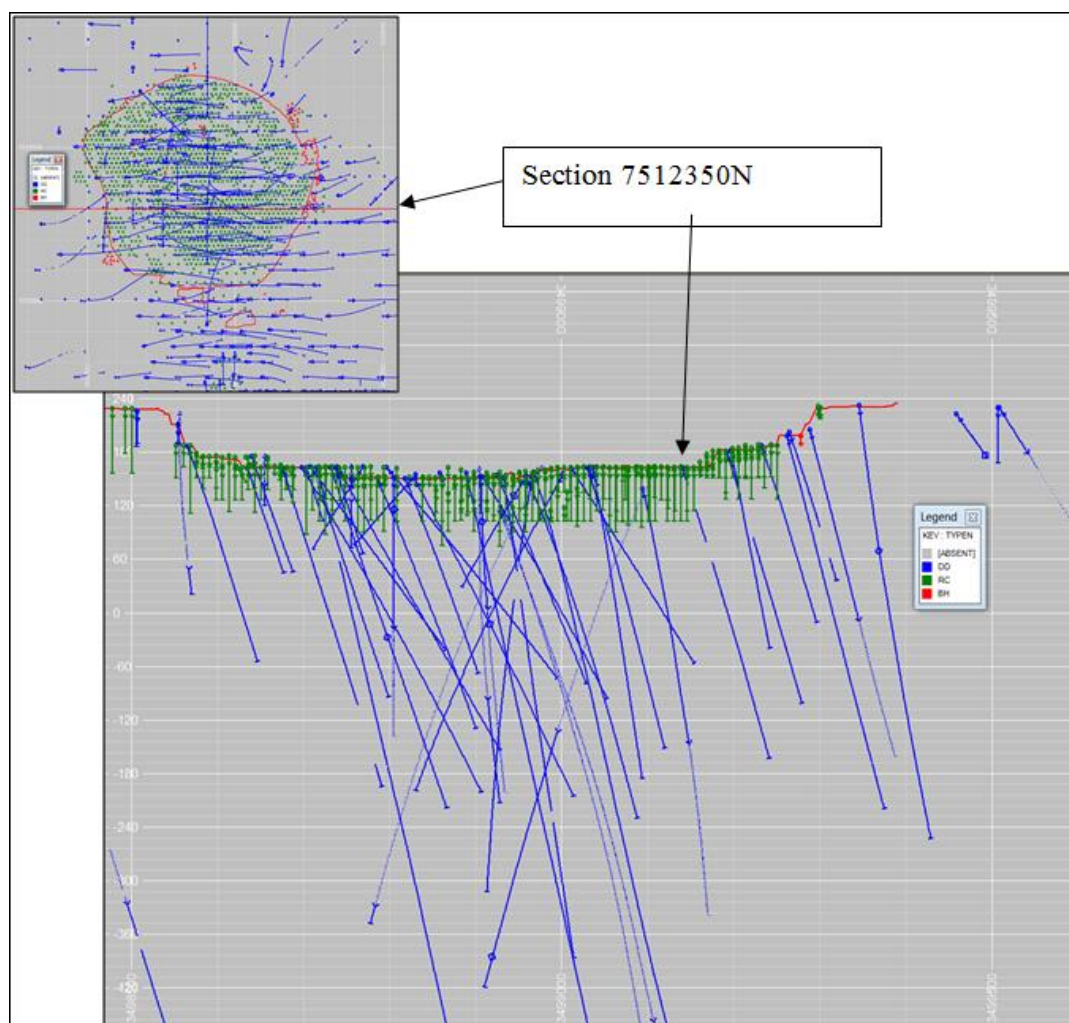


Due to the use of both diamond core and RC chip samples in this estimate, a bias study was completed in order to ensure that resulting assay results were equally representative of prevailing mineralisation. The study used holes which were spatially coincident and may be considered as twin holes. Twinning samples between RC and diamond holes eliminates risks of variability from dissimilar styles of mineralisation. The study highlighted that there was no significant difference between the RC and diamond sample results. At the 50th percentile, RC samples had a value of 0.15% which compares very well with the 0.16% of the diamond samples (Figure 10-9). As the grade increases minor deviation starts to favour diamond samples. However, higher grade samples only constitute a small proportion of the overall dataset.  It is the QP's opinion that there is limited risk associated with bias between the two datasets that would adversely affect block grade estimates.

**Figure 10-9      QQ plot of NiS RC vs DD data for the Kevitsa ore body, above 0.15% NiS there starts to be a slight bias in favour of diamond data.**
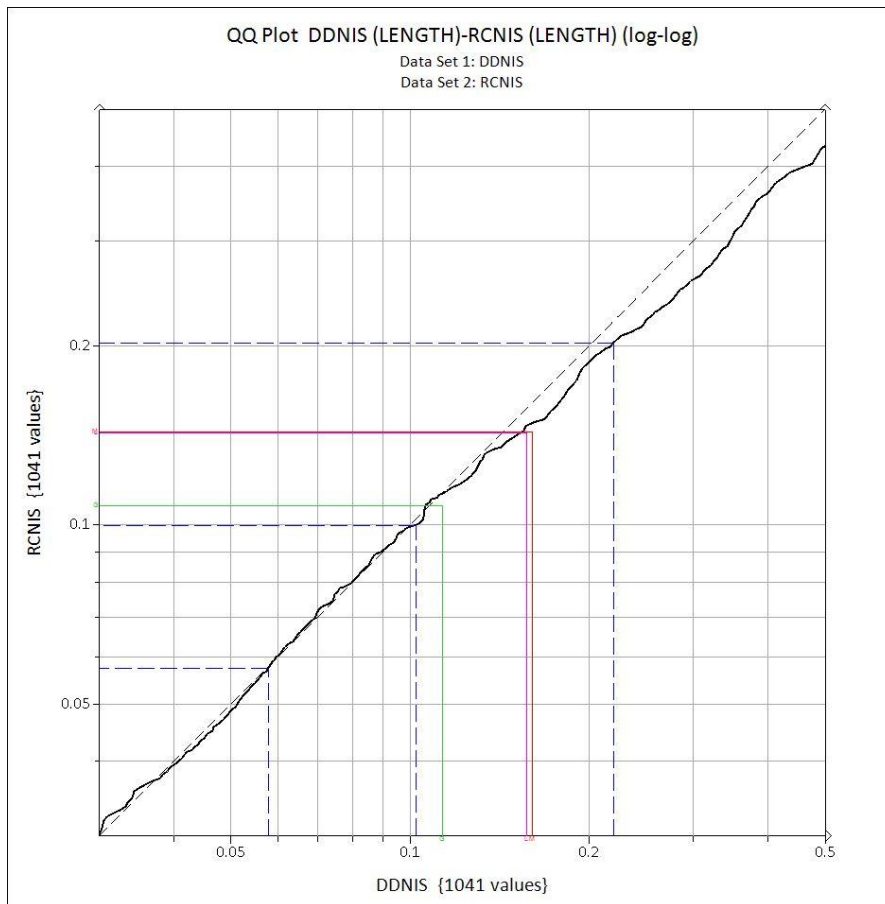
# ITEM 11  **SAMPLE PREPARATION, ANALYSES AND SECURITY**

Sample preparation and analysis has good evidence of been 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 and RC drilling by reputable companies and suitably trained persons.  First Quantum Minerals has practised Quality Assurance and Quality Control (QAQC) for the duration of their diamond drilling and RC grade control drilling.

Labtium and GTK laboratories (local) and OMAC, ALS (international) laboratories were used for sample preparation and analysis, with results electronically uploaded into a secure database system. Most samples were prepared and analysed in Finland, with check samples and exploration samples sent overseas. The primary laboratory used by FQM was Labtium, which has International Standard ISO/IEC 17025:2005 accreditation. Regular laboratory visits and audits were completed by the geological team from Kevitsa mine since 2009.

In December 2010, all geological data held by Kevitsa mine was migrated from a Microsoft Access database to an SQL database with a DataShed front end, bringing Kevitsa in line with the FQM group database standards. Regional exploration data, outside the remit of this report, is stored in a separate database maintained by FinnEX. There are links between the two databases to allow for collective viewing of both datasets at the same time. Table 11-1 below summarises the main components of the Kevitsa DataShed database.

**Table 11-1**      **Details of the Kevitsa Mine Geology SQL database (DSKevitsaGC) accessed through DataShed**

| Database | Details |
|---|---|
| Name | DSKevitsaGC |
| Format | The data is stored in an SQL database accessed by most users through a DataShed front end. |
| Front end | Data is loaded, viewed and exported through a DataShed front end. <br> Core content and format of database tables, and some views, are controlled at Global level to ensure consistency across FQM sites. <br> Site based modifications to these, additional and combinations of data are done in views, which link to source tables. <br> Data is loaded using locked templates and predefined import layouts. Modifications are strictly controlled. |
| Back end | The SQL database can be accessed through SQL Server Management Studio (SSMS). Only one Geologist on site and selected senior Database Administrators (DBAs) have access to the back end. |
| Location | Kevitsa Server based on site. |
| Security / Access | Access is controlled through user groups, with different user groups having specific read / write / edit access. Each user is assigned to a group based on their role and requirements. DataShed and Windows user groups are reviewed monthly and updated as required. DataShed user groups are controlled through ConfigManager, access to which is limited. |
| Backups | Back up to tape is done nightly (incremental), weekly, monthly and annual (full). Once a tape reaches the monthly or longer storage state it is taken off site for long term storage. |
| Day to day | Kevitsa does not have a full time DBA. Data entry is managed and validated by the |

| management | Senior Geologists who have experience in database management; all geologists have rights to load their own data. |
|---|---|
| Support | Support for any design based structural change or modification to the database comes from consultants working who work closely with the Global Data Manager in Perth. |

The QP, Mr David Gray is not aware of any data losses and has investigated and verified the sample preparation, sample storage and security practices during recent site visits and has deemed results to be adequate for the use in this Mineral Resource estimate. The analytical techniques employed by the respective laboratories for analyses of the prepared samples were similarly deemed adequate for the purposes of this Mineral Resource estimate. Database data, QAQC results, geostatistical analysis and comparison of different generations of sample data highlighted very few risks to data quality or its representative nature. The sample data was believed representative of the prevailing styles of mineralisation and therefore suitable for use in this Mineral Resource estimate update.

## 11.1    Sample preparation

### 11.1.1  Diamond drilled core

Core from all campaigns was logged and marked with sample intervals and photographed before the core was split and divided into the pre-defined sample intervals. Both GTK and SGL applied systematic 2 m sampling downhole. FQML also 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 test work. Most of the core drilled by GTK is at the Finnish national core warehouse at Loppi. 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 and FQML drilling are stored on site. The Sample Handling Foreman maintains a map with the location of each drill hole and the corresponding coarse and pulp reject, stored per batch number.

Sample preparation for both the GTK and SGL drilling campaigns was done by the Geological Survey of Finland. The core was cut using a diamond saw and bagged, weighing approximately 4 kg. Samples were crushed to 90% passing 2 mm and riffle split to 150 g. This material was then milled to 90% passing 100μm. Pulp material was sent to GTK laboratory in Rovaniemi for analysis.

Core drilled by FQM was cut by either employees or a subcontractor (GTK). Half 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 90 individual samples, inclusive of QC samples. QC samples included blanks, 3 commercial standards and quarter core duplicates. Sample lists sent to the preparation laboratory included details of which samples should have a coarse duplicate prepared after crushing. Once two batches of samples were ready for analysis samples were despatched to the sample preparation facilities at the relevant laboratory. Chain of custody forms were sent with the samples and a copy retained on site for reference. Half core samples were then prepared by the receiving

laboratory. The majority of samples drilled by the Kevitsa mine were prepared and analysed by Labtium Laboratories in Rovaniemi. Labtium prepared the samples by drying (method 10), crushing (method 30), splitting (method 35) and grinding (method 50). Samples are 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. The pulp was split into sub-samples for the various analytical techniques; one sub-sample was returned with the remaining coarse reject to Kevitsa. At the end of 2014 Labtium in Rovaniemi was closed down and equipment and expertise was split between the companies other laboratories. Since this time, no diamond core has been submitted for analysis by the Kevitsa mine team. A second laboratory, OMAC Laboratories Ltd (Alex Stewart Group Geochemical & Assays), Ireland, was used briefly in 2009 for a limited number of primary assay results. The sample preparation and analysis techniques were comparable with those used at Labtium laboratories. OMAC laboratories has ISO/IEC 17015 accreditation. Personnel from Kevitsa made a laboratory visit and audit in 2009. 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 FinnEx were logged on site by exploration geologists according to the established FQM standards. FinnEx geology technicians cut the core on site after which half core samples were sent for further preparation to ALS Minerals' laboratory in Outokumpu, Finland. In Outokumpu the 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 microns (lab code PREP-31). Each core sample batch included blank and standard samples inserted in the sequence by FinnEx technicians. The blank samples were "silica gravel" (crushed quartzite) while the standards were OREAS' commercial CRM products OREAS 14P and OREAS 13b. These were inserted in the sample batches in random order so that each batch contained 2-3 blanks and at least one standard of both types. In addition, every batch had 1-2 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.

## 11.2    Reverse Circulation samples

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 – 90 samples. Samples were dried at 110 degrees 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 QAQC samples were inserted by the Sampling Supervisors.  Two commercial standards and two blank samples were inserted per batch. Coarse duplicates (3 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 2kg 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.

## 11.3     Sample analysis

Apart from drilling conducted by FinnEx all diamond drilling pulp samples have used the same digest method for total Nickel (Ni) and Copper (Cu); Aqua Regia. The main drilling program completed by FQM used an Aqua Regia digest followed by Inductively Coupled Plasma (ICP-AES) analysis. Additional element analysis included cobalt, chromium, iron, manganese, lead and sulphur.

Nickel and copper sulphide results were available for a subset of the SGL drilling and all of the FQM drilling. This method was introduced to give a more accurate analysis of Ni in sulphides as opposed to Ni in silicates. Ni in silicates would not be liberated during metallurgical processing. Labtium method 240P is an ammonium citrate hydrogen peroxide leach with ICP-AES finish and is comparable to bromine-methanol leach method. Labtium and OMAC laboratories used this method of analysis of Ni and Cu in sulphide.

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

After preparation at ALS Outokumpu, FinnEx samples were sent to ALS Loughrea, Ireland, and consisted of near-total leach (four acid) multi-element ICP-MS method (lab code ME-MS61) as well as lead fire assay with ICP-AES finish (lab 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 (lab code ME-ICP09). All ALS Minerals' laboratories have been accredited to the ISO 17025 standard and so are the above mentioned assay methods. The Irish laboratory is also an INAB accredited testing laboratory (Reg. No. 173T).

All RC drilling samples have been routinely analysed at Labtium Sodankylä laboratory for total and sulphide nickel and copper plus gold, platinum and palladium. Between 2010 and 2012 total Ni and Cu results were assessed using an Aqua Regia digest followed by an ICP-AES finish, Labtium method 510P.  In 2012 the decision was made to change the total Ni and Cu analysis to XRF, Labtium method 195X. This method also provided results for calcium, chromium, iron, magnesium and sulphur. Labtium were using a bench top XRF and a loose powder sample. NiS and CuS was consistently analysed using method 240P, an ammonium citrate-H2O2 leach.  Gold, platinum and palladium was analysed from a 25 g sample using lead collection fire assay with ICP-OES finish, Labtium method 704P.

A summary of methods used for the respective campaigns is detailed in Table 11-2 below.

**Table 11-2        Summary of the analytical methods per drilling campaign.**

| Campaign | Primary Laboratory | Aqua Regia Total Ni, Cu[3] | XRF Total Ni, Cu, S | Selective Leach[1] Sulphide Ni, Cu, Co | Multi element Ni, Cu etc | Fire Assay[2] Au, Pt, Pd |
|---|---|---|---|---|---|---|
| Diamond drilling | | | | | | |
| GTK | GTK | ✓ | | | | ✓ |
| SGL | GTK, Labtium[4] | ✓ | | ✓ | | ✓ |
| KMOY | Labtium Rovaniemi | ✓ | | ✓ | | ✓ |
| FXOY | ALS Loughrea | | | | ✓ | ✓ |
| RC drilling | | | | | | |
| KMOY | Labtium Sodankyla | ✓ | ✓ | ✓ | | ✓ |

A total of 329 holes within the resource area have density data. This has been collected by a conventional gravimetric (Archimedes) method as well as with down hole geophysics methods.

- Gravimetric method (database table DH Specific Gravity) = 254 holes
- Down hole Geophysics method (database table DH Geophysics) = 30 holes
- Both methods = 45 holes

The previous TR resource estimate used assigned average density values from the respective mineralised zones. This 2016 estimate has used the available density data in an Inverse Distance Weighting (IDW) estimation method. Where there was insufficient data to estimate, an average grade from the whole dataset was assigned. Figure 11-1 below shows the spread of available data across the resource area coloured by methodology.

The Archimedes method (found in DH Specific Gravity), whole core was weighed in air and then 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 done on 10 cm intervals representing a 5 m down hole length.

The down hole petrophysics dataset (DH Geophysics) 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 drill holes 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.

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[1] Ammonium Citrate–H2O2-leach; Labtium method 240P. CoS only analysed for in diamond drilling.
[2] The majority of samples were analysed using lead collection fire assay
[3] Full set of elements analysed; Ag, As, Cd, Co, Cr, Cu, Fe, Mn, Mo, Ni, Pb, Sb, S
[4] SGL switched from using GTK Rovaniemi to using Labtium Rovaniemi Laboratory in September 2007. Some of the drill holes were submitted for analysis by FQML after acquiring SGL in 2008.

**Figure 11-1       Drill holes with density readings within the Resource area, coloured based on the method and corresponding table in the database.**
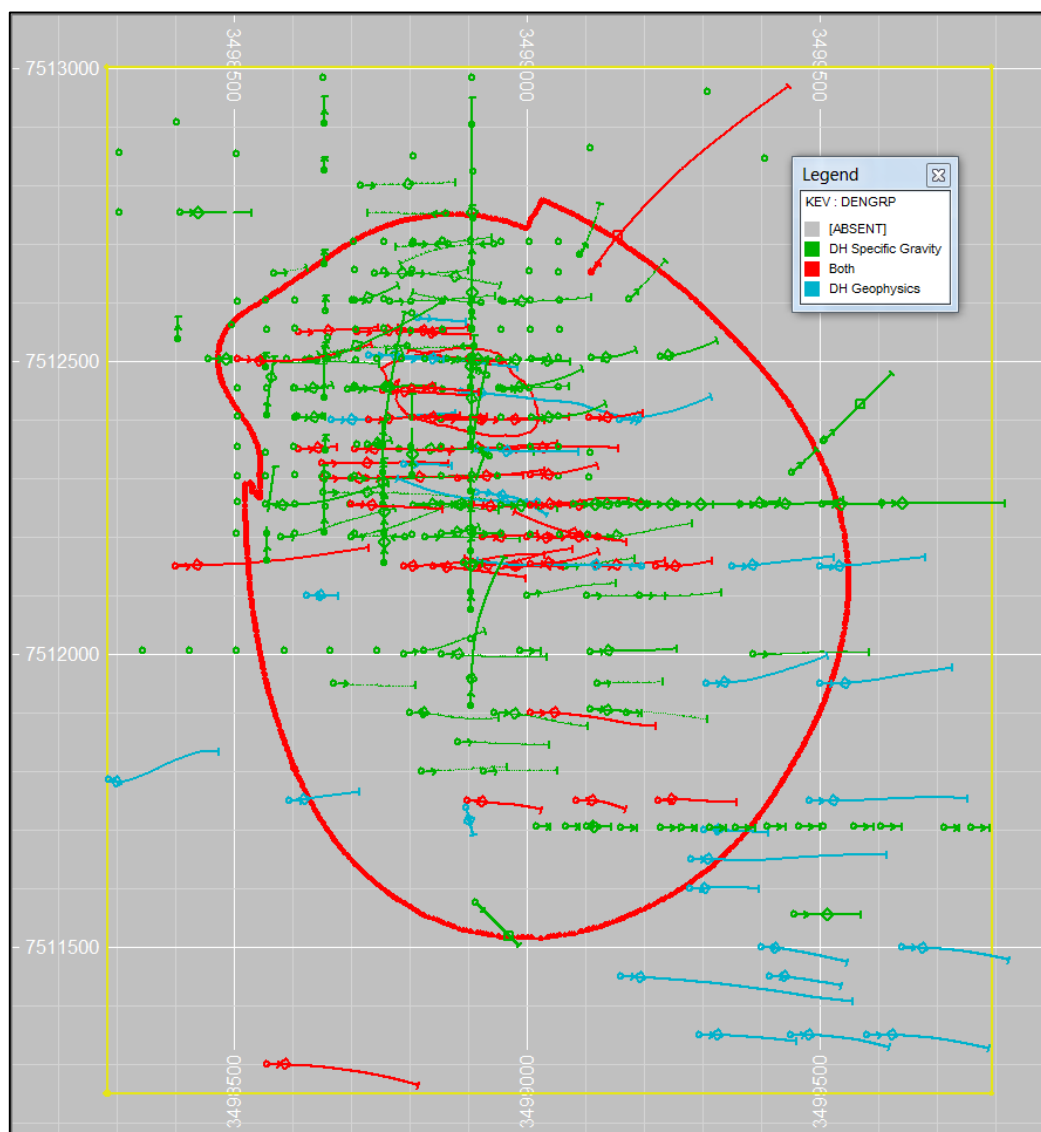


## 11.4    Quality Assurance and Quality Control

FQM applied systematic QAQC practices on all diamond and grade control RC drillhole sampling. Historically, there was no documentation of QAQC work done by GTK. SGL introduced QAQC 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 consultants 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 of these verification programs raised any concerns pertaining to the quality of the data used for this Mineral Resource Estimate.

FQM applied a comprehensive QAQC program throughout its diamond drilling campaign. QAQC samples include certified reference material (CRM), blanks and duplicates randomly inserted within a batch.  Within a batch of 90 samples, 9% were QC samples; three standards (CRMs), three

duplicate samples (1/4 core duplicate, coarse crush duplicate and a pulp duplicate) and two blank samples. QAQC data was actively monitored for failure with re-assaying of samples requested when results fell outside of acceptable limits. Umpire pulp check samples (approximately 1%) were also sent to an independent laboratory as an umpire program designed to confirm the precision of the primary laboratory. ALS Chemex laboratory in Perth was used as the umpire laboratory.

Similarly, the RC grade control program had routine QAQC samples inserted every batch (80-90 samples); 3 field duplicates (FDP), 2 coarse duplicates (CDP), 2 blanks and 2 commercial standards. Monthly QAQC reports were compiled by the mine geology team in order to control results. Failures were discussed directly with Labtium laboratory management and where required were re-analysed. Labtium also produced a monthly QAQC report summarising their performance according to their internal QC samples.

Across the campaigns, a number of different commercial standards have been used to assess analytical accuracy. These were chosen to have a similar matrix to the Kevitsa mineralisation and rock types and are detailed in Table 11-3 and Table 11-4

The inserted CRM results were investigated using standard control charts and results were compared to the certified values and deviations. Results for three of the standards are presented in Figure 11-2, Figure 11-3 and Figure 11-4, and demonstrate that the primary laboratory accuracy was well within two standard deviations of the accepted values for Ni and Cu. The primary laboratory accuracy was deemed appropriate for using the sample analysis results in this Mineral Resource estimate.

**Table 11-3      Summary of the Certified Reference Material (CRM) / Standards used during the QAQC programs at Kevitsa.**

| CRM | Kevitsa DB alias | Program usage | Ni Aqua Regia (ppm) | Cu Aqua Regia (%) | Ni XRF (ppm) | Cu XRF (%) |
|-----|-----|-----|-----|-----|-----|-----|
| AMIS0056 | KEV-7 | SGL | 1940 | 0.1401 | | |
| AMIS0124 | KEV-1 | KMOY (DD) | 1917 | 0.1324 | | |
| AMIS0060 | KEV-2 | KMOY (DD) | 2909 | 0.3308 | | |
| AMIS0073 | KEV-3 | KMOY (DD, RC), FXOY | 5316 | 0.237 | | |
| AMIS0107 | KEV-A | KMOY (DD, RC), FXOY | 701 | 0.0423 | | |
| AMIS0127 | KEV-low | KMOY (DD, RC) | 1366 | 0.0777 | | |
| AMIS0093 | KEV-med | KMOY (DD, RC), FXOY | 2596 | 0.2859 | | |
| AMIS0061 | KEV–high | KMOY (DD) | 34950 | 1.278 | | |
| AMIS0099 | KEV-99 | KMOY (DD, RC) | 362 | 0.025 | 427 | |
| AMIS0170 | KEV-170 | KMOY (DD, RC) | 1033 | 0.0705 | 1099 | 0.069 |
| AMIS0316 | KEV-316 | KMOY RC | 5531 | 0.2093 | 5949 | 0.2097 |
| AMIS0318 | KEV-318 | KMOY RC | 1529 | 0.1552 | 1653 | 0.1495 |

The inserted CRM results were investigated using standard control charts and results were compared to the certified values and deviations. Results for three of the standards are presented in Figure 11-2, Figure 11-3 and Figure 11-4, and demonstrate that the primary laboratory accuracy was well within two standard deviations of the accepted values for Ni and Cu. The primary laboratory

accuracy was deemed appropriate for using the sample analysis results in this Mineral Resource estimate.

**Table 11-4**     **Summary of the Certified Reference Material (CRM) / Standards used during the QAQC programs at Kevitsa per year and per campaign.**

| Program | 2007-2008 | 2009 | 2010 | 2011 | 2012 | 2013 | 2014 | 2015 |
|---------|-----------|------|------|------|------|------|------|------|
| SGL_DD | KEV-7 | | | | | | | |
| KMOY_DD | KEV-1 KEV-2 KEV-3 | KEV-1 KEV-2 KEV-3 | KEV-2 KEV-low KEV-Med | KEV-3 KEV-A KEV-Med | KEV-3 KEV-A KEV-Med KEV-99 KEV-170 | KEV-99 KEV-170 | KEV-99 KEV-170 | |
| KMOY_RC | | | KEV-Low KEV-Med | KEV-3 KEV-Med KEV-A | KEV-Med KEV-A KEV-99[5] KEV-170 | KEV-99 KEV-170 | KEV-99 KEV-170 KEV-316 KEV-318 | KEV-316 KEV-318 |
| FXOY_DD | | | | | KEV-3 KEV-A KEV-Med | | FX-1 FX-2 | FX-1 FX-2 |

**Figure 11-2**     **Results for AMIS0093 (KEV-med) total Ni by Aqua Regia for diamond drilling samples.**



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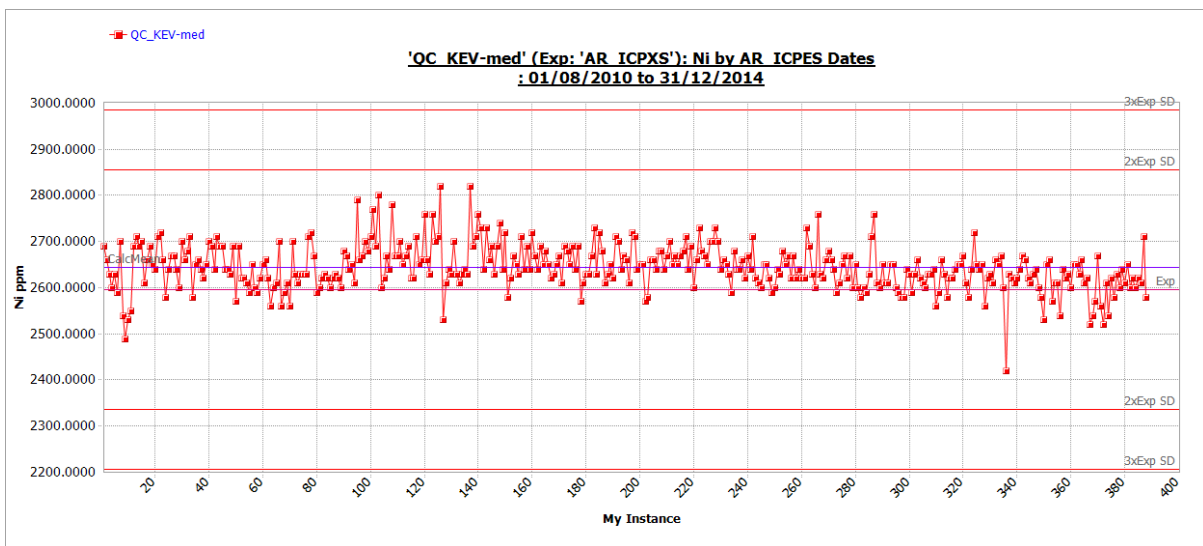
[5] Standards used when the total Ni and Cu method changed to XRF

**Figure 11-3        Results for AMIS0073 (KEV-3) total Cu by Aqua Regia for diamond drilling sample.**
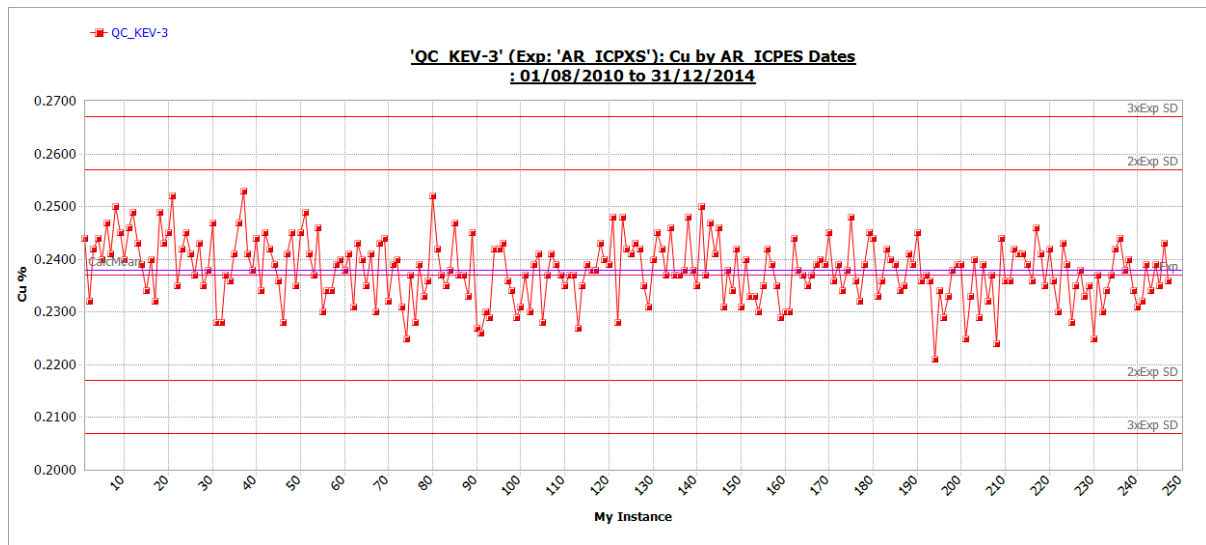


**Figure 11-4        Results for AMIS0099 total Ni by Aqua Regia – main use in 2012 for diamond and RC drilling.**
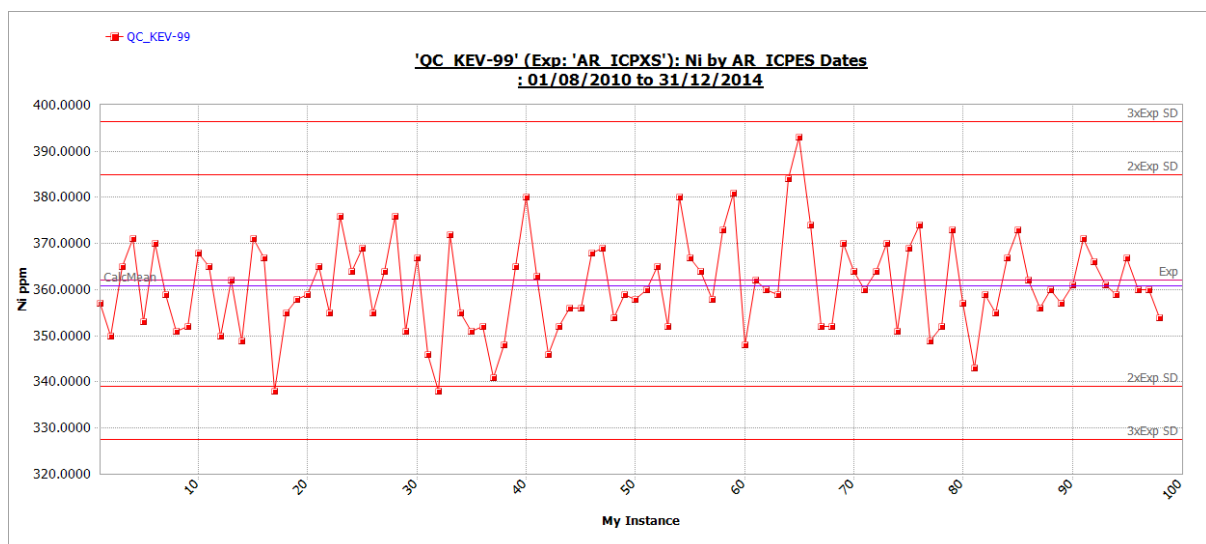


In 2012 the RC analytical method was changed from Aqua Regia to bench top XRF (Labtium method 195X) for the analysis of total nickel and copper. It is noted that the detection limit on XRF is higher than Aqua Regia; 0.01% for Ni and 0.02% for Cu. As such, in 2015, FQM completed a comparison of methods for total Ni and Cu. XRF was compared to Peroxide Fusion and Aqua Regia. The XRF results were comparable with Aqua Regia and were deemed suitable for grade control purposes. Standards inserted during the RC XRF sample analysis did not raise any concerns with regards analytical accuracy (Figure 11-5).

