Trident Project

North West Province, Zambia
NI 43-101 Technical Report

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ITEM 1 SUMMARY

This Technical Report on the Trident Project (the property) has been prepared by Qualified Persons David Gray, Michael Lawlor and Andrew Briggs of First Quantum Minerals Pty Ltd (FQM, the issuer or the Company). This Technical Report prepared by FQM as an issuer, follows a previous report filed in May 2015 (FQM, May 2015) in respect of the two then current development projects at Trident, namely Sentinel and Enterprise.

The purpose of this Technical Report is to:

- document updated Mineral Resource and Mineral Reserve estimates for the property
- provide a commentary on the current status of operations at Sentinel, especially in regard to a proposed copper ore processing expansion
- provide an update on the proposed resumption of site development works at Enterprise and the concepts for commencement of nickel ore processing

The effective date for the Mineral Resource and Mineral Reserve estimates is 31st December 2019.

Development and construction activities for the processing plant (ie, designed to process cupriferous ore from Sentinel and nickeliferous ore from Enterprise) had commenced in the second half of 2012. Construction of the copper processing circuit was substantially completed in late 2014, with commissioning under way in the first half of 2015. Beyond the commissioning period, the eventual rate of annual ore processing was intended to reach 55 Mtpa. 62 Mtpa processing is now proposed, commencing from 2022, and with the increased concentrate production to be sold into the Zambian Copperbelt and also supplementing the feed into a proposed contemporary expansion of smelting capacity at the Company's Kansanshi operation.

The 4 Mtpa nickel processing circuit within the processing plant was also substantially completed in late 2014, and has remained on care and maintenance since that time. The rougher and cleaner flotation cells within the nickel processing circuit have been used at times for the recovery of copper from Sentinel Pit feed. After the suspension of development work at the Enterprise mining site in late 2015, involving the commencement of clearing and initial mining pre-strip, technical investigations were resumed in 2018 in anticipation of improved metal price circumstances.

It is now proposed that development activities will continue for a period from 2020, but with the start-up date for nickel ore mining and processing still to be determined.

1.1 Project location and ownership

The Trident Project is located in the north-west province of Zambia, approximately 150 km west of the town of Solwezi, which is the capital of the Zambian North West Province. Chingola, a major town in the Zambian Copperbelt, is approximately 180 km to the east of Solwezi.

First Quantum Minerals Ltd has a 100% interest in the Trident Project, through a subsidiary operating entity, Kalumbila Minerals Ltd (KML).

1.2 Project background

The Trident Project comprises an existing copper mining/processing operation and a nickel mining development project at Sentinel and Enterprise, respectively. The broader Trident Project includes regional exploration targets including the Intrepid copper deposit. As yet, Mineral Resources have not been defined for these exploration areas and they are not addressed in this Technical Report.

The process plant design is based on a conventional sulphide ore flotation circuit, with a separate circuit designed to process future nickel ore feed from Enterprise, or additional copper ore feed from Sentinel. Concentrate from Sentinel is road-hauled to Solwezi, to the Company's Kansanshi smelter (KCS), for further treatment, or is sold to third party facilities in the Zambian Copperbelt.

Once operational, the Enterprise Pit will be an adjunct to the Sentinel Pit, with mining operations at both sites feeding the common processing plant. Nickel ore will be road-hauled to a ROM pad at the plant wherefrom it will be primary crushed and then comminuted in a SAG-ball milling circuit ahead of a conventional flotation circuit. The projected annual average nickel production is around 35,000 tonnes of metal in concentrate.

Figure 1-1 is a plan of the Trident Project showing the extent of current operational areas at Sentinel, the development site at Enterprise, and the existing tailings storage facility.

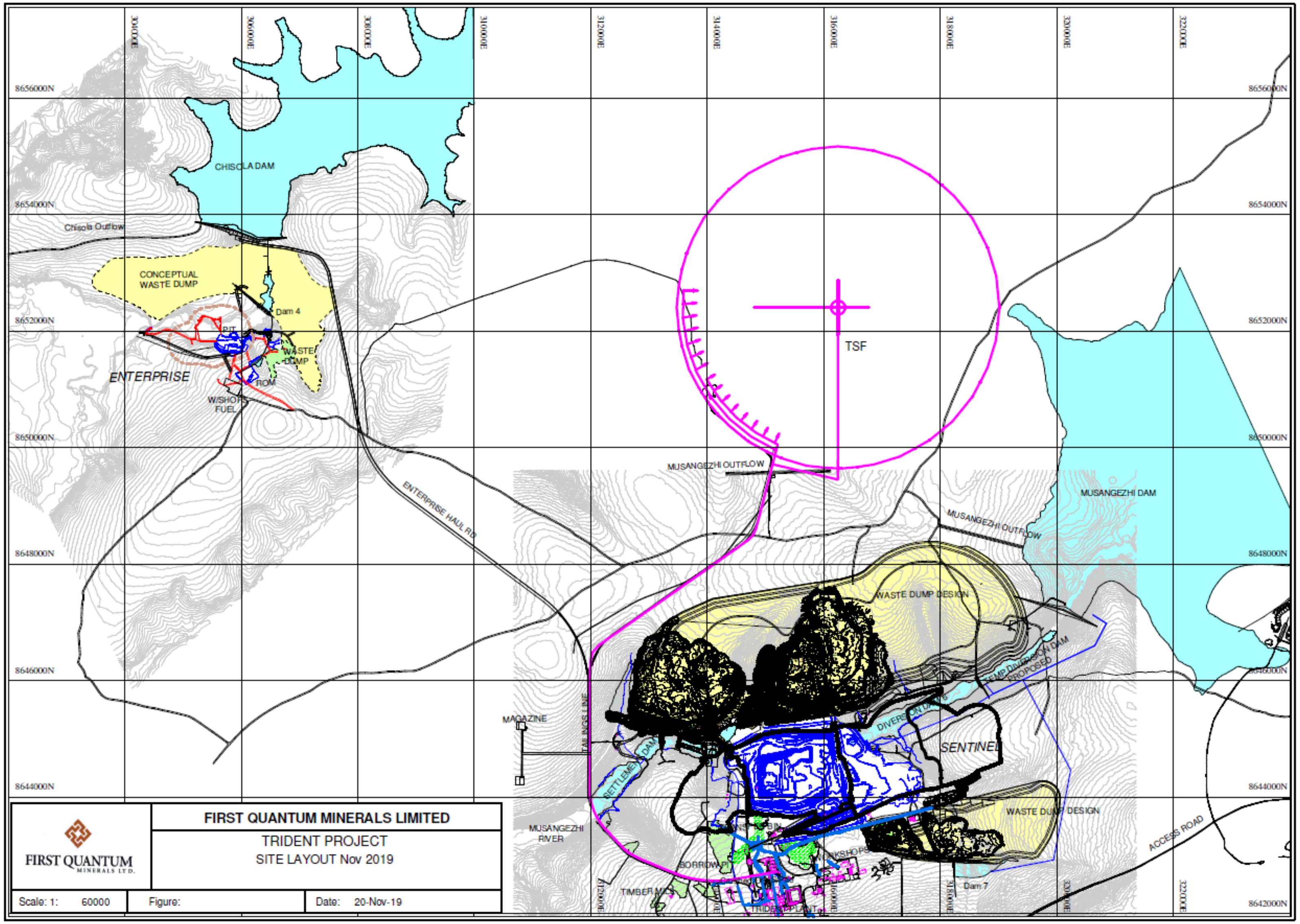
1.2.1 Sentinel

Mining at Sentinel is carried out using conventional open pit methods, with electric shovels, hydraulic excavators and a fleet of ultra class trucks enhanced by trolley-assisted haulage technology. Initial mine development commenced in mid 2013 to establish two starter pits within which in-pit crushing and conveying (IPCC) infrastructure was installed. These starter pits were located in the first of a number of along-strike mining phases, extending east to west over an ultimate length of 5.4 km. Since 2013, the initial Phase 1 pit has been mined along its full length of 1.8 km, to a width of 1.5 km, and to a current depth of approximately 200 m.

Three IPCC installations are operational in this first mining phase, with conveyors extending from the crushers to a common surface transfer bin and thence overland to a crushed ore stockpile at the processing plant. Mining commenced eastwards into the along-strike Phase 2 pit in 2018 and has reached a depth of approximately 25 m in this phase. Since 2015, total mined material movement has been in the range of 80 to 110 Mtpa.

Construction of the 55 Mtpa capacity copper circuit in the processing plant was completed in late 2014, with commissioning and progressive production ramp-up underway through 2015. According to the 2015 Technical Report (FQM, May 2015), and reflective of preliminary metallurgical testwork results, ore processing was expected to ramp up to 55 Mtpa by 2016, at an average copper (Cu) grade of 0.5% and an average copper recovery of 90%, to produce an annual copper metal output (in concentrate) of 247,000 to 297,000 t.

Figure 1-1 Trident Project site, Q4 2019



Despite the processing expectations, Sentinel start-up operations suffered from power supply interruptions, and recoveries from the initial feed sources of near surface non-primary ore were less than had been projected. Nevertheless, recoveries did improve to over 70% in 2016, and to 85% in 2017, when the plant feed became increasingly primary sulphide ore. In 2019 the plant processed 49.4 Mt of ore with a recovery of 89.5% and a concentrate grade of 26.6% Cu. Annual ore processing will continue to increase to +55 Mt in 2020 and then expand to a now proposed 62 Mtpa from 2022.

In order for the expanded production target to be met:

- a fourth IPC is now planned to be installed in the upper Phase 2 horizon for commissioning in 2021,
- screening of feed to the secondary crushers may be required to optimise performance,
- installation of two additional flotation columns is required to provide additional concentrate cleaning capacity, and
- operation of both secondary crushers is required to crush an increased tonnage from the expanded operations.

1.2.2 Enterprise

Environmental approval has been granted for the start of nickel ore mining at Enterprise, and limited site clearing, pre-stripping and development work was carried out in 2015 and 2016 before being deferred pending more favourable metal price circumstances.

Minimal infrastructure is proposed for the Enterprise site, as the processing facilities and support functions are located across at Sentinel. The main site components will be a small satellite administration office, a run-of-mine (ROM) ore pad, a fuel storage facility and a workshop area. Some of the proposed surface waste haul roads are already in place and the ore haul road connecting the ROM Pad with the existing main access road to the processing plant is partially complete. The top of the proposed starter pit was exposed in 2015 and this has afforded an opportunity for confirmatory drilling, geotechnical/hydrogeological investigations, and metallurgical test work sampling to be carried out when the development programme was resumed in 2018.

The Chisola River flows over the top of the Enterprise deposit and since 2015 has been dammed upstream (the Chisola Dam) to prevent inundation of the future pit and to supply process water to the processing plant. In addition to this major dam, a number of lesser surface water management structures have been completed for the catchment between the Chisola Dam and the proposed pit crest. When the mining pre-strip is resumed, one other surface water control dam is required to be constructed, in addition to completion of the haul road from Enterprise to the processing plant. A power line extension into the site will also be constructed, as will be the required satellite administration and mining facilities.

1.3 Project tenure

KML is the holder of five Large-scale Mining Licences. 15868-HQ-LML covers the Sentinel deposit, processing plant and supporting infrastructure, whilst 15869-HQ-LML covers the Enterprise deposit. 15870-HQ-LML, 15871-HQ-LML and 15872-HQ-LML cover exploration areas, sites for Project infrastructure and buffer zones to prevent encroachment of local settlements.

1.4 Geology and mineralization

The Trident Project is located on the western end of the Lufilian Arc, extending from northern Zambia across the Katanga Province of the Democratic Republic of Congo, and into northeast Angola. The Sentinel copper deposit and Enterprise nickel deposit are located along the margins of the Mesoproterozoic Kapombo Dome within a thick succession of Katanga Supergroup (c.880 to 550 Ma) sedimentary rocks.

1.4.1 Sentinel

The Sentinel copper deposit is a structurally modified, sediment hosted copper deposit with a strike extent of about 11 km and a dip extent of approximately 800 m. Copper mineralization is hosted within a phyllite package and occurs as a series of layered continuous sheets having a northerly 20^{o} to 30^{o} dip. Copper mineralization is dominated by chalcopyrite which occurs as fine to coarse disseminations and or veinlets.

1.4.2 Enterprise

The Enterprise deposit is a hydrothermal nickel deposit with mineralization hosted in a sequence of shale and siltstone units. These units have been preferentially mineralized due to rheological and geochemical interactions with mineralizing fluids. Enterprise mineralization has an unusual lack of spatial control from mafic intrusives and the primary source of nickel remains unclear. Structural deformation (faulting and folding) was modelled from core logging, early pit mapping and multi-element data. The deposit is characterised by a series of relatively shallow dipping bodies covering an area of 1,000 m by 500 m in the main area and approximately 800 m by 300 m in the south-west area. Nickel mineralization occurs mainly as sulphide disseminations and or veinlets.

1.5 Exploration status

1.5.1 Sentinel

Drilling at Sentinel dates back to 1959, where 31 holes were drilled up until 1964 by Mwinilunga Mines Ltd, a subsidiary of Roan Selection Trust Technical Services Ltd (RST). Anglo American Corporation Central Africa (AACCA) completed three diamond drilled holes and 34 reverse circulation holes between 1993 and 2000, exploring for nickel mineralization at Sentinel. However, only data from holes drilled by Kiwara PLC (Kiwara) (2007 to 2010) and FQM (2010 to 2019) have been used in the Mineral Resource estimate update for this Technical Report. Data from historical holes drilled by RST and AACCA were deemed to be of sub-standard quality to support robust Mineral Resource estimates.

1.5.2 Enterprise

Enterprise mineralization was discovered by AACCA in 1996. Kiwara acquired the prospect in 2007 and completed geochemical soil sampling in 2008-2009, which highlighted deposit targets for nickel mineralization. FQM acquired Enterprise in January 2010 and subsequent exploration work included airborne and ground electromagnetic (EM), and magnetic geophysical surveys covering the entire Trident Project area. Soil geochemical sampling was completed, followed by an intensive exploration and resource definition infill diamond (DD) drill programme completed during 2011 and 2012. 41 additional diamond drill holes with a closer grid were drilled in 2018-19 in the area of the potential starter pit.

1.6 Metallurgical summary

1.6.1 Sentinel

The Sentinel ore is crushed in-pit and conveyed overland onto a crushed ore stockpile ahead of two milling trains, each comprising a SAG mill and a single ball mill. Each train consists of two parallel banks of flotation cells, comprising seven cells operating in series. Final concentrate is thickened and filtered in a dedicated concentrate handling facility.

At the outset of mining and processing, near surface mining areas limited the opportunity to provide a blended feed and hence non-primary sulphide ore provided the predominant plant feed type at that time. During the plant commissioning phase, additional measures and processing improvements were investigated to assist in treating the unblended non-primary sulphide ore. Since 2015, there have been test work programmes and trials to improve flotation performance by means of pyrite rejection, chalcocite recovery and suppression of gangue minerals in the final concentrate. There has also been considerable plant development completed since commissioning. The start-up processing difficulties experienced with non-primary ore as the sole feed type may not be experienced to the same extent, going forward.

Based on newly derived variable recovery equations, which account for actual recoveries recorded in 2018 and 2019, and in view of the aforementioned on-going metallurgical improvements, the overall average metallurgical parameters for mine planning and Mineral Reserve estimation are 91.2% recovery for primary sulphide, and 79.0% recovery for the significantly smaller proportion of near-surface non-primary sulphide.

1.6.2 Enterprise

The crushed Enterprise ore will be milled in a SAG-ball milling circuit, and the ground product floated in a circuit comprising talc pre-float, nickel rougher flotation and three stages of cleaning. Final concentrate will be thickened and filtered in a dedicated concentrate handling facility.

The Enterprise circuits will share the processing plant infrastructure. Tailings will be discharged to the Sentinel tailings thickeners and thence to a common tailings storage facility.

Based on test work to date, the recommended metallurgical parameters for mine planning and Mineral Reserve estimation are 85% recovery for primary sulphide, and 60% recovery for the relatively smaller proportion of non-primary sulphide.

Further testwork is planned to confirm recoveries and reagent consumptions. Results are expected in late 2020.

1.6.3 Tailings storage

The Trident Project features a single tailings storage facility (TSF), designed to accommodate the combined tailings generated from processing of Sentinel and Enterprise ores. The circular TSF is 5.5 km in diameter and will eventually be around 40 m high. It has a capacity of approximately 1,000 million tonnes of tailings, ultimately impounded in four quadrants. The TSF was commenced with an earthen perimeter embankment and over time, will be upstream raised with tailings.

1.7 Environmental approvals and status

The granting of the large scale mining licences (LMLs) for the Trident Project was conditional upon approval by the Zambian Environmental Management Agency (ZEMA) of the Sentinel Environmental and Social Impact Assessment (ESIA), which was submitted in early February and approved in July 2011. The Sentinel Addendum ESIA, which included modifications to the Project (eg, amendments to the TSF, waste dump design and process water facilities) was approved by ZEMA in August 2013. The separate Enterprise ESIA was approved in September 2014. In addition, the Trident Project Resettlement Action Plan (RAP) has also been approved by ZEMA.

Each approval is accompanied by a list of environmental and social commitments, which vary depending on the project and perceived impact. The environmental commitments typically require the implementation of a number of control measures and adherence to related Zambian legislation, effluent and emissions limits. The social commitments typically require honouring the conditions of the RAP agreements and ensuring all resettled communities have received their entitlements.

All environmental and social commitments from these approvals have been captured in a site legal register and a Consolidated Environmental Management Plan.

1.8 Operations and development status

The construction phase for the Trident Project commenced during 2012, focussing first on the building of the Sentinel processing circuits and common infrastructure (eg, the Chisola Dam process water reservoir and the TSF). There followed the first of the in-pit crushers and conveyors within the Sentinel Pit from late 2013 through into 2014, and subsequently the building of the Enterprise processing circuits (including a dedicated primary crusher).

During this period, construction proceeded on the new Kalumbila town site for the housing of the operations personnel and their families. The town site now caters for approximately 7,500 people accommodated in over 1,300 houses, and is serviced by schools, a clinic, shopping/banking and related facilities. A 30 km long bitumen surfaced road has been constructed to connect the town site with the existing national road. A new airstrip has been built adjacent to the town site and a 600 km long power line has been constructed to service the Project and the town site.

The Musangezhi River has been diverted to allow the Sentinel Pit to be developed. Likewise, the Chisola River has been diverted to allow the Enterprise Pit to be developed. In the case of the former, the dammed river forms a recreational lake alongside the Kalumbila town site whilst the Chisola River dam is a source of process water.

At the time of the 2015 Technical Report, commercial production of copper metal had not been declared. Commercial production was eventually declared in November 2016.

Although substantially completed in 2014, some elements of the Enterprise plant require refurbishment and/or reinstatement to enable processing of nickel ore to commence.

1.8.1 Sentinel

In relation to Sentinel, noteworthy changes from the 2015 Technical Report (FQM, May 2015) production plan are:

- the proposed expansion of cupriferous ore processing to 62 Mtpa, commencing in 2022
- a commensurate increase in total mining movement capacity to about 180 Mtpa from 2022, and 190 Mtpa from 2026
- installation of a fourth IPC during 2021, near-surface, in the Phase 2 pit

Capital costs have been estimated to account for the expansion in mining and processing capacity, and these costs are included in the pre-tax cashflow model produced for this Technical Report.

In addition to the expanded production target, other technical aspects which have changed since the 2015 Technical Report (FQM, May 2015) are as follows:

- revision from a five-phase pit to a four-phase pit design, with revised future IPC relocation positions to suit
- modified overall pit slope angles based on updated geotechnical design criteria
- changes in the relative annual feed proportions of primary and non-primary (OTA and OTB types) plant feed, and high grade / low grade feed (where low grade feed is primary WTA and non-primary WTB types)
- increased recovery rate for non-primary plant feed (OTB feed type)
- feeding of all ore above the marginal cut-off grade

1.8.2 Enterprise

Currently there are few changes relative to the 2015 Technical Report (FQM, May 2015) development plan and concepts. In the lead up to the proposed resumption of the Enterprise development programme, the status of site works and technical investigations is as follows:

- during 2018, minor geotextile stabilisation works were carried out along the upper south wall of the starter pit, exposed since 2015
- remediation works were completed on two of the existing surface water control dams and on the Chisola diversion channel
- ore control drilling was completed as the basis of an updated and improved geological and Mineral Resource model
- eight geotechnical holes were drilled and logged to provide laboratory test work samples and to enable an update to be made to the pit slope geotechnical design parameters
- six out of ten planned groundwater pilot boreholes were drilled, with piezometers subsequently installed
- representative samples were selected from newly drilled cores and submitted for metallurgical testing (although results will not be available until late Q1 2020)
- a development programme has been determined, with an unspecified processing start time, and accounting for:
    - a resumption in pre-strip mining
    - completion of the haul road connecting Enterprise to the processing plant
    - work required to enable the processing plant to be commissioned for nickel ore processing
    - construction of another surface water control dam, and
    - installation of other infrastructure items including power lines, office, workshops, pumps and waterlines

Capital costs have been estimated to account for the resumption of site development at Enterprise, and these costs are included in the pre-tax cashflow model produced for this Technical Report.

1.9 Mineral Resource estimate summary

1.9.1 Sentinel

Block model grade estimates for the Sentinel deposit were updated in 2019 by Carmelo Gomez of FQM, and the site geologists, under the supervision of the QP, David Gray of FQM (Group Mine and Resource Geologist). The Sentinel Mineral Resource estimates were supported by a grid of diamond drilled holes covering the extents of the deposit with a closer grid of reverse circulation drilled holes across the production area. Assay data from an additional 38 infill diamond drilled holes and 17,553 reverse circulation drilled holes has been included in this estimate since the 2015 Technical Report estimate (FQM, May 2015). The drillhole sample assay results and diamond core geology logging data have been used for interpretation of geology models that relate to the spatial distribution of copper mineralization. Block grade estimation was by ordinary kriging into a panel block size of 60 mE by 36 mN on 12 m benches and is considered appropriate for the distribution of sample data and the deposit type. Post processing by localised uniform conditioning of the copper panel estimates used a smallest mining unit block size of 6 mE by 6 mN by 12 m high flitches.

Sentinel Mineral Resource estimates were classified according to drill grid spacing, geological confidence and accuracy in the panel grade estimate, as well as in consideration of the sampling and preparation methods, analytical techniques, the associated data quality and as having reasonable prospects for eventual economic extraction. Additional data and confidences have provided 68% of Mineral Resources classed as Measured and 32% classed as Indicated.