**Figure 11-5        Results for AMIS0170 total Ni by XRF used mainly in 2012 and 2014 in the RC grade control drilling.**
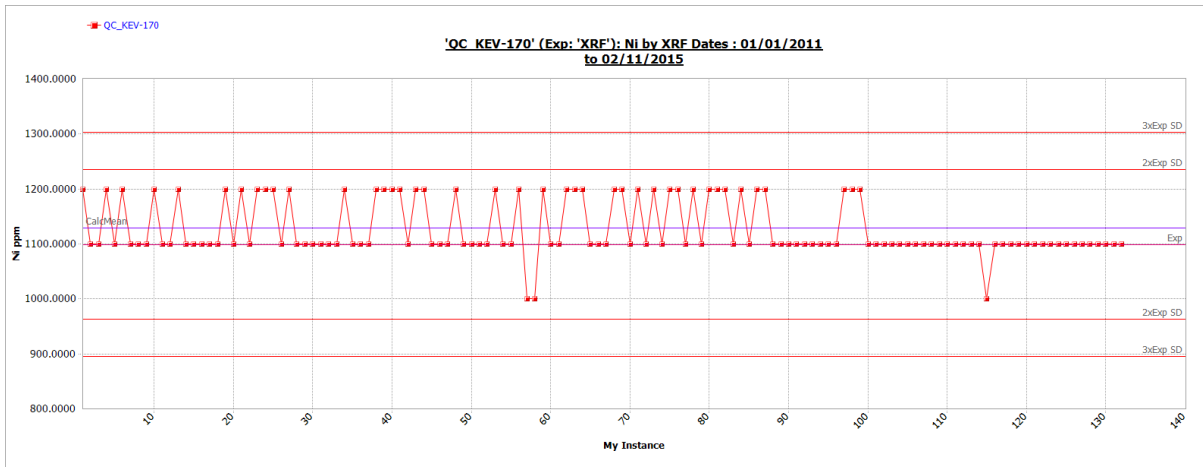


There were no certified standards available for the Ni and Cu sulphide analysis using ammonium citrate–H2O2-leach (a0.15g charge weight) and the ICP-OES analysis (240P). In order to support analytical accuracy for the 240P method, Labtium determined its own upper and lower limits for the standards used. The limits have been developed over time and were based upon a large dataset of results. These were the best measure of accuracy for this analytical method. AMIS0192 was used by Labtium from mid-2013 to late 2015, the results for NiS and CuS are presented in Figure 11-6 and Figure 11-7.

**Figure 11-6        Results for AMIS0192 NiS by selective leach for RC grade control samples**
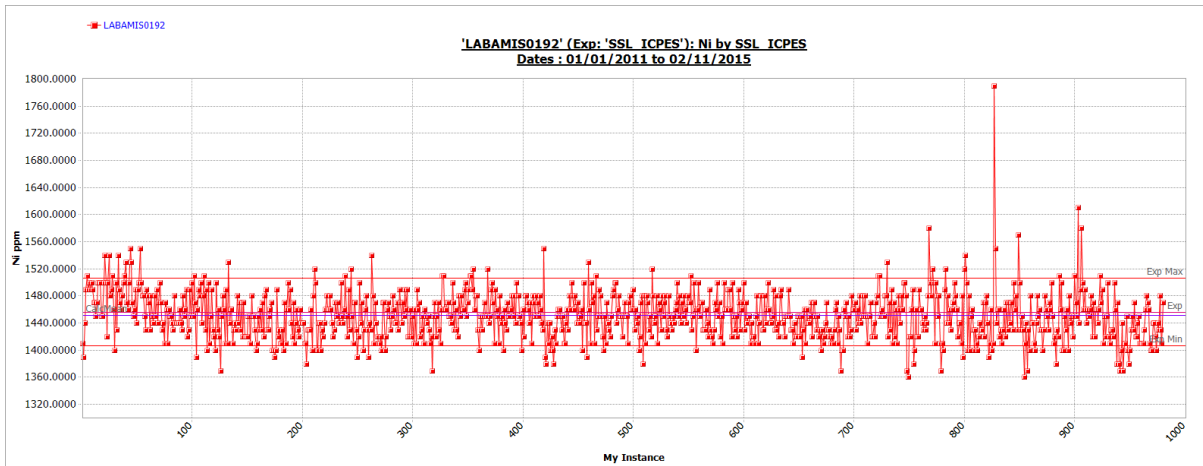
**Figure 11-7        Results for AMIS0192 CuS by selective leach for RC grade control samples.**
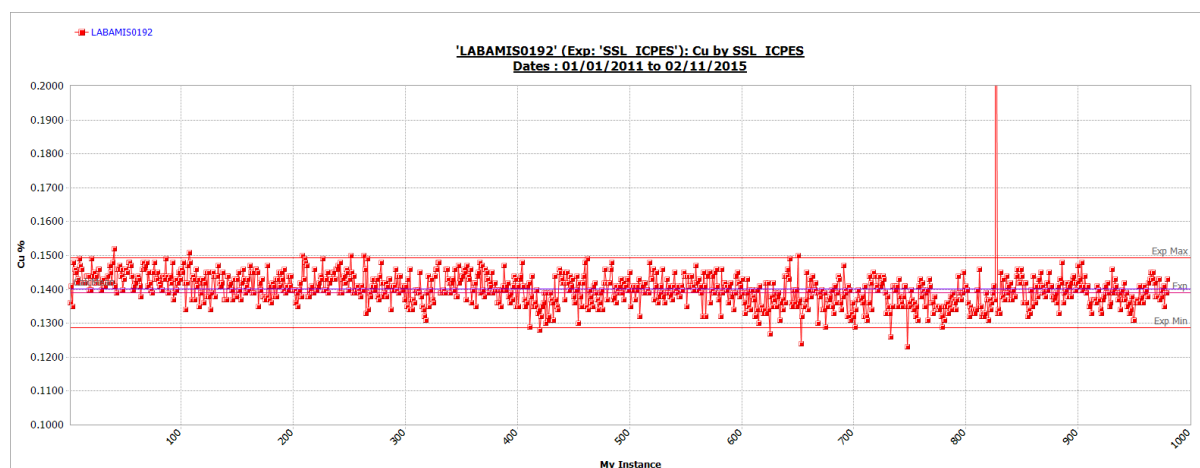


The analysis of blank material shows no major concern regarding the control of contamination in the sample preparation process. The mean value of all results was well below the grade of mineralisation. Minor outliers were routinely investigated. Figure 11-8 shows an example of the blank analysis for Ni by Aqua Regia for the diamond drilling.

**Figure 11-8        Results for blank material samples analysed for total Ni by Aqua Regia.**
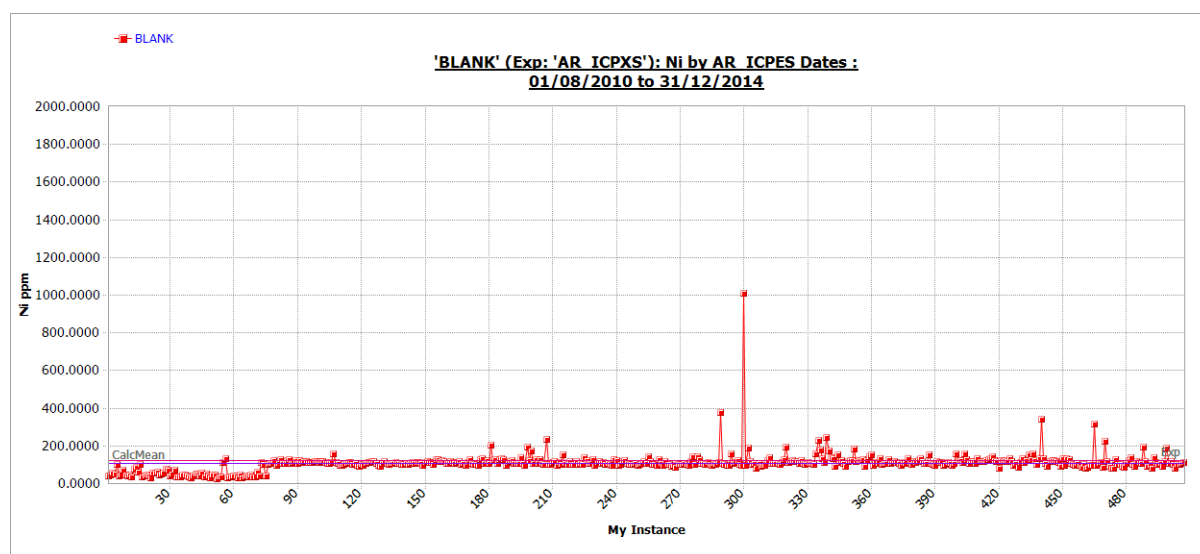


Duplicate samples were inserted in order to assess precision associated with sample preparation and include core, coarse crush, pulp and laboratory duplicates. The graphs below (Figure 11-9 to Figure 11-12) show the results of the duplicate samples inserted into the batches of diamond drilling samples. The results highlight good precision for both nickel and copper for all duplicate methods. The results were comparable with the RC sample results having approximately 98% of data with a MAPD% value below 10% for Ni and approximately 96% of data with a MAPD% value below 10% for Cu.

**Figure 11-9**       **MAPD% plot showing results for total Nickel diamond drilling duplicate samples by Aqua Regia.**
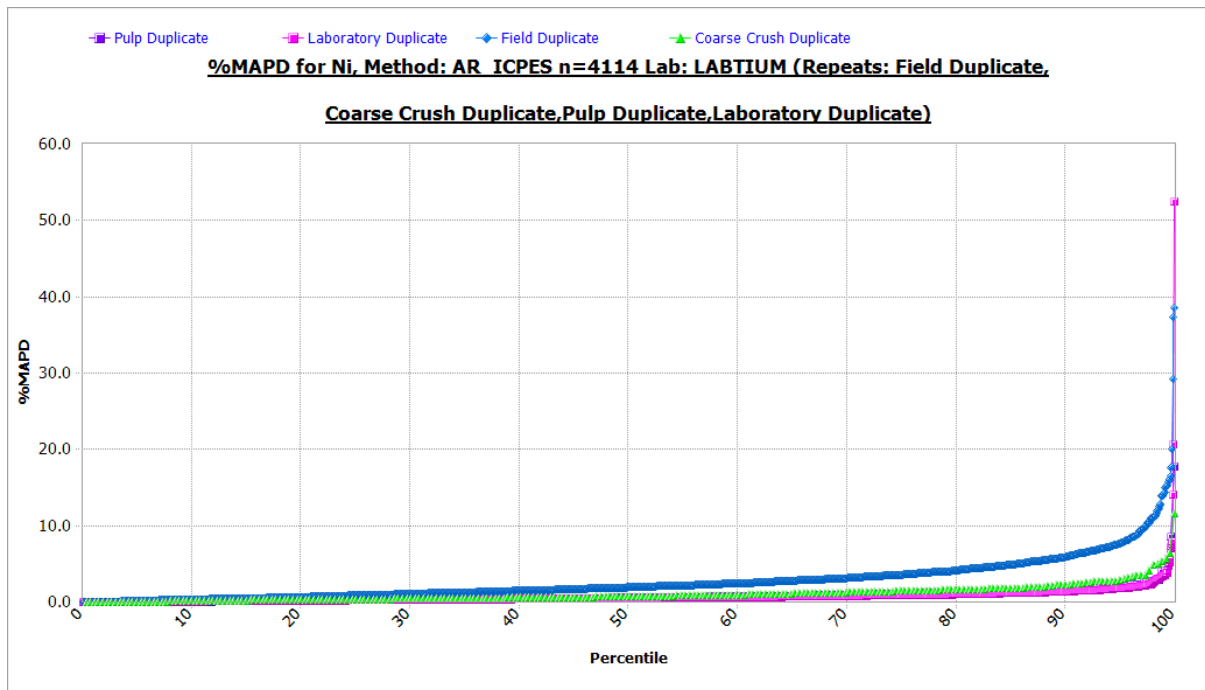


**Figure 11-10**       **MAPD% plot showing results for total Copper diamond drilling duplicate samples by Aqua Regia.**
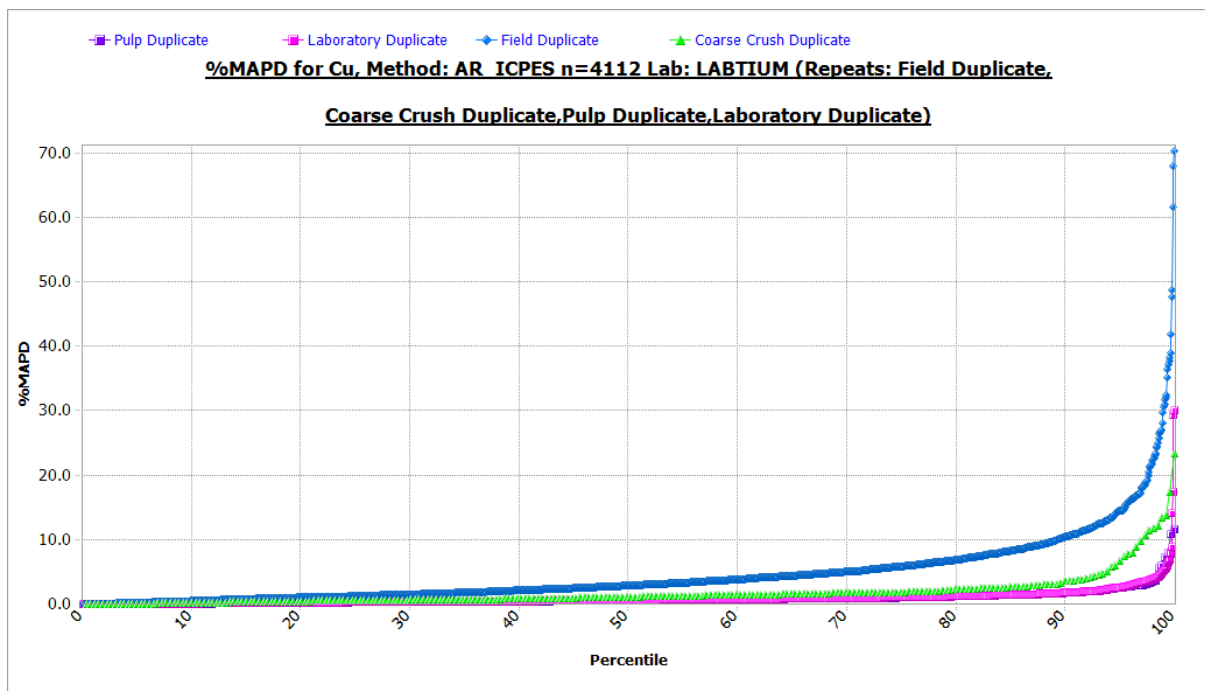
**Figure 11-11**     **MAPD% plot showing results for Nickel sulphide diamond drilling duplicate samples by Ammonium Citrate–H2O2-leach.**
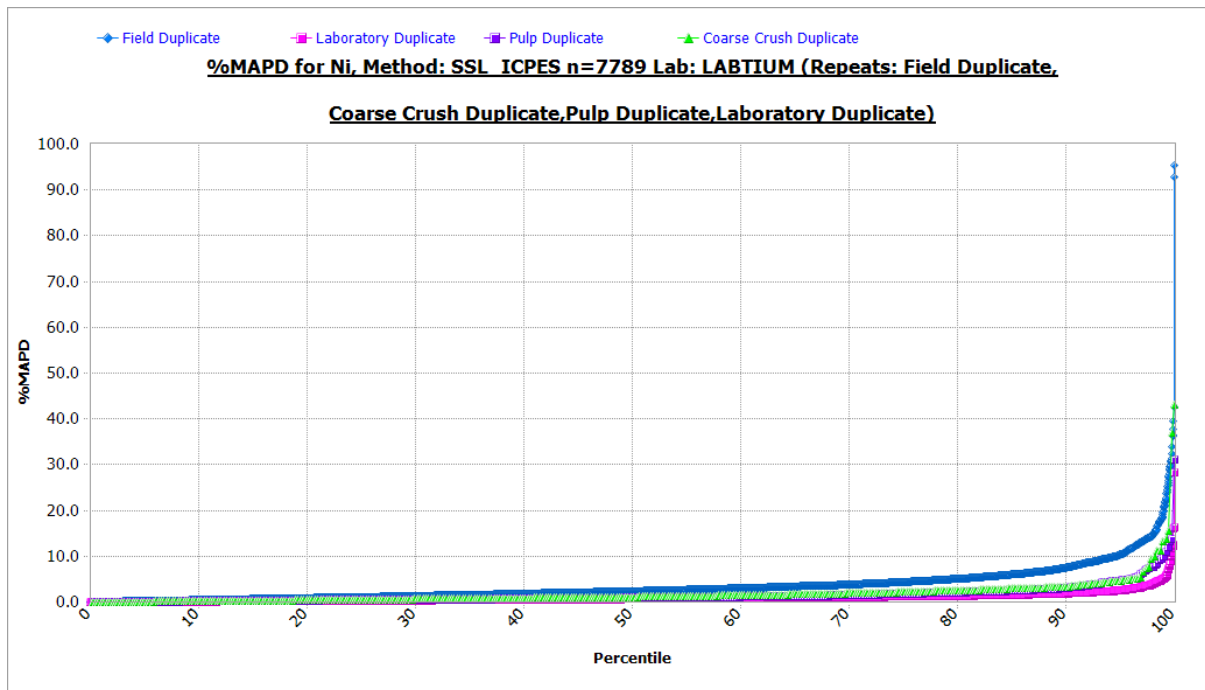


**Figure 11-12**     **MAPD% plot showing results for Copper sulphide diamond drilling duplicate samples by Ammonium Citrate–H2O2-leach.**
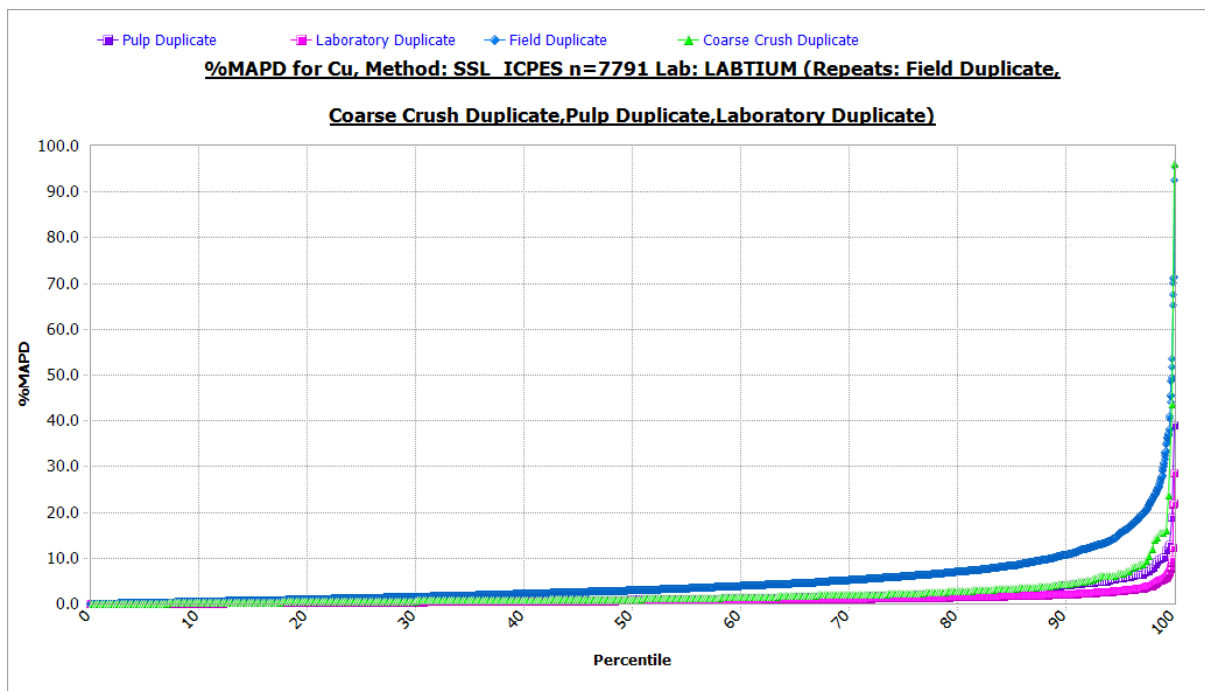


## 11.5    Conclusions

Numerous programmes of QAQC sampling have been undertaken at Kevitsa by previous owners and FQM.  Whilst a systemised programme of QAQC sampling was not fully implemented until FQM, programmes of check analysis were undertaken to verify historic drilling completed by previous owners. FQM is currently importing and validating all Kevitsa diamond and RC drillhole data into a

corporate SQL database where QAQC results are analysed using DataShed software. Errors identified during QAQC reviews, were investigated and corrected with re-assaying or database corrections.

The nickel and copper QAQC results indicate that:

- the 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 degrees of precision
- coarse crush duplicates display low bias and high degrees of precision
- umpire check samples display low bias and good degrees of precision
- twinned drillholes display correlations between assays which are considered acceptable.

It is considered that the QAQC results reviewed for this Technical Report indicate that the Kevitsa deposit's drillhole assays are suitable for Mineral Resource estimation.

# ITEM 12  **DATA VERIFICATION**

The Qualified Person, David Gray, has visited Kevitsa Operations on four occasions over the last two years.  During these visits, subsequent studies and this resource estimate, the QP has gained good familiarity and confidence in the available diamond and reverse circulation drillhole data, the geology models and understanding of the prevailing mineralisation.  Mr Gray believes the geological understanding and data available for this Kevitsa Mineral Resource estimate update is of good quality and is representative of the prevailing mineralisation relevant to the deposit.

Several verifications are hereby confirmed by Mr Gray.

1.  Diamond and RC drillhole collar coordinates were verified through visual observation and digital checks against database data.
2.  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.
3.  A small and random selection of original laboratory assay results was verified against those in the database.
4.  QAQC data was investigated together with the process used for analysis and were verified as robust for assuring assay accuracy, precision and controlling contamination.
5.  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.
6.  Mining and run of mine stockpiling of mineralised material was verified through visual checks, grade control and reconciliation processes.
7.  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. It is the Qualified Person's opinion that the data used for this Mineral Resource estimate update is adequate for the purposes of this report.

# ITEM 13 MINERAL PROCESSING AND METALLURGICAL TESTING

The following information is reproduced in part from the January 2011 Technical Report (CSA and DumpSolver, 2011), with an update provided by the site team and reviewed by Andrew Briggs (QP).

## 13.1 Overview

The mineral processing facilities at Kevitsa have undergone several modifications and an expansion since commissioning in 2012. The details of the modifications are summarised in Item 17.1 and the current capacity of the Kevitsa processing plant is 9.0 Mtpa.

## 13.2 Metallurgical Testing Programs

Historical test work in the 1990´s and early 2000´s indicated that by flotation a bulk sulphide concentrate containing copper (Cu) and nickel (Ni) could be produced successfully.

The grades of the bulk concentrate produced during these metallurgical studies did not meet the requirements for downstream processing and the test-work for producing separate saleable concentrates of copper and nickel was not successful.

From 2004 to 2009 metallurgical testing was carried out at the laboratories of The Geological Survey of Finland (GTK - formerly VTT) in Outokumpu, Finland with the focus being on developing a flotation process to produce separate smelter-grade copper and nickel concentrates. This work was carried out at bench scale and in a pilot plant campaigns.

Results for the three phases of piloting performed in 2006/07 are presented in Table 13-1 and Table 13-2.

**Table 13-1        Copper Concentrate Grades and Recoveries for the Pilot Plant Campaigns**

| Test-work | Phase | 1 | 2 | 3 |
|---|---|---|---|---|
| Mass | % Weight | 1.42 | 1.44 | 1.18 |
| Cu Grade | % Cu | 28.48 | 30.50 | 29.8 |
| Cu Recovery | % | 78.6 | 80.3 | 77.3 |
| Ni Grade | % Ni | 0.810 | 0.504 | 0.612 |
| Ni Recovery | % | 3.0 | 1.8 | 2.0 |
| Au Grade | g/t Au | 4.37 | 4.41 | 3.92 |
| Au Recovery | % | 37.8 | 39.3 | 25.0 |
| Pt Grade | g/t Pt | 3.91 | 2.50 | 2.96 |
| Pt Recovery | % | 8.1 | 10.0 | 10.7 |
| Pd Grade | g/t Pd | 4.42 | 3.17 | 3.19 |
| Pd Recovery | % | 16.0 | 13.3 | 11.6 |
| S Grade | % | 30.1 | | 34.1 |
| Fe Grade | % | 30.6 | | 30.9 |
| MgO Grade | % | 2.8 | | 2.3 |

**Table 13-2        Nickel Concentrate Grades and Recoveries for the Pilot Plant Campaigns**

| Test-work | Phase | 1 | 2 | 3 |
|---|---|---|---|---|
| Mass | % Weight | 3.21 | 4.21 | 1.97 |
| Cu Grade | % Cu | 1.45 | 1.25 | 1.85 |
| Cu Recovery | % | 9.5 | 12.3 | 11.2 |
| Ni Grade | % Ni | 8.11 | 6.9 | 12.16 |
| Ni Recovery | % | 63.2 | 70.5 | 65.3 |
| Au Grade | g/t Au | 14.2 | 21.9 | 15.0 |
| Au Recovery | % | 0.98 | 0.85 | 1.56 |
| Pt Grade | g/t Pt | 5.06 | 4.16 | 10.69 |
| Pt Recovery | % | 45.1 | 47.9 | 58.1 |
| Pd Grade | g/t Pd | 4.44 | 4.20 | 8.04 |
| Pd Recovery | % | 41.0 | 50.9 | 42.8 |
| S Grade | % | 26.9 | | 35.8 |
| Fe Grade | % | 40.3 | 44.0 | 44.3 |
| MgO Grade | % | 7.9 | 6.5 | 2.7 |
| Fe/MgO | Ratio | 5.5 | 6.8 | 16.3 |

The autogenous (AG) and pebble milling circuit, and dewatering of concentrates was demonstrated during the pilot scale testing, providing data for the process designs.

Only minimal metallurgical test programs have been done since 2012 however numerous operational testwork programs have been run in the site laboratories. Plant recoveries are now broadly in line with the pilot plant results shown in Table 13-1 and Table 13-2 above.

## 13.3      Plant Recovery

The KMO metallurgical team predicts recoveries using a regression equation of recovery as a function of head grade, which has been derived from historical plant performance data.

The latest iteration for Ni recovery is:  $y = 5.3095\ln(x) + 73.754$

and

The latest iteration for Cu recovery is $y = 3.2305\ln(x) + 93.451$.

These equations are reviewed regularly and modified to reflect performance gains when deemed to be consistently achieved.

## 13.4      Modifications since Commissioning

### 13.4.1  Comminution circuit:

The installation in 2014 of additional crushing and milling power, in the form of a second MP800 cone crusher and a second motor on the Secondary Mill, was supplemented in 2015 by the upgrade of the Primary Screen (increased top deck aperture) and the increase of blasting energy. Combined, these allowed more reduction through blasting and crushing and a reduction in lump fraction presented to the Primary AG Mills. Overall this provided for efficiency gains on the Primary AG Mills and a 15% increase in overall milling rate at a controlled flotation feed grind (coarser).

The installed, and proven, capability to convert the Secondary Mill from pebble to ball mill duty supports the consideration of additional throughput capacity through full utilization of the installed Secondary Milling power.

### 13.4.2  Cu Flotation Circuit:

The flowsheet reconfiguration executed in Q4 2013 was supplemented by the installation, in 2014, of the Column Cell and the expansion of the depressant dosing system. Combined, these allowed for improved rejection of non-sulphide gangue and circuit control. The commissioning in 2015 of the TCe500 flotation cell (as first Cu rougher) and the Cu Regrind Mill, provided for improved liberation and selective recovery of Cu versus Ni. Overall, the Cu recovery to Cu concentrate increased by 5% and total Cu recovery by 1.5%, for marginally higher concentrate grade.

The addition of the TCe500 and column cell effectively increased Cu flotation capacity and supports the consideration of processing at higher throughput rates.

### 13.4.3  Ni Flotation Circuit:

The 2014 flowsheet reconfiguration and expansion of the depressant dosing system allowed for improved rejection of non-sulphide gangue and circuit control. The inclusion of a new reagents suite in 2015, developed for rejection of iron-sulphide gangue, benefitted the selective recovery of Ni in the Cleaner circuit. This in turn allowed utilization of flowsheet changes and acid dosing capability both installed in 2014. Overall, the Ni recovery improved by 4%, for higher concentrate grade, with reduced sensitivity to feed grade.

The reduction in circulating loads of both non-sulphide and sulphide gangues have resulted in spare Cleaner circuit capacity and supports the consideration of processing at higher throughput rates.

# ITEM 14 **MINERAL RESOURCE ESTIMATES**

## 14.1 Introduction

The 2016 Kevitsa Mineral Resource estimate was prepared in January 2016 by the Qualified Person, Mr David Gray, together with assistance from the mine geology personnel at Kevitsa Operations. Grade estimates were interpolated into a 3-dimensional (3D) geology block model using ordinary kriging and commercially available software packages (Datamine Studio version 3.0, Vulcan version 8.2.1 and Snowden Supervisor version 8.3). The project limits and coordinates were based upon the Finnish National Grid Zone 3. Most of the deposit was delineated with holes drilled at approximately 70 degrees to the west with several holes drilled as scissors to the east in order to verify and improve understanding of structure and deposit mineralisation. Drillholes were spaced at 25 to 100 m along drill lines that are approximately 50 m apart. Drillhole grid spacing increases with increasing depth below surface. Mined areas have comprehensive coverage from reverse circulation drilled holes used for grade control and short term planning.

The resource estimate has used an updated drillhole database as at 15 November 2015 which includes all drill hole sample assay results together with interpretations of the prevailing geology that relates to the structure, lithology, alteration and the spatial distribution of nickel and copper mineralisation. This update benefits from additional diamond holes as well as the RC grade control holes. Interpolation parameters were based upon the geology, styles of mineralisation, drill hole spacing and geostatistical analysis of the data. Mineral Resource estimates were classified according to geological continuity, QAQC, density data, drillhole grid spacing, grade continuity and confidence in the panel grade estimate and have been reported in accordance with the guidelines of the Australasian JORC Code (JORC, 2012), which in turn complies with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014).

## 14.2 Geological and Mineralisation model

The Kevitsa deposit is a layered ultramafic Ni-Cu-PGE style of mineralisation. Mineralisation is characterised by amorphous volumes which appear to be influenced in part by structure, alteration and intrusive layering (Figure 14-1). Controls on mineralisation are a combination of these. As such, various data sets, including a 3D seismic survey (Figure 14-2) have been utilised to improve the understanding of geology controls on domains of mineralisation and so support the estimation approach. In addition, styles of sulphide mineralisation (Ni-Cu and Ni-PGE) have been considered together with the gradational contacts between mineralised and non-mineralised material.