The resulting Mineral Resources have been reported using a copper cut-off grade of 0.13% as at December 2019, as per Table 1-1. The reported Mineral Resources are inclusive of the Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-1 Sentinel Mineral Resource statement using a 0.13% copper cut-off grade and depleted for mining as at 31st December 2019

Classification	Material	Tonnes (Mt)	Density	TCu (%)	Cu metal (kt)
Measured	Non-primary sulphide	78.9	2.71	0.36	287.16
Measured	Primary sulphide	578.8	2.79	0.49	2,852.77
	Measured subtotal	**657.7**	**2.78**	**0.48**	**3,139.94**
Indicated	Non-primary sulphide	26.0	2.77	0.30	76.82
Indicated	Primary sulphide	277.0	2.81	0.42	1,163.38
	Indicated subtotal	**303.0**	**2.81**	**0.41**	**1,240.20**
	Measured and Indicated total	**960.7**	**2.79**	**0.46**	**4,380.14**
Inferred	Non-primary sulphide	5.3	2.73	0.27	14.05
Inferred	Primary sulphide	57.0	2.80	0.37	212.41
	Inferred subtotal	**62.3**	**2.80**	**0.36**	**226.46**

1.9.2 Enterprise

The Enterprise Mineral Resource estimate was completed for the Main Deposit using additional drilled data together with improved lithological and structural models. The estimate included mapping, drilling and sampling data, which adequately defined geological and grade continuity suitable for the estimation of Measured, Indicated and Inferred Mineral Resources. The estimation technique used ordinary kriging with post processing of parent block estimates into a smallest mining unit block dimension, providing grade and tonnage data relevant at the scale of mining.

The Mineral Resource statement (Table 1-2) uses a 0.15% total nickel cut-off grade as at 31st December 2019 and is inclusive of Mineral Reserves. Measured Resources comprise 24% and Indicated 76% of the resources available for conversion into Mineral Reserves. Inferred Mineral Resources have some opportunity to be converted into Indicated Mineral Resources with future infill drilling and improved geological understanding. Reported Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-2 Enterprise Mineral Resource statement as at 31st December 2019 and using a 0.15% Ni cut-off grade

Classification	Material	Tonnes (Mt)	Density (t/m^3)	Ni (%)	Ni metal (kt)
Measured	Non-primary sulphide	3.7	2.56	1.07	39.3
	Primary sulphide	5.5	2.87	1.60	88.1
Measured subtotal		**9.2**	**2.75**	**1.38**	**127.4**
Indicated	Non-primary sulphide	2.7	2.71	0.53	14.4
	Primary sulphide	25.7	2.87	0.95	244.3
Indicated subtotal		**28.5**	**2.86**	**0.91**	**258.7**
Measured and Indicated total		**37.7**	**2.83**	**1.03**	**386.1**
Inferred	Non-primary sulphide	1.1	2.71	0.60	6.7
	Primary sulphide	8.2	2.87	0.73	59.8
Inferred subtotal		**9.3**	**2.85**	**0.71**	**66.5**

1.10 Mineral Reserve estimate summary

The detailed mine planning work described in this Technical Report, including conventional optimisation processes, open pit designs, life of mine (LOM) production scheduling and resulting Mineral Reserve estimates, was completed by FQM staff under the supervision of Michael Lawlor (QP).

The planning detail included comprehensive and updated life of mine phase pit designs for Sentinel, incorporating current and proposed future in-pit crusher positions and conveyor routes, and the scheduling of ore and waste mining to suit an orderly sequence and progression through these pit phases.

In the optimisation and planning for both Sentinel and Enterprise mining extents, the mining and other unit operating costs have been reviewed and updated.

1.10.1 Sentinel

The reported Mineral Reserve for Sentinel (inclusive of stockpiles) is listed in Table 1-3 and is based on an economic cut-off grade, which accounts for a longer-term copper metal price projection of $3.00/lb ($6,615/t).

Table 1-3 Sentinel Pit Mineral Reserve statement, depleted for mining as at 31st December 2019, and based on a $3.00/lb Cu price

Mineral Reserve statement as at end of December 2019 at $3.00/lb Cu				
Classification	**Material**	**Tonnes (Mt)**	**TCu (%)**	**Cu metal (kt)**
Proven	Non-primary sulphide	63.7	0.40	254.0
Proven	Primary sulphide	539.1	0.49	2,663.9
	Proven subtotal	**602.8**	**0.48**	**2,917.9**
Probable	Non-primary sulphide	20.9	0.31	65.1
Probable	Primary sulphide	224.3	0.43	960.3
	Probable subtotal	**245.2**	**0.42**	**1,025.3**
	Proven and Probable subtotal	**847.9**	**0.47**	**3,943.3**
Probable	Stockpiles	28.9	0.24	69.1
	Proven and Probable total	**876.8**	**0.46**	**4,012.4**

The waste associated with this Mineral Reserve estimate is 1,633.3 Mt, with an overall average strip ratio of 1.9 : 1. The mine life is 15 years to 2034.

1.10.2 Enterprise

The reported Mineral Reserve for Enterprise is listed in Table 1-4 and is based on an economic cut-off grade which accounts for a longer-term nickel metal price projection of $7.50/lb ($16,535/t).

Table 1-4 Enterprise Mineral Reserves statement, in-pit inventory at 31st December 2019, and based on a $7.50/lb Ni price

Mineral Reserve statement as at end of December 2019 at $7.50/lb Ni				
Classification	**Material**	**Tonnes (Mt)**	**Ni (%)**	**Ni metal (kt)**
Proven	Non-primary sulphide	3.8	0.98	37.45
Proven	Primary sulphide	5.7	1.46	84.0
	Proven subtotal	**9.6**	**1.27**	**121.4**
Probable	Non-primary sulphide	2.0	0.44	8.68
Probable	Primary sulphide	23.1	0.93	214.4
	Probable subtotal	**25.1**	**0.89**	**223.1**
	Proven and Probable total	**34.7**	**0.99**	**344.5**

The waste associated with this Mineral Reserve estimate is 286.7 Mt, with an overall average strip ratio of 8.3 : 1. The mine life, after an initial pre strip year, is 11 years.

1.11 Production schedule

1.11.1 Sentinel

Table 1-5 lists the Sentinel life of mine production schedule that is associated with the updated ultimate and pit phase designs, and with the Mineral Reserve statement as at the end of December 2019. The tonnage and grade differences between the mined ore and the crusher feed are attributable to the starting ore stockpile balance.

Table 1-5 Sentinel life of mine production schedule

Year	Mined Ore (Mt)	Mined Ore (%TCu)	Mined Waste (Mt)	Total Mined (Mt)	Total Mined (Mbcm)	Strip ratio	Crusher feed (Mt)	Crusher feed (%TCu)	Insitu Cu (kt)	Rec'd Cu (kt)	Recovery (%)
2020	63.2	0.48	97.6	160.8	60.2	1.5	54.4	0.52	283.0	245.1	86.6%
2021	64.4	0.46	98.3	162.7	60.5	1.5	58.0	0.50	292.4	259.2	88.6%
2022	61.6	0.54	120.5	182.0	68.3	2.0	61.9	0.54	336.0	302.4	90.0%
2023	63.9	0.51	116.0	179.8	64.9	1.8	62.0	0.52	320.7	292.4	91.2%
2024	66.1	0.46	118.6	184.8	68.6	1.8	62.0	0.47	293.9	261.2	88.9%
2025	63.5	0.47	125.5	189.0	68.5	2.0	62.0	0.48	297.3	268.7	90.4%
2026	69.4	0.46	120.6	190.0	72.7	1.7	62.0	0.49	305.6	277.3	90.7%
2027	64.6	0.50	125.2	189.9	70.0	1.9	62.0	0.52	320.4	291.5	91.0%
2028	67.6	0.46	122.3	189.8	68.2	1.8	62.0	0.49	301.5	272.6	90.4%
2029	47.8	0.37	121.9	169.7	61.1	2.6	62.0	0.34	211.1	182.8	86.6%
2030	40.6	0.38	120.9	161.5	58.0	3.0	62.0	0.32	197.8	173.0	87.5%
2031	33.4	0.50	124.6	158.0	56.8	3.7	62.0	0.34	211.3	191.1	90.4%
2032	60.0	0.37	119.6	179.6	64.7	2.0	62.0	0.37	227.6	205.2	90.2%
2033	52.3	0.44	88.8	141.1	50.5	1.7	53.7	0.43	230.6	210.2	91.1%
2034	29.5	0.60	13.1	42.5	15.1	0.4	28.9	0.61	175.3	160.0	91.3%
LOM (diluted)	**847.9**	**0.47**	**1,633.3**	**2,481.3**	**908.3**	**1.9**	**876.8**	**0.46**	**4,004.6**	**3,592.6**	**89.7%**

Features of this mining and plant feed schedule are as follows:

- As at the end of December 2019, the Project life is 15 years (ie, to 2034), whereas the scheduled mine life reported in the 2015 Technical Report (FQM, May 2015) was 18.5 years to 2033. This is largely a function of a changed mining cut-off grade strategy, from an elevated 0.20% Cu in the 2015 Mineral Reserve plan, to a marginal (economic) cut-off grade approach.
- The total material mined amounts to 2,481.3 Mt (908.3 Mbcm), of which 847.9 Mt is ore (primary and non-primary) and 1,633.3 Mt is waste. This is on a mining/diluted recovered basis, assuming a mining dilution factor of 103% (at a nil diluent grade) and a mining recovery factor of 97%.
- Throughout the Project life, the total ore mined to stockpiles amounts to 56.0 Mt at an average grade of 0.19% Cu, whilst that reclaimed is 84.8 Mt at an average grade of 0.21% Cu. The reclaim inventory includes 28.9 Mt from the currently existing stockpile balance.
- The maximum size of the long term stockpiles is 67.8 Mt in 2028.
- The 62 Mtpa processing rate is essentially achieved from 2022, and is maintained until 2032.

Figure 1-2 shows a comparison between the life of mine production schedule reported in 2015 and that listed in Table 1-5. To note are:

- the pit phases and mining areas differ spatially between the 2015 and 2020 versions of the schedule
- the recovered metal spike in 2022 to 2023 is due to the throughput increase; the head grades are similar
- the significant recovered metal spike in 2025 to 2028 is due to a combination of increased throughput and increased head grades

Figure 1-2 Comparison between Technical Report production schedules

1.11.2 Enterprise

Table 1-6 lists the Enterprise life of mine production schedule that is associated with the ultimate and pit phase designs, and with the Mineral Reserve statement as at the end of December 2019.

Table 1-6 Enterprise life of mine production schedule

Year	Mined Ore		Mined Waste	Total Mined		Strip	Crusher Feed		Insitu Ni	Rec'd Ni	Recovery
	(Mt)	(%Ni)	(Mt)	(Mt)	(Mbcm)	ratio	(Mt)	(%Ni)	(kt)	(kt)	%
Prestrip	0.01	0.20	11.9	11.9	4.8						
Prestrip/Yr1	1.0	0.50	15.8	16.8	7.2	16.5	1.0	0.50	4.8	2.9	60.0%
Year 2	3.3	0.90	24.7	28.0	11.7	7.5	3.3	0.90	29.8	18.4	61.8%
Year 3	2.3	1.09	38.9	41.2	15.9	17.0	2.3	1.09	25.0	18.5	74.2%
Year 4	3.2	1.44	43.2	46.4	17.0	13.6	3.2	1.44	45.8	38.4	84.0%
Year 5	3.9	1.29	42.3	46.2	17.0	11.0	3.9	1.29	49.8	42.3	84.8%
Year 6	3.1	1.03	39.6	42.7	15.4	12.9	3.1	1.03	31.8	27.0	85.0%
Year 7	4.0	0.71	26.8	30.8	10.8	6.7	4.0	0.71	28.3	24.0	84.8%
Year 8	4.0	1.13	17.6	21.6	7.5	4.4	4.0	1.13	45.0	38.1	84.7%
Year 9	4.0	1.02	14.7	18.6	6.5	3.7	4.0	1.02	40.5	34.4	85.0%
Year 10	4.0	0.70	8.7	12.6	4.4	2.2	4.0	0.70	27.6	23.4	85.0%
Year 11	2.1	0.76	2.4	4.6	1.6	1.1	2.1	0.76	16.3	13.8	85.0%
LOM (diluted)	**34.7**	**0.99**	**286.7**	**321.4**	**119.7**	**8.3**	**34.7**	**0.99**	**344.5**	**281.3**	**81.7%**

Features of this mining and plant feed schedule are as follows:

- Beyond the pre-strip period commencing in 2020, the production schedule timeframe is notional; the commencement time for nickel ore processing is yet to be determined.