**Figure 14-1**      **A south-north vertical section through the centre of the Kevitsa deposit illustrating dislocated amorphous mineralisation shapes with some layering control at depth. The December 2015 pit floor is delineated in red.**
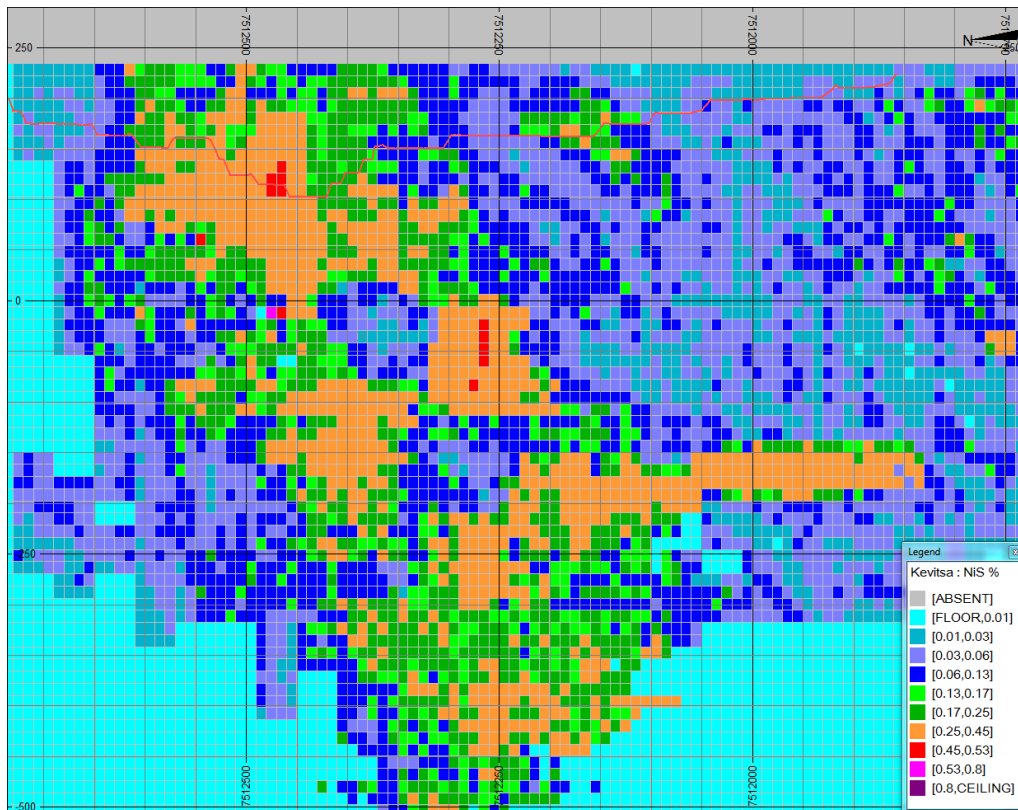


**Figure 14-2**      **An oblique 3D view of the 3D seismic data looking north west. Three north east dipping faults were interpreted and utilised to support mineralisation domain definition.**
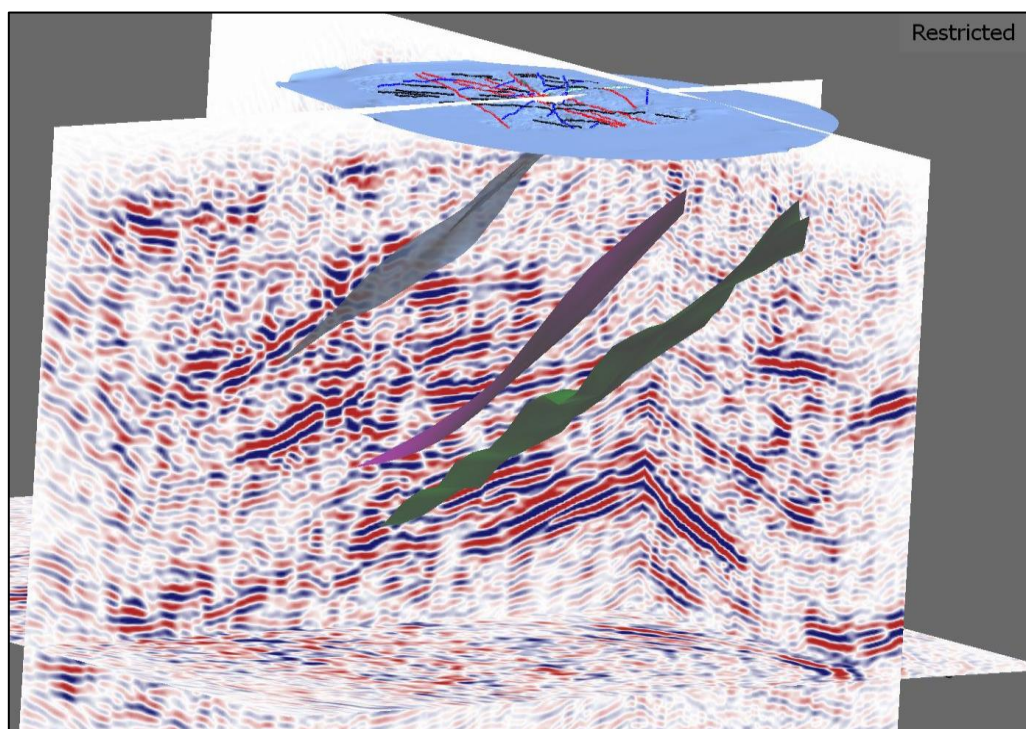
The deposit is overlain by a thin layer of soils and glacial moraine. Weathering and oxidation is minimal, with outcropping mineralisation generally exhibiting only marginal tarnishing of primary sulphides.  After removing the overlying soils and glacial moraine with mining, a bedrock survey (Figure 14-3) highlighted some of the deeper (>12 m) weathering along linear features. Overlying soils and moraine widths were, however, on average only 3-4 m thick. All near surface mineralisation has been mined out and therefore overburden width has little relevance to this Mineral Resource estimate. However, the linear weathered features have provided some confidence to the 3D seismic fault interpretation, which was used to validate the mineralisation domains selected for this estimate.

**Figure 14-3        A plan view of the surveyed bed rock surface highlighting deeper zones of linear weathering together  with drillhole collar positions.**
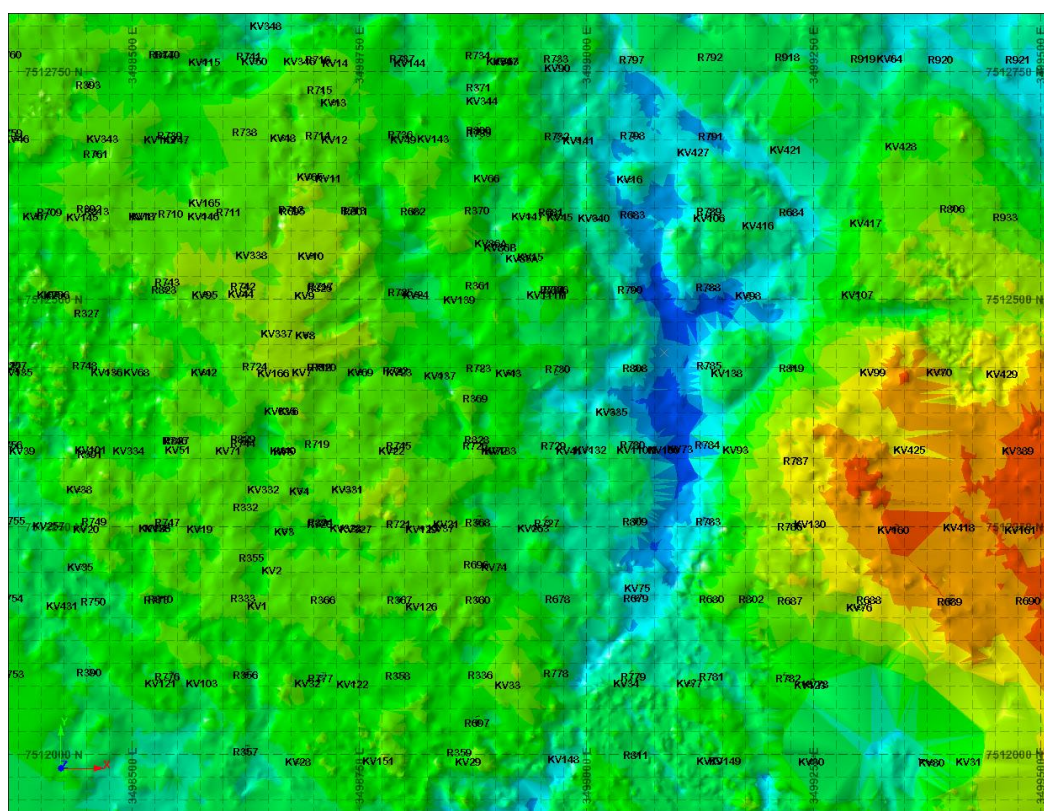


The different domains of mineralisation were guided by the prevailing geology, structure, alteration, univariate and bivariate statistics.  However, due to the complex mineralisation relationships, a more subjective approach to domaining was employed. The key elements (NiS, CuS, S and Fe) from the available drillhole sampling data were analysed using self-organising feature maps (SOFM). Self-organising feature maps are a pattern recognition technique based upon neural networks. A set of maps resulted and which allow features (elements) that tend to be distributed in a similar way to be examined. The resulting groupings (Figure 14-4) were statistically analysed and then used for defining the respective domains of mineralisation.

There was good spatial continuity of the derived domains (Figure 14-5), which together with favourable univariate statistics (reduced coefficients of variation and near normal data distributions per element and per domain) provided confidence in the respective domain's characteristics. Specifically, seven domains were identified with this method:

Kevitsa Nickel Copper Mine Technical Report                               March 2016

1. Group 1 – Low Ni, low Cu or poorly mineralised outer domain
2. Group 2 – Non-mineralised material
3. Group 3 – Low Ni, moderate Cu located within Group 1
4. Group 4 – Moderate Ni and high Cu located within Group 3
5. Group 5 – Low Cu, low Ni, but with elevated sulphides – possibly late stage intrusives
6. Group 6 – High Ni, moderate Cu with elevated PGE and apparent structural control
7. Group 7 – Moderate to low Cu and Ni, but with elevated Fe and S (pyrrhotite).

**Figure 14-4    Resulting SOFM for NiS, CuS, S and Fe illustrating domains where elements were distributed in a similar way.**
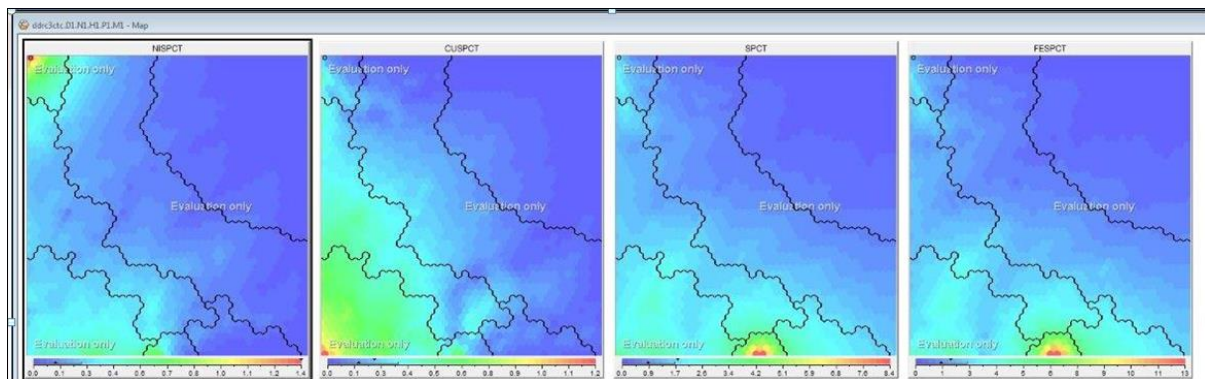


**Figure 14-5    An oblique north west 3D view of drillhole data coded according to the SOFM domains and illustrating good spatial continuity.**
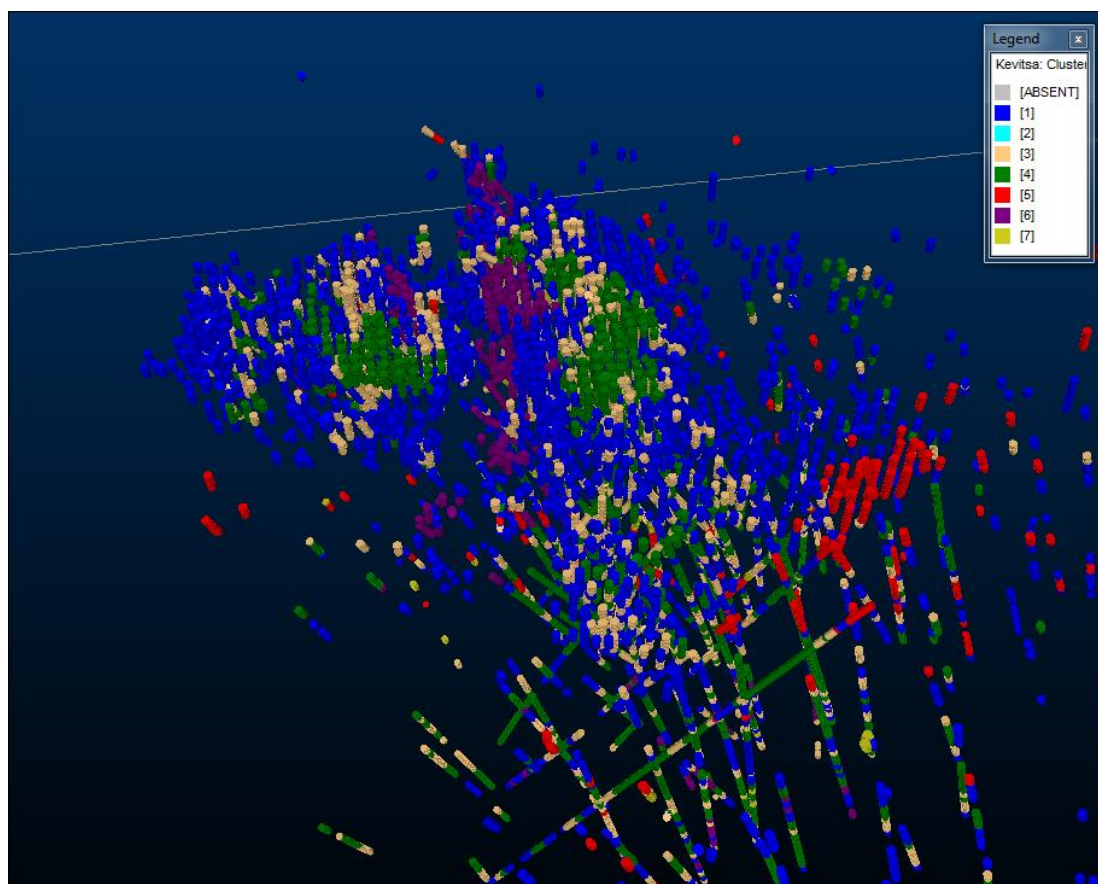
Each of these domains of mineralisation was identified in the drillhole data and then used to develop variography for a categorical indicator of each domain. Indicators were estimated into the empty block model. The resulting categorical indicator estimates had values between 0 and 1, with higher values implying a greater probability the block belongs to that domain. The sequence of indicator estimation was selected so as to honour the mineralisation pattern i.e. domain 1 was background with domain 3 overprinting followed by domain 4 and then domain 5, 6 and 7.  Domains 5, 6 and 7 were all relatively small domains compared to 1, 3 and 4. The resulting block model domain volumes were then used to code the drillhole data into its respective mineralisation domains for spatial analysis and estimation.

## 14.3     Available Data

Upper limits to the 3D block model were defined by pre-mining and 31$^{st}$ December 2015 mined and detailed topographic surface.  The surveyed pit floor (31sts December 2015) was used to define the upper limit of unmined Mineral Resources (Figure 14-6).

**Figure 14-6      A 3D view (an oblique northeast view) of the prevailing surveyed topography and pit as at 31$^{st}$ December 2015, which was used for reporting the unmined Mineral Resources.**
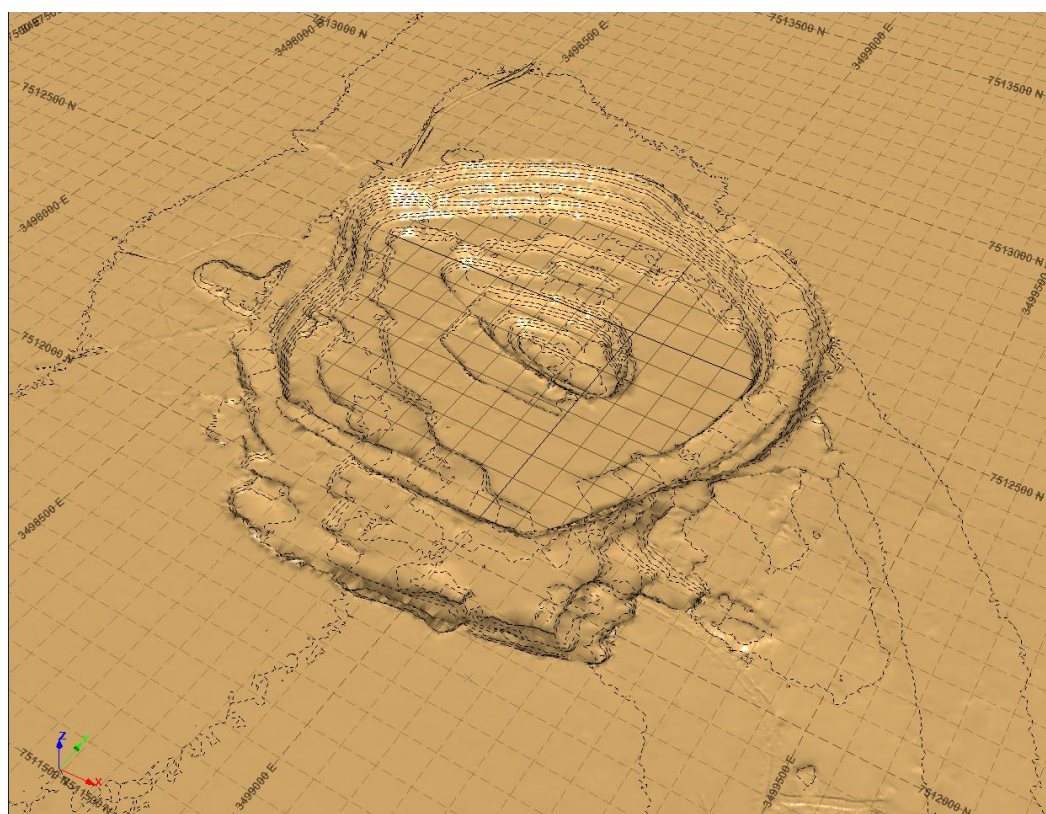


549 holes, for a total of 155,559 m, were drilled and available for this estimate.  Of this, 137,365 m was sampled and used to define mineralisation volumes and estimate block model grades.  Holes that did not intersect mineralisation or were drilled for geotechnical reasons were not sampled.

In addition, a total of 2,806 reverse circulation holes were included.  A general summary of the underlying drillhole statistics per drilling method and element is presented in Table 14-1.

**Table 14-1        Table of drillhole statistics per drilling method and element of interest**

| Type | Element | No of samples | Minimum | Maximum | Mean |
|------|---------|--------------:|--------:|--------:|-----:|
| DD | Au ppm | 72089 | 0.0003 | 4.80 | 0.04 |
| DD | Co ppm | 73335 | 0.5000 | 3020.00 | 92.30 |
| DD | CoS ppm | 44799 | 0.5459 | 2000.00 | 61.03 |
| DD | Cr ppm | 72150 | 1.2000 | 11100.00 | 436.89 |
| DD | Cu % | 73334 | 0.0000 | 12.00 | 0.14 |
| DD | CuS % | 45877 | 0.0001 | 11.00 | 0.12 |
| DD | Fe % | 72185 | 0.0025 | 64.90 | 5.22 |
| DD | Ni % | 73336 | 0.0003 | 4.50 | 0.13 |
| DD | NiS % | 46089 | 0.0003 | 3.08 | 0.10 |
| DD | Pd ppm | 70580 | 0.0002 | 8.70 | 0.06 |
| DD | Pt ppm | 60185 | 0.0001 | 9.00 | 0.10 |
| DD | S % | 72160 | 0.0007 | 33.40 | 0.85 |
| RC | Au ppm | 52738 | 0.0050 | 3.19 | 0.07 |
| RC | Co ppm | 24774 | 0.5000 | 3210.00 | 101.55 |
| RC | CoS ppm | 25676 | 5.0000 | 1590.00 | 79.41 |
| RC | Cr ppm | 60380 | 6.7000 | 12100.00 | 1632.57 |
| RC | Cu % | 60380 | 0.0002 | 6.50 | 0.16 |
| RC | CuS % | 60291 | 0.0002 | 4.86 | 0.14 |
| RC | Fe % | 60380 | 0.0025 | 65.00 | 6.79 |
| RC | Ni % | 60380 | 0.0003 | 3.51 | 0.16 |
| RC | NiS % | 60291 | 0.0003 | 3.51 | 0.13 |
| RC | Pd ppm | 52738 | 0.0050 | 6.78 | 0.11 |
| RC | Pt ppm | 52738 | 0.0050 | 9.00 | 0.16 |
| RC | S % | 60380 | 0.0023 | 17.20 | 0.76 |

A series of data validations were completed prior to desurveying the drillhole data into a 3D format. These included:

- Visual checks of collar elevations against to pit survey and original topographic surface digital terrain models. No corrections were required.
-  The logging, sampling and assay data were investigated for overlaps, gaps or duplication. No deviations were detected.
- Assay value checks were completed for samples having values outside of expected limits.
- Downhole survey data was investigated for excessive deviations. No risks were identified.

The database also includes logged lithology codes and bulk density measurements. Individual sample lengths range from 0.1 m to 54.3 m. 99.5% of diamond core samples were taken at intervals between 0.5 and 4 m with occasional rare long lengths. Reverse circulation sample lengths had 99.9% of samples taken at intervals between 0.5 and 3 m. As a result, a 3 m composite length was used.

Core recovery measurements were available for most sample intervals with more than 70% of drilled metres having a recovery greater than 90%. Poorer core recoveries were mostly restricted to the first 20 m of drilling. No sample lengths were adjusted or removed from the database data.

The desurveyed assay drillhole file was coded according to each of the mineralisation domains for estimation. The coded drillhole data was used for compositing, statistical and geostatistical analysis, and grade estimation.

## 14.4    Sample Compositing

Downhole compositing of drillhole samples was completed in order to reduce the effect that varying sample length may have on grade values. Compositing was completed in order to support robust statistical analysis. A 3 m downhole composite length was chosen in order to honour the dominant reverse circulation hole sample length and the current smallest mining unit bench height of 12 m. Drillhole samples were composited according to their respective sample lengths (length weighted) and were combined down-the-hole so as not to cross geology boundaries. Composites begin at the top of each hole and are generated at 3 m intervals down the length of the hole. A small number of composite lengths less than 0.5 m were removed. Several holes were randomly selected and composite values validated against the input sample data. No metal loss resulted from the compositing process and no errors were identified.

## 14.5    Statistical Analysis

Initial univariate statistical analysis (Table 14-1) highlights that approximately half of the diamond drilled samples were not analysed for Ni and Cu in sulphide (NiS and CuS). Missing metal sulphide analysis was restricted to the GTK and SGL diamond drilling campaigns. Total metal analysis, as opposed to metal in sulphide analysis, has some risk of elevating the grade of recoverable metal due to a small proportion of Ni and Cu been associated with silicate minerals. The samples with missing NiS and CuS were twinned with samples having both total and sulphide analysis (KMOY/FQM campaign). This was completed by:

- Spatially twinning samples having absent sulphide analysis (GTK and SGL) with samples that have both total and sulphide analysis (FQM/KMOY). The twinning study was completed for both GTK and SGL samples separately.
- The respective element statistics of the twinned KMOY/FQM samples was analysed for both sulphide and total metal values. Data used for correlation was restricted to samples having a similar metal tenor and was grouped into ultra-low, low and moderate grade populations.
- Strength of correlation was evaluated and a regression formula was established per metal per grade domain and per campaign.

The twinning process ensures minimal domain mixing with samples from the respective drilling campaigns been spatially coincident. Correlation coefficients were above 0.91 demonstrating strong positive correlations. Regression formulae are presented in Table 14-2 below.

**Table 14-2        Table of NiS and CuS regression formulae per campaign and mineralisation domain**

|       | Ni % range       | NiS formulae    | Cu % range       | CuS formulae     |
|-------|------------------|-----------------|------------------|------------------|
| GTK   | >0.21            | 0.921*Ni-0.003  | >0.4             | 0.922*Cu-0.001   |
| GTK   | >0.025 <=0.21    | 0.911*Ni-0.015  | >0.006  <=0.4    | 0.928*Cu-0.002   |
| GTK   | <=0.025          | 0.69*Ni         | <= 0.006         | 0.89*Cu          |
| SGL   | >0.22            | 0.986*Ni-0.021  | >0.4             | 0.948*Cu+0.004   |
| SGL   | >0.025 <=0.22    | 0.901*Ni-0.013  | >0.006  <=0.4    | 0.962*Cu-0.003   |
| SGL   | <=0.025          | 0.73*Ni         | <= 0.006         | 0.91*Cu          |

Exploratory data analysis was completed on the regressed and composited sample data. Analysis involved summarizing the statistics of the composite sample values in order to understand the nature of each domains grade distribution in order to ensure sufficient difference in values between respective domains (Figure 14-7).  Statistical analysis was undertaken to ensure minimal mixing of sample grade values for a particular domain.  The data analysis highlighted that multiple populations were not evident per domain. The proposed domains limit mixing of populations and therefore improve estimate accuracy. Domain statistics were assessed per domain per element (Table 14-3) using a series of boxplots, histograms and log probability curves. Nickel sulphide distribution examples are presented in Figure 14-7.  While each of these domains has distinctly different mean values and grade ranges, they each have good distributions with low coefficients of variation (CV). Detail statistics per metal and per domain are tabled in Table 14-3 below.

**Figure 14-7        NiS histograms and descriptive statistics for the main mineralisation domains (1, 3, 4, and 6).**
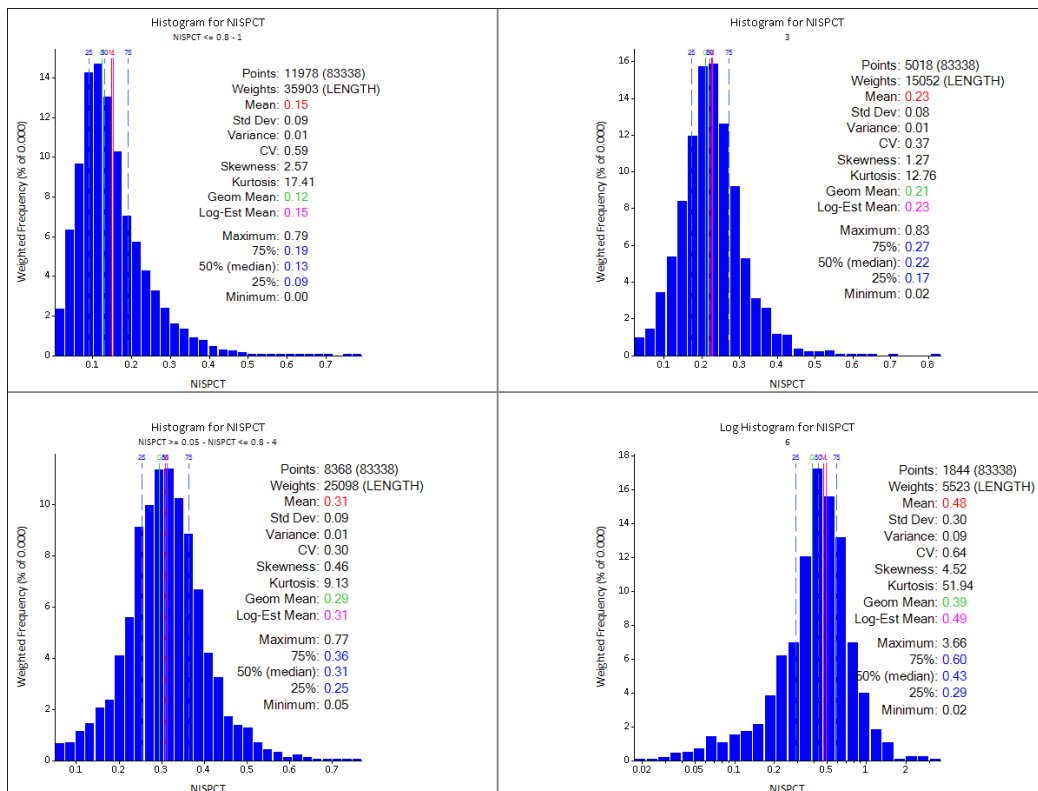
**Table 14-3**     **Sample composite statistics per metal and domain. Domain 0=waste; 1=Low Ni and Cu; 3=Low Ni and Mod Cu; 4=Mod Ni and High Cu; 5=Low Ni and Cu(intrusive); 6=High Ni and Mod Cu; 7=Low Ni & Cu, High Fe and S.**

| | Domain number | 0 | 1 | 3 | 4 | 5 | 6 | 7 |
|---|---|---|---|---|---|---|---|---|
| NiS % | Samples | 54287 | 12319 | 4736 | 8490 | 1335 | 1664 | 248 |
| | Minimum | 0.0003 | 0.0040 | 0.0210 | 0.0081 | 0.0016 | 0.0174 | 0.0025 |
| | Maximum | 1.00 | 0.80 | 0.83 | 0.80 | 0.30 | 2.50 | 0.92 |
| | Mean | 0.05 | 0.15 | 0.23 | 0.31 | 0.05 | 0.50 | 0.18 |
| | Standard deviation | 0.06 | 0.09 | 0.08 | 0.10 | 0.04 | 0.30 | 0.19 |
| | CV | 1.08 | 0.60 | 0.36 | 0.31 | 0.81 | 0.61 | 1.06 |
| CuS % | Samples | 54180 | 12303 | 4736 | 8490 | 1335 | 1664 | 240 |
| | Minimum | 0.0001 | 0.0021 | 0.0031 | 0.0027 | 0.0016 | 0.0011 | 0.0039 |
| | Maximum | 1.50 | 1.00 | 1.00 | 1.50 | 0.40 | 1.00 | 2.00 |
| | Mean | 0.05 | 0.18 | 0.33 | 0.51 | 0.09 | 0.19 | 0.24 |
| | Standard deviation | 0.07 | 0.12 | 0.13 | 0.19 | 0.06 | 0.15 | 0.28 |
| | CV | 1.37 | 0.69 | 0.39 | 0.36 | 0.66 | 0.80 | 1.16 |
| Fe % | Samples | 53770 | 12225 | 4724 | 8457 | 1331 | 1664 | 248 |
| | Minimum | 0.3182 | 0.4901 | 1.2262 | 1.7591 | 0.7401 | 0.8210 | 1.4836 |
| | Maximum | 20.00 | 15.00 | 13.00 | 14.63 | 17.00 | 11.00 | 39.92 |
| | Mean | 5.56 | 6.46 | 7.41 | 8.26 | 7.64 | 5.02 | 11.17 |
| | Standard deviation | 2.04 | 1.93 | 1.72 | 1.65 | 2.02 | 1.81 | 6.81 |
| | CV | 0.37 | 0.30 | 0.23 | 0.20 | 0.26 | 0.36 | 0.61 |
| S % | Samples | 53760 | 12220 | 4722 | 8457 | 1331 | 1664 | 248 |
| | Minimum | 0.0011 | 0.0050 | 0.0050 | 0.0040 | 0.0050 | 0.0050 | 0.0918 |
| | Maximum | 11.00 | 5.00 | 4.30 | 4.50 | 4.50 | 3.70 | 16.00 |
| | Mean | 0.38 | 1.03 | 1.65 | 2.06 | 2.10 | 1.16 | 4.37 |
| | Standard deviation | 0.50 | 0.57 | 0.57 | 0.57 | 0.70 | 0.66 | 3.22 |
| | CV | 1.33 | 0.56 | 0.35 | 0.28 | 0.33 | 0.57 | 0.74 |
| Au ppm | Samples | 49015 | 12154 | 4688 | 8438 | 1220 | 1664 | 248 |
| | Minimum | 0.0005 | 0.0010 | 0.0032 | 0.0015 | 0.0009 | 0.0038 | 0.0013 |
| | Maximum | 0.50 | 0.60 | 0.45 | 1.50 | 0.15 | 1.20 | 0.40 |
| | Mean | 0.02 | 0.07 | 0.12 | 0.17 | 0.02 | 0.15 | 0.04 |
| | Standard deviation | 0.03 | 0.06 | 0.07 | 0.09 | 0.02 | 0.12 | 0.06 |
| | CV | 1.58 | 0.85 | 0.57 | 0.54 | 0.92 | 0.83 | 1.39 |
| Pd ppm | Samples | 48494 | 12149 | 4665 | 8404 | 1214 | 1664 | 248 |
| | Minimum | 0.0002 | 0.0007 | 0.0010 | 0.0010 | 0.0009 | 0.0050 | 0.0009 |
| | Maximum | 3.00 | 1.50 | 0.70 | 1.50 | 0.20 | 3.50 | 0.50 |
| | Mean | 0.03 | 0.11 | 0.16 | 0.20 | 0.02 | 0.51 | 0.07 |
| | Standard deviation | 0.09 | 0.13 | 0.10 | 0.12 | 0.03 | 0.45 | 0.09 |
| | CV | 2.74 | 1.14 | 0.65 | 0.59 | 1.34 | 0.88 | 1.25 |
| Pt ppm | Samples | 43659 | 11176 | 4387 | 7857 | 925 | 1621 | 245 |
| | Minimum | 0.0002 | 0.0041 | 0.0023 | 0.0050 | 0.0046 | 0.0050 | 0.0026 |
| | Maximum | 4.00 | 1.70 | 1.10 | 1.50 | 0.40 | 4.96 | 0.50 |
| | Mean | 0.06 | 0.18 | 0.23 | 0.30 | 0.03 | 0.70 | 0.07 |
| | Standard deviation | 0.13 | 0.19 | 0.14 | 0.16 | 0.05 | 0.59 | 0.10 |
| | CV | 2.38 | 1.05 | 0.61 | 0.52 | 1.48 | 0.84 | 1.35 |

## 14.6     Boundary Analysis

Boundary analysis examines the rate of grade change across the contact between domains. Contact profiles with rapid grade changes are referred to as hard boundaries.  Conversely, boundaries having a gradual change in grade are referred to as soft.  Kevitsa domain boundary analysis highlights soft grade boundaries across 5 to 10 metres. Key examples are illustrated in Figure 14-8 .