- The total material mined is 321.4 Mt (119.7 Mbcm), of which 34.7 Mt is ore (primary and non-primary) and 286.7 Mt is waste. This is on a mining diluted/recovered basis, assuming a mining dilution factor of 105% (at a nil diluent grade) and a mining recovery factor of 95%.
- There are no stockpiles; all ore mined is hauled to surface for ROM reclaim into road transport across to the processing plant.
- Due to the stockpiling limitation, the geometry of the pit, and combined with sensible vertical mining advance rate constraints, it is not possible to achieve a consistent 4 Mtpa processing rate.

1.12 Capital and operating cost estimates

1.12.1 Sentinel

Table 1-7 to Table 1-9 list the capital, sustaining and closure cost provisions in the cashflow model supporting the Sentinel Mineral Reserve estimate.

Mining and other unit operating cost estimates have been updated since the 2015 Technical Report (FQM, May 2015). The updated unit costs are:

- overall average waste drill, blast, load and haul cost = \$5.05/bcm (\$1.86/t)
- overall average ore drill, blast, load and haul cost = \$5.48/bcm (\$1.96/t)
- overall average process operating cost = \$4.85/t
- overall average general and administration (G&A) charge = \$1.00/t

Between the Mineral Reserve pit optimisation process and cashflow modelling, the combined overall average process operating plus G&A cost of \$5.85/t was revised upwards to \$6.02/t (a 3% increase). The unit mining costs were also adjusted to account for ore hauls direct to IPCs, and to and from surface stockpiles (the latter without a drill and blast cost component).

In terms of metal costs, updated estimates account for transport charges to the Kansanshi smelter (KCS), smelting costs, anode transport costs and subsequent refining charges. Account was also taken of an assumed proportion of Sentinel concentrate being delivered to an external smelter on the Zambian Copperbelt. The updated overall estimates equate to:

- TCRCs = \$0.34/lb for concentrate to the KCS, or \$0.45/lb for concentrate delivered elsewhere
- average TCRCs = \$0.38/lb, assuming 63% of the Sentinel concentrate is delivered to the KCS and the balance is delivered elsewhere
- royalties = 7.5% of \$3.00/lb = \$0.23/lb

Table 1-7 Sentinel capital cost provisions

	2020	2021	2022	2023	2024	TOTAL
Mining capital ($M)						
Inpit crusher IPC4A	$27.0	$32.0				**$59.0**
Trolley assist phases 4 & 5	$6.5					**$6.5**
Additional electric shovel (+10% duty incl.)		$27.5				**$27.5**
2 X additional haul trucks (+10% duty incl.)		$13.2				**$13.2**
1 X additional dozers (+10% duty incl.)		$2.2				**$2.2**
1 X additional drill rigs / diesel PVs (+10% duty incl.)		$3.3				**$3.3**
Geotech - IPC4A ground support						**$0.0**
subtotal ($M)	**$33.5**	**$78.2**	**$0.0**	**$0.0**	**$0.0**	**$111.7**
Processing capital ($M)						
Aggregate crusher upgrade	$0.1					**$0.1**
Float column	$1.7					**$1.7**
Front end rectification (complete)						**$0.0**
Replacement pebble magnet			$10.0			**$10.0**
IPC and conveyor reloc. plus mods and new splitter					$20.0	**$20.0**
SAG discharge screen	$2.0	$2.0	$2.0	$2.0	$2.0	**$10.0**
Tails piping (complete)						**$0.0**
Punch list rectification (complete)						**$0.0**
subtotal ($M)	**$3.8**	**$2.0**	**$12.0**	**$2.0**	**$22.0**	**$41.8**
Other capital ($M)						
Site Projects						**$0.0**
Dam 6B Construction						**$0.0**
Carry over (From 2019)	$0.5	$1.0				**$1.5**
Other Items	$2.2					**$2.2**
subtotal ($M)	**$2.7**	**$1.0**	**$0.0**	**$0.0**	**$0.0**	**$3.7**
Total ($M)	**$40.0**	**$81.2**	**$12.0**	**$2.0**	**$22.0**	**$157.2**

Table 1-8 Sentinel sustaining cost provisions

	2020	2021	2022	2023	2024	>2024	TOTAL
Mine sustaining ($M)							
Mining eq't - planned maintenance parts	$39.5	$37.8	$37.1	$38.1	$18.0		**$170.5**
MD6640 drill mast	$1.9	$0.0	$0.0	$0.0	$0.0		**$1.9**
Refurbishment of HD1500	$1.0	$0.0	$0.0	$0.0	$0.0		**$1.0**
CAT 7945 dipper handle	$0.0	$1.2	$0.0	$0.0	$0.0		**$1.2**
New bucket and dump trays welding area	$1.0	$0.0	$0.0	$0.0	$0.0		**$1.0**
Mine services - 1 X motivator	$0.0	$1.5	$0.0	$0.0	$0.0		**$1.5**
4 X additional drill rigs	$3.8	$0.0	$0.0	$0.0	$0.0		**$3.8**
Road sweeper for treated haul roads	$0.7	$0.5	$0.0	$0.0	$0.0		**$1.2**
Pit dewatering - 2 X multiflow pumps and MCCs	$0.0	$1.5	$1.0	$1.0	$0.1		**$3.6**
Progressive sustaining costs (at 5% of opex)						$111.0	**$111.0**
subtotal ($M)	**$47.8**	**$42.5**	**$38.1**	**$39.1**	**$18.1**	**$111.0**	**$296.7**
Process sustaining ($M)							
SAG and ball mill segment coils	$0.0	$0.0	$1.1	$0.5	$0.0		**$1.6**
XRF and fusion system	$0.0	$1.0	$0.0	$0.0	$0.0		**$1.0**
Installation of new clear water tank for chillers	$0.0	$0.0	$1.3	$0.0	$0.0		**$1.3**
Progressive sustaining costs (at 5% of opex)						$125.8	**$125.8**
subtotal ($M)	**$0.0**	**$1.0**	**$2.4**	**$0.5**	**$0.0**	**$125.8**	**$129.8**
Other sustaining ($M)							
Capital components (not PCR)	$3.6	$4.4	$4.4	$4.4	$4.4		**$21.2**
ETP plant relocate and new strategy	$0.0	$15.0	$0.0	$0.0	$0.0		**$15.0**
New fabrication shop	$0.0	$1.2	$0.0	$1.2	$0.0		**$2.4**
Light vehicles	$0.6	$0.6	$0.0	$0.0	$0.0		**$1.2**
Carry over (From 2019)	$1.5	$0.0	$0.0	$0.0	$0.0		**$1.5**
Other Items	$5.7	$5.9	$3.1	$4.5	$1.8		**$21.0**
subtotal ($M)	**$11.4**	**$27.1**	**$7.5**	**$10.1**	**$6.2**		**$62.3**
Total ($M)	**$59.2**	**$70.6**	**$48.0**	**$49.7**	**$24.3**	**$236.9**	**$488.7**

Table 1-9 Sentinel closure cost provisions

	Progressive 2020 to 2031	Final 2032 to 2034	($M)
Closure components			
Infrastructural aspects:			
dismantling of structures	$0.0	$12.3	$12.3
rehabilitation of roads	$0.0	$0.2	$0.2
removal of linear structures	$0.0	$1.5	$1.5
disposal of demolition waste	$0.0	$2.2	$2.2
subtotal	**$0.0**	**$16.3**	**$16.3**
Mining aspects:			
pit rehabilitation	$0.4	$4.6	$5.0
waste dump rehabilitation	$27.0	$0.0	$27.0
surface water pond rehabilitation	$0.0	$0.8	$0.8
TSF rehabilitation	$12.6	$0.0	$12.6
subtotal	**$40.0**	**$5.4**	**$45.5**
General surface rehabilitation	$0.0	$3.0	$3.0
Water management	$0.0	$0.3	$0.3
Subtotal closure costs	**$40.0**	**$25.0**	**$65.0**
Post-closure components			
Surface water monitoring	$0.0	$0.1	$0.1
Groundwater monitoring	$0.0	$0.0	$0.0
Rehabilitation monitoring	$0.0	$0.1	$0.1
Care and maintenance	$0.0	$1.3	$1.3
Contingencies for post-closure	$0.0	$0.2	$0.2
Subtotal post-closure costs	**$0.0**	**$1.7**	**$1.7**
Additional allowances			
Preliminary and general	$0.0	$9.8	$9.8
Contingencies	$0.0	$0.0	$0.0
Additional studies	$0.0	$1.6	$1.6
Subtotal additional costs	**$0.0**	**$11.4**	**$11.4**
Total closure costs	**$40.0**	**$38.2**	**$78.2**

1.12.2 Enterprise

Table 1-10 lists the capital cost provisions in the cashflow model supporting the Enterprise Mineral Reserve estimate. A provision has been made for sustaining capital allowances equal to 5% of the operating costs in each year (excluding contracted mining costs). A closure cost provision of $6.5 M has also been made.

Mining and other unit operating cost estimates have been updated since the 2015 Technical Report (FQM, May 2015). The updated unit costs are:

- ore and waste drill and blast cost = $2.45/bcm ($0.90/t)
- overall average waste load and haul cost = $3.96/bcm ($1.49/t)
- overall average ore load and haul cost = $3.81/bcm ($1.36/t)
- overall average process operating cost (fixed plus variable) = $8.46/t
- overall average general and administration charge = $0.00/t
- ore reclaim and surface road haulage to processing plant = $1.50/t

In addition to royalties (5% of $7.50/lb = $0.375/lb Ni), the envisaged Project metal costs will include concentrate road transport charges to Walvis Bay, Namibia, and separate ocean freight charges for transport thereon. As advised by the Company's metals marketing group (MCT), the total TCRCs would be in the order of $2.88/lb.

Table 1-10 Enterprise capital cost provisions

		2019	2020	2021	2022	2023	TOTAL
Site capital ($M)							
	Powerline and transformers		$0.1	$0.6	$0.0	$0.0	**$0.8**
	Civil works on existing dams		$0.1	$0.0	$0.0	$0.0	**$0.1**
	Dam 4 earthworks		$1.2	$0.0	$0.0	$0.0	**$1.2**
	Dam 4: 6 x 90 kW pumps + 1.8 km x 630 mm pipe		$0.8	$0.0	$0.0	$0.0	**$0.8**
	Dam 2: 2 x 90kW + pipes		$0.0	$0.1	$0.0	$0.0	**$0.1**
	Fuel storage facility		$1.0	$1.0	$0.0	$0.0	**$2.0**
	subtotal ($M)	**$0.0**	**$3.3**	**$1.8**	**$0.0**	**$0.0**	**$5.0**
Mining capital ($M)							
	Mining pre-strip		$20.7	$31.5	$1.8	$0.0	**$54.0**
	Haul road upgrade		$1.8	$0.0	$0.0	$0.0	**$1.8**
	Pit dewatering bores		$0.4	$1.1	$0.0	$0.0	**$1.5**
	Geotechnical: slope monitoring radar		$0.0	$0.5	$0.0	$0.0	**$0.5**
	Two-way radio network upgrade		$0.2	$0.0	$0.0	$0.0	**$0.2**
	subtotal ($M)	**$0.0**	**$23.1**	**$33.1**	**$1.8**	**$0.0**	**$58.0**
Processing capital ($M)							
	Trident plant works		$0.4	$0.0	$0.0	$0.0	**$0.4**
	Other						**$0.0**
	subtotal ($M)	**$0.0**	**$0.4**	**$0.0**	**$0.0**	**$0.0**	**$0.4**
Other capital ($M)							
	Re-approved technical work		$1.0	$0.0	$0.0	$0.0	**$1.0**
	Resource drilling and assaying		$0.9	$0.0	$0.0	$0.0	**$0.9**
	Metallurgical testwork		$0.1	$0.0	$0.0	$0.0	**$0.1**
	subtotal ($M)	**$0.0**	**$1.9**	**$0.0**	**$0.0**	**$0.0**	**$1.9**
	Total ($M)	**$0.0**	**$28.7**	**$34.9**	**$1.8**	**$0.0**	**$65.4**

1.13 Economic analysis

1.13.1 Sentinel

An economic analysis in the form of a pre-tax cashflow model to support the Sentinel Mineral Reserve estimate is summarised in Table 1-11 (the cashflow is listed annually in Item 22). The model shows the indicative cashflow and does not replace a more comprehensive financial model that exists for the operations.

The annual revenues are calculated from late 2019 metal price projections, for which the adopted median long term price is $3.00/lb ($6,614/t). The modelled overall average processing recovery is 90.0% and the modelled payable metal factor is 96.6%. For an undiscounted life of mine cashflow of $7,989.8 M, the indicative net present value (NPV) at a 10% discount rate is $4,356.8 M. At an 8.5% discount rate, the indicative NPV is $4,713.9 M.

1.13.2 Enterprise

An economic analysis in the form of a pre-tax cashflow model to support the Enterprise Mineral Reserve estimate is summarised in Table 1-12 (the cashflow is listed annually in Item 22). The model shows the indicative cashflow and does not replace a more comprehensive financial model that exists for the operations.

The annual revenues are calculated from late 2019 metal price projections, for which the adopted long term price is $7.50/lb ($16,535/t). The modelled overall average processing recovery is 81.7% and the modelled payable metal factor is 75%. Assuming that there would be continuous cashflows over eleven years, and commencing from the pre-strip phase, the notional NPV at a 10% discount rate would be $818.5 M. At an 8.5% discount rate, the notional NPV would be $906.8 M.

Table 1-11 Sentinel Mineral Reserve cashflow model summary

		UNITS	TOTAL	2019	2020 TO 2024	2025 TO 2029	2030 TO 2034
MINING (after mining dilution and recovery)							
	Total ore	Mt	**847.9**		319.2	312.9	215.8
	Total waste	Mt	**1,633.3**		550.9	615.4	467.0
	Total mined	Mt	**2,481.3**		870.1	928.4	682.8
	Strip ratio	t : t	**1.9**		1.7	2.0	2.2
Reclaim							
	Active (OTA/OTB) reclaim	Mt	**6.2**		4.2	1.9	0.0
	Long Term (WTA/WTB) reclaim	Mt	**78.7**		0.6	20.6	57.5
	Total reclaim	Mt	**84.8**		4.8	22.6	57.5
PIT TO IPCs DIRECT							
	Total ore	Mt	**792.0**		293.5	287.4	211.1
	Grade	%	**0.50**		0.53	0.50	0.46
PIT TO STOCKPILE							
	Total ore	Mt	**56.0**		27.0	25.5	3.5
	Grade	%	**0.19**		0.22	0.17	0.16
STOCKPILE TO IPCs							
	Total ore	Mt	**84.8**		4.8	22.6	57.5
	Grade	%	**0.21**		0.45	0.23	0.18
STOCKPILE BALANCE							
	Total ore	Mt		28.9	51.1	54.0	0.0
	Grade	%		0.24	0.21	0.18	0.00
TOTAL FEED TO PLANT (before mining dilution and recovery)							
	Total ore	Mt	**876.8**		298.2	310.0	268.6
	Grade	%	**0.47**		0.53	0.48	0.40
	Insitu metal	**kt**	**4,127.9**		**1,573.3**	1,480.2	**1,074.4**
TOTAL FEED TO PLANT (after mining dilution and recovery)							
	Total ore	Mt	**876.8**		298.2	310.0	268.6
	Grade	%	**0.46**		0.51	0.46	0.39
	Insitu metal	**kt**	**4,001.7**		**1,523.2**	**1,435.8**	**1,042.7**
AVERAGE RECOVERIES							
		%	**90%**		**89%**	**90%**	**90%**
METAL RECOVERED							
	Rec. metal	kt	**3,592.6**		1,360.2	1,292.8	939.5
		Mlb	**7,920.3**		**2,998.8**	**2,850.2**	**2,071.3**
CONCENTRATE GRADE							
	26.5%	%	26.5%		26.5%	26.5%	26.5%
METAL PAYABLE							
	96.5%	**kt**	**3,468.6**		**1,313.3**	**1,248.2**	**907.1**
		$*10^6$ lbs	**7,646.9**		**2,895.3**	**2,751.8**	**1,999.8**
GROSS REVENUE							
	Cu $/t 3.00	**$M**	**$22,940.7**		**$8,685.9**	**$8,255.4**	**$5,999.4**
CAPITAL COSTS							
	Mining capital	$M	**$111.7**	$0.0	$111.7	$0.0	$0.0
	Processing capital	$M	**$41.8**	$0.0	$41.8	$0.0	$0.0
	Other capital	$M	**$3.7**	$0.0	$3.7	$0.0	$0.0
	Closure cost provisions	$M	**$78.2**		$16.7	$16.7	$44.8
	subtotal	**$M**	**$235.3**	**$0.0**	**$173.8**	**$16.7**	**$44.8**
SUSTAINING CAPITAL							
	Mining sustaining, initial	$M	**$185.6**	$0.0	**$185.6**	**$0.0**	**$0.0**
	Process sustaining, initial	$M	**$3.9**	$0.0	**$3.9**	**$0.0**	**$0.0**
	Other sustaining, initial	$M	**$62.3**	$0.0	**$62.3**	**$0.0**	**$0.0**
	Sustaining, ongoing	$M	**$237.1**		**$0.0**	**$171.6**	**$65.5**
	subtotal	**$M**	**$488.9**	**$0.0**	**$251.8**	**$171.6**	**$65.5**
OPERATING COSTS							
	Mining:						
	Ore $/t mined	$/t	**$1.66**		$1.63	$1.63	$1.76
		$M	**$1,409.4**		**$519.4**	**$510.0**	**$380.0**
	Waste $/t mined	$/t	**$1.86**		$1.83	$1.83	$1.95
		$M	**$3,044.6**		**$1,008.9**	**$1,123.7**	**$912.1**
		$/t	**$1.80**		$1.76	$1.76	$1.89
	subtotal mining	**$M**	**$4,454.0**		**$1,528.3**	**$1,633.7**	**$1,292.0**
	Processing (incl. G&A):						
		$/t	**$5.83**		$5.89	$5.80	$5.80
	subtotal processing	**$M**	**$5,111.3**		**$1,755.6**	**$1,798.0**	**$1,557.7**
	Stockpile reclaim allow:						
		$/t	**$1.00**		$0.99	$1.02	$1.00
	subtotal processing	**$M**	**$85.1**		**$4.7**	**$22.9**	**$57.5**
	subtotal operating costs	**$M**	**$9,650.5**		**$3,288.7**	**$3,454.6**	**$2,907.2**
METAL COSTS							
		$/lb	**$0.37**		$0.37	$0.37	$0.37
	TCRCs	**$M**	**$2,855.6**		**$1,081.2**	**$1,027.6**	**$746.8**
		$/lb	**$0.22**		$0.22	$0.22	$0.22
	royalty	**$M**	**$1,720.6**		**$651.4**	**$619.2**	**$450.0**
		$/lb	**$0.60**		$0.60	$0.60	$0.60
	subtotal metal costs	**$M**	**$4,576.2**		**$1,732.6**	**$1,646.8**	**$1,196.7**
UNDISCOUNTED CASHFLOW		**$M**	**$7,989.8**	**$0.0**	**$3,238.9**	**$2,965.8**	**$1,785.1**

Table 1-12 Enterprise Mineral Reserve cashflow model summary

		UNITS	TOTAL	PRESTRIP / Year 1		Year 2 to 6	Year 7 to 11
MINING (after mining dilution and recovery)							
	Total ore	Mbcm	**12.3**	0.01	0.4	5.6	6.3
		Mt	**34.7**	0.0	1.0	15.7	18.0
	Total waste (incl. min. saprolite)	Mbcm	**107.4**	4.8	6.7	71.4	24.5
		Mt	**286.7**	11.9	15.8	188.8	70.3
	Total mined	**Mbcm**	**119.7**	**4.8**	**7.2**	**77.0**	**30.8**
		Mt	**321.4**	**11.9**	**16.8**	**204.5**	**88.3**
	Strip ratio	bcm:bcm	**8.7**	**872.7**	**16.2**	**12.8**	**3.9**
TOTAL FEED TO PLANT (after mining dilution and recovery)							
	Non-primary ore	Mt	**5.8**	0.0	1.0	4.8	0.1
		% Ni	**0.79**	0.00	0.50	0.85	0.77
	Primary ore	Mt	**28.9**	0.0	0.0	10.9	17.9
		% Ni	**1.03**	0.00	0.00	1.29	0.87
	Total ore	**Mt**	**34.7**	**0.0**	**1.0**	**15.7**	**18.0**
	Grade	% Ni	**0.99**	**0.00**	**0.50**	**1.16**	**0.87**
	Insitu metal	**Ni kt**	**344.5**	**0.0**	**4.8**	**182.1**	**157.6**
METAL RECOVERD							
	Non-primary ore (60%)	Ni kt	**27.7**	0.0	2.9	24.4	0.4
	Primary ore (85%)	Ni kt	**253.6**	0.0	0.0	120.3	133.4
	Total recovered metal	**Ni kt**	**281.3**	**0.0**	**2.9**	**144.6**	**133.8**
GROSS REVENUE							
	At $7.50/lb Ni	**$M**	**$4,651.5**	**$0.0**	**$47.4**	**$2,391.7**	**$2,212.3**
CAPITAL COSTS							
	Mining prestrip	$M	**$54.0**	$21.7	$32.3	$0.0	$0.0
	Site capital works	$M	**$5.0**	$3.3	$1.8	$0.0	$0.0
	Mining capital	$M	**$4.0**	$2.4	$1.6	$0.0	$0.0
	Processing capital	$M	**$0.4**	$0.4	$0.0	$0.0	$0.0
	Other capital	$M	**$1.9**	$1.9	$0.0	$0.0	$0.0
	Closure cost provisions	$M	**$6.5**	$0.0	$0.0	$0.0	$6.5
	Total development capital	**$M**	**$71.9**	**$29.7**	**$35.7**	**$0.0**	**$6.5**
SUSTAINING CAPITAL COSTS							
	Mine sustaining	$M	**$0.0**	$0.0	$0.0	$0.0	$0.0
	Process sustaining	$M	**$10.4**	$0.0	$0.4	$6.6	$3.4
	Other sustaining	$M	**$0.0**	$0.0	$0.0	$0.0	$0.0
	Total sustaining capital	**$M**	**$10.4**	**$0.0**	**$0.4**	**$6.6**	**$3.4**
OPERATING COSTS							
	Mining ore	$M	**$78.5**	$0.0	$2.0	$32.4	$44.1
	Mining waste	$M	**$627.9**	$0.0	$0.0	$439.9	$188.0
	Processing variable costs	$M	**$215.4**	$0.0	$5.9	$97.5	$112.0
	Processing fixed costs	$M	**$78.1**	$0.0	$2.2	$35.3	$40.6
	Reclaim and overland ore haul	$M	**$52.0**	$0.0	$1.4	$23.6	$27.1
	Total operating costs	**$M**	**$1,051.9**	**$0.0**	**$11.5**	**$628.7**	**$411.7**
METAL COSTS (TCRCs AND ROYALTIES)							
	Nickel concentrate	kt	**1,875.4**	0.0	19.1	964.3	892.0
		% Ni	**15.0**	15.0	15.0	15.0	15.0
	Concentrate transport	$M	**$527.6**	$0.0	$5.4	$271.3	$250.9
	TCRCs (75% payability)	$M	**$1,058.2**	$0.0	$10.8	$544.1	$503.3
	Royalties	$M	**$232.6**	$0.0	$2.4	$119.6	$110.6
	Total metal costs	**$M**	**$1,818.4**	**$0.0**	**$18.5**	**$935.0**	**$864.9**
UNDISCOUNTED CASHFLOW		**$M**	**$1,698.9**	**-$29.7**	**-$18.7**	**$821.4**	**$925.9**

1.14 Conclusions and recommendations

1.14.1 Sentinel

1.14.1.1 Geology and Mineral Resource estimates

Sentinel mineralization occurs as a series of continuous horizons locally separated and influenced by sub-parallel structural detachment zones. Close spaced RC grade control drilling provides additional detail relevant to the scale of mining for these features. Diamond drilled core samples have good recovery and coverage across the extents of Sentinel mineralization. QAQC demonstrates sample results are adequate for use in this Mineral Resource estimate and supports the applied Mineral Resource classification. There is sufficient density data for applying robust mean values per geological domain. Geological domains relevant to grade and oxidation provide good detail for mining and processing. The Sentinel grade estimates validate well against the input data and are believed representative of the prevailing in-situ mineralization.