**Figure 14-8      Nickel sulphide grade profiles with increasing distance from the contact between respective key domains at Kevitsa.**
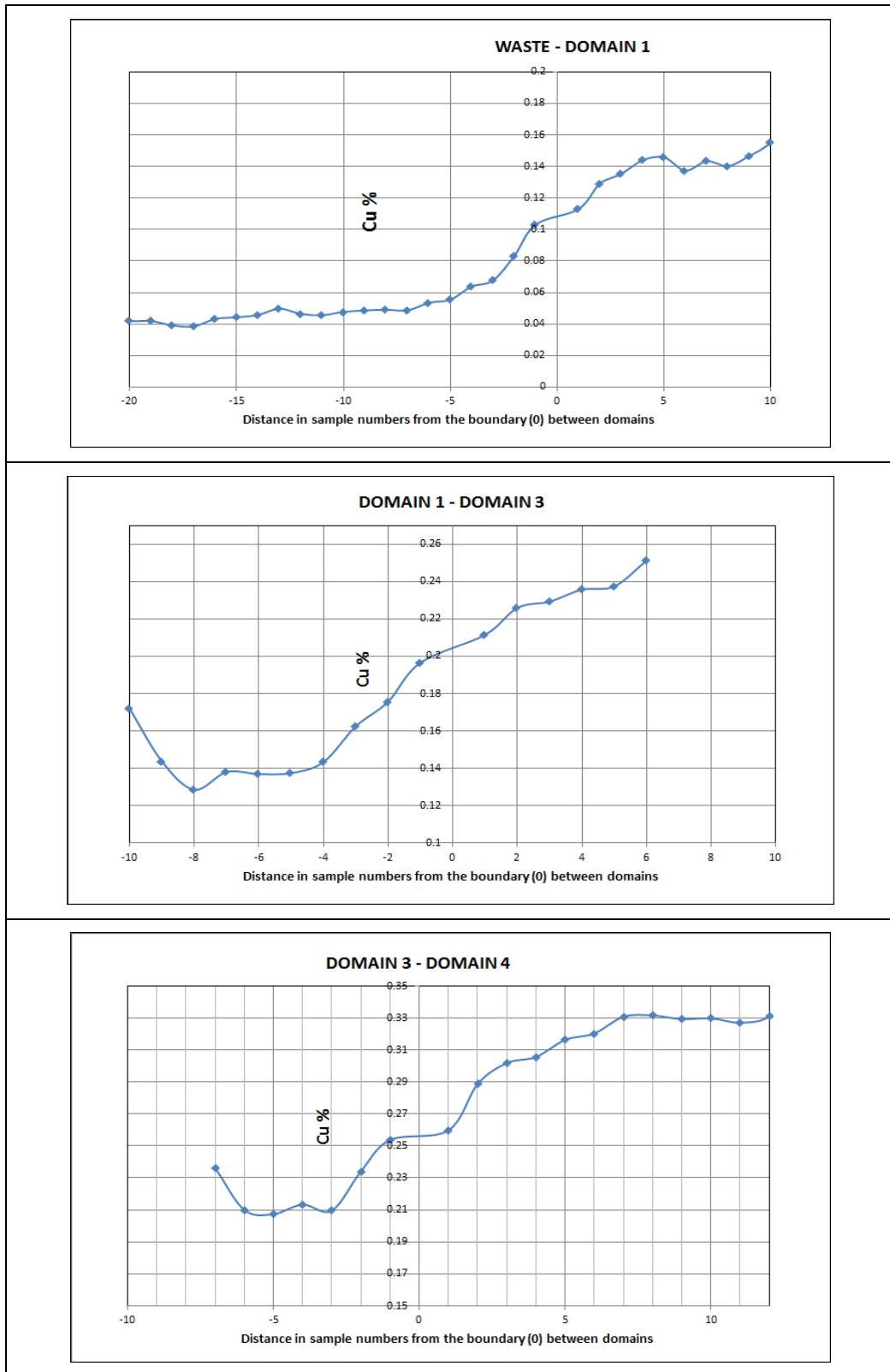


Apart from Domain 5, soft boundary conditions were applied to all domain contacts for each element estimated into the block model. The respective domains estimated are summarized in Table

14-4 below and in general are zoned, with domain 4, surrounded by domain3 and with domain 1 around the margins (Figure 14-5). Domain 6 is mostly within domain 1 and appears to be structurally controlled.  Domain 5 is randomly located and likely to be associated with late stage intrusives or xenoliths. Domain 2 was renamed to 0 and represents the waste or ultra-low grade domain volume.

**Table 14-4        Summary of domain names used for block grade estimation**

| Domain (INDGRP) | Comments |
|:---:|:---|
| 0 | Ultra-low grade or waste rock |
| 1 | Low Ni, low Cu – outer zone of mineralisation |
| 3 | Low Ni, moderate Cu – located within domain 1 |
| 4 | Moderate Ni, high Cu – located within domain 3 |
| 5 | Low Ni, low Cu – late stage intrusives or xenoliths |
| 6 | High Ni, moderate Cu with a strong PGE tenor and apparent structural control |
| 7 | Moderate to low Ni and Cu, but with high Fe and S from pyrrhotite veins |

## 14.7    Density Data

Density measurements were made on 299 diamond drilled holes.  In addition, geophysical downhole wireline logging has captured density values for 75 (45 of which already had density data) holes with measurements taken every 2 to 5 cm down the hole.  In total, 329 drillholes have density values. The geophysical measurements were averaged to 1 m intervals.  Gravimetric data had a range of assigned widths which were modified to best represent the closest metre.  Density data was then combined and used for variography and estimation. Density data is strongly normal with a very low coefficient of variation (CV=0.04) and minimal evidence for mixed populations (Figure 14-9).  Density has a mean value of 3.14 and tends to have higher values within the more strongly mineralised domains. Apart from a top cut of 3.9 t/m$^3$ this data has been used to determine a variogram, which was used to estimate density into the block model.

**Figure 14-9      Histogram of combined density data composited to 1 m sample lengths. Data is strongly normal with a low CV and minimal evidence for mixed populations.**
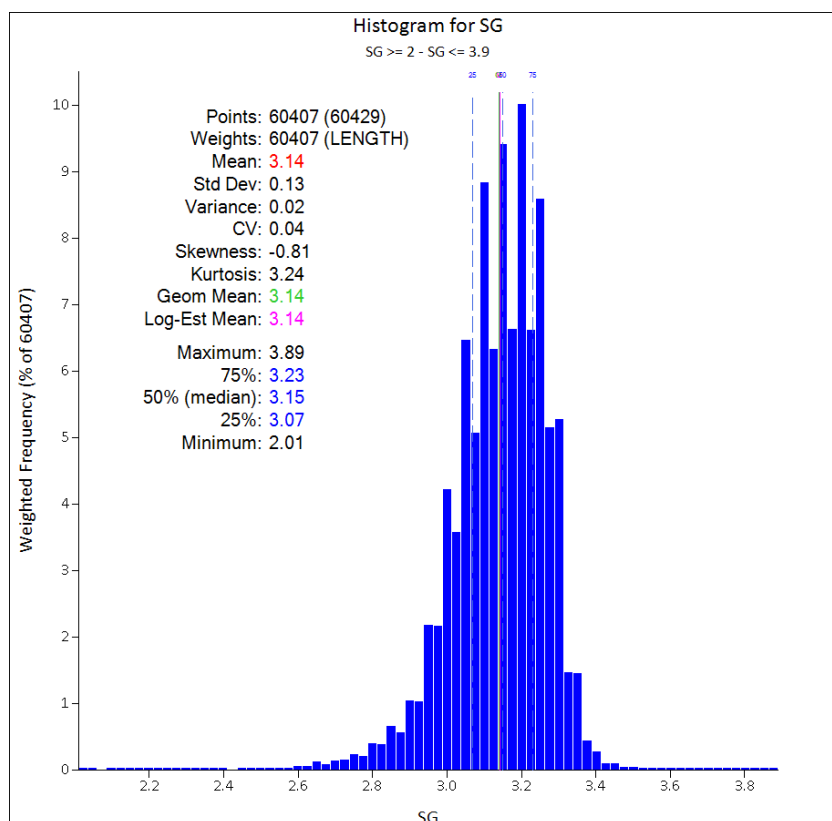


## 14.8    Bivariate element relationships

The correlation coefficient for Pt and Pd is good for most domains of mineralisation. While copper, nickel, gold, iron and sulphur all occur within similar mineralisation volumes, correlation between these elements is moderate to poor. Estimation has been completed using similar sample selection routines, particularly for Pt and Pd. This ensures that element correlations from block estimates are similar to those of the input data.

## 14.9    Top Cutting

Histograms and log probability plots were used to identify the presence of anomalous outlier grades for the sample composites of each element per domain.  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. No outlier samples were located in areas with low data support. Top capping of outlier samples was employed in order to reduce the population variance and so minimise the risk of high grade samples affecting poorly informed block estimates. The low number of samples and the marginal impact upon mean values has allowed the following top cuts to be applied (Table 14-5).

**Table 14-5        Summary table of top cut values applied**

| Domain | Ni | NiS | Cu | CuS | CoS | Fe | S | Au | Pd | Pt | Fe |
|--------|-----|-----|-----|-----|-----|-----|-----|------|-----|-----|------|
| 1 | 0.9 | 0.8 | 1 | 1 | 450 | 15 | 5 | 0.6 | 1.5 | 1.7 | 8 |
| 3 | | | 1 | 1 | 300 | 13 | 4.3 | 0.45 | 0.7 | 1.1 | 7 |
| 4 | | 0.8 | 1.5 | 1.5 | 350 | | 4.5 | 1.5 | 1.5 | 1.5 | 6.5 |
| 5 | 0.4 | 0.3 | 0.4 | 0.4 | 300 | 17 | 4.5 | 0.15 | 0.2 | 0.4 | 8 |
| 6 | 2.5 | 2.5 | 1 | 1 | | 11 | 3.7 | 1.2 | 3.5 | | 6 |
| 7 | | | 2 | 2 | 900 | | 16 | 0.4 | | 0.5 | 25 |
| 0 | | 1 | 1.5 | 1.5 | 400 | 20 | 11 | 0.5 | 3 | 4 | 15 |

## 14.10    **Variography**

Variography, which represents the continuity of grade in 3D space, was generated per element and per domain. Variography was analysed and variogram models were determined for each element in each domain using Snowden Supervisor v8.3 software.  The following methodology was applied:

- data was declustered where required prior to variogram modelling so as to remove the effect of closely spaced samples
- the principal axes of anisotropy were determined using variogram fans based on normal scores variograms
- directional normal scores variograms were calculated for each of the principal axes of anisotropy
- downhole normal scores variograms were modelled for each domain to determine the normal scores nugget effect
- variogram models were determined for each of the principal axes of anisotropy using the nugget effect from the downhole variogram
- the variogram parameters were standardised to a sill of one
- the variogram models were back-transformed to the original distribution using a Gaussian anamorphosis and used to guide search parameters and complete ordinary kriging estimation
- the variogram parameters were standardised to the population variance for each domain to permit post-processing of the copper panel estimates to SMU estimates.

The multidirectional variogram model results are summarised in Table 14-6 for nickel sulphide. While each element per domain had similar anisotropy, ranges and sill differentials were different. Variogram models had low nugget values which were clearly defined by the close spaced RC data. Each of the domains variograms were modelled using one to three spherical structures.  The ranges of influence were clearly visible from the variograms, providing confidence in domain data selections and grade continuity (Figure 14-10).

**Table 14-6        Variogram parameters per domain for nickel sulphide**

| Domain | Nugget | Structure | Sill differential | Bearing | Plunge | dip | Major axis | Semi axis | Minor axis |
|---|---|---|---|---|---|---|---|---|---|
| 1 | 0.11 | 1 | 0.49 | 20 | 130 | 90 | 16 | 16 | 10 |
| 1 | | 2 | 0.4 | | | | 137 | 117 | 45 |
| 3 | 0.201 | 1 | 0.55 | 75 | 80 | 100 | 52 | 37 | 36 |
| 3 | | 2 | 0.25 | | | | 159 | 169 | 73 |
| 4 | 0.15 | 1 | 0.36 | 80 | 10 | 180 | 13 | 13 | 13 |
| 4 | | 2 | 0.18 | | | | 45 | 45 | 45 |
| 4 | | | 0.31 | | | | 269 | 182 | 112 |
| 5 | 0.109 | 1 | 0.891 | 25 | 160 | -180 | 408 | 239 | 37 |
| 6 | 0.19 | 1 | 0.66 | 90 | 90 | 90 | 25 | 21 | 10 |
| 6 | | 2 | 0.14 | | | | 66 | 30 | 15 |
| 0 | 0.16 | 1 | 0.33 | 40 | 120 | 150 | 9 | 14 | 9 |
| 0 | | 2 | 0.22 | | | | 88 | 55 | 24 |
| 0 | | 3 | 0.28 | | | | 327 | 224 | 197 |

**Figure 14-10        An example of robust variography for NiS % across domain 4.**
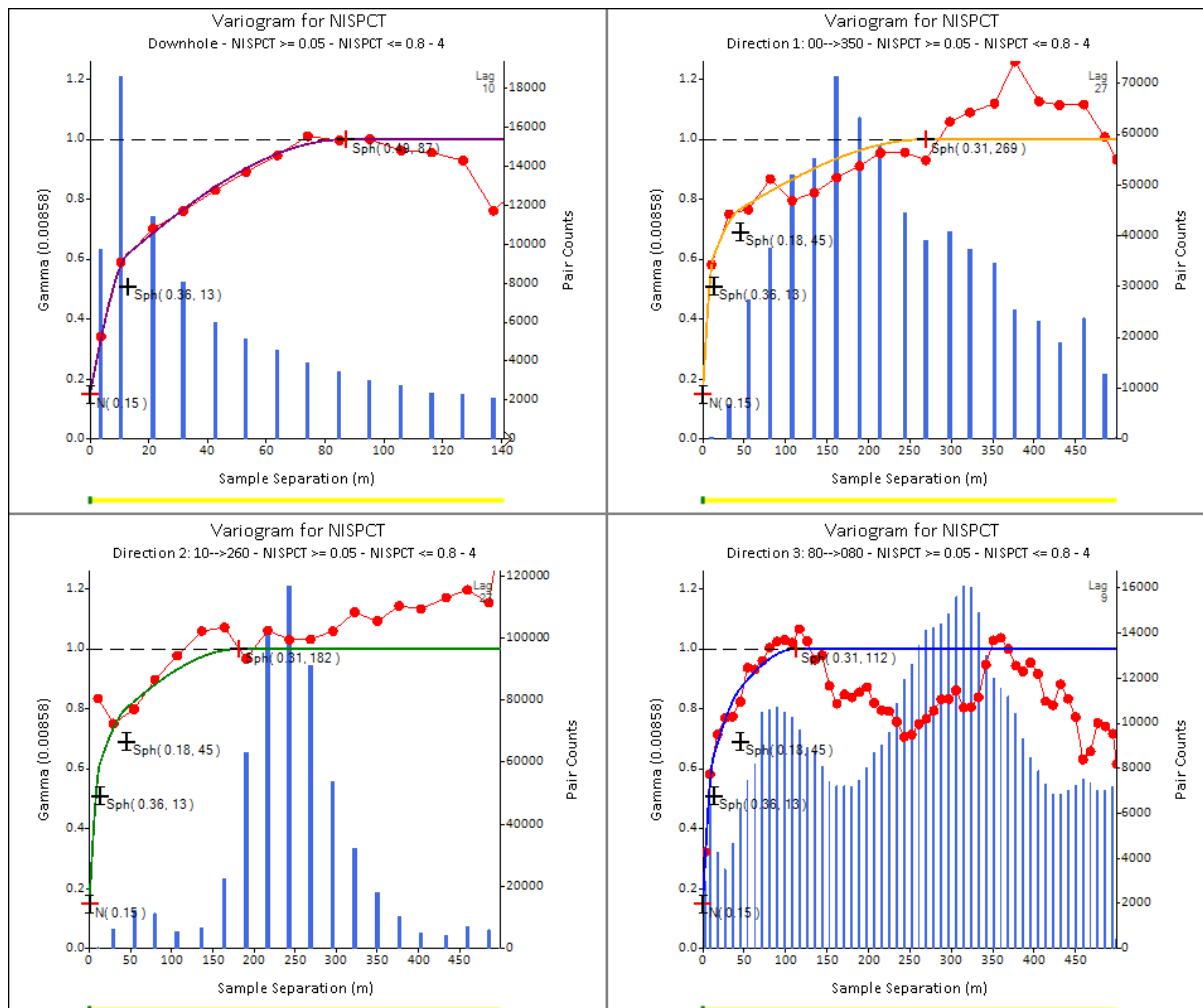
## 14.11    Kriging Neighbourhood Analysis

A kriging neighbourhood analysis (KNA) was completed to determine optimal block size, ellipse dimensions, minimum and maximum numbers of samples to be used for grade estimation as well as the discretization parameters.  Correctly defined sample selection parameters are essential to optimising the estimation process and help reduce the risk of conditional bias (overestimation of high grades and underestimation of low grades).  An optimum parent block size of 30 m by 40 m by 24 m was selected (Figure 14-11) together with a minimum of 10 samples and a maximum of 40 for the estimation routine.  KNA was completed using Snowden Supervisor's KNA analysis tools.

**Figure 14-11    Kriging neighbourhood analysis highlighting optimal kriging efficiency for the selected block size**
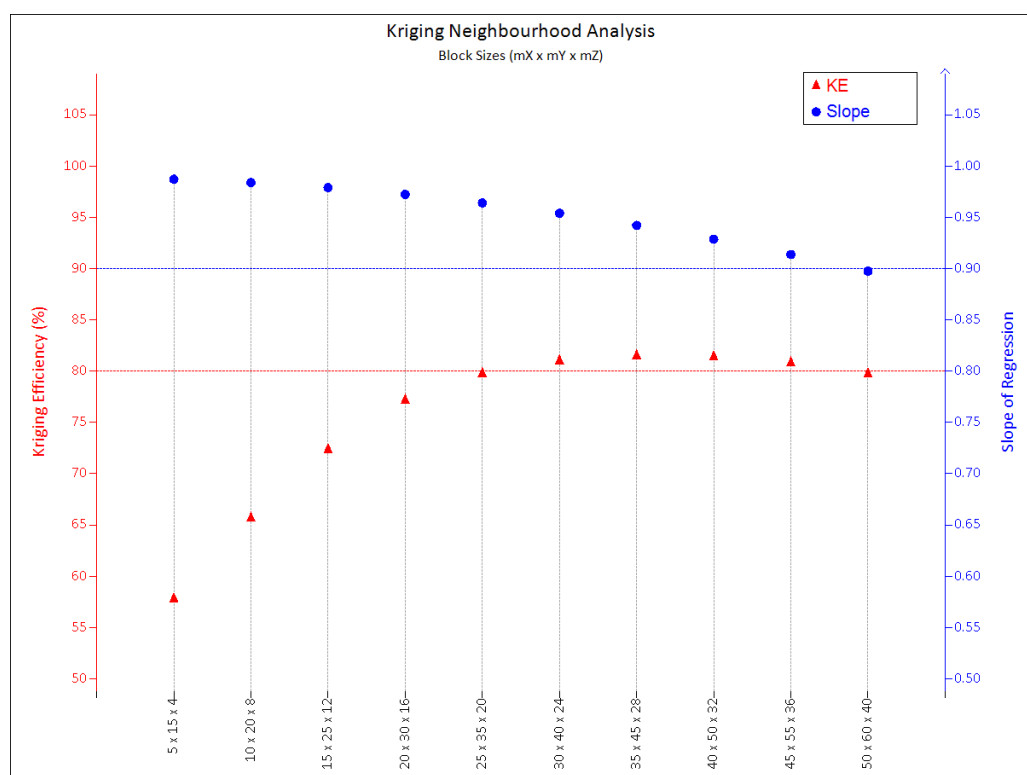


## 14.12    Block Model Setup and Limits

A 3D empty block model was developed in Datamine to cover the Kevitsa deposit extents. Dimensions and coordinate origins of this model are defined in Table 14-7.  The parent block size was set to 30 m x 40 m x 24 m as per the KNA study and a sub-block size of 10 m x 10 m  x 12 m was used in order to ensure accurate representation of mineralised volumes and to honour the smallest mining unit (SMU) dimensions.  The block model was not rotated and there is currently no justification for the requirement of unfolding.

**Table 14-7        Block model parameters – limits and dimensions**

| Direction | Minimum | Maximum | Parent Block Size (m) | SubBlock Size (m) | No. SubBlocks |
|-----------|---------|---------|----------------------:|------------------:|--------------:|
| East | 3498285 | 3499305 | 30 | 10 | 102 |
| North | 7511250 | 7512130 | 40 | 10 | 88 |
| Elevation | -1014 | 306 | 24 | 12 | 110 |

Block centroids were assigned domain code values according to the categorical indicators per domain.

## 14.13    Interpolation Parameters

Ni, NiS, Cu, CuS, Pt, Pd, Au, Fe, S and CoS were each estimated into the parent block model using ordinary kriging (OK).  OK was deemed an appropriate estimation technique due to the near normal distributions and limited domain grade mixing of the respective domains input data.   The interpolation parameters are summarized by domain in the Table 14-8.  Sample selection routines were the same for each element and domain in order to ensure that similar samples were used per block estimate in order to minimise the risk of distorting metal correlations. Estimation into parent blocks used a discretisation of 3 (X points) by 3 (Y points) by 3 (Z points) to better represent the block volume shape.  Apart from domain 5, each domain was estimated using soft boundaries where a single sample from the adjacent domain was used during the estimation. Most mineralisation blocks were estimated using the first search pass.

**Table 14-8        Estimate sample selection parameters. The second pass was adjusted to between 3 and 10 times the first search range allowing for all blocks to be estimated.**

| Domain | Search ellipse range (m) | | | No of composite samples | | |
|:------:|:---:|:---:|:---:|:---------:|:---------:|:--------:|
| | X | Y | Z | Min/block | Max/block | Max/hole |
| **1-7** | 90 | 80 | 90 | 10 | 40 | 4 |
| **0** | 100 | 100 | 70 | 14 | 38 | 4 |

## 14.14    Post-processing by Localized Uniform Conditioning

A localised uniform conditioning estimate (LUC) was completed for each element per domain. Uniform conditioning (UC) provides an estimate of the proportion of smallest mining unit blocks (SMU) inside the parent block that are above a cut-off grade and their corresponding average grade. UC does not provide spatial information pertaining to these SMU grades.  LUC provides the spatial grade estimates of the blocks that are smaller (SMU) than the parent block size.  LUC results in an assessment of recoverable resources available per domain at the scale of mining and is particularly relevant in the more widely drilled areas. The parent block size used was 30 m by 40 m by 24 m and the SMU block size was 10 m by 10 m by 12 m.  LUC models were validated by:

- Visual comparisons with drillhole sample grades and the OK sample grades
- Checks that the SMU average grade is the same as the Parent grade
- Checking that contained metal at a zero cut-off grade is the same for the OK estimates and the LUC estimates.

LUC was only completed for those blocks not supported by RC grade control drilling. Blocks located within the RC grade control drilled areas were estimated directly into the SMU block size. No deviations or anomalies were noted by the QP, Mr. David Gray.

## 14.15    Validation of Block Model Estimates

The results of the modeling process were validated through several methods including a thorough visual review of the model grades in relation to the underlying drillhole sample grades; comparisons with the change of support model and grade distribution comparisons using swath plots.

### 14.15.1        Visual Validations

Detailed visual validations comparing block model and input data grades was conducted along northsouth, eastwest and vertical sections.  The validation included confirmation of the correct domain coding of blocks.  Figure 14-12 illustrate a vertical cross section validation representing the LUC block (SMU) nickel sulphide grades and the input drillhole composite (points) nickel sulphide grades.

**Figure 14-12        An east-west vertical section through central portion of current pit outlined in red. Model estimates visually validate well with input drillhole data.**
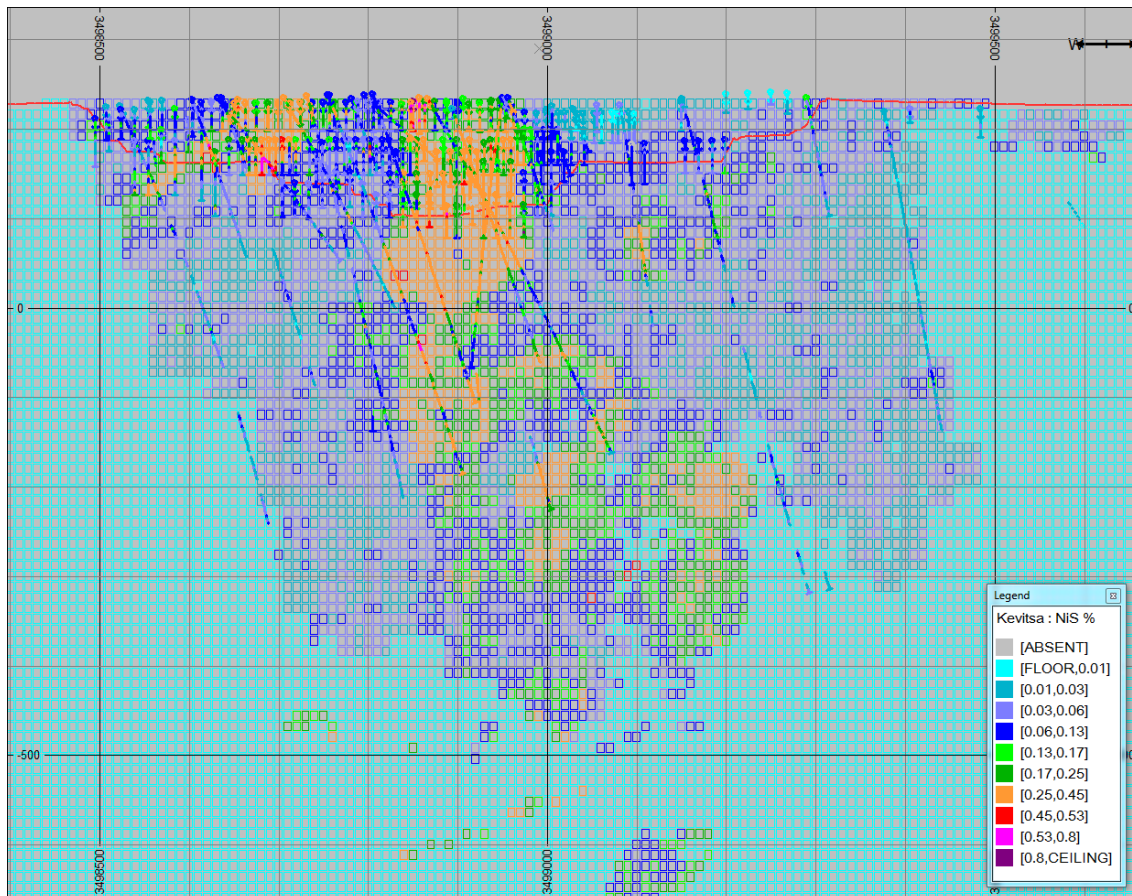
### 14.15.2        Model Checks for Change of Support

The parent (OK) and the LUC (SMU) estimates were compared at various cut-off grades using grade/tonnage curves (Figure 14-13).   Overall, there is very good correlation between models and overall copper metal was preserved (the same) in both model estimates at zero cut-off.   As expected, the LUC estimates provide greater resolution (more tonnes) at higher grades.  The LUC estimate better honours the input data grade distribution than the parent (OK) estimate, which tends to be smoothed.

**Figure 14-13      Grade and tonnage distribution of LUC (green) and OK (blue) copper sulphide block estimates for domain 3.**
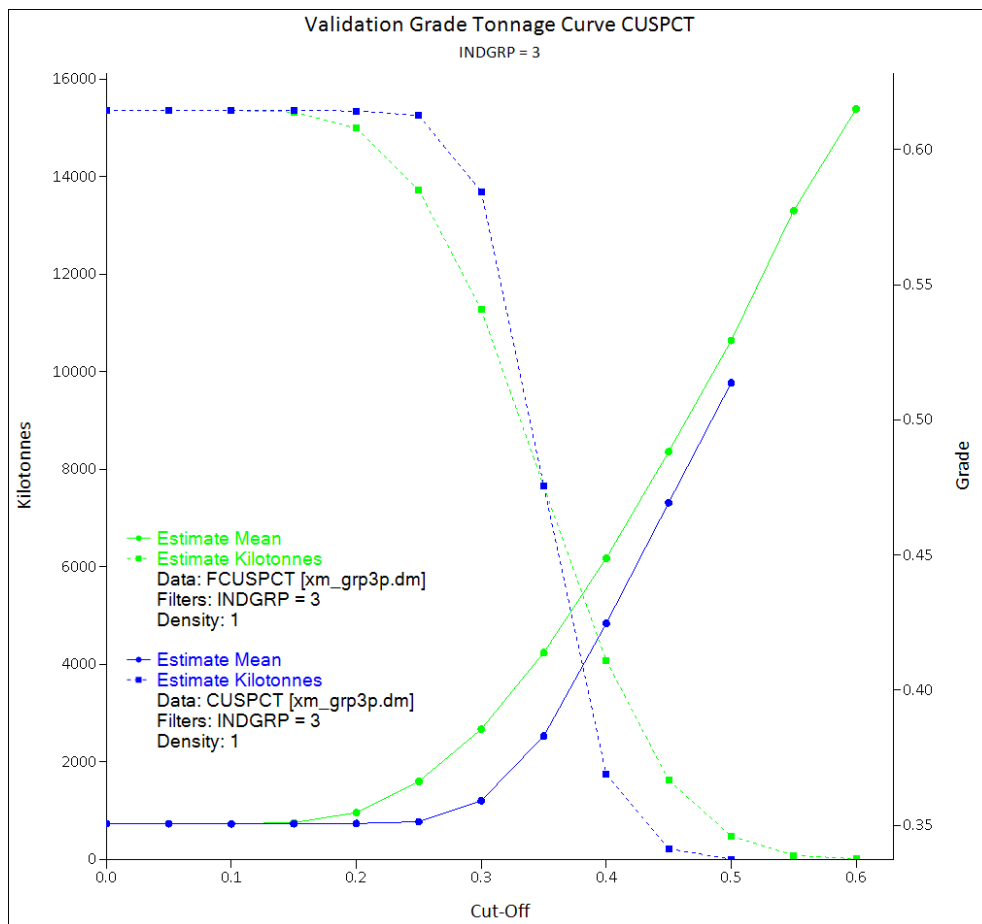


### 14.15.3        Swath Plots (Drift Analysis)

Swath plots compare the mean grades of the input data and block estimates for consecutive widths in a particular direction (easting, northing or vertical).  Grade variations from the OK model are compared to those derived from the declustered input grade data.  Examples of nickel sulphide parent and SMU block estimates for domain 3 are shown in Figure 14-14. Parent and SMU estimates validate well against the input data.

87

**Figure 14-14     Validation slices comparing domain 3 nickel sulphide estimates and input data for easting and northing.**
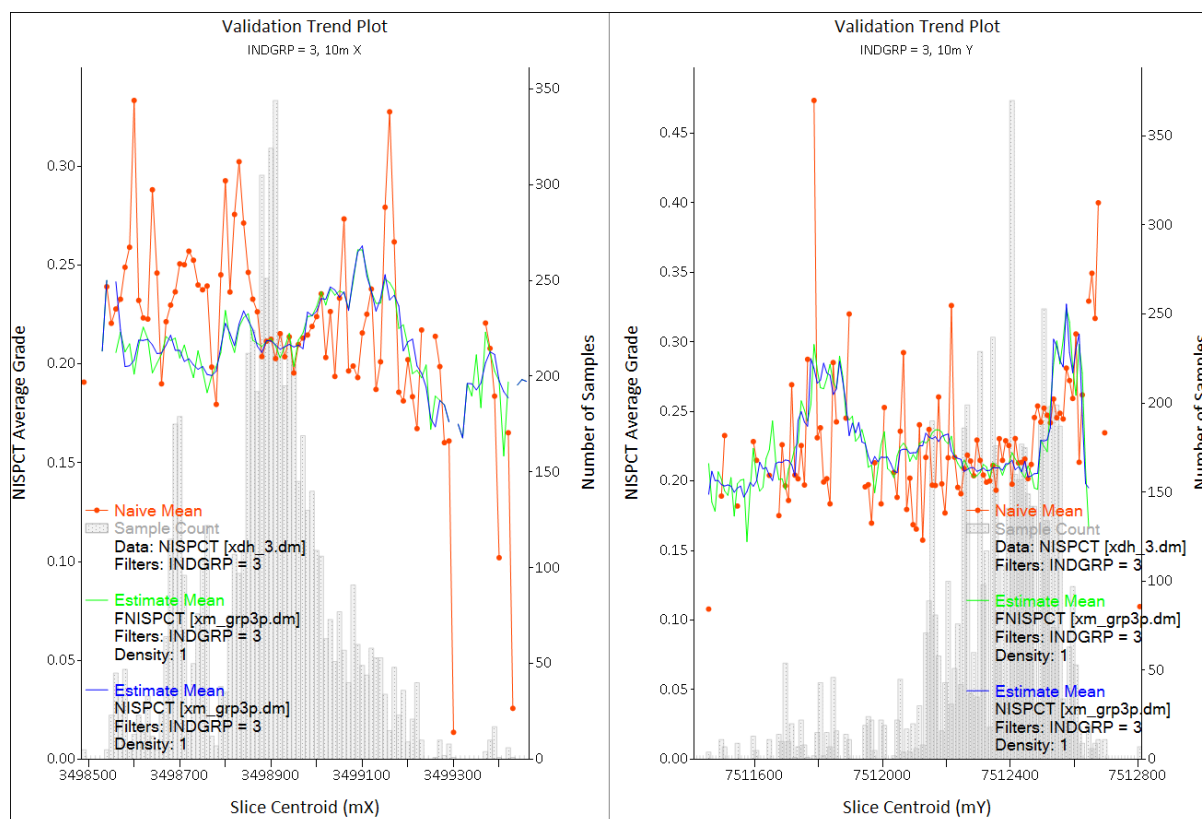


## 14.15.4     Summary statistics

Summary statistics comparing each domain's mean input composite grade, the mean parent block estimate, highlights good results reflecting the respective domains mineralisation. The comparison between parent mean estimate and LUC mean estimate has no differences.

In conclusion, the summary statistics, visual validations and swath plots, the OK parent and LUC SMU estimates are consistent with the input drillhole composite data, and are believed to constitute a reasonable representation of the respective domains of mineralisation.

## 14.16     Mineral Resource Classification and reporting

Mineral Resource estimates of the Kevitsa deposit have been classified and reported using the guidelines of the JORC Code (JORC, 2012), which in turn comply with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014). Classification (Figure 14-15) 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, QAQC of sample data, kriging efficiency and regression slope values.

Measured Mineral Resources (Figure 14-15) were generally deemed appropriate in areas where the drill grid spacing was less than 25 m.  Kriging efficiency was greater than 80% and regression slope values were greater than 0.8.   Indicated Mineral Resources were assigned to block estimates where

the drill grid was between 25 m to 75 m, where the kriging efficiency was between 60 % to 80% and where regression slope values were greater than 0.6. Block estimates that did not meet the measured or indicated criteria and that were within 100 m of a single drillhole with geological continuity, were assigned to the inferred category. Typically, inferred block estimates had a kriging efficiency greater than 40% and regression slope value greater than 0.4.

**Figure 14-15     A 3D oblique view looking northeast and illustrating the Mineral Resource classification of the Kevitsa deposit with the December 2015 mining surface as reference.**
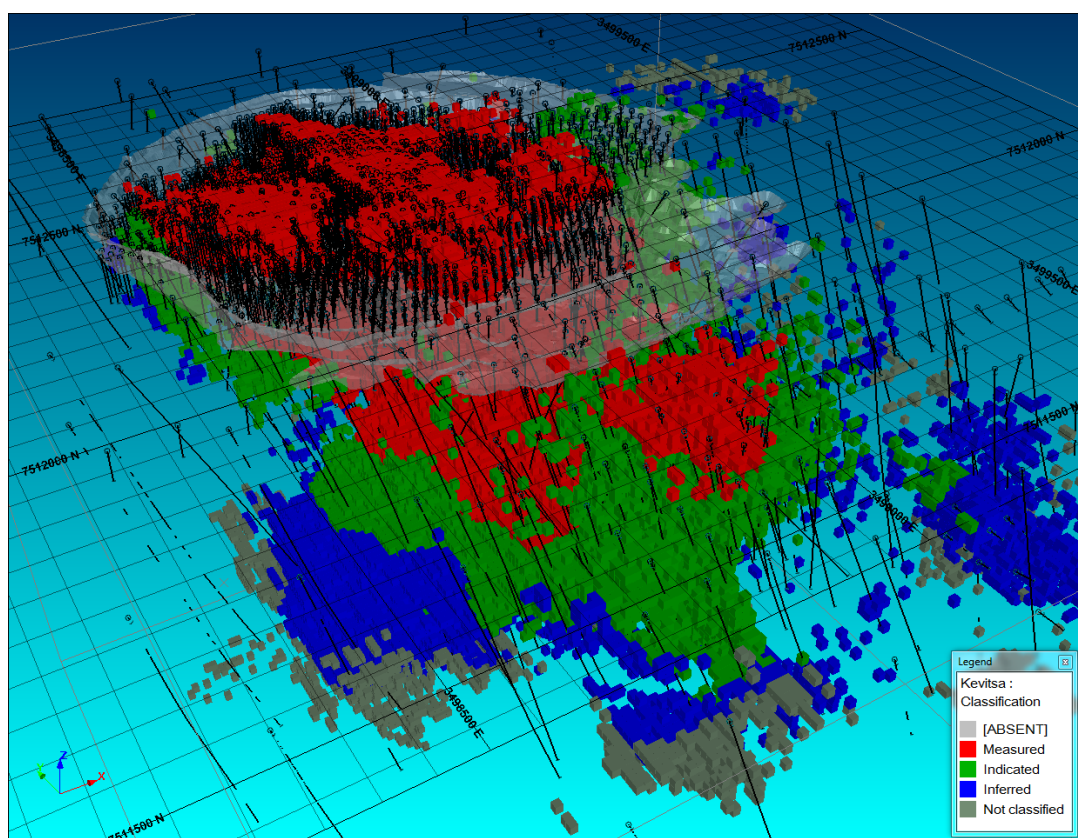


Indicated Mineral Resource areas were suggested as potential locations for new drilling in order to assure geological and grade continuity for accurate estimates.

The Mineral Resource statement depleted of mining as at the end of December 2015 is presented in Table 14-9 below. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. A NiSEq cut-off grade of 0.22% was used for reporting Mineral Resources and was guided by this reports Mineral Reserve break even cut-off. The nickel sulphide equivalent formula is:

NiSEq = NiS % + (0.722* CuS %) + (0.081 * Pt ppm) + (0.056 * Pd ppm) + (0.219 * Au ppm)

The nickel sulphide equivalent formula uses feasibility level mining costs, recoveries and metal prices.

There are no known factors related to environmental, permitting, legal, title, taxation, socio-economic, marketing or political issues which could materially affect this Mineral Resource statement.

**Table 14-9     January 2016 Kevitsa Mineral Resource statement depleted of mined material as at 31 December 2015 and using a 0.22% NiSEq cut-off grade.**

| Category | Density | Tonnes (Mt) | Ni (%) | NiS (%) | Cu (%) | Au (ppm) | Pt (ppm) | Pd (ppm) |
|---|---|---|---|---|---|---|---|---|
| Measured | 3.17 | 108.7 | 0.25 | 0.23 | 0.34 | 0.10 | 0.21 | 0.13 |
| Indicated | 3.16 | 167.0 | 0.26 | 0.24 | 0.34 | 0.09 | 0.16 | 0.11 |
| **Total Measured and Indicated** | **3.16** | **275.7** | **0.26** | **0.23** | **0.34** | **0.09** | **0.18** | **0.12** |
| Inferred | 3.16 | 57.1 | 0.22 | 0.20 | 0.31 | 0.06 | 0.12 | 0.07 |

## 14.17   Mineral Resource Estimate Comparisons

Compared to the previous estimate, the 2016 updated Mineral Resource estimate has increased total available nickel sulphide (NiS) metal by 6% and has increased total available copper metal by 4%.  Metal increases have resulted from expanded mineralised volumes which have been guided by:

1. on-mine reconciliation data of the 23 Mt mined since the previous estimate,
2. additional RC drilling, which has improved domain delineation and geology detail,
3. in-pit mapping with improved geology and deposit understanding,
4. updated 3D seismic structural interpretations,
5. a comprehensive data set of XRD data supporting mineralisation, lithology and alteration definition, and
6. alignment and improvement of the employed estimation methods  to improved geology model, mineralisation domains and added data.

The Kevitsa geology and mining team have completed a due diligence internal review of the input data, estimation methods, block estimate results and classification.  Findings of this review support this updated estimate to be robust in methodology and representative of the input data and current mining performances.

# ITEM 15  **MINERAL RESERVE ESTIMATE**

The Reserves for Kevitsa are a subset of the Mineral Resource estimated in Item 14 of this report and are contained within the ultimate pit as detailed in this section. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

## 15.1    **Pit Optimisation**

The latest pit optimisation was carried out using Whittle Four-X software using the LG algorithm and the Mineral Resource block model constructed by FQM in December 2015. The optimisation was undertaken assuming 9 Mtpa of ore feed.

The first objective of the pit optimisation was to define the economic limits and generate a final pit outline to check against the original ultimate pit designs. The second objective was to review the push-back strategy using the current pit surface and check whether the series of staged pits smooth the stripping ratio and provide a rational development of the mine and to maximise NPV.  The stage pit shells output from Whittle were used to guide the dressed designs which incorporate haul roads, ramps, and safety berms.

Actual project operating cost data was obtained from Kevitsa's production records and the 2016 operating budget and adjusted to take account of projected improvements from current capital projects.  Load and haul costs were adjusted to take account of the different haul profiles from now until the end of mining.

As the optimisation was for the purposes of Reserve estimation, Inferred category mineralisation was treated as waste and therefore had no impact on the optimisation results.

In addition to the selection of the ultimate pit and two interim stages under base case conditions for pit design, a suite of sensitivity analyses were carried out for ± 20 % variation in metal prices as well as variable mining and process operating costs.

### 15.1.1  **Optimisation Parameters**

The block model (FQM/KMO 2015) was supplied with block grade estimates for total nickel, sulphide nickel, total copper, sulphide copper, platinum, palladium, gold, and cobalt.

**Table 15-1        Input Parameters for Kevitsa Open Pit Optimisation – December 2015**

| Optimisation Parameter | Description |
| --- | --- |
| Production limit | 9 Mt/a Process Feed |
| Nickel price | US$7.50/lb |
| Copper price | US$3.00/lb |
| Platinum Price | US$1,100/oz |
| Palladium Price | US$700/oz |
| Gold Price | US$1,200/oz |
| Rock types treated | Measured and Indicated Sulphide Mineralisation only |
| Slope parameters | North and East                                      43°<br>South and West                                      46° |

| Optimisation Parameter | Description |
|---|---|
| Mining cost (default waste) | Average: US$1.84/t<br>Surface Reference: US$1.40/t<br>Mining Positional Cost Adjustment Factors are used to model variation with depth and rock type. |
| Ore mining cost difference | Block Specific |
| Mining recovery | 97 % |
| Mining dilution | 3 % at zero grade |
| Processing cost | US$7.50/t ore processed |
| Fixed cost | US$3.52/t ore processed |
| Treatment and Transport | Ni US$ 7/lb<br>Cu US$ 0.64/lb<br>Pt  US$507.67/oz<br>Pd US$331.59/oz<br>Au US$488.88/oz |
| Royalties | Zero |
| Recoveries | Ni 66%<br>Cu 90%<br>Pt  53%<br>Pd 60%<br>Au 52% |
| Discount Rate | 10 % |

Nickel Equivalent calculation:

Following the economic parameters and averaged long term consensus pricing described above, the nickel equivalent calculation which takes into account copper, platinum, palladium, and gold credits was as follows:

- Net Value Nickel          109.35 $/10kg
- Net Value Copper        52.01 $/10kg
- Net Value Platinum        14.71  $/g
- Net Value Palladium     9.21 $/g
- Net Value Gold           22.25 $/g
- Nickel Equivalent(%) = NiS% +0.659*cu%+0.078*pt(g/t)+0.053*pd(g/t)+0.169*au(g/t)

It is noted here that although cobalt credits contribute a small amount to the annual revenue (approximately US$2.5 million per year), cobalt grade is very low in-situ and is not tracked in grade control or through the process plant. Hence cobalt is not included in the nickel equivalent calculation.

## 15.1.2  Optimisation Results

Initial analysis of pit optimisation with the base case parameters and comparison with the current Stage 4 design showed that the current Final Pit (Stage 4) matches with Shells 6 and 7 from the optimisation (around 0.7 revenue factor)

It was noted that the revenue factor 1 pit shell, Shell 12 was much larger than the Stage 4 pit (Figure 15-3 shows Shell 11 and Stage4). However, considering the revenue 1 factor pit is based on long term consensus price forecasts, it was decided that given current economic conditions, a more conservative approach towards pricing should be used with regards to pit design and reserves estimation. Accordingly, the decision was made to keep the current Stage 4 design as the ultimate pit for Reserves estimation at this point in time.  The results are shown in Figure 15-1, Figure 15-2 and Figure 15-3.

In order to assess the idea of pit expansion in the future, Shell 11 (0.95 Revenue Factor), which contains up to 315 Mt of potential mining inventory at the current long term consensus metal prices, was selected for comparison purposes. It was also noted that given Shell 11 extends to a depth of over 650 metres and the resource model indicates large zones of mineralisation outside the shell at depth, it may be necessary in future to compare open pit mining against underground mining options for the deeper mineralisation zones.

The selection of Shell 7 as the ultimate pit effectively fixes the optimisation shell at approximately $US5.25/lb Ni which is higher than current spot prices but significantly lower than the long term consensus forecasts.

**Figure 15-1        December 2015 - Base Case Pit Optimisation Results**
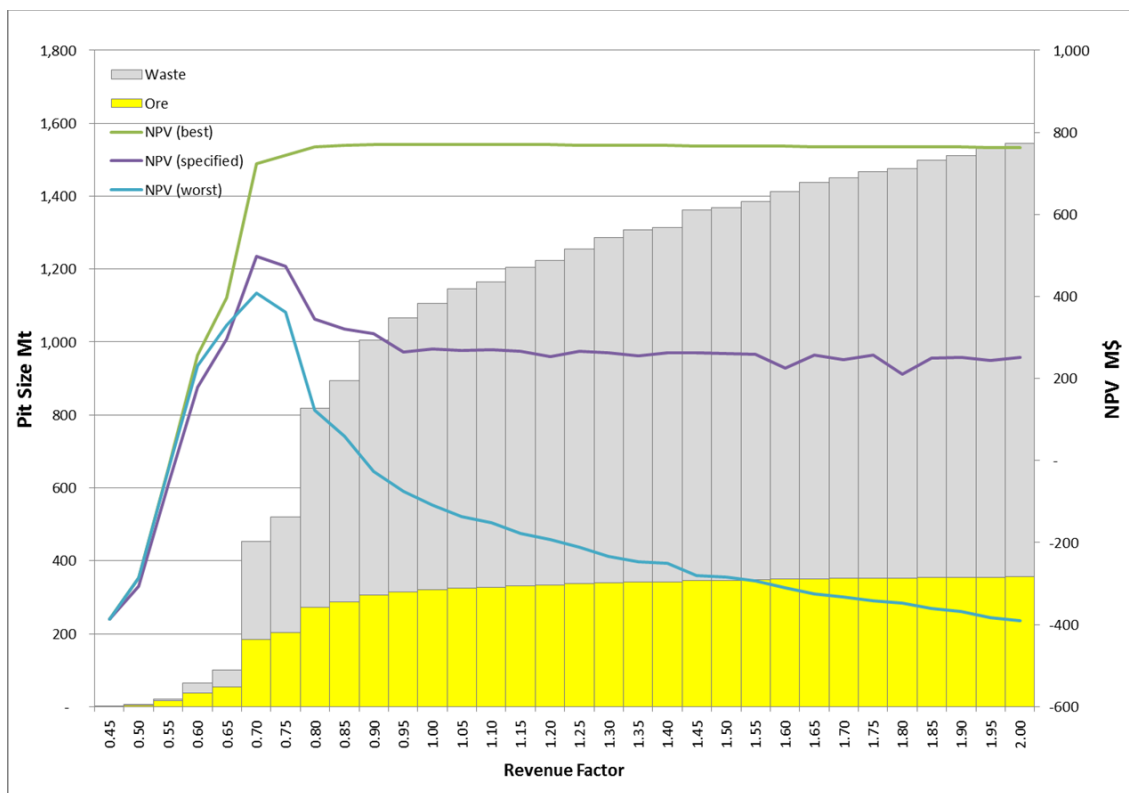
**Figure 15-2        Optimisation Shell 7 (silver) overlaid with Stage 4 Ultimate Pit (gold)**
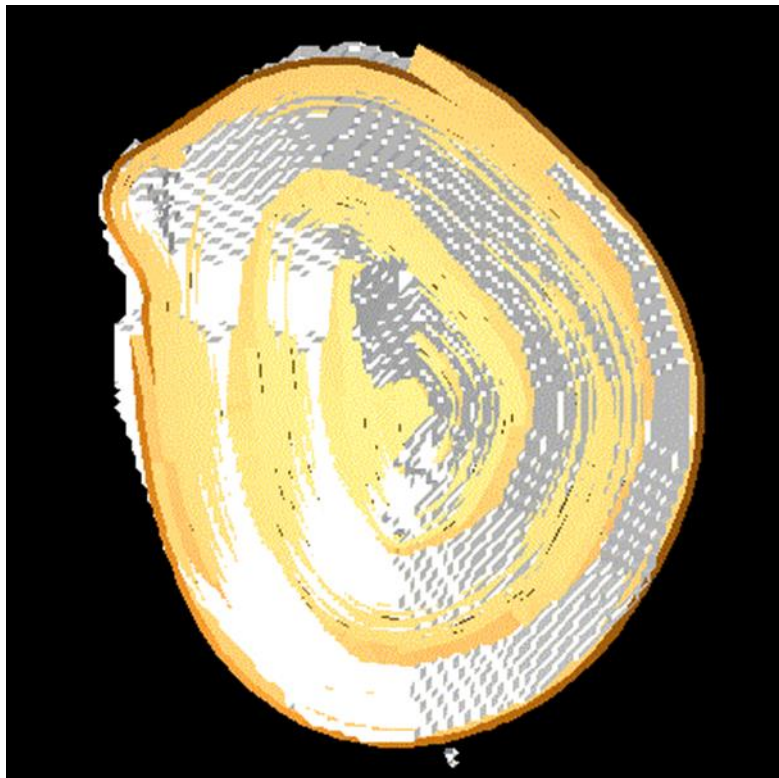


**Figure 15-3        Optimisation Shell 11 (silver) overlaid with Stage 4 Ultimate Pit (gold)**
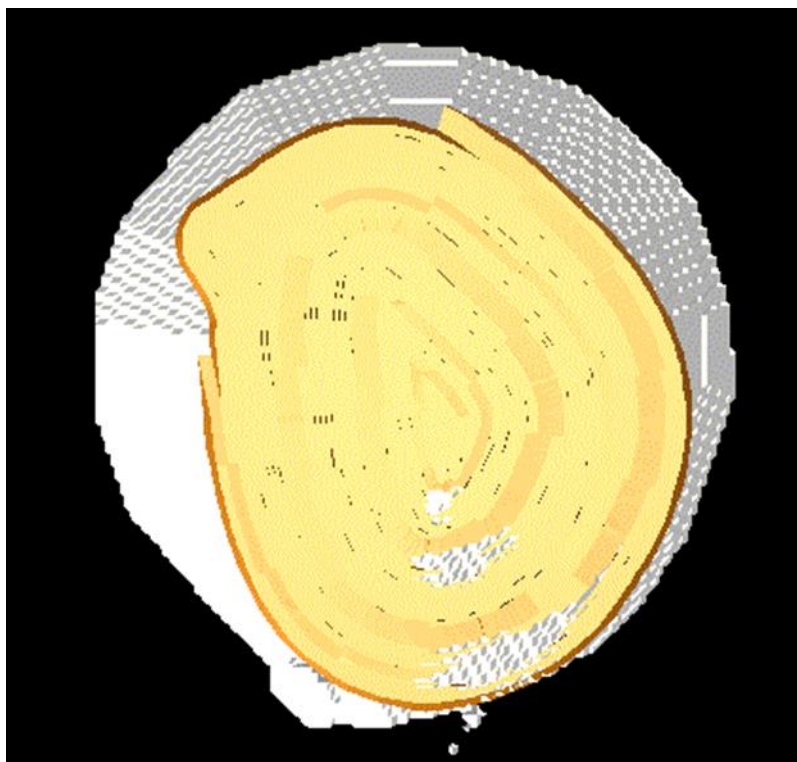
## 15.2    Marginal cut-off grades

Whittle uses the following simplified formula to calculate the marginal cut-off grade as listed in Table 15-2.

- Marginal COG = (PROCOST x MINDIL)/(NR)
- where PROCOST is the sum of the processing cost plus the ore mining cost differential, and
- MINDIL is the mining dilution factor

**Table 15-2       Kevitsa marginal cut-off grades (indicative averages)**

|                          | Units      | Oxide  |
|--------------------------|------------|--------|
| **Marginal Cut-Off Grade:** |         |        |
| PROCOST                  | $/t ore    | 11.02  |
| MINDIL factor            |            | 1.03   |
| TOTAL NET RETURN         | $/10kg     | 77.39  |
| **C/O GRADE**            | **%Ni**    | **0.15** |

It should be noted here that this is the Whittle cut-off grade calculation using the input parameters at the revenue factor of 1. It is not the cut-off grade that was used for the reserves estimate.

## 15.3    Mine Design

### 15.3.1  Open Pit Design and Planning Parameters

The pit slope parameters (Table 15-3) used for the  pit designs referred to in this report are based on the design recommendations provided by independent Geotechnical Engineer, Mike Turner of Turner Mining and Geotechnical Pty Ltd (TMG), who visits the site annually and also receives monthly data from site for review. The most recent review by TMG was in October 2015. A second geotechnical consultant, WSP, has also provided assistance in terms of collecting, analysing data, and providing reports.

Completed pits and current Stage 1 to 3 designs use the original slope design parameters of 80° batters mined in 12 metre high benches with a 10 metre wide berm every two benches (24 m).

In October 2014 a recommendation was made by TMG to modify the slope design parameters for the final pit. This revised slope parameters are shown in Table 15-3 below.

**Table 15-3       Updated Slope Design Guidelines (Source – TMG)**

| | Stage and Domain | Inter-ramp slope angle | Bench Face Angle | Bench Height | Bench width |
|---|---|---|---|---|---|
| **Overburden (average depth = 5mbs)** | | Remove overburden | | | |
| **Weathered zone (varies across pit)** | | 32° | 40° | 10m | 0m |
| | | | 90° | 12m | 20m |
| **Stage 1** | All domains | 59° | 90° | 24m | 12m* |
| **Stage 2** | All domains | 59° | 90° | 24m | 12m* |
| **Stage 3** | All domains | 59° | 90° | 24m | 12m* |
| **Stage 4** | All domains | 59° | 90° | 24m | 12m* |

- *12m berm widths for 3 benches then 24m berm width (or width to suit designs)

Ramps are all designed at 10% gradient for all Stage designs and the final pit. The ramps have different design widths depending on usage as shown Table 15-4 below.

**Table 15-4       Current Ramp Design Parameters**

| Kevitsa Ramps | | | |
|---|---|---|---|
| Gradient | | 1 in 10 | |
| Width | Trolley Assist | 55 | m |
| | Non Trolley Assist - Haulage | 40 | m |
| | Service | 25 | m |

The site is currently trialling the 90° batters described in Table 15-3 above on a section of wall in the Stage 2 pit. The purpose of this trial is to test the slopes both form an operational and stability perspective prior to committing to the design parameters for the final pit walls.

## 15.4    Pit Designs

As pointed out in section 15.1 above, the current Stage 4 pit matched well with Shell 7 of the optimisation and the decision was made to retain the Stage 4 design as the ultimate pit. When completed, the Stage 4 Kevitsa open pit will be approximately 1.2 long, 1 km wide and 500 metres deep. As of December 31st 2015, there remains 531 Mt of material (ore and waste) to be mined from the Stage 4 pit.

**Figure 15-4     Kevitsa - Stage 4 Ultimate Pit**
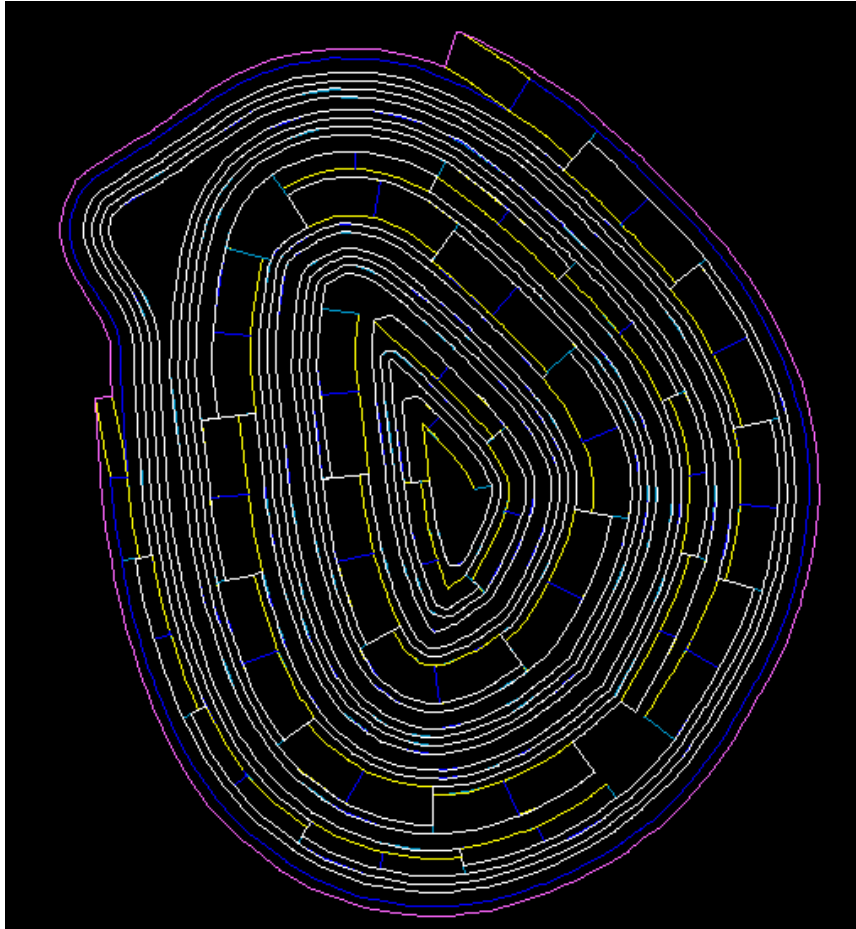


It is noted that the Stage 4 pit design will need some modification for Trolley Assist ramps in the future. The main issue being the radius of curves allowed for trucks while connected to the Trolley Assist line. FQM is currently testing curved sections of TA haulage ramps at its Kansanshi Mine in Zambia.

**Figure 15-5        Kevitsa – Current Stage 2 Design**
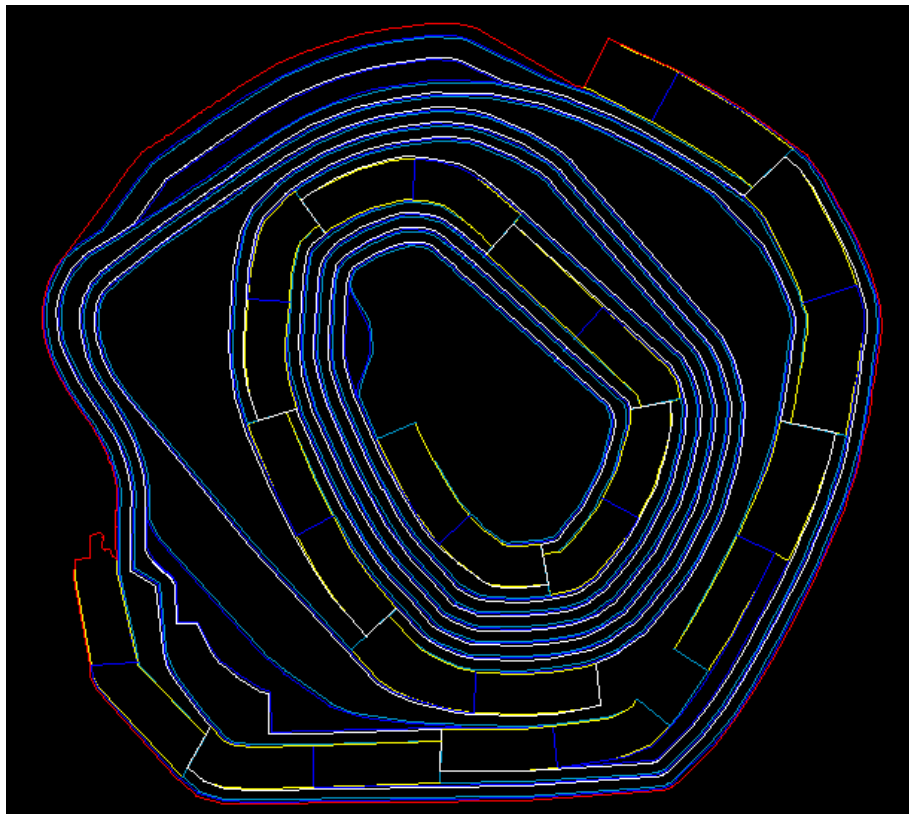


**Figure 15-6        Kevitsa – Current Stage 3 Design**
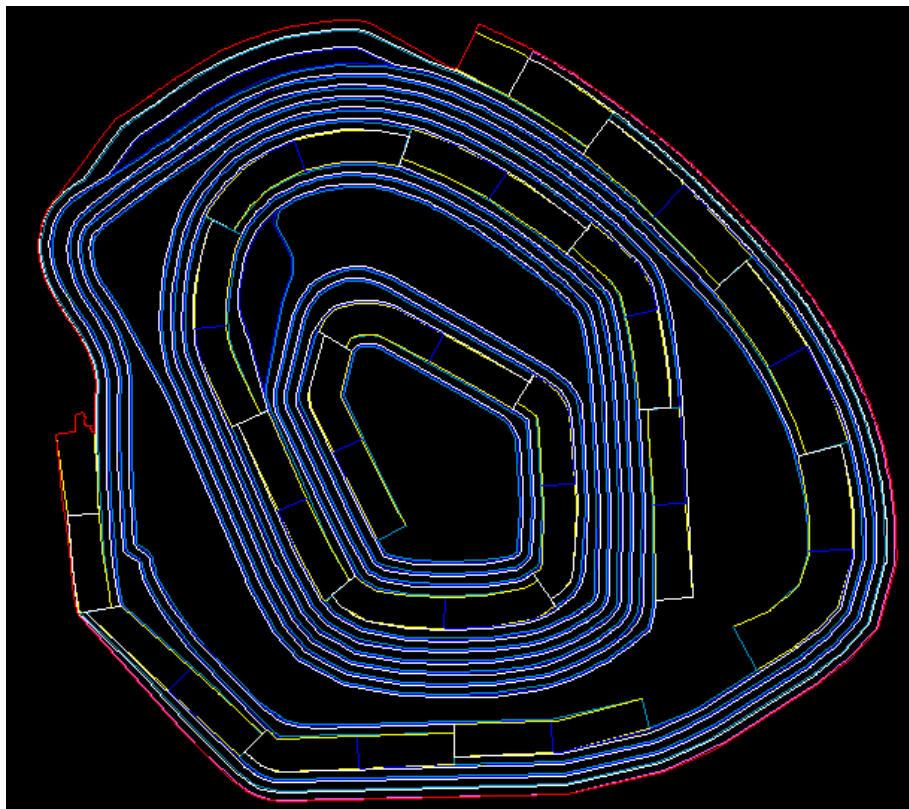
Stages 2 and 3 are site developed designs that are in use for guiding current mining activities and short to medium term scheduling. These designs are also in the process of being modified for Trolley Assist.

As noted in section 15.1 of this report, there is a possibility that the pit could be extended to Stage 5 in the future. This concept is still being developed and design options and plant throughput rates have not been finalised hence are not ready for publication or use in LOM schedules.

## 15.5    Stockpile Inventories

As at the end of December 2015, a total of 133 kt tonnes of ore at an average grade of 0.19% NiS and 0.25% Cu was on the ROM Stockpile.

## 15.6    Life of Mine Schedule

The life of mine production schedule was prepared using Gemcom Minesched mining software and the results are summarised shown in Table 15-5 below.

This schedule was developed using the FQM/KMO 2015 block model and the pit designs detailed in section 15.2 of this report. The process plant feed rate was set at 7.6 Mtpa for 2016/17 ramping up to 9 Mtpa from 2018 onwards. The mining fleet capacity was capped at 46 Mtpa until the end of 2018 and then increased to 50 from 2019 onwards. The required mining movement decreases from 2022 onwards once the main orebody is exposed in Stage 4 and the pit deepens.

The main focus of this round of scheduling was to try and increase the grades in the early years and also delay waste stripping as much as possible without adversely affecting metal production. Previous schedules had been run in different economic conditions and had brought waste forward in order to allow more flexibility in ore processing.

**Table 15-5        Kevitsa Life-of-Mine Production Schedule**

| | | UNITS | | TOTAL | 2016 | 2017 | 2018 | 2019 | 2020 | 2021 - 2025 | 2026-2030 | 2030-2033 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **MINING** | | | | | | | | | | | | |
| | Waste | Tonnes | Mt | **376.7** | 36.3 | 38.3 | 36.3 | 41.7 | 40.4 | 151.2 | 24.8 | 7.6 |
| | Ore | Tonnes | Mt | **154.37** | 9.8 | 7.7 | 9.7 | 8.1 | 9.7 | 46.5 | 45.1 | 17.8 |
| | Total Mined | Tonnes | Mt | **531.0** | 46.1 | 46.0 | 46.0 | 49.7 | 50.1 | 197.7 | 70.0 | 25.4 |
| | Strip ratio | | t:t | **2.4** | 3.7 | 5.0 | 3.7 | 5.2 | 4.2 | 3.2 | 0.5 | 0.4 |
| **PROCESSING SUMMARY** | | | | | | | | | | | | |
| | Feed to Plant | Tonnes | Mt | **154.5** | 7.6 | 7.6 | 9.0 | 9.0 | 9.0 | 45.0 | 45.0 | 22.30 |
| | | Ni | % | **0.24%** | 0.25% | 0.25% | 0.24% | 0.23% | 0.23% | 0.23% | 0.25% | 0.27% |
| | | Cu | % | **0.35%** | 0.32% | 0.34% | 0.34% | 0.31% | 0.33% | 0.32% | 0.38% | 0.38% |
| | | Pt | g/t | **0.22** | 0.31 | 0.30 | 0.30 | 0.24 | 0.23 | 0.24 | 0.18 | 0.14 |
| | | Pd | g/t | **0.15** | 0.21 | 0.19 | 0.19 | 0.15 | 0.14 | 0.17 | 0.12 | 0.09 |
| | | Au | g/t | **0.11** | 0.14 | 0.14 | 0.13 | 0.10 | 0.11 | 0.11 | 0.10 | 0.09 |
| | | | | | | | | | | | | |
| | Plant Recovery | Ni | % | **66.1%** | 66.4% | 66.4% | 66.3% | 65.9% | 65.9% | 65.9% | 66.3% | 66.0% |
| | | Cu | % | **89.9%** | 89.8% | 89.9% | 90.0% | 89.7% | 89.8% | 89.7% | 90.3% | 89.8% |
| | | Pt | % | **53.0%** | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% |
| | | Pd | % | **61.0%** | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% |
| | | Au | % | **52.0%** | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% |
| | Metal in Concentrate | | | | | | | | | | | |
| | | Ni | Kt | **247.3** | 12.6 | 12.6 | 14.6 | 13.4 | 13.5 | 67.4 | 73.5 | 39.7 |
| | | Cu | Kt | **479.6** | 21.8 | 23.0 | 27.8 | 25.1 | 26.3 | 127.7 | 152.5 | 75.4 |
| | | Pt | koz | **574.1** | 40.1 | 38.5 | 45.6 | 36.6 | 34.6 | 185.4 | 140.5 | 52.8 |
| | | Pd | koz | **445.8** | 30.7 | 28.3 | 33.5 | 27.2 | 24.1 | 152.8 | 109.1 | 40.2 |
| | | Au | koz | **278.7** | 17.8 | 17.2 | 20.2 | 14.5 | 16.5 | 83.0 | 76.6 | 32.8 |

More details of the LOM schedule are discussed in Section 16.4 of this report.