RC drilling coverage of the eastern areas of the Sentinel deposit outcrop should be considered in order to best define the horizons of weathering and oxidation. Continued mine to mill reconciliation will provide support for future updates.

1.14.1.2 Grade control and reconciliation

RC grade control drilling, in-pit mapping and 3D geology modelling of Sentinel mineralization volumes continue to focus on improving accuracy in position, volume and grades. Continued mine to mill reconciliation will focus on the improvement of key value areas.

1.14.1.3 Mineral processing and metallurgy

Metallurgical sampling and testing for Sentinel is performed on a continuous basis using internal and external laboratories. Additionally, samples of near surface non-primary ore from the Phase 2 pit cut-back have been tested and plant trials for this material are being undertaken to better understand the grade/recovery relationships for this material.

1.14.1.4 Mine planning and Mineral Reserve estimate

The Sentinel Mineral Reserve estimate reflects an achievable mining plan to support the proposed production expansion to 62 Mtpa, and production sequencing taking into account phased mining progression and reasonable increased material movement profiles (and hence equipment usage). The long term mine plan takes account of current locations and proposed future in-pit crusher relocations and trolley-assisted haulage routes.

Unit mining cost estimates have been updated to reflect detailed modelling of life of mine ore and waste haul routes, and under trolley-assist where applicable. Updated geotechnical pit slope design criteria have been followed in the optimisation and detailed pit design process and use has been made of new variable process recovery relationships which reflect operational performance. Phase pit designs and IPCC layouts have changed since the 2015 Technical Report (FQM, May 2015). It is recommended that the proposed future crusher locations and their timeframes for repositioning and installation continue to be reviewed and optimised.

1.14.2 Enterprise

1.14.2.1 Geology and Mineral Resource estimates

The Enterprise estimates have considered structural deformation with the use of dynamic anisotropy estimation methods. QAQC supports analysed samples as being representative of the in-situ mineralization. Density data is adequate within fresh material; however, additional data is required for saprolite and saprock in the Enterprise South West area. The data used for this update has allowed reasonable Measured and Indicated inventories to be classified, with opportunity for Inferred Mineral Resources to be upgraded to Indicated status with additional drilling. The estimated model validates well against the input data and is believed representative of the in-situ mineralization.

Recommendations include planning for a relevant RC grade control grid spacing for initial mining. Additional density data is required for saprolite and saprock in the Enterprise SW area.

1.14.2.2 Mineral processing and metallurgy

Subsequent to the initial sampling and test work for Enterprise nickel ores, further metallurgical sample drilling was carried out in 2018. Laboratory testing is currently in progress and findings from that work will allow refinements, as required, to be considered prior to commencement of operations. This work will continue to define the optimum flotation conditions for the various mineralogical suites present in the Enterprise plant feed.

1.14.2.3 Mine planning and Mineral Reserve estimate

The Enterprise Mineral Reserve reflects an achievable contracted mining plan, under the current consensus long term nickel metal price projection. Whilst there is no commitment, as yet, to a timeframe for the commencement of ore mining and processing, it is proposed that site development activities will commence in 2020 with the mining of pre-strip waste.

Beyond the initial phases of mining there is a waste stripping burden to access deeper ore in the later cutbacks. Metallurgical uncertainty is currently being addressed in a testwork programme that remains in progress through 2020. In conjunction with an improved metal price outlook, improved primary ore recovery projections arising from this testwork would enhance the value of the later pit cutbacks.

Analysis and modelling of recent hydrogeological drilling investigations are currently in progress. It is recommended that the pit designs and dewatering costs (as applicable) be reviewed and updated when that information becomes available.

ITEM 2 INTRODUCTION

2.1 Purpose of this report

This Technical Report on the Trident Project (the property) has been prepared by Qualified Persons (QPs) David Gray, Michael Lawlor and Andrew Briggs of First Quantum Minerals Pty Ltd (FQM, the issuer).

The purpose of this Technical Report is to document updated Mineral Resource and Mineral Reserve estimates for the property, to provide a commentary on the status of the Sentinel operations and a proposed expansion of these operations, and to provide an update commentary on the proposed Enterprise development project.

2.2 Terms of reference

This Technical Report has been written to comply with the reporting requirements of the Canadian National Instrument (NI) 43-101 guidelines: 'Standards of Disclosure for Mineral Properties' of April 2011 (the Instrument) and with the 'Australasian Code for Reporting of Mineral Resources and Ore Reserves' of December 2012 (the 2012 JORC Code) as produced by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, the Australian Institute of Geoscientists and the Minerals Council of Australia (JORC).

The effective date for the Mineral Resource and Mineral Reserve estimates is 31st December 2019.

2.3 Qualified Persons and authors

The Mineral Resource estimates were prepared by Mr Carmelo Gomez (FQM), a Qualified Person, under the direction and supervision of David Gray (QP). Mr Gray of FQM 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 of Michael Lawlor (QP), with the assistance of FQM staff. Mr Lawlor of FQM meets the requirements of a Qualified Person according to his Certificate of Qualified Person attached in Item 28. Mr Lawlor takes responsibility for those items not addressed specifically by the other QPs. Metallurgical testing, mineral processing/process recovery and infrastructure aspects of this Technical Report were addressed by 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.

Table 2-1 identifies which items of the Technical Report have been the responsibility of each QP.

2.4 Principal sources of information

Information used in compiling this Technical Report was derived from previous technical reports on the property, and from the reports and documents listed in the References item (Item 27).

2.5 Personal inspections

The Qualified Persons (QPs) have visited the site and carried out personal inspections, as follows:

- David Gray last visited the Trident Project in August 2019. Mr Gray inspected drill core and drilling sites, reviewed geological, data collection and sample preparation procedures, and carried out independent data verification.

- Michael Lawlor last visited the Trident Project in November 2019. Mr Lawlor visited accessible areas of the site, particularly the Sentinel mining operations, the Enterprise development site and the Enterprise processing circuits within the processing plant.
- Andrew Briggs last visited the Trident Project in November 2019. Mr Briggs visited accessible areas of the site, particularly the processing plant inclusive of the Enterprise circuits, the tailings storage facilities and the Enterprise development site.

Table 2-1 QP details

Name	Position	NI 43-101 Contribution
David Gray *BSc Hons (Geology), MAusIMM, FAIG*	Group Mine and Resource Geologist FQM (Australia) Pty Ltd	Author and Qualified Person Items 7 to 12, 14
Carmelo Gomez *BSc Hons (Geology), EurGeol*	Principal Resource Geologist FQM (Zambia)	Contributing author and Qualified Person Items 7 to 12, 14
Michael Lawlor *BEng Hons (Mining), MEngSc, FAusIMM*	Consultant Mining Engineer FQM (Australia) Pty Ltd	Author and Qualified Person Items 1 to 6, 15 and 16, 18 to 26
Andrew Briggs *BSc (Eng), ARSM, FSAIMM*	Group Consulting Metallurgist FQM (Australia) Pty Ltd	Author and Qualified Person Items 13, 17 and 21 (in respect of processing and G&A costs only)

2.6 Conventions and definitions

Reference in this Technical Report to dollars or $, relates to United States dollars. Copper metal production is reported in (metric) tonnes and (imperial) pounds, where the conversion factor is 1 tonne (t) = 2,204.62 pounds (lb). Gold production is reported in troy ounces (toz).

The conventional chemical abbreviation for copper of Cu is used throughout this report, whilst the abbreviation for nickel is Ni and for gold is Au. ASCu is used to denote Acid Soluble Copper and TCu is used to denote Total Copper.

Where not explained in the text of this report, specific terms and definitions are as listed in Table 2-2.

Table 2-2 Terms and definitions

Term	Definition	Term	Definition
bcm	bank cubic metres	µm, mm, cm, m, km	microns, millimetres, centimetres, metres, kilometres
csv	coma separated value	NPV	net present value
g, kg	grams, kilograms	oz	ounces
g/t, kg/t	grams per tonne, kilograms per tonne	P_{80}	80% passing
ha	hectares	pH	potential of hydrogen
IRR	internal rate of return	Q1, Q2, Q3, Q4	quarter 1 to 4
kWh/t	kilowatt hours per tonne	t, kt, Mt	tonnes, thousands of tonnes, millions of tonnes
lb	pounds	tpa	tonnes per annum
LOM	life of mine	MW, LG, MG, HG	mineralised waste, low grade, medium grade, high grade
Ma	mega annum (million years)	V, kV	volts, kilovolts
masl	metres above sea level	W, MW	watts, megawatts
mE, mN	coordinates: metres East, metres North	WGS	Western Geodetic System

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 DESCRIPTION, LOCATION AND TENURE

4.1 Project description

The Trident Project comprises the operating Sentinel mine, the Enterprise development project, a common processing plant and tailings storage facility, in addition to a number of common infrastructure elements. The Sentinel mining and processing operations have achieved an increasing level of recovered metal production since commissioning in 2015; the 2019 production level was about 220,000 tonnes of recovered copper.

Conventional open pit mining has been carried out at Sentinel, initially from a single phase of the ultimate 5.4 km long pit, and now expanding into an adjoining second phase. The mine has benefited from the successful implementation of ultra-class mining equipment, in-pit crushing and conveying, and trolley assisted truck haulage. The processing plant, also conventional, features one of the larger copper ore milling circuits in the world.

The Enterprise development project, once operational, has the ability to produce an annual average of 34,000 tonnes of recovered nickel metal through purpose built circuits within the processing plant.

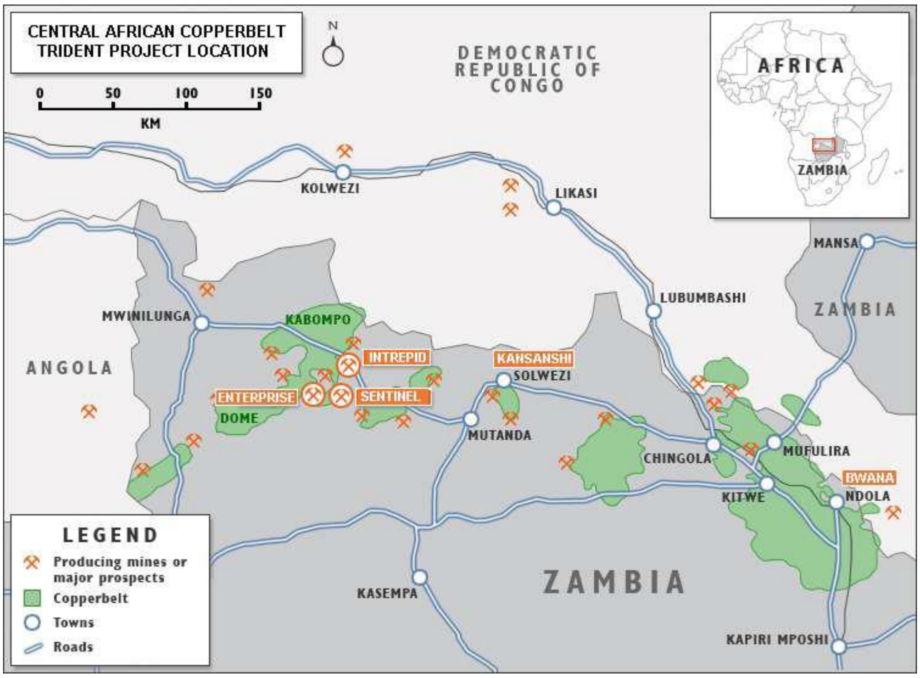
The remaining mine life at Sentinel is estimated as fifteen years (to 2034), whilst the expected mine life at Enterprise is of the order of eleven years from the commencement of processing.

Proposed expanded mining and processing at Sentinel, and the proposed resumption of development activities at Enterprise, are the subject of this Technical Report.

4.2 Project location

The Trident Project is located in the North-West Province of Zambia, approximately 150 km southwest of the town of Solwezi (Figure 4-1).

Figure 4-1 Location of Trident Project (Sentinel and Enterprise), North-West Province, Zambia



4.3 Tenure and property area

FQM acquired its 100% interest in the Trident Project from Kiwara PLC in January 2010. The site operating entity is Kalumbila Minerals Ltd (KML).

In April 2011, a Large-scale Mining Licence (15868-HQ-LML) was granted to KML under the provisions of the Mines and Minerals Development Act, 2008. This LML covers an area of 248.07 km^2 over the Sentinel deposit, processing plant and supporting infrastructure and is valid for 25 years, expiring in April 2036. At the same time, an LML (15869-HQ-LML) was granted covering an area of 243.69 km^2 over the Enterprise deposit. Additional licenses (15870-HQ-LML, 15871-HQ-LML and 15872-HQ-LML) cover exploration areas, sites for project infrastructure and buffer zones to prevent encroachment of local settlements.

These licences confer an exclusive right to mine copper, nickel, cobalt, gold, platinum group minerals, silver, iron and selenium. The coordinates of the Sentinel LML are listed in Table 4-1, the coordinates of the Enterprise LML are listed in Table 4-2, whilst a plan of their location is shown in Figure 4-1.

Table 4-1 Coordinates of Sentinel Mining Licence, 15868-HQ-LML (coordinates are in ARC1950 SUTM35)

Point Number	Northing	Easting
15868-001	8,644,914.07	310,189.43
15868-012	8,645,876.82	312,359.63
15868-013	8,655,094.81	312,300.95
15868-014	8,655,136.63	319,012.76
15868-015	8,651,449.57	319,035.34
15868-020	8,651,128.78	327,017.79
15868-042	8,646,508.26	325,050.19
15868-055	8,640,233.75	323,999.78
15868-062	8,639,492.04	323,279.00
15868-070	8,638,568.01	322,922.02
15868-071	8,637,279.81	323,292.38
15868-105	8,634,103.11	316,423.30
15868-110	8,634,458.11	314,245.71
15868-124	8,633,141.81	310,266.05

Surface rights are vested in the President of the Republic of Zambia and may be converted to State land under the provisions of the Lands Act, 1995. Upon conversion, land is held under leasehold title for a maximum period of 99 years.

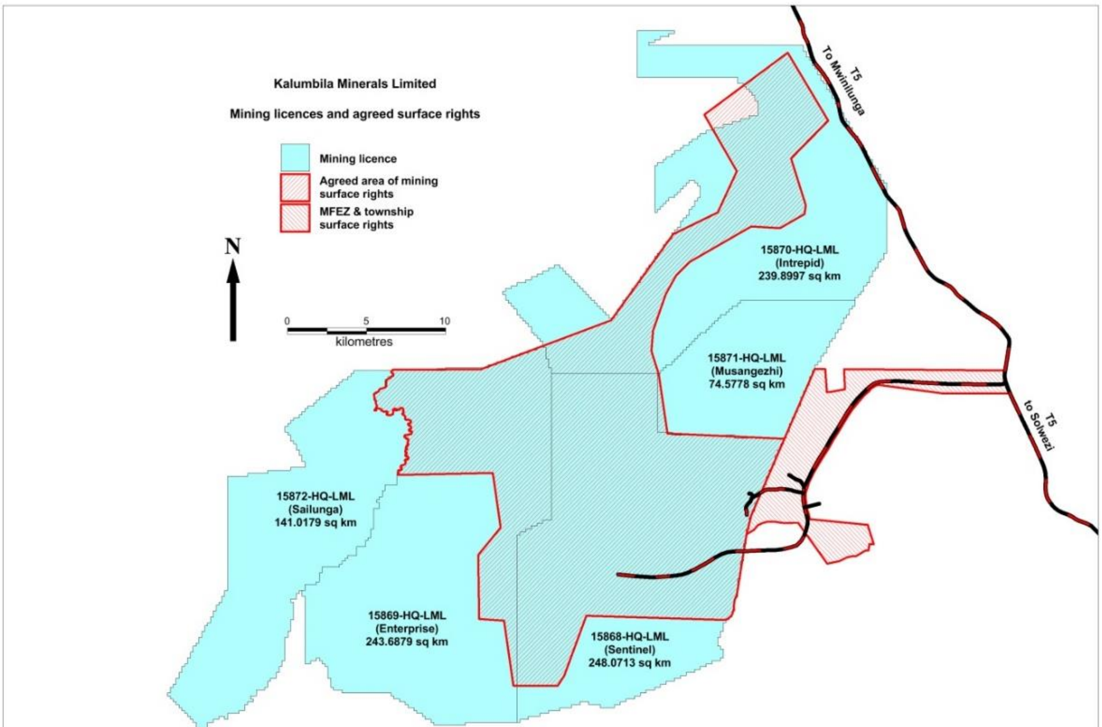
In October 2013, the Zambian government and KML agreed upon a surface rights area of 383.36 km^2 for conversion to State land for the mining operations and infrastructure at both Sentinel and Enterprise. This land lies almost entirely within the five LMLs as shown in Figure 4-2. An Invitation to Treat was received from the Commissioner of Lands in February 2018 and was followed by a revision of the consideration fee in June 2018, which fee is currently under consideration by KML.

In October 2013, the Zambian government and KML agreed upon an area of 50.64 km^2 for conversion to State land, for the establishment of a Multi Facility Economic Zone (MFEZ) containing the Kalumbila township development, industrial zone and airport. The MFEZ area lies entirely outside of the mining licences.

Table 4-2 Coordinates of Enterprise Mining Licence, 15869-HQ-LML (coordinates are in ARC1950 SUTM35)

Point Number	Northing	Easting
15869-002	8,655,463.61	312,298.60
15869-006	8,656,751.74	311,927.63
15869-007	8,656,748.28	311,383.40
15869-020	8,655,438.02	308,307.64
15869-023	8,655,213.02	302,141.00
15869-036	8,653,730.80	301,062.45
15869-037	8,653,361.99	301,064.93
15869-043	8,652,446.27	301,978.08
15869-044	8,652,448.70	302,340.83
15869-045	8,652,817.40	302,338.36
15869-046	8,652,821.04	302,882.56
15869-049	8,652,048.85	303,068.91
15869-061	8,649,872.44	303,083.67
15869-082	8,646,540.37	301,110.98
15869-083	8,645,249.75	301,119.71
15869-099	8,642,838.05	298,959.92
15869-100	8,642,830.56	297,871.89
15869-106	8,641,716.79	296,791.52
15869-111	8,636,738.67	296,826.12
15869-116	8,637,113.79	297,730.04
15869-123	8,635,826.91	298,282.88
15869-129	8,634,911.33	299,195.67
15869-130	8,634,932.41	302,277.62
15869-151	8,632,922.61	305,010.49
15869-154	8,633,141.77	310,266.05
15869-155	8,644,941.08	310,190.63
15869-166	8,645,876.82	312,359.63

Figure 4-2 Plan showing mining licences and surface rights areas



Since the 2013 agreement, the Zambian government has excluded the residential area and limited the MFEZ area to 97 ha designated for the Kalumbila Industrial Park. A revised Invitation to Treat from the Commissioner of Lands for the reduced area is awaited. Stand demarcation within the township and industrial area is advanced and, to date, eight Certificates of Title have been issued to third parties.

4.4 Royalties, rights, payments and agreements

In the 2015 Technical Report (FQM, May 2015), it was reported that the Zambian government had proposed (January 2015) an amendment to the mining royalty regime by increasing revenue based royalties from 6% to 20%. In April 2015, the government revised the royalty increase to 9%, effective from 1st July 2015.

However, effective from 1st June 2016, the government made another amendment, involving a reduction in the mining royalty rate from 9% to a sliding scale of 4% to 6% depending on the London Metal Exchange (LME) monthly average copper price.

Further changes announced in September 2018, were implemented by the government from January 1st 2019:

- the sliding scale mineral royalty rate on copper was increased by 1.5% to between 5.5% and 7.5% depending on the LME monthly average price
- an 8% royalty rate is applicable if the LME monthly average price is greater than $7,500 per tonne (ie, $3.40/lb) and less than $9,000 per tonne (ie, $4.08/lb)
- a 10% royalty rate is applicable if the LME monthly average price is greater than $9,000 per tonne

Effective 1st June 2016, a 10% export duty was suspended in relation to ores and concentrates for which there are no processing facilities in Zambia (eg, nickel concentrate). The royalty rate on nickel is currently 5% of the LME price. In January 2019, a 15% export levy was re-imposed on precious metals, including gold, and an import duty of 5% imposed on copper and cobalt concentrates.

The Trident Project is not subject to any other known third-party royalties, rights, payments, agreements or encumbrances.

4.5 Environmental liabilities

There are no known pre-existing environmental liabilities associated with the property.

The primary future environmental liabilities at Trident will arise at closure and are related to the dismantling and closure of the process plants and ancillary infrastructure, and the rehabilitation of the tailings dams, open pit mine(s) and waste rock dumps.

The closure plan is reviewed annually, and as at the 31st December 2019, the asset retirement obligation (ARO) for the Sentinel operations was estimated by consultants to be $78.2 M. The internally estimated ARO for Enterprise is $6.5 M.

No material environmental incident was reported at Trident up to the end of December 2019 and the Company has not been subject to any penalties arising because of water pollution or contamination of land beyond the boundaries of its operations.

To the Company's knowledge, KML is not considered by any applicable environmental regulatory authority to be an imminent threat to the environment.

4.6 Required permits

KML holds all necessary Zambian permits required to carry out its operations.

In addition to the permitted rights of tenure, and In terms of other permits required to enable work to be conducted on the property, KML has the following:

- Musangezhi Dam water abstraction rights:
- rate of 3,000 m^3/day
- approved in April 2019
- confirmation of validity period awaiting issue of certificate
- Chisola Dam water abstraction rights:
- rate of 98,000 m^3/day
- approved in April 2019
- confirmation of validity period awaiting issue of certificate
- Sentinel pit dewatering permit:
- rate of 2,100,000 m^3/day
- approved in April 2019
- confirmation of validity period awaiting issue of certificate

4.7 Factors and risks which may affect access or title

To the extent known, and other than the geographical and political risks that are described in the Company's 2019 Annual Information Form (AIF, FQM, March 2019), there are no significant technical factors and risks that may affect access, title to, or the rights or ability to perform work on the property.