## 15.7    Mineral Reserve Tabulation

Table 15-6 shows the NI 43-101 compliant reserve estimate for sulphide mineralisation contained within the final pit (Figure 15-4) after applying modifying factors to the Measured and Indicated Resources as defined in the 2015 Mineral Resource statement.

**Table 15-6    Kevitsa Mineral Reserve Estimate depleted as at end December 2015 using a 0.22% NiSEq cutoff grade.**

| Kevitsa Ore Reserve Estimate December 31 2015 | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | Tonnes (Mt) | Ni % | Ni(S) % | Cu % | Cu(S) % | Au g/t | Pd g/t | Pt g/t |
| Pit Only | | | | | | | | |
| Proven Reserve | 88.1 | 0.25 | 0.23 | 0.36 | 0.34 | 0.11 | 0.14 | 0.22 |
| Probable Reserve | 66.3 | 0.28 | 0.25 | 0.36 | 0.34 | 0.11 | 0.15 | 0.22 |
| Total | 154.4 | 0.27 | 0.24 | 0.36 | 0.34 | 0.11 | 0.15 | 0.22 |
| Stockpile | | | | | | | | |
| Proven Reserve | 0.1 | 0.23 | 0.19 | 0.25 | 0.24 | 0.09 | 0.16 | 0.21 |
| Probable Reserve | 0.0 | - | - | - | - | - | - | - |
| Total | 0.1 | 0.23 | 0.19 | 0.25 | 0.24 | 0.09 | 0.16 | 0.21 |
| Total Pit + Stockpile | | | | | | | | |
| Proven Reserve | 88.2 | 0.25 | 0.23 | 0.36 | 0.34 | 0.11 | 0.14 | 0.22 |
| Probable Reserve | 66.3 | 0.28 | 0.25 | 0.36 | 0.34 | 0.11 | 0.15 | 0.22 |
| Total | 154.5 | 0.27 | 0.24 | 0.36 | 0.34 | 0.11 | 0.15 | 0.22 |

This Mineral Reserve estimate detailed in Table 15-6 has been produced using this Mineral Resource estimates block model, appropriate modifying factors, and a 0.22% NiSeq cut-off grade which is based on a nickel price of $6.05/lb, a copper price of $2.50/lb, a gold price of $1,159/oz, a palladium price of $732/oz and a platinum price of $1,118/oz (average consensus prices for 2016 to 2018). This cutoff grade is used to define the updated reserve contained within the ultimate pit design up to stage 4. Consideration of stage 5 remains an opportunity as already stated.

Using these prices along with the input parameters in Table 15-2, the nickel equivalent formula was adjusted for Resources and Reserves reporting to:

$Ni\_eq = NiS\% + 0.722*CuS\% + 0.081*Pt(g/t) + 0.056*Pd(g/t) + 0.219*Au(g/t)$.

# ITEM 16   **MINING METHODS**

Portions of the following information regarding the history are sourced from the 2011 Technical Report (CSA/Dumpsolver , May 2011).  The remainder has been updated to reflect the current status as at December 2015.

## 16.1     Mining Details

### 16.1.1  Mine Site Layout

**Figure 16-1        Plan view of the Kevitsa mine site, illustrating the pits, dumps and tailings facilities.**



### 16.1.2  Mining Method

The Kevitsa mine is a moderate-sized open pit mining operation using conventional truck and shovel operations.   KMO has a base mining fleet that is owned by the company and uses a mining contractor to assist mining ore and waste. KMO employs a technical group to supervise the mine contractor.  In 2015 the mine delivered 6.6 Mt of ore to the ROM pad and Process Plant. In 2016/17 onwards the mine is expected to produce 7.6 Mt tonnes of ore and from 2018 onwards, production will be 9 Mtpa.

Mine development required a pre-production phase of almost 12 months for pre-stripping to expose sufficient ore to ensure steady ore production.  The open pit mining method utilizes drilling and blasting, loading with hydraulic excavators, and transport by trucks. The overall pit slope angle will

be 43° to 46° including ramps. Trucks haul the ore to the Process plant where it is either direct tipped into the primary crusher or stockpiled on the ROM Pad for blending.

The waste dump is located to the north-east of the pit (refer figure 16-1). During the feasibility phase of the project, waste rock was analysed for acid generating potential. The resultant dumping guidelines were developed and are being adhered to: -

- Where Sulphur is >0.8% the rock is treated as PAF and is encapsulated with NAF waste. Aprroximately 20% of the total waste rock is expected to be PAF
- Where Sulphur content is between 0.3% and 0.8%, the waste rock is treated as normal waste rock and dumped in accordance with the design and regulations.
- Low sulphur waste (<0.3%S) is classified as NAF and used for encapsulation and construction.

The waste dump design has been modified recently to remove a small eastern section that would have covered a swamp area. Apart from that small modification, the design and parameters have not changed since the Feasibility study.

The stages are designed such that they generally comprise wide benches of between 50 m and 200 m in width, providing several mining horizons to satisfy the feed requirements for blending. In general, ore is hauled to a ROM Pad located immediately south of the pit, whereas waste is hauled to the waste dump located to the north and east of the ultimate pit.

The stages are designed such that they generally comprise wide benches of between 50 m and 200 m in width, providing several mining horizons to satisfy the feed requirements for blending. In general, ore is hauled to a ROM Pad located immediately south of the pit, whereas waste is hauled to the waste dump located to the north and east of the ultimate pit.

KMO commenced mining operations in 2012. As at end of December 2015, the pit was approximately 130 metres deep, 900 m long, and 800 m wide. Mining has proceeded from initial excavations in Stage 1 through to stage 2 of a planned sequence of 4 stages. At the projected mining rate and based on the latest updated mineral reserves as of December 2015 the pit will be completely mined out by early 2033 and the mining operation will be scaled down for stockpile rehandle and processing until mid-2033 with minimum mining fleet and staff.

### 16.1.3  Mining Fleet

KMO has a mixed fleet comprising of large efficient owner operated equipment to undertake the base movement load and mining contractors with smaller more flexible mining equipment to mine areas and materials that don't allow the KMO equipment to operate efficiently.

KMO plans to install Trolley Assist on the main ramp in mid-2016 in order to take further advantage of the larger low unit cost truck fleet.

**Figure 16-2      Kevitsa Mine – December 2015**



The current primary earthmoving fleet at Kevitsa comprises the following:

• 1 x Komatsu PC8000 and 1 x Komatsu PC5500 Electric/Hydraulic Shovels
• 4 x Caterpillar 795F AC and 6 x Caterpillar 793F rigid bodied haul trucks
• 1 x Komatsu WA1200 FEL

The support fleet consists of conventional equipment such as graders (16), tractor dozers (D11 and D-10 sizes), RTDs (Cat 854) and FELs (Cat 980), service and lube trucks, etc.

Drilling is performed by 2 x Atlas Copco Pit Viper 271 Electric drill rigs and 4 x Atlas Copco SmartROC D65 Deisel powered drill rigs. Dust suppression on haul roads is performed with two water tankers.

KMO employs a number of Contractors in conjunction with its owned fleet. These include:

• Mining Contractors; Hartikaienen ( about 3 Million Bcm/an). Mainly ore mining and restricted area:
  o Maintenance Contractors:
  o WITRAKTOR for Caterpillar Equipment
  o SRO for Komatsu ( with support for KMG )
  o Atlas Copco for Drills
• Various Contractors for fabrication works, welding ,bucket and body works

Blasting Contractor- Orica.

## 16.2     Mining Sequence

The mining sequence broadly follows the sequence of events as follows:

• RC grade control drill holes will delineate the ore zones
• Blast patterns designed to reduce material throw and ore dilution – and a Blast Master planning process controls sequence of operation

- Ore and waste blasted and mined separately as fragmentation requirements vary significantly
- Waste removed on each 12 metre bench subsequent to the mining of ore
- The removal of waste in the successive cut-backs utilises planned bulk systems of operation
- Trim blasts and perimeter blasting will be utilised to ensure pit wall profiles are cut to the correct angle and wall damage minimised
- Face shovels will load rock into the 225 tonne class trucks and ore will be hauled from the pit to the crusher and will be tipped directly into the crusher
- Finger stockpiles will also be an integral part of the feed sequence to ensure ore blending can be achieved, haulage efficiencies can be maximised and operational flexibility enhanced at all times

### 16.2.1 Grade Control Drilling and Modelling

In order to assist with medium term planning, all grade control drilling is done using RC drill rigs on a staggered pattern with the aim of getting coverage to at least 30 metres below the working benches. Details of the grade control process can be found in section 10.2 of this report.

### 16.2.2 Drilling and Blasting

Rock fragmentation is a critical part of the mining process. Benches are 12 metres high in high strength rock. Large diameter holes are drilled with down-the-hole hammer machines (Pit Vipers and D65 SmarrROC).

A preliminary KUZRAM fragmentation model is also used to estimate the fragmentation distribution.

Where possible, ore is blasted separately from waste. Perimeter blasting is undertaken on final walls to prevent blast damage and to maintain wall control.

## 16.3    Mine Planning Considerations

### 16.3.1 Mine Design Parameters

The Mine design parameters are discussed and presented in section 15.2 of this report.

### 16.3.2 Production Reconciliations

As per the grade control process, ore mark outs are assigned evaluated grades as per the grade control block model and are assigned to the digging and truck despatches for the respective ROM stockpiles and waste dumps.  Similarly, each ROM stockpile tonnage and grade is monitored via depletions to the processing facility and additions from the mined material.  Regular (monthly) pit surveying allows for accurate assignment of tonnes and grades mined and despatched to the respective destinations.  Accordingly, feed to the plant is known and is verified with a milled measurement for reconciliation of tonnes and grade.  Final metal generated as concentrate is reconciled back to the declared tonnes and grades.  As the Resource model has recently been updated, there has been no reconciliation of new production carried out however reconciliation of the December 2015 Resource model against historical production indicates a reasonable correlation.

### 16.3.3  Mining Dilution and Recovery

Where possible, ore and waste are blasted and mined separately to minimise loss and dilution. Loss and dilution (estimated at 3-4%) has been built in to the geological model by the use of block sizes matched to the SMU size.

Dilution and loss should be continuously reviewed annually against reconciliation data between the Mineral Resource estimate and the actual mine and plant production.

### 16.3.4  Geotechnical Engineering

As noted in Section 15.3.1, the pit design parameters used during for the  Kevitsa pit designs rare based on the latest update from independent Geotechnical Engineer Mike Turner of Turner Mining Consulting who visits site annually and reviews data from site on a monthly basis. A second geotechnical consultant WSP has also collected and reviewed geotechnical data which has been utilised by Turner Mining when making design and operational recommendations.

As well as the regulated review, KMO monitors the pit slopes as follows: -

1. Topographic – GPS based – Post processed - checking control points using one GPS as the base point and the other as a moving one going to set positions. Weekly
2. Peizometric and Clinometric network. Checked weekly by KMO geotechnical employees.
3. 3D station used as a 3D scanner.

The pit walls show some minor bench failures in the upper overburden levels (mostly related to blast damage) and there has also been of two larger wedge failures that were both investigated by the KMO team and the geotechnical consultants.

### 16.3.5  Mine Dewatering

To date water encountered in the pit has come from the intersection of perched water tables. The pit is dewatered in advance using sumps in order to reduce pore pressures in the pit wall and maintain a safe and efficient mining operation.  Plans are in place to try to use a combination of ex-pit and in-pit dewatering bores with in-pit sumps are used to collect the water. Semi-permanent pipes, booster pumps, and distribution lines would then be used to remove the water from the pit area.

### 16.4      Mining and Processing Schedules

Kevitsa mining started with a 12 month pre-stripping program that exposed the initial ore in the pit, in an area of relatively high-grade mineralisation.  Ore production commenced in the middle of 2012 and the plant achieved commercial production at the initial 5 Mtpa capacity in August 2012. The mine has been operating for almost 4 years and since much of the plant was designed and built for higher throughput rates, additional equipment and process facilities have been installed since commissioning to allow the plant to treat up to 9.0 Mtpa. This additional equipment includes:

- Additional secondary MP800 crusher
- Primary screen upgrade
- Second 7MW motor on pebble mill

- Ball charging on pebble mill.

It should be noted that achieving the 9.0 Mtpa process rate requires the use of steel balls to maximise power draw. This has been scheduled for 2018 when the consensus metal price forecasts indicate that that additional cost of using steel balls will be economically viable. An option to increase to the plant capacity to 11 Mtpa requires further capital expenditure of US$67.2 MM.

The current LOM schedule runs until mid-2033 and has been developed using a series of 4 stages, designed such that ore will be exposed without requiring excessive stripped waste inventories. Currently Stage 2 is almost developed to the point that sufficient ore is exposed and available to be able to target higher grade ore feed to the plant from late 2016 onwards. Waste movement in Stage 3 can also be delayed if necessary to conserve cash now without adversely affecting the ore feed later on. The geometry of the deposit is sufficiently well known that, provided waste mining is maintained in accordance with the schedules, ore exposure is not likely to be a problem.

The primary drivers of the schedule are maximizing the grade and meeting the ore feed targets.  The mine and plant production schedule for the remaining life of the current proved and probable reserve is 18 years and shown in Table 16-1.

**Table 16-1        Kevitsa Life of Mine Schedule.**

| | | UNITS | | TOTAL | 2016 | 2017 | 2018 | 2019 | 2020 | 2021 - 2025 | 2026-2030 | 2030-2033 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **MINING** | | | | | | | | | | | | |
| | Waste | Tonnes | Mt | **376.7** | 36.3 | 38.3 | 36.3 | 41.7 | 40.4 | 151.2 | 24.8 | 7.6 |
| | Ore | Tonnes | Mt | **154.37** | 9.8 | 7.7 | 9.7 | 8.1 | 9.7 | 46.5 | 45.1 | 17.8 |
| | Total Mined | Tonnes | Mt | **531.0** | 46.1 | 46.0 | 46.0 | 49.7 | 50.1 | 197.7 | 70.0 | 25.4 |
| | Strip ratio | | t:t | **2.4** | 3.7 | 5.0 | 3.7 | 5.2 | 4.2 | 3.2 | 0.5 | 0.4 |
| **PROCESSING SUMMARY** | | | | | | | | | | | | |
| | Feed to Plant | Tonnes | Mt | **154.5** | 7.6 | 7.6 | 9.0 | 9.0 | 9.0 | 45.0 | 45.0 | 22.30 |
| | | Ni | % | **0.24%** | 0.25% | 0.25% | 0.24% | 0.23% | 0.23% | 0.23% | 0.25% | 0.27% |
| | | Cu | % | **0.35%** | 0.32% | 0.34% | 0.34% | 0.31% | 0.33% | 0.32% | 0.38% | 0.38% |
| | | Pt | g/t | **0.22** | 0.31 | 0.30 | 0.30 | 0.24 | 0.23 | 0.24 | 0.18 | 0.14 |
| | | Pd | g/t | **0.15** | 0.21 | 0.19 | 0.19 | 0.15 | 0.14 | 0.17 | 0.12 | 0.09 |
| | | Au | g/t | **0.11** | 0.14 | 0.14 | 0.13 | 0.10 | 0.11 | 0.11 | 0.10 | 0.09 |
| | | | | | | | | | | | | |
| | Plant Recovery | Ni | % | **66.1%** | 66.4% | 66.4% | 66.3% | 65.9% | 65.9% | 65.9% | 66.3% | 66.0% |
| | | Cu | % | **89.9%** | 89.8% | 89.9% | 90.0% | 89.7% | 89.8% | 89.7% | 90.3% | 89.8% |
| | | Pt | % | **53.0%** | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% |
| | | Pd | % | **61.0%** | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% |
| | | Au | % | **52.0%** | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% |
| | Metal in Concentrate | | | | | | | | | | | | |
| | | Ni | Kt | **247.3** | 12.6 | 12.6 | 14.6 | 13.4 | 13.5 | 67.4 | 73.5 | 39.7 |
| | | Cu | Kt | **479.6** | 21.8 | 23.0 | 27.8 | 25.1 | 26.3 | 127.7 | 152.5 | 75.4 |
| | | Pt | koz | **574.1** | 40.1 | 38.5 | 45.6 | 36.6 | 34.6 | 185.4 | 140.5 | 52.8 |
| | | Pd | koz | **445.8** | 30.7 | 28.3 | 33.5 | 27.2 | 24.1 | 152.8 | 109.1 | 40.2 |
| | | Au | koz | **278.7** | 17.8 | 17.2 | 20.2 | 14.5 | 16.5 | 83.0 | 76.6 | 32.8 |

# ITEM 17   RECOVERY METHODS

This section provides a brief description of the ore processing techniques, process plant equipment operation, and process plant support facilities.   Process plant capital and operating costs are discussed in Item 22. Information regarding the history of the metallurgical test work and processing was presented in the 2011 Technical Report (CSA/Dumpsolver, May 2011) and is summarised in Item 13 of this report.

## 17.1     Processing Overview

The plant design and operating philosophy follows that of typical operations in cold weather climates, and particularly the newer operations in northern Finland.

The plant capacity is based on the treatment of 9.0 Mtpa of ore, treated over 8,000hrs per year (91.3% running time), giving a nominal milling and treatment rate of 1,125 tph.  The front end of the plant – primary crushing, screening, stockpile and conveying - is designed for a higher throughput of 10Mtpa to cater for future expansions.

As shown in the reserves table (Table 16-1), the average metal contents of Kevitsa ore are 0.24% Ni (as a sulphide) and 0.34% Cu (also as a sulphide).  Whilst a small portion of the nickel in the ore occurs in a non-floatable form in silicates and goethite, the main sulphide minerals are pyrrhotite, chalcopyrite and pentlandite, and the main gangue minerals are Clinopyroxene, triolite, olivine, serpentine and talc.

The current production level is about 130,000 tons per annum of nickel concentrate with a nickel content of 11% and about 115,000 tons per annum of copper concentrate with a copper content of 25%.

The recoveries are 66% of the sulphide nickel in feed, and 90% of the sulphide copper in feed (78% reports to the copper concentrate and an additional 12% copper recovery is accounted for in the nickel concentrate).

## 17.2     Process Description

The circuit is currently capable of treating 9 Mtpa of ore, which is equivalent to 1,125 tph at 91.3% (8,000hrs per year) availability. A block flowsheet summarizing the process is provided below (Figure 17-1).

The following unit processes comprise the Kevitsa Metallurgical facility:

- Primary crushing of run-of mine (ROM) ore from the open pit (delivered by dump truck)
- Screening of the primary crushed ore to produce three products – coarse lumps and fines as feed to the AG mills, and a mid-size product for the pebble mill.
- Pebble storage bin 750t live capacity
- Crushing of excess pebbles
- A single stockpile of the mixed coarse and fine ore, with 15,000t live capacity (16.7h).
- Two 7MW AG mills operating in parallel on material fed from the stockpile

- The AG mills operate in partial closed circuit with hydrocyclones, and with transfer of AG mill discharge slurry to the pebble mill by pump.  Cyclone overflow is finished product
- A single pebble mill in closed circuit with cyclones to produce a final product ($P_{80}$) size of 75µm.
- Sequential flotation of copper and nickel concentrates
- Copper flotation cleaning in four stages with regrind of scavenger concentrates product.
- Nickel flotation cleaning in five stages with regrind of the 2nd cleaner concentrate product.
- Flotation of sulphide rich concentrate from the nickel scavenger flotation tails to produce a low sulphur content tailings with low acid forming capacity.
- Dewatering of Cu and Ni concentrates by thickening and filtration
- Deposition of primary tailings into conventional (unlined) tailings storage facility (TSF)
- Deposition of sulphide rich concentrate into a dedicated lined tailings storage facility.
- Reagent make-up, storage and dosing systems.
- Water services and reticulation systems.
- Compressed and instrument air generation and reticulation systems.

The circuit is described in more detail in Figure 17-1 below:

**Figure 17-1        Kevitsa Process Flowsheet**
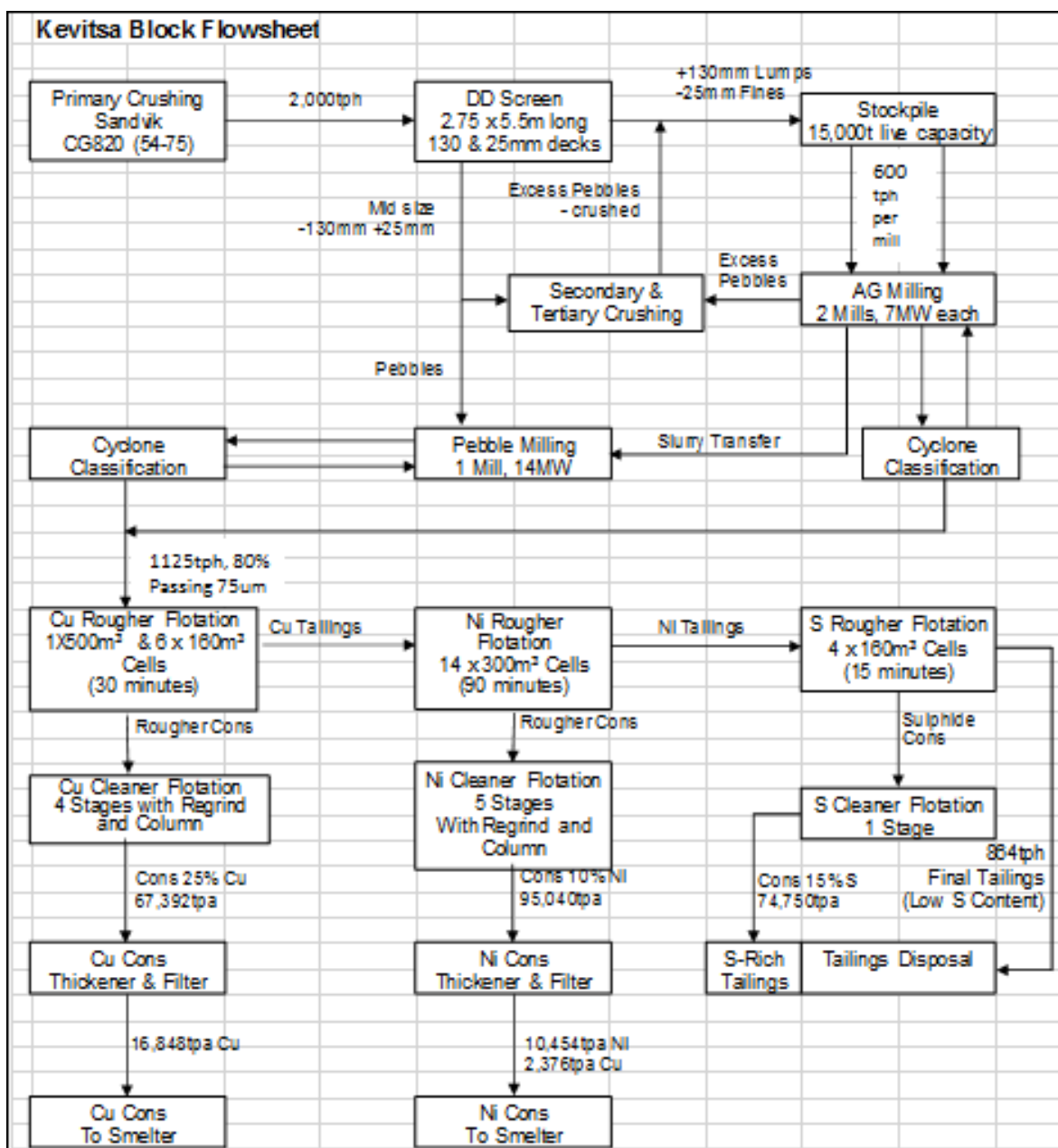


### 17.2.1  Primary Crushing

Ore Feed:

Run-of-mine ore is delivered to the crusher dump pocket by mine trucks with a capacity of 240 tonnes.  The live capacity of the primary crusher dump pocket is 400 tonnes - or two truckloads, and sufficient for 17 minutes operation of the crusher circuit.

The dumping of ore is regulated from the control room using green 'Dump' and red 'No Dump' traffic lights.

Primary Crusher:

The primary crusher is a Sandvik CG820 gyratory crusher with 28 millimetre eccentric throw. This crusher is sized to handle a maximum rock size of about 1,200 millimetres and is equipped with a 535 kW drive.

The gyratory crusher product discharges directly into a surge pocket immediately below the crusher. Crushed ore is withdrawn from the surge pocket by a variable speed feeder onto a 1,200mm wide crusher discharge conveyor, which conveys the primary crushed ore to the screening plant.

Any tramp iron in the ore is removed from the conveyor by a self-cleaning magnet. A weightometer is also installed on this conveyor, for metallurgical accounting purposes.

Services in the crusher area include a compressor to provide high pressure (850 kPa) air for instrumentation and the dust collection system, and a crane for maintenance purposes.

The crusher is installed underground, with the top of the feed pocket being at ground level. This reduces the elevation to which the trucks have to drive to, thereby reducing fuel requirements and noise.

### 17.2.2  Pebble Screening & Crushing

The primary screen is 2.8 m wide by 5.5 m long, and is fed directly by the crushed ore conveyor. The screen is a double deck screen and has two functions – production of a coarse lump for grinding media in the AG mills, a mid-size pebble for grinding media in the pebble mill. The bottom deck undersize is conveyed to the AG mill stockpile with the coarse lumps, and the mid-size material – typically being between 25 and 130 mm in size is conveyed to a pebble bin.

The quantity of top size material from the screen can be varied through controlled blasting (mine to mill optimization) and by changing the screen's top deck aperture. The quantity of mid-size product generated is controlled by the secondary crusher circuit.

Mid-size pebbles discharge onto the secondary crushing feed conveyor, which delivers the pebbles to a combined pebble surge and secondary crusher feed bin. This 1200 m wide conveyor discharges material onto a vibrating screen located above the bin, which is equipped with a 60mm aperture deck, to provide segregated feed to the secondary and tertiary crushers.

This bin is designed to provide approximately 750 tonnes of live capacity for pebble feed to the pebble mill, allowing excess pebbles to be crushed and returned to the main circuit. The bin also receives pebbles generated from the AG milling circuit.

Pebbles are drawn from the bin by conveyor, at a rate set by the power draw on the pebble mill. Excess pebbles are crushed in two MP800 Cone crushers equipped with a 600 kW drive each. A secondary and tertiary crusher are employed for 9 Mtpa, operated at a closed side setting of 22 mm and 12 mm respectively, to give a product size of 80% passing 25 mm.  When the pebble mill is operated as a ball mill, all pebbles will be crushed in this circuit.

Discharge from the secondary and tertiary crushing is collected on a single conveyor belt and returned to the stockpile feed conveyor, thus rejoining the lump and fine material from the primary screen. The stockpile feed conveyor is a 1,200 mm wide belt sized for a capacity of 2,600 tph (for future operations at a throughput targeting 11 Mtpa).

Weightometers on the main belts are used to reconcile tonnages of crushed ore and pebbles feeding the milling circuit.

Services in the area comprise a 20 t bridge crane above the primary screen and a similar crane over the secondary crushers for maintenance purposes.  There is also a dust collection system by the primary screen.

The apron feeder on the gyratory crusher discharge and all conveyors are equipped with the usual operational and safety devices.

### 17.2.3  Crushed Ore Stockpile

The crushed ore stockpile is 62 m in diameter and 24 m high, and has a live volume of about 8,000 m$^3$, equivalent to 15,000 tonnes, providing approximately 16.7 hours of ore storage.

The stockpile is covered for dust control and to prevent the ingress of snow during the winter.

Ore is withdrawn from the stockpile using 4 variable speed apron feeders – two per AG mill.  The feeders discharge ore onto the mill feed conveyors; each of which has a mill feed weightometer to control the speed of the feeders, and to totalize the mill feed tonnage.

The mill feed conveyors are 1,000 mm wide and equipped with 55kW drives.

The stockpile discharge feeders and mill feed conveyors are equipped with the usual operational and safety devices described above.

### 17.2.4  AG & Pebble Milling

The milling circuit comprises 2 primary AG mills and a single pebble mill.  Each mill is 8.5 m in diameter (inside shell) and 8.5 m long (EGL).  The primary mills are equipped with single pinion 7 MW drives; one of the AG mills is variable speed, and the other mill is fixed speed.  The fixed speed mill runs at 75% critical, and the variable speed mill has a range of 65 to 80% critical. The pebble mill is equipped with two 7 MW drives.

All mills are grate discharge mills.  The mills are lined with rubber liners with integral steel caps.  A liner handler is used for relining these mills.

AG Milling:

Each AG mill is fed by a fixed speed 1,000 mm wide belt; the mill feed rate is controlled from a weightometer on each belt varying the speed of the respective stockpile discharge feeder.  Water addition to the mill feed is controlled from the weightometer as a direct ratio of the mill feed.

AG mill discharge slurry normally has a density of between 68% and 72% solids.  Discharge slurry gravitates over a vibrating screen (one screen per mill) into a discharge sump, where it is diluted with process water to a density of approximately 55% solids and pumped to the hydrocyclones.

Cyclone pump speed is controlled by the level in the mill discharge sump; dilution water is added to maintain a constant cyclone feed density.  The pressure at the inlet to the cyclones is in the range of 100 kPa to 130 kPa.

A second variable speed pump on each AG mills discharge sump is used to transfer slurry from the AG milling circuit to the pebble mill.  The speed of this pump controls the amount of material transferred to the pebble mill, and this is dictated by the power draw of the AG's and the pebble mills.

The classification circuits operate in closed circuit with each of the two AG mills and consist of nests of 8 x 380 mm diameter cyclones.  The number of operating cyclones is automatically controlled to maintain a steady distributor feed pressure, and is normally 6, with 2 on standby.

Cyclone underflow slurry at a density of approximately 70% solids is returned to the AG mills. Cyclone overflow slurry at 30 to 35% solids gravitates to the flotation feed surge tank.

The AG mills discharge vibrating screen is equipped with 8mm wide by 25 mm long slots -screen oversize is recycled to the secondary crusher feed bin via a common pebble conveyor.

Pebble Milling

Secondary grinding take place in a grate discharge pebble mill.  The mill is identical in size to the AG mills and is driven by two 7 MW motor at 75% critical speed.

Mill feed comprises AG mill discharge slurry, transferred by centrifugal pumps on the AG mill discharge sumps, and pebbles for grinding media, from the pebble bins.  When the mill is operated as to a ball mill, grinding media will be fed via an indexer and high lift ball charging conveyor (already installed).

Currently, pebbles are drawn from the pebble bin by a variable speed belt feeder discharging onto the pebble conveyor, which is equipped with a weightometer for process control and metallurgical accounting.  Pebble requirements for the mill are dictated by mill power draw or cyclone overflow size distribution

The pebble mill is operated in closed circuit with its own dedicated hydrocyclones, in much the same way as the AG mills.  Pebble mill cyclone overflow slurry joins that from the AG mill cyclones and gravitate to the flotation feed surge tank, whilst underflow slurry is recycled to the mill.

An on-stream particle size analyzer PSI 500 with sampling equipment monitors the fineness of hydrocyclone overflow of all three grinding circuits; the required grind size is 80% passing 75 micron.

There is provision to add lime in grinding circuit for alkalinity control, as well as collector for conditioning of the freshly liberated minerals surfaces.

The mills are equipped with lubrication systems comprising high pressure lubrication pumps on the trunnion bearings for mill start-up, low pressure trunnion lubrication pumps, and a lubrication oil heating and cooling system

A 75 tonne overhead crane provides maintenance support the grinding area and a 10 tonne crane services the cyclone clusters.

The addition of steel balls to the pebble mill will maximize power draw and allow the plant to achieve the target rate of 9 Mtpa. The expanded facilities and equipment to achieve this throughput rate are in place.

### 17.2.5 Flotation

The flotation circuit is configured for copper flotation first, followed by nickel flotation, and then pyrrhotite flotation.

Copper flotation is accomplished in a single bank of 7 rougher and scavenger cells followed by 4 stages of cleaning in 17 cells and 1 column.  Because of the slow flotation kinetics of pentlandite, nickel flotation requires 2 banks of 7 cells for nickel rougher and scavenger flotation, followed by 5 stages of cleaning in a total of 25 cleaner cells and 1 column.  Residual sulphide flotation has 4 rougher cells followed by 4 cleaner cells.

Copper Flotation:

The design of the copper flotation circuit is based on a mass balance derived from the following copper grades:

- Feed grade – 0.32% CuS (Ore Reserve average is 0.34% CuS)
- Final concentrate grade – 25% Cu at 78% Cu recovery
- Cu Flotation tailings grade – 0.05% Cu

It is important to minimize the nickel content of the copper concentrate since there is no payment for nickel in the copper concentrate, and above 0.8% Ni, penalties are applied by the smelter

Cyclone overflow from the three mills gravitates through a launder sampler (for feed grade determination) and to a rubber lined 900 m$^3$ surge tank.  Slurry is pumped at a controlled rate from this tank to the copper flotation circuit.

The rougher flotation circuit comprises one 500 m³ cell, equipped with a 480 kW drive, and five 160 m³ TankCells® equipped with 160 kW drives.  The cells have an arrangement of 1+4+2, with each bank separated by intermediate boxes.  The retention time in rougher flotation is approximately 30 minutes.  Copper rougher-scavenger tailings are pumped to the head of nickel flotation.

Concentrate from all rougher cells is combined into a single rougher concentrate which is pumped to the copper regrind circuit to liberate and reject nickel concentrates that float in the copper circuit.

The regrind mill is an Outotec HIG (high intensity grinding) mill, utilizing 3mm ceramic media.  The circuit operates in closed circuit with 10 hydrocyclones; copper concentrate feed is pumped into the mill discharge hopper and combined with the HIG mill discharge prior to separation of fines in the cyclones.  Cyclone underflow is returned to the mill, and overflow pumped to copper cleaner flotation.

Concentrate from the first cleaners is pumped to the second cleaners.  Second cleaner concentrate is pumped to the third cleaners, etc.  The fourth cleaner cells or the flotation column produces a final copper concentrate which is pumped through a pipe sampler to the copper concentrate thickener.

Tailings from the fourth cleaner cells will gravitate to the third cleaners, and tailings from the third cleaners will gravitate to the second cleaners, etc. Cleaner tailings from the final cell in the bank (the first cleaners) are pumped to Ni rougher feed hopper.

Flotation reagents are distributed to the flotation circuit on a ring main system and fed to each addition point through a solenoid valve and programmable timer.

The collector is sodium dithiophosphinate (Aerophine 3418A) – it is added to the rougher feed tank, the second and the first cleaner at a total addition rate of 12 g/t ore.

Nasfroth 240 is used as the frother and is added to the first rougher and to the first cleaner flotation cells. Total addition of Nasfroth to copper flotation is 7 g/t ore. Both the frother and the collector solutions are added at the as-supplied strength.

Carboxy methyl cellulose (CMC) is used as a depressant to minimize gangue flotation – the consumption is about 40 g/t, and it is added as a 1% solution.

Lime is required for pH control, with a pH of 11 and 12 being required in the rougher and cleaner flotation banks respectively. Lime added to the first cleaner cell and final cleaner cells or column at a total consumption of 76 g/t ore.

Nickel Flotation:

The design basis for the nickel flotation circuit assumes the following grades and recoveries:

- Feed grade – 0.22% NiS (Average grade of Ore Reserve is 0.24% NiS)
- Final concentrate grade – 11% Ni at 66.0% recovery of Ni sulphides
- Flotation tailings grade – 0.05% Ni, as a sulphide

The nickel flotation circuit comprises rougher/scavenger flotation and five-stage cleaning of the rougher concentrate, with intermediate regrind. The retention time in rougher/scavenger flotation is approximately 90 minutes. Feed to nickel rougher flotation comprises copper rougher and first cleaner flotation tails.

There are two banks of rougher/scavenger cells operating in parallel, each comprising seven 300m$^3$ TankCells® with individual level control - i.e. in an arrangement of 2+1+2+2. The cells are equipped with 315kW drives, and each cell is be separated by an intermediate box containing 2 dart valves to control the flow from the cell for level control. The first two cells of each row are rougher cells with the third and fourth cells being swing cells, and the last 3 cells are designated as scavenger cells.

Tailings from the two rougher/scavenger circuits is collected in a tailings hopper and pumped to the sulphide flotation circuit.

Rougher concentrate is pumped directly to the third cleaners. The scavenger concentrate reports to first cleaners. First cleaner concentrate is pumped to the second cleaners. The second cleaner concentrate feeds the regrind mill whose product goes to the third cleaner cells. Third cleaner concentrate is pumped to the fourth cleaners. Fourth cleaner concentrate is pumped to the fifth and final cleaners.

Flotation reagents are distributed to the flotation circuit on a ring main system and fed to each addition point through a solenoid valve set to open for a pre-determined time.

Sodium Ethyl Xanthate (SEX) and Sodium Isopropyl Xanthate (SIPX) are used as collector; the combined consumption is approximately 80 g/t

Nasfroth is used as the frother in rougher/scavenger and cleaner flotation banks: consumption is about 23.7 g/t.

The pH in nickel rougher/scavenger flotation is decreased gradually to desired levels using hydrochloric acid. Lime is added to the cleaner cells to maintain selectivity.

Residual Sulphide Flotation:

For environmental reasons the remaining sulphides are removed from the flotation tailings to decrease the sulphide content to produce a benign tailings with no acid generating potential. These reduced sulphide tailings are able to be deposited in an unlined tailings impoundment dam.

The sulphide flotation circuit comprises rougher flotation followed by one stage of cleaning.

Rougher flotation has a residence time of approximately 30 minutes and is achieved in three 160 m³ TankCells®, with 160 kW drives, in a 2+1 configuration. Tailings from the last rougher cell gravitates through a sampler to a 40m³ sump from where they are pumped to tailings disposal area.

Rougher concentrate gravitates to the cleaner flotation circuit, which comprises four 20 m³ TankCells® with 45 kW drives. The cleaner concentrate (sulphide rich tailings) is pumped through a pipe sampler to a designated area in the main tailings pond, which is provided with a plastic liner to prevent seepage. Cleaner tailings are pumped back to the first sulphide rougher cell.

Potassium Iso Amyl Xanthate (PIAX) is used as the collector in residual sulphide flotation and Nasfroth 350 as frother. Both reagents are added to the first rougher flotation cell only. PIAX consumption is about 20g/t ore, and Nasfroth consumption 20 g/t ore.

All flotation cells are controlled on level using dart valves on the cell discharge. Cameras are used to monitor the froth appearance and to measure the velocity of the froth overflowing the cells – froth velocity is controlled through the air addition to the cells.

Three multistage centrifugal air blowers produce the air required for the large flotation cells (namely the 160 m³, 200 m³ and 300 m³ cells). These blowers are rated at about 7,000 m$^3$/h per blower with an outlet pressure of 80 kPa. Two blowers generate 7,000 m$^3$/h of air at 50 kPa for the smaller flotation cells.

All major concentrate and tailings streams are sampled through pressure pipe samples, which forward samples to an-stream analysers (OSAs). Two Courier 6SL on-stream analysers monitor the copper, nickel and sulphide flotation circuits. Copper, nickel, iron and slurry density are analysed for each sample. The OSAs provides real time assay information for control of the circuit, and provides a sample for analysis in the laboratory for metallurgical accounting.

Spillage and wash down in each area is handled by floor sump pumps which return spillage to the appropriate place in the circuit.   These pumps are automatically stopped and started by level switches in the sumps.

The flotation area is equipped with a 25 t overhead crane for maintenance purposes.

Any fumes are collected and vented to outside of the building through a ventilation system.

## 17.2.6  Concentrate Handling

Final concentrates from the copper and nickel flotation circuits are dewatered prior to dispatch to the smelters.  Two identical circuits are provided – one for the copper concentrates, and one for the nickel concentrates.

The final dewatered products average approximately 9% moisture by weight, and are discharged into a storage shed.  Copper concentrates are trucked off site in bulk, whilst nickel concentrates are bagged prior to transport to minimize self-heating.

The following process description is generic and applies to either circuit.

Concentrate Thickening:

An 11m diameter thickener is used to dewater each concentrate to between 60 and 70% solids prior to pressure filtration.  This thickener also collects filter cloth washings, spillage and wash down in the area, etc.

Thickener underflow is pumped at a slurry density of 60 to 70% w/w solids by one of the two peristaltic pumps to a small de-gritting screen ahead of an agitated  filter feed surge tank.

Overflow water from each concentrate thickener overflows to a dedicated spray water tank for re-use in the corresponding flotation section as sprays in the concentrate launders.

The thickeners are provided with the standard interlocks and controls, including interface level detection for flocculent addition control, etc.  Density measurement on the pump discharge lines controls the underflow pump speed.

Concentrate Filtration:

Each filter feed surge tank has a live volume of 250 m$^3$ and provides approximately 24 hours surge capacity ahead of the filter. One of two installed horizontal-centrifugal pumps equipped with variable speed drive, pumps the thickened slurry from the surge tank to the pressure filter.

The pressure filter is an automatic Larox filter press, operating in a batch mode.  The filtration cycle comprise 6 stages, lasting approximately 10 minutes in total.

The filtrate and cloth wash water discharging from each filter press flows to a hopper where a pump returns the solution to the respective concentrate thickener feed box.

Each filter is located over a concentrate storage area, separated from each other by a concrete wall. The filter cake falls directly to the respective bay in the storage area; copper concentrate is loaded

by front-end loader on to trucks for transportation to offsite smelters, whilst some nickel concentrate which is shipped overseas is bagged to minimise self-heating.

### 17.2.7  Tailings Disposal

The pumping distance to the tailings disposal area is about 1,600 m.  Two sets of two slurry pumps operating in parallel (with one common stand-by set) pump the final tailings (low sulphur flotation tailings) from the flotation tailings hopper through two separate tailings lines to the tailings pond. The pump hopper is provided with level measurement and control: the pumps are equipped with variable speed drives.

The tailings lines is high-density polyethylene (HDPE), approximately 350 mm ID, with allowances for expansion, because of the difference between summer and winter temperatures.

Flow meters and pressure transmitters are installed on the pipeline to assist in operation, and warn of blockages or other disturbances in the pumping system.

The sulphide rich tailings (concentrate from the sulphur flotation circuit) are pumped to a separate lined tailings storage facility adjacent to the main tailings pond.

### 17.2.8  Preparation and Dosing of Reagents

Reagents are stored in a separate heated building next to the flotation building whilst preparation takes place inside the floatation building.  The storage facility is capable of holding approximately one month's supply of each reagent except lime.

Two tonne overhead cranes are used to move bags or drums of reagents to each reagent mixing system and to assist in maintenance

Reagent make-up systems are fully automated using tank levels and timers; the only operator involvement is feeding the reagents to the mixing tanks or feed hoppers.

Control of flotation reagents to each individual feeding point is based on a grams per ton of ore feed basis.  Dosing pumps are controlled by ratio from the flotation feed mass calculator.

Each reagent mixing area has a dedicated floor sump and pump for collection of spillage and pumping to the final tailings pump hopper.

The following reagents are used at the Kevitsa concentrator:

Copper Flotation:
- Collector - sodium dithiophosphinate (Aerophine 3418A)
- Frother - Glycol ether (Nasfroth 240)
- Depressant - carboxy methyl cellulose (CMC)
- pH modifier – lime

Nickel Flotation:
- Collector - sodium ethyl xanthate (SEX) and sodium iso-propyl xanthate (SIPX)
- Frother - (Nasfroth 240)
- Depressant - carboxy methyl cellulose (CMC)

- Dispersant – Sodium metabisulphite (SMBS)
- Depressant – Tri-ethyl tetra amine (TETA)
- pH modifier – lime (odour control).

Sulphide Flotation:
- Collector - potassium iso-amyl xanthate (PIAX)
- Frother – Glycol ether (Nasfroth 350)
- pH modifier – not normally required, but there is provision for sulphuric acid to be added.

Concentrate Thickening:
- Flocculant.

### 17.2.9 Water Systems

Raw Water Supply:

The primary source of fresh water for the Kevitsa concentrator is the Vajukoski Pond, which is approximately 7 km to the west of the mine site.

A water intake and two submersible pumps are installed in the pond, and these supply water as required to the 1,200 m³ raw water tank in the plant.

The estimated total demand for raw water is about 2,500m$^3$/day.

Process Water Supply:

A Process water storage pond, located to the north-east of the plant has a capacity of about 328,000 m³ of water. This reservoir collects surface run-off from the site and from the waste rock dump area. Mine water from the open pit, consisting of ground water and rainwater, is also pumped to the reservoir, and excess water from the plant, overflowing the process water tank gravitates to the reservoir.

Water from the reservoir is pumped by one of two installed submersible pumps to the process water tank in the plant, or to the effluent treatment plant, for eventual disposal into the wetlands and onto the Vajukoski Pond.

Process Make-Up (TSF Decant) Water:

The secondary water source for process water make-up is the decant water from the tailings pond. This water return is the main source of process water used in the concentrator.

Two pumps will return decant water from the tailings pond to the process water tank

Total consumption of process water is approximately 2,500 m$^3$/h.

Potable Water:

Potable water is reticulated throughout the plant for use in safety showers and the plant buildings. The daily consumption of potable water is estimated to be 100 m$^3$.

Raw water is pumped from the raw water tank to a dedicated water treatment plant were it is treated and filtered to reduce turbidity and bacteria content.

The potable water production system is a dual-media pressurised sand filter water treatment system, comprising filter feed and backwash pumps, an automatic nano-filtration system, and a UV sterilisation system.

The potable water tank has a capacity of 50 m$^3$ of water.  Potable water is distributed throughout the plant by one of two installed potable water pumps at the rate of 25 m$^3$/h.  Minor off-takes are satisfied by a potable water jockey pump and accumulator.

Fire Fighting Water:

Fire protection water is distributed around the plant area by a dedicated reticulation system to a number of fire hydrants and hose reels.

Fire water is sourced from the raw water tank, with the fire water pump suction lines connected to the bottom of the tank.  Since all other pump suction lines connected to this tank are located at an elevated point on the tank wall, a minimum quantity of water is guaranteed for firefighting purposes at all times.

## 17.2.10       Air Systems

Three distinct air systems are provided throughout the plant:

- Plant air
- Instrument air
- Flotation low pressure air

In addition, because of its remote locations, a dedicated compressor and air receiver is installed in the crusher circuit.

Plant Air:

High pressure compressed air (750 kPa) is generated for general use throughout the plant and for instrument air supply.  Air is supplied at a rate of 3,000 Nm$^3$/h by two air compressors, one running and one standby.

Plant air is stored in an air receiver to ensure a continuous air pressure in the system.  In addition a mill air receiver located in the grinding plant is used to ensure a continuous supply of air to the mill lubrication systems.

A similar system comprising a compressor rated at 150 m$^3$/h and 850 kPa is provided in the crusher area, with compressed air being filtered through an in-line filter, prior to the crusher air receiver.

Instrument Air:

Instrument air is distributed throughout the plant for use in actuation of valves and for use by several instrument systems.  Instrument air is generated by the plant air compressors.

Compressed air is cleaned in a series of filters and dryers to produce compressed air of instrument air quality for the plant:

- instrument air dryer pre-filters
- one of two installed instrument air dryers
- instrument air post-dryer filters

Instrument air is stored in the instrument air receiver to ensure continuous pressure in the instrument air supply to the plant.

Flotation Low Pressure Air:

The flotation cells require forced air addition at a gauge pressure of 80 kPa for the 300 m³ and 160 m³ cells and at 50 kPa for the smaller cells.  Two separate systems are installed to minimize power requirements.

Three blowers, each rated at 10,000 Nm³/h and equipped with 560 kW drives are installed for the higher pressure duty.  Lower pressure air is also supplied by three blowers rated at 15,000 Nm³/h and fitted with 315 kW drives.

## 17.2.11        Process Control Philosophy

A significant level of process control and instrumentation was incorporated into the process design, to minimize the number of process operators required.  Process control and stop/start of equipment is managed from a central control room.

The original level of process control has been further increased with the addition of expert control systems on both the grinding and flotation circuits, with the following objectives:

- To maximize the throughput of the plant at desired grinding fineness to achieve sufficient liberation of valuable minerals.
- To produce high-grade copper and nickel concentrates of commercial grade at maximum recovery.

The control loops used are standard for milling and concentrator circuits, with variable speed pumps controlling sump and tank levels, density control through water addition, reagent addition control is ratioed to feed tonnage and/or head grades, and expert control of flotation through cameras controlling cell levels and air flow addition.

In addition, the plant is equipped with:

- an Outotec on-stream particle size-measuring instrument (PSI 500) for control of fineness of grind.
- Outotec on-stream X-ray analysers (Courier 6 SLs) to monitor and control the flotation process by analyses for copper, nickel, iron and density of selected process slurries.

Cameras are used to monitor the froth speed and bubble size and stability from the control room.

The Process Flow Diagrams for the current Kevitsa plant configuration are shown in the Figure 17-2 (Crushing and Milling) and Figure 17-3 (Flotation).

**Figure 17-2        Kevitsa Crushing and Milling Flowsheet**
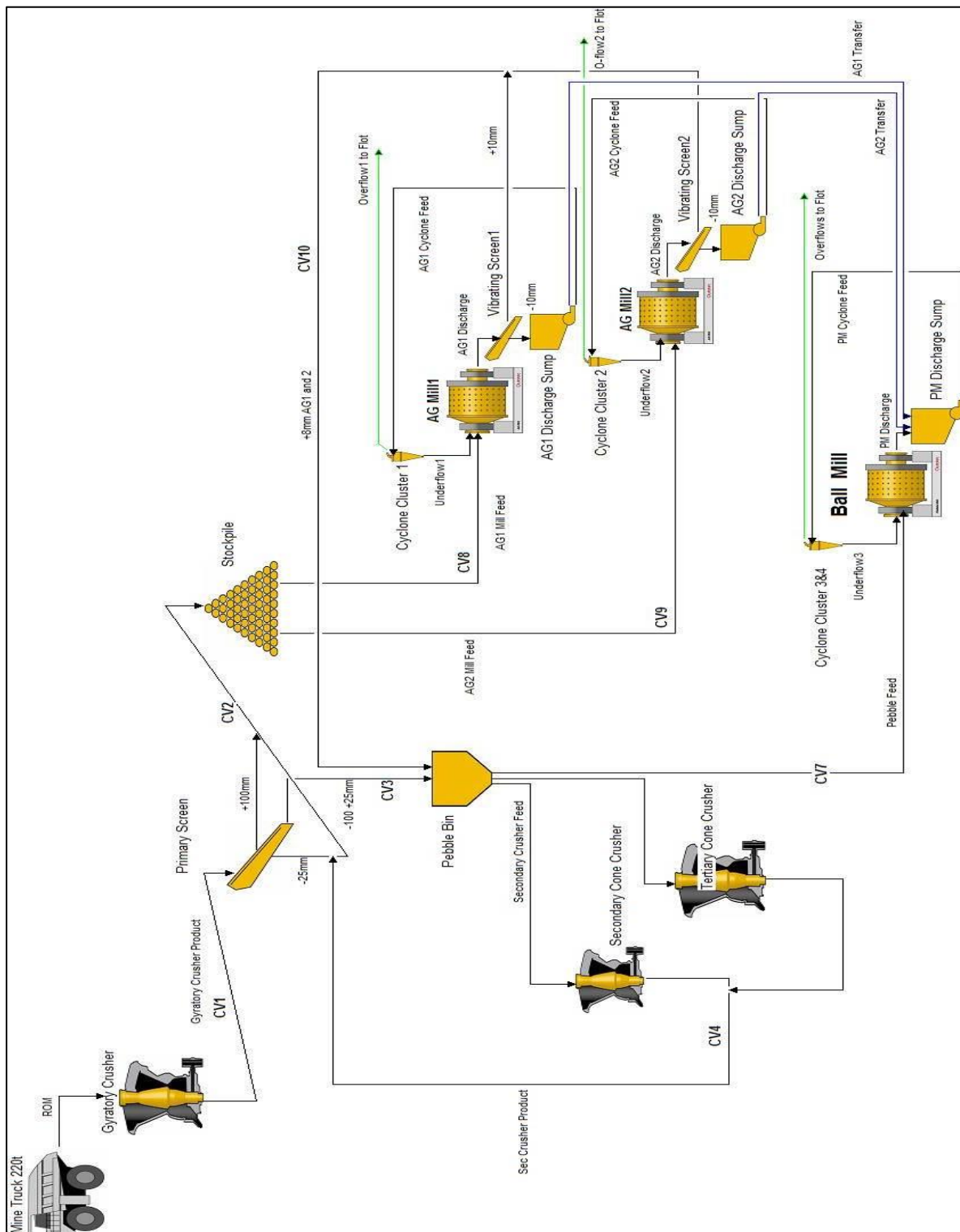
**Figure 17-3        Kevitsa Flotation Flowsheet**
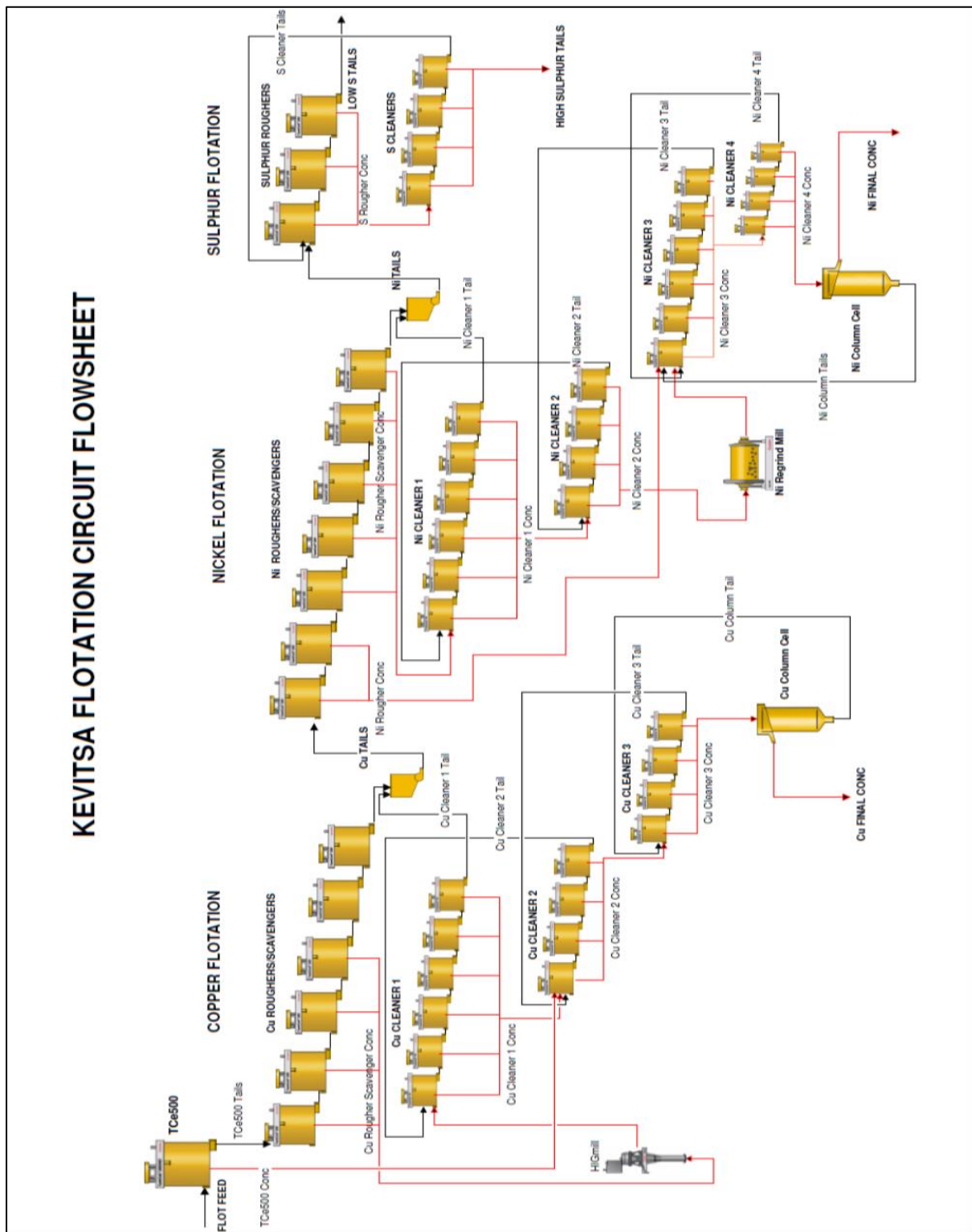


## 17.3    Tailings Facility

The tailings from the process are delivered to two storages facilities located adjacent to the plant to the south.  Both storages are designed as no release facilities and provide permanent storage of all the tailings produced by the Kevitsa plant.

The bulk of the tailings (flotation tailings), are delivered to the main flotation tailings storage (TSF A) which is formed by the staged construction of the peripheral embankments. This material represents around 97% of the total ore processed through the treatment plant, and is discharged at approximately 32% solids. These tailings are inert, and TSF A is unlined.

A smaller fully lined storage (TSF B) contains the sulphide concentrate produced by the last stage of flotation  This material represents around 1 % of the total ore processed through the treatment plant, and is discharged at approximately 20% solids.

The retaining embankments which form the storage will be progressively raised, by 3 meter increments and roughly every 2 years, over the life of the mine to maintain freeboard over the deposited tailings.

The Stage 1 embankment, built during the original project construction phase, provided sufficient storage for the first 18 months of the mining operation. The Stage 2 embankment, built throughout the 2013 and 2014 summer seasons, involved extending of the peripheral starter wall (to the extent allowed by the natural ground elevation), and uplifting of the Stage 1 embankment by downstream construction.

Subsequent staged raises of the embankment (Stage 3 and onwards) are done by upstream construction, supported by the tailings beaches directly upstream of the active embankment. These structures are constructed from run-of-mine waste rock, with an upstream filter. This is a free-draining structure and incorporates a seepage collection system in the downstream shoulder zone. The collected seepage waters are pumped back into the storage facility.

The Stage 3 embankment construction was initiated in 2015 and will be completed in 2016.

The main flotation storage is equipped with a number of monitoring instrument sections, installed at regular intervals along the embankment length. Each instrument section comprises of:

- Vibrating Wire Piezometers to monitor the evolution of the pore pressure within the tailings beach;
- Standpipe Casagrande Piezometers to monitor pressure within the foundation of the starter wall;
- Settlement Plates to monitor deformations of the embankment.

**Figure 17-4          Kevitsa Tailings Plan – December 2015**
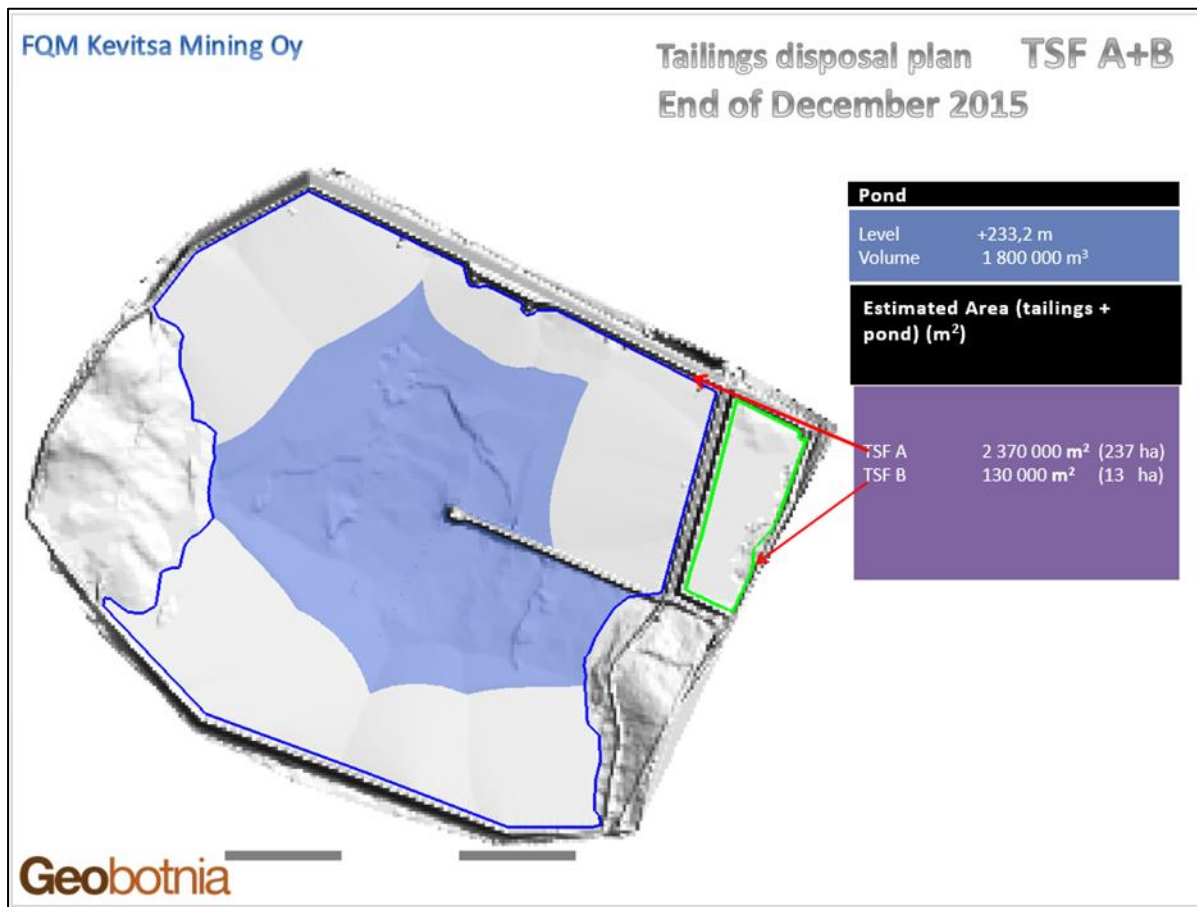
# ITEM 18   **PROJECT INFRASTRUCTURE**

Kevitsa is a mature operation that has been mining and processing ore for more almost 4 years. All infrastructure required by the Mine is in place including sealed roads, power lines and substations, process plant, site offices, workshops, tailings dams, and waste storage facilities.

## 18.1     **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. There are no supply issues and price competition is working to the advantage of KMO.

## 18.2     **Water Supply and Water Systems**

Water supply and water management systems are in place. A description can be found in section 17.2.9.

## 18.3     **Air Systems**

Water supply and water management systems are in place. A description can be found in section 17.2.10.

## 18.4     **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.

## 18.5     **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 in order to allow year round operation and access.

## 18.6     **Mine services**

The KMO 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 ancilliary contractors have their own workshops and maintenence facilities in areas allocated to them

## 18.7     **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 8km west of the property.

## 18.8    **Waste dumps, tailings dams and pipelines**

Information on the site waste dumps is provided in Item 16.1 with an accompanying plan shown in Figure 16-1. This same plan shows the location of the tailings dams, a commentary on which is provided in Item 17.3.

# ITEM 19  MARKET STUDIES AND CONTRACTS

## 19.1    Markets for Kevitsa Products

Two different concentrates are produced at Kevitsa each having different metal quality and hence are transported to separate smelting facilities. The first is a Nickel-Copper-PGE-Gold concentrate grading close to 11% nickel. The second is a Copper-PGE-gold concentrate grading close to 25% copper. The nickel concentrate is loaded into double skin recycled canvas bags before transportation to the smelters via sea-freight, however alternative local markets which are being investigated would be able to receive this product in bulk configuration. The copper concentrate is transported by road/rail in bulk configuration.

Commercial and sales arrangements are handled by Metal Corp trading, a Subsidiary of First Quantum Minerals Ltd. These off-take arrangements are subject to commercial confidentiality however the base terms of these agreements have been utilized to develop the long term Mineral Reserves and cash flows.

## 19.2    Contracts

As an operating mine Kevitsa has a series of on-going contracts for the supply of various services, materials and consumables. These contracts are tendered and renewed as required on a regular basis.

The concentrate off-take contracts identified in section 19.1 are re-tendered and negotiated as needed to maintain an optimal commercial position for the mine.

# ITEM 20  ENVIRONMENTAL STUDIES, PERMITTING, SOCIAL AND COMMUNITY IMPACT

## 20.1    Environmental Setting

Environmental issues for Kevitsa are typical for the mining industry regarding noise, dust, and ongoing environmental management.  Of these, the waste rock and water management will have an impact on cash flow.  Costs for both have been adequately factored into the economic analysis.

## 20.2    Status of Environmental Approvals

The 10Mtpa environmental permit was issued on 11th July 2014. However the permit appeals process is still ongoing due to various minor anticipated appeals and KMO expects this process to be successfully completed in 2017.

The Kevitsa mine is fully compliant with all final effluent discharge limits and air quality standards. There have been some challenges with internal compliance points with regards to nickel in drainage from the mine dumps prior to treatment in the neutralisation facility. The site water discharge pumping capability was upgraded in 2015 in line with the increased effluent discharge volumes approved in the new 10Mtpa permit. Work has started on enhancing mine drainage treatment capacity for the expanded mine. Kevitsa has applied for postponement of the deadline for increased treatment capacity from 1st January 2016 to 30th April 2017. Kevitsa's view is that postponing compliance of the stipulation does not present a significant risk of environment deterioration. The application was sent to the permitting authority at the end September 2015 and the application was published 4th February 2016. No decision has been made and is likely as early as the end of 2016.

**Figure 20-1       Kevitsa plant seen from the water reservoir**



## 20.3    Environmental Monitoring

All environmental monitoring is done according to the approved monitoring program of operational efficiency, emissions, and environmental impacts of the Kevitsa mine. Monitoring includes operational efficiency, water and airborne emissions, waste fractions and environmental impacts. The monitoring (**Error! Reference source not found.** and Figure 20-2 and Figure 20-3**Error! Reference source not found.**) of environmental impacts includes surface waters (physicochemical water quality, sediment properties and biological monitoring), fish and fishery, groundwater,

biological monitoring of terrestrial organism (soil contamination, fungi, ants, pine needle birds, moor frogs and vegetation count square lines), air quality (dust and PM10), noise and vibration.

**Figure 20-2      Monitoring of benthic communities at brook Mataraoja - September 2015**



**Figure 20-2      Monitoring of moss (*Pleurozium schreberi*)**



All monitoring results are reported to the supervising authority monthly and an annual report is submitted by the end of February each year. The 2015 annual report was submitted in accordance with this requirement.

## 20.4    Environmental Issues

The main issues faced by Kevitsa since commencing operations have been in relation to the water management and the preparation and submission of the application to expand the processing

capacity from 5.5 Mtpa to 10 Mtpa. Other minor issues included the requirement to construct a noise bund to protect the Natura and the modification of the waste dump design.

### 20.4.1 Natura

The original EIA area included a small part of the extensive Koitelainen conservation area (Figure 9-3) that belongs to the European Union Natura 2000 network. This was removed and KMO was required to construct a noise bund on the eastern boundary of the mining concession in order to minimize disturbance of the wildlife residing in the Natura.

### 20.4.2 Waste Dump Modification

In 2015, a small section on the eastern side of the waste dump was modified in order to ensure that the dump did not encroach upon a swampy area known to be the habitat of endangered species of frogs. The modification caused a minor reduction of the dump capacity but does not affect the mining or waste dumping schedules.

## 20.5    Safety Performance

Generally the workforce has a good safety attitude and Kevitsa has a good safety system and record. Finland has a culture of reporting issues, which has been utilised in training programs to develop and reinforce safe behaviour on and around the mine site.

## 20.6    Social and Community Related Requirements

The site has a good relationship with local communities, Sodankylä municipality and reindeer herders, and that also applies to the relationships with the supervising and permitting authorities. At the time of writing this report, the issuer was not aware of any material social or community risks at Kevitsa.

## 20.7    Archaeology

All archaeological sites identified prior to the commencement of the Mine have been investigated by a registered archaeological company (Mikroliitti Oy). A single insignificant site was identified inside the mining lease area which the mine's activities will not affect. A few other minor sites have been identified outside of the mining lease area, with operations having no affect to these either.

## 20.8    Mine Closure Provisions

The reclamation phase, which will follow once the extraction and mineral processing phase has been completed, will involve dismantling buildings and constructions, decommissioning the mine, and landscaping. The reclamation and landscaping of the waste disposal sites will begin in stages as soon as the final dump heights have been reached. Monitoring of the site will continue for several years after mining has been completed. Kevitsa's Asset Retirement Obligation was US $21,239,062 on 31st December 2015. To date, KMO has deposited Euros 17,068,848 with the Lapland Environmental Authority as a guarantee for restoration of the mine site. Environmental spend in 2015 was US $1,757,915.

**Figure 20-3          Whooper Swan Nesting at the Mine Site - Summer 2015**

# ITEM 21  CAPITAL AND OPERATING COSTS

## 21.1  Capital Costs

The 2011 Technical Report stated that US$400 million was required to construct and commission the Kevitsa Project to the initial capacity of 5 Mtpa. In 2011, US$303.4 million was spent and between 2012 and 2014 a further US$352 million was spent completing the 5 Mtpa plant and then expanding it to 9 Mtpa.

**Figure 21-1        Kevitsa Capital Expenditure 2012 - 2014**
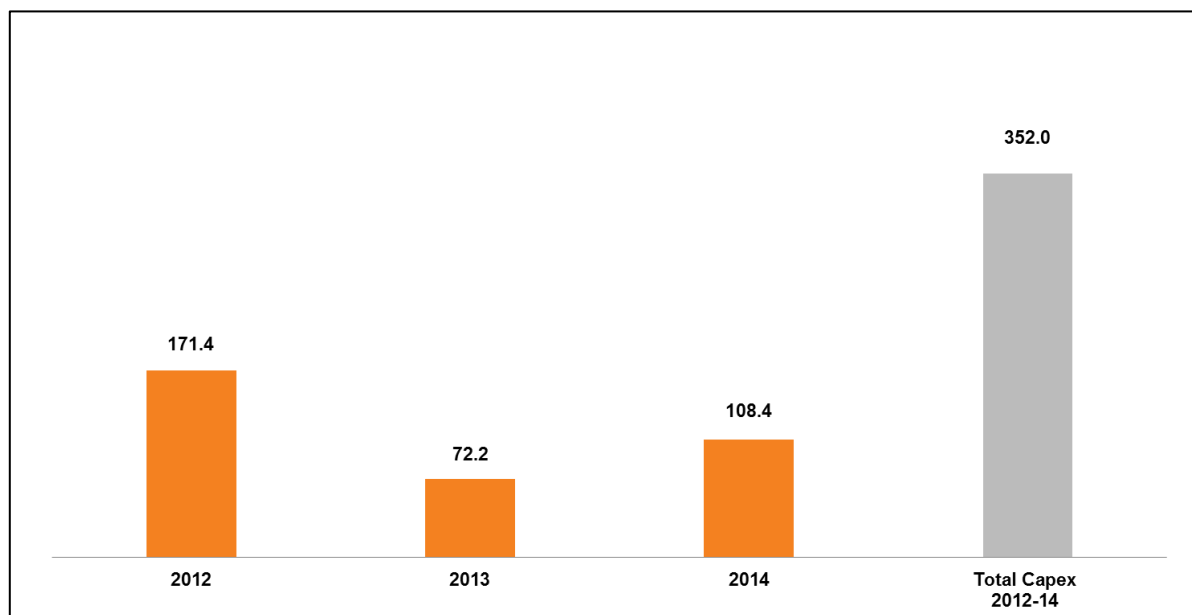


KMO have estimated that US$368 million is required for capital expenditure over the remaining life of mine.

**Table 21-1        Kevitsa Capital Estimate – Life of Mine**

| 9Mtpa Case | Total | 2016 | 2017 | 2018 | 2019 | 2020 | 2021 | 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Sustaining | 56,000 | | | | | | 7,000 | 7,000 | 7,000 | 7,000 | 7,000 | 7,000 | 7,000 | 7,000 | | | | |
| Expansion Capex | - | | | | | | | | | | | | | | | | | |
| Stripping | 235,975 | 24,471 | 37,664 | 23,142 | 42,102 | 30,238 | 23,720 | 20,962 | 23,486 | 10,191 | - | - | - | - | | | | |
| Mining Fleet* | 56,000 | | | | | | 14,000 | | 14,000 | | 14,000 | | 14,000 | | | | | |
| Tailings Uplift | 12,148 | 2,922 | 2,830 | 2,113 | 2,141 | 2,143 | | | | | | | | | | | | |
| Effluent Treatment Plant Updgrade | 3,103 | 1,813 | 1,137 | 58 | 53 | 42 | | | | | | | | | | | | |
| Trolley Assist Maintenance | 5,000 | | | | | 1,000 | | 1,000 | | 1,000 | | 1,000 | | 1,000 | | | | |
| Total | 368,227 | 29,205 | 41,630 | 25,314 | 44,295 | 33,423 | 44,720 | 28,962 | 44,486 | 18,191 | 21,000 | 8,000 | 21,000 | 8,000 | - | - | - | - |

The mining fleet expansion allows for an additional face shovel (in 2021) and the increase in truck fleet (three units for each $14M in 2023, 2025 and 2027).  As these are projections, allowance has been included for additional drill and ancillary equipment.

In addition, a total USD21.2 million has been budgeted for closure and reclamation between 2033 and 2036. This amount includes the reclamation bond which is currently US$15 million.

## 21.2    **Operating Costs**

### 21.2.1  **Mining**

Forecast operating costs for mining were developed using the projected haul profiles and based on the assumption that Trolley Assist Haulage from the pit commences in 2017.

**Figure 21-2    Haul profiles**
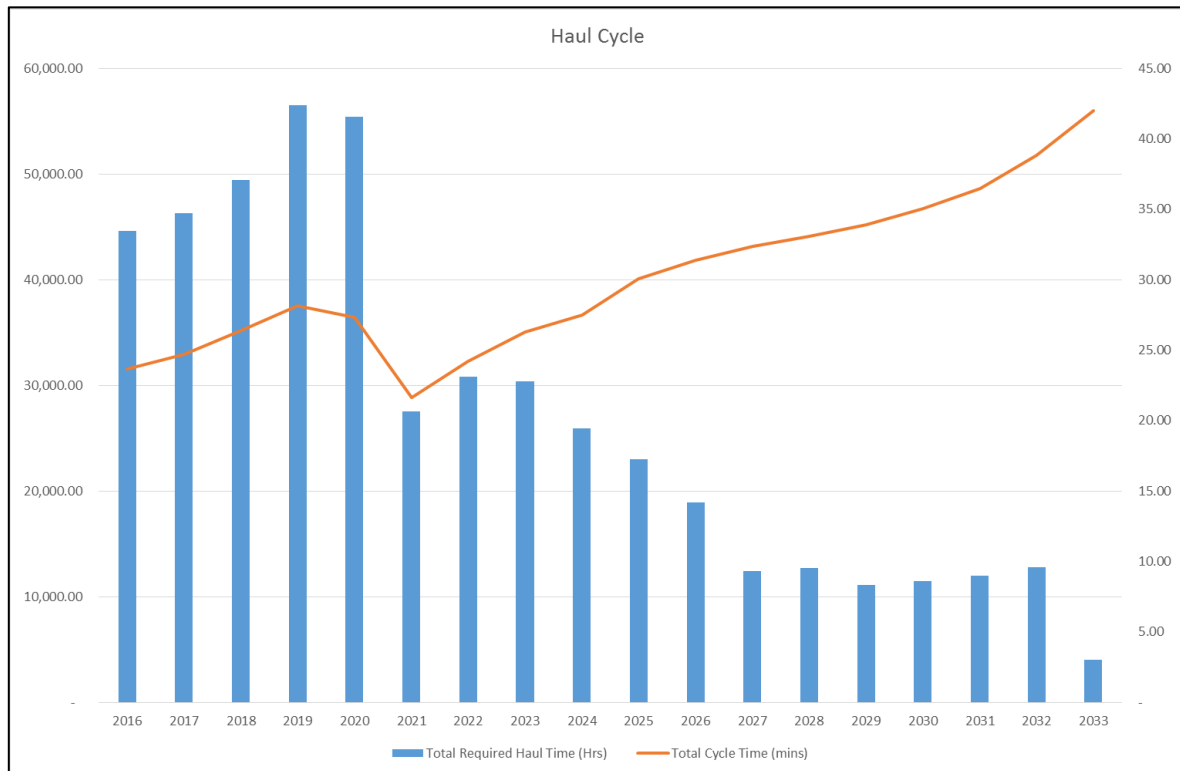


Forecast operating costs for the life of mine are based on 2013 to 2015 actuals along with the KMO 2016 operating budget are summarised in Table 21-2 below.

**Table 21-2        Summary of Operating Costs by Cost Area for Life of Mine**

| Ore processed | 7.6 | Mt |
| --- | --- | --- |
| Nickel produced | 14,000 | tonnes |
| Copper produced | 29,200 | tonnes |

| Cost Area | Annual Cost (US$1,000's) | Unit costs (US$/t ore) |
| --- | --- | --- |
| Mining | 49,720 | 6.93 |
| Processing | 32,250 | 8.05 |
| G&A (incl Environment) | 28,600 | 3.66 |
| Metal Costs | 70,600 | 7.91 |
| Total | 181,170 | 26.55 |

Note: Some numbers may not add up due to rounding

# ITEM 22 **ECONOMIC ANALYSIS**

Item 22, Economic Analysis is not required for operating properties, however a cashflow has been developed using the revenue and cost estimates derived from 2015 actual costs and the 2016 operating budget.

**Table 22-1 Kevitsa LOM Cashflow Summary – December 31st, 2015.**

| | UNITS | | TOTAL | 2016 | 2017 | 2018 | 2019 | 2020 | 2021 - 2025 | 2026-2030 | 2030-2033 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **MINING** | | | | | | | | | | | |
| Waste | Tonnes | Mt | **376.7** | 36.3 | 38.3 | 36.3 | 41.7 | 40.4 | 151.2 | 24.8 | 7.6 |
| Ore | Tonnes | Mt | **154.37** | 9.8 | 7.7 | 9.7 | 8.1 | 9.7 | 46.5 | 45.1 | 17.8 |
| Total Mined | Tonnes | Mt | **531.0** | 46.1 | 46.0 | 46.0 | 49.7 | 50.1 | 197.7 | 70.0 | 25.4 |
| Strip ratio | | t:t | **2.4** | 3.7 | 5.0 | 3.7 | 5.2 | 4.2 | 3.2 | 0.5 | 0.4 |
| **PROCESSING SUMMARY** | | | | | | | | | | | |
| Feed to Plant | Tonnes | Mt | **154.5** | 7.6 | 7.6 | 9.0 | 9.0 | 9.0 | 45.0 | 45.0 | 22.30 |
| | Ni | % | **0.24%** | 0.25% | 0.25% | 0.24% | 0.23% | 0.23% | 0.23% | 0.25% | 0.27% |
| | Cu | % | **0.35%** | 0.32% | 0.34% | 0.34% | 0.31% | 0.33% | 0.32% | 0.38% | 0.38% |
| | Pt | g/t | **0.22** | 0.31 | 0.30 | 0.30 | 0.24 | 0.23 | 0.24 | 0.18 | 0.14 |
| | Pd | g/t | **0.15** | 0.21 | 0.19 | 0.19 | 0.15 | 0.14 | 0.17 | 0.12 | 0.09 |
| | Au | g/t | **0.11** | 0.14 | 0.14 | 0.13 | 0.10 | 0.11 | 0.11 | 0.10 | 0.09 |
| | | | | | | | | | | | |
| Plant Recovery | Ni | % | **66.1%** | 66.4% | 66.4% | 66.3% | 65.9% | 65.9% | 65.9% | 66.3% | 66.0% |
| | Cu | % | **89.9%** | 89.8% | 89.9% | 90.0% | 89.7% | 89.8% | 89.7% | 90.3% | 89.8% |
| | Pt | % | **53.0%** | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% | 53.0% |
| | Pd | % | **61.0%** | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% | 61.0% |
| | Au | % | **52.0%** | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% | 52.0% |
| Metal in Concentrate | | | | | | | | | | | |
| | Ni | Kt | **247.3** | 12.6 | 12.6 | 14.6 | 13.4 | 13.5 | 67.4 | 73.5 | 39.7 |
| | Cu | Kt | **479.6** | 21.8 | 23.0 | 27.8 | 25.1 | 26.3 | 127.7 | 152.5 | 75.4 |
| | Pt | koz | **574.1** | 40.1 | 38.5 | 45.6 | 36.6 | 34.6 | 185.4 | 140.5 | 52.8 |
| | Pd | koz | **445.8** | 30.7 | 28.3 | 33.5 | 27.2 | 24.1 | 152.8 | 109.1 | 40.2 |
| | Au | koz | **278.7** | 17.8 | 17.2 | 20.2 | 14.5 | 16.5 | 83.0 | 76.6 | 32.8 |
| **REVENUE SUMMARY** | | | | | | | | | | | |
| Metal Prices | | | | | | | | | | | |
| (Source Dec 2015 Consensus) | Ni | $/lb | | 5.40 | 5.92 | 6.82 | 7.31 | 7.50 | 7.50 | 7.50 | 7.50 |
| | Cu | $/lb | | 2.40 | 2.44 | 2.65 | 2.81 | 2.93 | 3.00 | 3.00 | 3.00 |
| | Pt | $/oz | | 1,063 | 1,100 | 1,100 | 1,100 | 1,100 | 1,100 | 1,100 | 1,100 |
| | Pd | $/oz | | 693 | 700 | 700 | 700 | 700 | 700 | 700 | 700 |
| | Au | $/oz | | 1,158 | 1,159 | 1,159 | 1,153 | 1,200 | 1,200 | 1,200 | 1,200 |
| **REVENUE** | | $M | **8,311.1** | 350.1 | 370.8 | 478.6 | 447.5 | 468.6 | 2,369.6 | 2,546.4 | 1,279.6 |
| | | | | | | | | | | | |
| **CAPITAL COSTS** | | | | | | | | | | | |
| Development capex | | $M | **312.2** | 29.2 | 41.6 | 25.3 | 44.3 | 33.4 | 122.4 | 16.0 | |
| Expansion capex | | $M | **0.0** | | | | | | | | |
| Sustaining capex | | $M | **56.0** | | | | | | 35.0 | 21.0 | |
| Closure and reclamation | | $M | **21.2** | | | | | | | | 21.2 |
| subtotal | | $M | **389.5** | **29.2** | **41.6** | **25.3** | **44.3** | **33.4** | **157.4** | **37.0** | **21.2** |
| **OPERATING COSTS** | | | | | | | | | | | |
| **Mining** | | | **771.7** | 69.6 | 52.2 | 64.9 | 54.5 | 62.3 | 281.5 | 134.1 | 52.6 |
| **Processing** | | | **1,241.4** | 65.1 | 67.1 | 79.1 | 77.7 | 70.7 | 353.3 | 353.3 | 175.0 |
| **Site Administration** | | $M | **177.6** | 9.8 | 9.8 | 9.9 | 9.9 | 9.9 | 49.4 | 49.4 | 29.6 |
| **Other Direct Costs** | | $M | **283.4** | 16.1 | 15.7 | 15.7 | 15.7 | 15.7 | 78.6 | 78.6 | 47.2 |
| subtotal | | $M | **2,474.1** | **160.6** | **144.9** | **169.5** | **157.9** | **158.5** | **762.8** | **615.4** | **304.4** |
| **METAL COSTS** | | | | | | | | | | | |
| Treatment Charges | | $M | **505.4** | 14.5 | 25.0 | 29.3 | 26.8 | 28.8 | 142.0 | 159.5 | 79.6 |
| Refining Charges | | $M | **673.0** | 13.4 | 35.2 | 41.7 | 37.8 | 38.5 | 191.3 | 210.9 | 104.3 |
| Transport/Freight Charges | | $M | **16.3** | 0.7 | 0.6 | 0.7 | 0.7 | 1.0 | 4.8 | 5.2 | 2.7 |
| subtotal | | $M | **1,194.7** | **28.5** | **60.8** | **71.6** | **65.3** | **68.3** | **338.0** | **375.7** | **186.6** |
| **CASHFLOW (Undiscounted)** | | $M | **4,252.9** | **131.8** | **123.5** | **212.1** | **180.1** | **208.3** | **1,111.4** | **1,518.3** | **767.4** |

The cashflow shown above does not include allowances for Interest, Taxes, Depreciation or Amortisation.

Economic model sensitivity analysis was completed on the metal prices, recoveries, as well as capital and operating cost estimates. The results indicate that the Mine is relatively robust and profitability is most sensitive to variations in metal price, recoveries, operating costs, and capital costs in that order.

# ITEM 23  **ADJACENT PROPERTIES**

As of December 2015, according to the public information provided by Ministry of Employment and the Economy, there are claims and claims reservations owned by Anglo American Exploration B.V. adjacent to the Kevitsa Mining concession and KMO's surrounding claims and applications. Anglo American has not publically disclosed any information regarding their activities on adjacent properties. As at the end of December 2015, there were no mining concessions granted or Resources identified or reported adjacent to the property.

# ITEM 24  **OTHER RELEVANT DATA AND INFORMATION**

The QP's are not aware of any relevant data or information not already presented in this report.

# ITEM 25   INTERPRETATIONS AND CONCLUSIONS

## 25.1    Mineral Resource Modelling and Estimation

The updated Kevitsa Mineral Resource estimate, subject of this report, uses all available drillhole data, including all diamond drill hole sample assay results, and the reverse circulation hole sample results. In addition, the estimate is under-pinned by an improved interpretation of the prevailing geology that is now more relevant to the spatial distribution of nickel, copper and PGE mineralisation. In-pit exposure of the different domains of mineralisation has supported the local geology understanding and models. Interpolation parameters used during block estimation were based upon geostatistical and spatial analysis (variography) of the data coded to the geology domains and their respective styles of mineralisation. The updated Mineral Resource estimates were classified according to geological continuity, QAQC, density data, drillhole grid spacing, grade continuity and confidence in the panel grade estimate. The December 2015 Mineral Resource statement has been reported in accordance with the guidelines of the Australasian JORC Code (JORC, 2012), which in turn complies with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014).

### 25.1.1  Uncertainty and risk

In respect of the Mineral Resource estimate, and the opinion of David Gray (QP), the classifications applied to the estimates at Kevitsa accurately reflect the confidences in the geological model and grade estimates.

Including reverse circulation samples in this estimate was supported by comparing sample values with twinned high quality diamond samples.  The comparison demonstrated these samples to have no bias. In addition, the RC sampling routines included good QAQC which highlighted precise and accurate sample analysis with contamination suitably controlled. The RC sampling data does however have a limited influence on the overall resource estimate as these holes were restricted to holes drilled from within the pit. The RC data has provided good support for shorter range geological and grade continuity. However, the bulk of the Mineral Resource is still reliant upon wider spaced diamond drilled holes which leaves areas of mineralisation open and others, albeit deep, poorly supported.

Mineralisation domains and their impact on mining and processing continue to be an opportunity for future refinement. The value of collected XRD data will be developed by relating the existing mineralisation domains to the rock mineralogy, alteration and the sulphide mineralogy.  This is likely to result in further refinement of the mineralisation domains which is likely to improve upon mining and processing efficiencies and overall mine reconciliation results.

In addition, continued in-pit mapping and 3D geology modelling incorporating the XRD data will add value to the robustness of future estimates through improved domain definition and delineation.

## 25.2 Mineral Reserve Estimation

The data is adequate to support the Mineral Reserve estimates using the categories of Mineral Reserve estimates permitted under NI 43-101 within the designed final pits and based upon Measured and Indicated Resources only.

### 25.2.1 Uncertainty and Risk

The Kevitsa mine has been running for more than three years and many of the operational parameters for open pit and process plant, including recoveries and costs, are known. In respect of the Mineral Reserve estimate, and the opinion of Tony Cameron (QP), the classifications applied to the estimates at Kevitsa accurately reflect the confidences in the Resource model and operational parameters and are deemed to be a low risk.

### 25.2.2 Mining

The mine footprint has expanded over time such that the KMO owner operated mining fleet has now got sufficient room to operate efficiently (which is reflected in the production numbers increasing quarterly through 2015). The mining contractor has been working with KMO since mining commenced and is generally responsible for mining areas deemed too small for efficient operation of the KMO fleet. Production targets are being met and KMO and the contractor have sufficient equipment on site to meet the future targets. The only issue in relation to mining is slope stability. Whilst this may have the potential to close the pit, the wall rock is competent and management systems are in place and have worked effectively to date. There is a low to medium mining risk.

### 25.2.3 Processing

The Kevitsa Nickel Copper mine is mature and is capable of successfully operating at design capacity. KMO geological, mining, and processing personnel have developed a good understanding of the mineralogical and blending requirements for plant feed as well as how to operate the plant efficiently. All concentrate produced in 2015 was within contract specifications.

Comminution circuit:

The installation in 2014 of additional crushing and milling power, in the form of a second MP800 cone crusher and a second motor on the Secondary Mill, was supplemented in 2015 by the upgrade of the Primary Screen (increased top deck aperture) and the increase of blasting energy. Combined, these allowed to apply more reduction through blasting and crushing and a reduction in lump fraction presented to the Primary AG Mills. Overall, this provided for efficiency gains on the Primary AG Mills and a 15% increase in overall milling rate at a controlled flotation feed grind (coarser).

The installed, and proven, capability to convert the Secondary Mill from pebble to ball mill duty supports the consideration of additional throughput capacity through full utilization of the installed Secondary Milling power.

Cu Flotation Circuit:

The flowsheet reconfiguration executed in Q4 2013 was supplemented by the installation, in 2014, of the Column Cell and the expansion of the depressant dosing system. Combined, these allowed for improved rejection of non-sulphide gangue and circuit control. The commissioning in 2015 of the

TCe500 flotation cell (as first Cu rougher) and the Cu Regrind Mill, provided for improved liberation and selective recovery of Cu versus Ni. Overall, the Cu recovery to Cu concentrate increased by 5 p.p. and total Cu recovery by 1.5 p.p., for marginally higher concentrate grade.

The addition of the TCe500 and column cell effectively increased Cu flotation capacity and supports the consideration of processing at higher throughput rates.

Ni Flotation Circuit:

The 2014 flowsheet reconfiguration and expansion of the depressant dosing system allowed for improved rejection of non-sulphide gangue and circuit control. The inclusion of a new reagents suite in 2015, developed for rejection of iron-sulphide gangue, benefitted the selective recovery of Ni in the Cleaner circuit. This in turn allowed utilization of flowsheet changes and acid dosing capability both installed in 2014. Overall, the Ni recovery improved by 4 p.p., for higher concentrate grade, with reduced sensitivity to feed grade.

The reduction in circulating loads of both non-sulphide and sulphide gangues has resulted in spare Cleaner circuit capacity and supports the consideration of processing at higher throughput rates.

Going forward there is ongoing work in relation to the mine to mill optimisation and the effect of mineralogy changes process plant performance (both process rates and recoveries). Processing is deemed to be low risk.

### 25.2.4  Environmental Compliance

Environmental compliance is a key performance area for KMO with water management, protection of the Natura 2000 area, and compliance with emissions targets being high priority . It is noted that KMO have been working closely with the authorities in order monitor and control responses with good quality records. This close contact has help develop good relationships that have resulted in KMO being able to negotiate changes to some of the requirements where they have been found to be incorrect. Environmental compliance is deemed a low to medium Risk.

### 25.2.5  Social and Political Risk Management

The site has a good relationship with local communities, Sodankylä municipality and reindeer herders, and that also applies to the relationships with the supervising and permitting authorities. At the time of writing this report, the issuer was not aware of any material social or community risks at Kevitsa.

### 25.2.6  Economic Sensitivity Analysis

Sensitivity analyses were run as part of the pit optimisation process as well as the economic analysis. The analysis during the optimisation showed that the pit is relatively robust and does not reduce in size until metal prices get below the current low levels. The economic analysis showed that short term modifications to the mining and process strategy may be required in order to keep ther project cashflow positive.  In the current economic environment, economic risk is considered to be a medium risk.

# ITEM 26  RECOMMENDATIONS

## 26.1    Geology and Mineral Resource estimation recommendations

The Kevitsa geology and mining team have completed a due diligence review of the input data, estimation methods, block estimate results and classification.  Findings of this review support this updated estimate to be robust in methodology and representative of the input data and current mining performances. The QP, David Gray notes the following recommendations to be considered for future Mineral Resource estimates

Recommendations include:

- Consider extensional diamond drilling in order to improve confidence in deposit extents and deeper zones of mineralisation.
- Consider increasing confidence of Indicated Mineral Resources through motivating for a program of infill diamond drilling particularly covering the next 5 years of Mineral Reserves.
- Complete correlation of the developed neural network mineralisation domains with lithology and alteration from XRD data.
- Continue to improve the definition of the respective mineralisation domains mineralogy and the spatial delineation thereof.
- Continued development of 3D structural geological models are important for improving the understanding of Kevitsa geology and potential controls on mineralisation.
- Improve upon current estimation methods in both resource and grade control estimation routines. Improved domaining will in turn improve spatial analysis and allow for more accurate block estimates.

## 26.2    Mineral Reserve Estimation and Mining

The QP, Tony Cameron notes the following recommendations to be considered for future Mineral Reserve estimates:

- Continue to refine mine designs based on the results from ongoing grade control drilling, reconciliation, and Resource model updates.
- Monitor ore reconciliation to obtain data on the new model in relation to unplanned dilution and loss. Update estimates if required.
- Review the optimisation results, pit designs, and process plant operational parameters if and when metal prices improve. At long term consensus metal prices, the optimisation indicated that Stage 5 of the pit could be developed thus increasing the life of mine.
- Continue monitoring of pit slopes and using data obtained to review design parameters. It is noted that it may be possible to steepen some slopes and reduce waste stripping costs however this needs to be evaluated carefully and weighed up against the risk of wall failure.

## 26.3    Processing

The QP, Andrew Briggs notes the following recommendations to be followed over the next 12 to 18 months:

- Progress the geo-metallurgical characterization of the resource, and develop an effective ore control system with the objective of further improving metal recoveries;
- Continue the blast optimization initiative with the objective of further improving throughput  performance;
- Continue debottlenecking programs in the process plant in order to improve efficiencies and reduce operating costs.

## 26.4    **Environment**

It was noted during the site visits that the close working relationship between KMO personnel and personnel employed by the regulatory authorities means that the Mine is being constantly monitored and reviewed. It is recommended that KMO continue to work with the authorities to show the commitment to meeting environmental targets.

## ITEM 27  **REFERENCES**

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# ITEM 28   CERTIFICATES

*David Gray*
*First Quantum Minerals Ltd*
*24 Outram St, West Perth, Western Australia, 6005*
*Tel +61 8 9346 0100; david.gray@fqml.com*

I, David Gray, do hereby certify that:

1. I am the Group Mine and Resource Geologist employed by First Quantum Minerals Ltd.

2. This certificate applies to the technical report entitled "Kevitsa Nickel Copper Mine, Lapland, Finland, NI 43-101 Technical Report", dated effective 30th March 2016 (the "Technical Report").

3. I am a professional geologist having graduated with a Bachelor of Science degree with Honours (1988) in Geology from Rhodes University in Grahamstown, South Africa.

4. I am a Member of the Australasian Institute of Mining and Metallurgy and a registered Professional Natural Scientist with the South African Council for Natural Scientific Professions (SACNASP).

5. I have worked as a geologist for a total of twenty six years since my graduation from university. I have over 16 years experience in production geology, over 5 years of exploration management of precious, base metal and copper deposits.  During the last ten years I have consulted to and held senior technical mineral resource positions in copper mining companies operating in Central Africa and worldwide.

6. I have read the definition of "qualified person" as set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.

7. I most recently personally inspected the Kevitsa property described in the Technical Report in January 2016.

8. I am responsible for the preparation of those portions of the Technical Report relating to geology, data collection, data analysis and verification and Mineral Resource estimation (namely Items 7 to 12 and 14).

9. I am not independent (as defined by Section 1.5 of NI 43-101) of First Quantum Minerals Ltd.

10. I have had prior involvement with the property that is the subject of the Technical Report.  The nature of my prior involvement has been in the assurance of sampling QAQC, optimisation of estimation methods and the development of geology and mineralisation models.

11. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.

12. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required for it to be disclosed and to make the Technical Report not misleading.

Signed and dated this 30th day of March, 2016 at West Perth, Western Australia, Australia.

*David Gray*

*Tony Cameron*
*Cameron Mining Consulting Ltd*
*20/F Central Tower, 28 Queen's Road, Central, Hong Kong*
*Tel +86 13521513186; tony@cameronmining.com*

I, Tony Cameron, do hereby certify that:

1. I am a Consultant Mining Engineer employed by Cameron Mining Consulting Ltd.

2. This certificate applies to the technical report entitled "Kevitsa Nickel Copper Mine, Lapland, Finland, NI 43-101 Technical Report", dated effective 30th March 2016 (the "Technical Report").

3. I am a professional mining engineer having graduated with an undergraduate degree of Bachelor of Engineering (Mining) from the University of Queensland in 1988. In addition, I have obtained a First Class Mine Manager's Certificate (No. 509) in Western Australia, a Graduate Dilpoma in Business from Curtin University (Western Australia) in 2000, and a Masters of Commercial Law from Melbourne University in 2004.

4. I am a Fellow of the Australasian Institute of Mining and Metallurgy.

5. I have worked as a mining engineer for a period in excess of twenty five years since my graduation from university. Over the last fifteen years I have worked as a consulting mining engineer on mine planning and evaluations for base metals operations and development projects worldwide.

6. I have read the definition of "qualified person" as set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.

7. I most recently personally inspected the Kevitsa property described in the Technical Report in January 2016.

8. I am responsible for the preparation of those portions of the Technical Report relating to Mineral Reserve estimation and Mining, namely Items 15 and 16, respectively. I am also responsible for the preparation of those items of the Technical Report not covered specifically by the other qualified persons which are Items 1 to 6 and 18 to 28.

9. I am independent (as defined by Section 1.5 of NI 43-101) of First Quantum Minerals Ltd.

10. I have not had prior involvement with the property that is the subject of the Technical Report.

11. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.

12. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required for it to be disclosed and to make the Technical Report not misleading.

Signed and dated this 30th day of March, 2016 at West Perth, Western Australia, Australia.

*Tony Cameron*

*Andrew Briggs*
*First Quantum Minerals Ltd*
*24 Outram St*
*West Perth, Western Australia, 6005*
*Tel +61 8 9346 0100; andrew.briggs@fqml.com*

I, Andrew Briggs, do hereby certify that:

1.  I am the Group Consulting Project Metallurgist employed by First Quantum Minerals Ltd.

2.  This certificate applies to the technical report entitled "Kevitsa Nickel Copper Mine, Lapland, Finland, NI 43-101 Technical Report", dated effective 30th March 2016 (the "Technical Report").

3.  I am a professional metallurgist having graduated in 1974 from the Imperial College (Royal School of Mines), London, with a BSc (Eng) First Class in Metallurgy.

4.  I am a Fellow of the Southern African Institute of Mining and Metallurgy, and am a Professional Engineer licenced by NAPEG (#L770) – the Association of Professional Engineers, Geologists and Geophysicists of the Northwest Territories (and Nanavut), Canada.

5.  I have worked as a process engineer and metallurgist since graduation in 1974 (41 years); the first 13 years of which were in operating positions up to Metallurgical Manager in the gold mining industry. This was followed by 19 years in engineering companies in Process Design for projects worldwide, and finally 9 years with First Quantum Minerals Ltd as a Process Consultant.

6.  I have read the definition of "qualified person" as set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.

7.  I most recently personally inspected the Kevitsa property described in the Technical Report in November 2014.

8.  I am responsible for the preparation of those portions of the Technical Report relating to mineral processing/metallurgical testing and recovery methods, namely Items 13 and 17, respectively.

9.  I am not independent (as defined by Section 1.4 of NI 43-101) of First Quantum Minerals Ltd.

10. I have been involved with the property that is the subject of the Technical Report, since inception. This work has included metallurgical testwork, process design for the plant and associated infrastructure, project planning, and engineering studies.

11. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.

12. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required for it to be disclosed and to make the Technical Report not misleading.

Signed and dated this 30th day of March, 2016 at West Perth, Western Australia, Australia.

Andrew Briggs