ITEM 5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

Access into the Trident Project is via a road of approximately 30 km length, sealed and leading from the national highway linking Solwezi to Mwinilunga. An airstrip is located close to the Kalumbila town site and is suitable for daily commuter plane traffic.

5.2 Climate

The local climate is characterised by warm wet summers and cool dry winters, ie, there is a distinct dry (April to October) season and a wet (November to March) season. Rainfall typically occurs as heavy thunderstorms with each event producing between 10 and 40 mm of rainfall.

The mean annual rainfall is around 1,400 mm. Mean temperatures range from between 5°C and 25°C in June and 14°C and 30°C in October. The climate has a minimal disruptive influence on continuous mining and processing operations.

5.3 Physiography

The Sentinel deposit lies at an elevation of 1,200 mRL in a setting of low relief, except for a number of ridges in the north, northeast and at Kalumbila Hill (1,397 mRL). The watershed is defined by the Musangezhi River which passes over the top of the Sentinel deposit, and which varies in altitude between 1,234 mRL and 1,204 mRL within the mining footprint (SWS, 2014).

The Enterprise deposit lies at an elevation of 1,150 mRL, in a relatively flat to gently undulating setting. The watershed is defined by the Chisola River which varies in altitude from 1,180 mRL to 1,140 mRL.

The topography and surface drainages of the Trident Project area are shown in Figure 5-1 as being dominated by three river systems which flow in a south-westerly direction (Schlumberger Water Services (SWS), 2014):

- the Musangezhi River; a perennial river that flows across the Sentinel deposit and which is the most southern of the three water courses
- the Kasombo River; located approximately 6 km to the north of the Musangezhi River
- the Chisola River; located approximately 5 km further north of the Kasombo River, and which flows across the Enterprise deposit

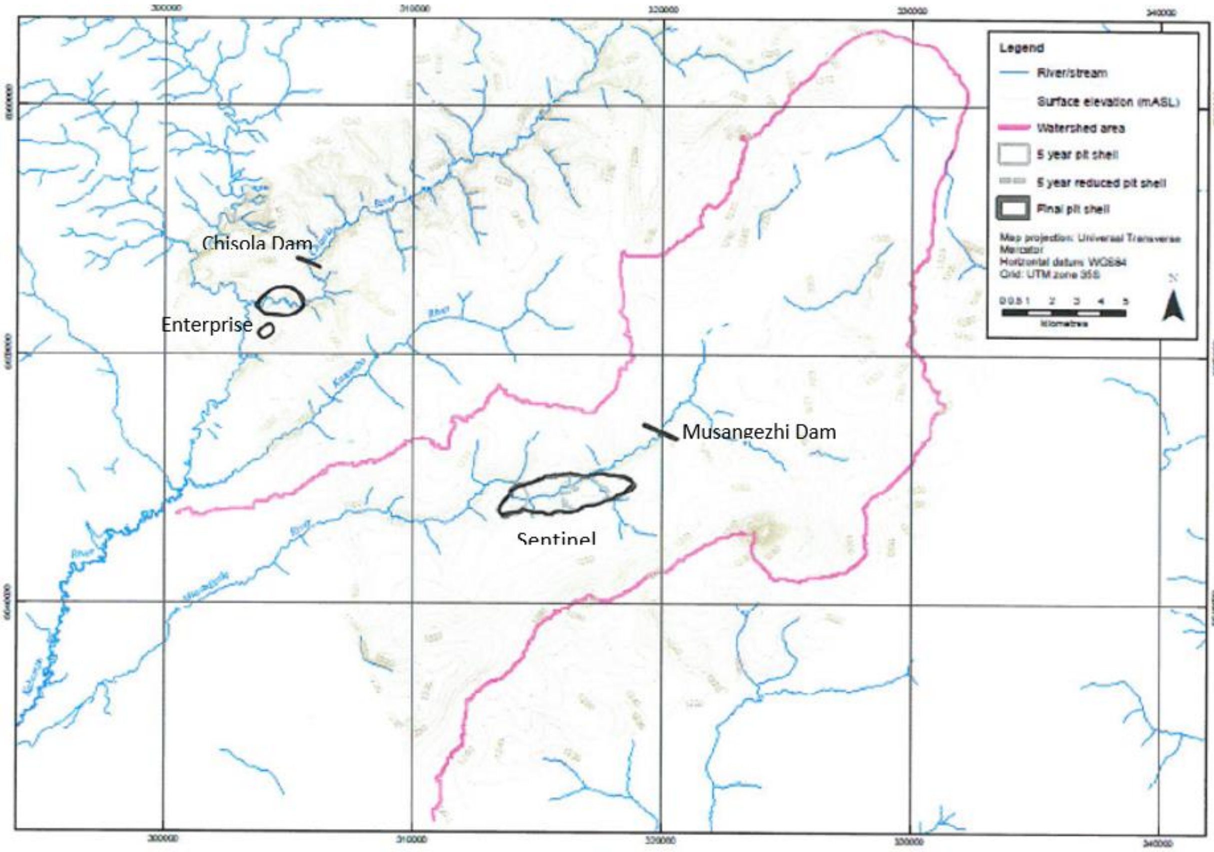
All three rivers are tributaries of the much larger Kabompo River which flows in a southerly direction to the west of the Project area.

5.4 Vegetation

The landscape consists of relatively flat to gently undulating miombo forest covered plains, interspersed with open grass covered phreatic dambos (ie, marshy wetland areas) which hold semi-permanent water.

The miombo forest is common throughout Zambia and especially in the North-West Province. Dambos and floodplains occur scattered throughout the site and often contain some riparian forest along the river banks. There is little vegetation within the Project site that is not already impacted upon by the local people and by agricultural activities.

Figure 5-1 Trident Project area, surface drainage (SWS, 2014)



5.5 Local Resources

Other than the new Kalumbila town site, the nearest major population centre is at Solwezi, 150 km away. The estimated population of Solwezi is approximately 200,000 people, most of whom live in rural areas surrounding that town. Personnel can be and are recruited from this local community. Whilst the majority of local people are unskilled and require training, skilled artisans and professional people can be and are recruited from throughout northern Zambia.

As at the end of December 2019, the Trident Project employed over 2,700 persons directly, and over 3,800 contractors. To date, over 1,300 houses have been constructed in the Kalumbila town site, with an additional 650 houses built for the resettlement of displaced families.

A number of light industrial and fabrication businesses exist in Solwezi and nearby Copperbelt towns, in addition to suppliers of contract mining services, mining/processing service providers, and suppliers of a wide range of consumable items. These suppliers service a number of mining companies locally and throughout northern Zambia. A supply chain is well established, with several road transport and air freight providers operating between Zambia and surrounding African countries.

5.6 Infrastructure

Prior to development, there was no infrastructure in the Trident Project area in the form of sealed roads, power supply, water supply or services.

The Company subsequently completed a number of infrastructure projects including the construction of a new town site (Kalumbila town) comprising personnel housing, roads, water supply and sewerage, electrification, schools and a medical clinic. A 30 km long sealed road was established into the Project site leading off the Solwezi to Mwinilunga national road. A bitumen surfaced airstrip was constructed close to the town site, replacing an older unsealed landing strip. A connection agreement with the national generating authority (ZESCO) enabled the construction of a 600 km long power line into the site.

5.7 Sufficiency of surface rights

The LML licence and surface rights boundaries cover a sufficient area to enable currently planned mining, waste dumping, and tailings storage activities to proceed.

ITEM 6 HISTORY

6.1 Sentinel

6.1.1 Prior ownership exploration and development work

The Trident Project area was originally prospected by Roan Selection Trust (RST) between 1959 and 1961, then by Anglo American Corporation Central Africa (AACCA) and Equinox in the 1980s and 1990s, to be followed by KML between 2007 and 2009. Over time, the exploration focus has varied from copper (RST) to nickel (AACCA) and back to copper (KML). On 29th January 2010, the Company acquired 100% of Kiwara PLC (Kiwara). Kiwara's main asset was a controlling interest in the Trident Prospecting Licence Area, covering the Sentinel copper deposit, the Enterprise nickel deposit and several other exploration targets.

RST drilled 31 wide spaced diamond holes over the Sentinel area and intersected widespread but relatively low grade copper mineralization. AACCA focussed on detailed drilling for localised nickel-copper mineralization around the Kalumbila Fault, whereas KML subsequently completed the first systematic drilling of the copper mineralization over an eight kilometre strike length. Following the acquisition of KML by the Company, geological data acquisition resumed at Sentinel in 2010 and has continued to date with mining, pit exposures and close spaced RC grade control drilling adding significant volumes of data and improved confidence in the Mineral Resource estimates.

6.1.2 Previous Mineral Resource estimates

Prior to ownership by the Company, there were no known Mineral Resource estimates for the Sentinel deposit. With completion of diamond drilling in 2011, FQM tasked CSA Global (UK) Ltd (CSA) to complete a Mineral Resource estimate for the Sentinel deposit. The estimate was the subject of the 2012 Technical Report and used the available geological data as at March 2012. The Mineral Resource statement, inclusive of Mineral Reserves (Table 6-1), included oxide mineralization, subsequently removed due to it being non-processable refractory copper. The total Measured and Indicated Resource (excluding oxide) was 998.2 Mt at an average grade of 0.51% Cu.

Table 6-1 May 2012 Mineral Resource statement for Sentinel deposit using a 0.2% Cu cut-off

Classification	Material	Tonnes (Mt)	TCu (%)	Cu metal (kt)
Measured	Oxide	4.1	0.67	27.5
Measured	Non-primary sulphide	20.7	0.51	105.6
Measured	Primary sulphide	489.2	0.55	2,690.6
	Measured sub total	**514.0**	**0.55**	**2,823.6**
Indicated	Oxide	25.0	0.53	132.5
Indicated	Non-primary sulphide	22.8	0.43	98.0
Indicated	Primary sulphide	465.5	0.47	2,187.9
	Indicated sub total	**513.3**	**0.47**	**2,418.4**
	Measured and Indicated total	**1,027.3**	**0.51**	**5,242.0**
Inferred	Non-primary sulphide	2.9	0.38	11.0
Inferred	Primary sulphide	162.3	0.42	681.7
	Inferred total	**165.6**	**0.42**	**695.5**

During the first half of 2015, FQM completed a Mineral Resource estimate (Table 6-2) prior to mining start-up. This estimate was the subject of the May 2015 Technical Report and included additional drilling data, improved 3D geology constraints and provided grade estimates into a smaller block size. These changes provided improved grade and tonnage estimates at the scale of mining and reflected the data available at the end of 2014. The total Measured and Indicated Mineral Resource was 1,027.7 million tonnes at a Cu grade of 0.53%.

Table 6-2 Sentinel Mineral Resource statement using a 0.2% copper cut-off grade and depleted for mining as at 31st May 2015

Classification	Material	Tonnes (Mt)	Density	TCu (%)	Cu metal (kt)
Measured	Non-primary sulphide	78.7	2.75	0.47	372.2
Measured	Primary sulphide	661.8	2.80	0.57	3,748.2
	Measured subtotal	**740.5**	**2.79**	**0.56**	**4,120.5**
Indicated	Non-primary sulphide	19.0	2.75	0.45	84.5
Indicated	Primary sulphide	268.2	2.80	0.46	1,239.9
	Indicated subtotal	**287.2**	**2.79**	**0.46**	**1,324.3**
	Measured and Indicated total	**1,027.7**	**2.79**	**0.53**	**5,444.8**
Inferred	Non-primary sulphide	6.9	2.78	0.29	19.7
Inferred	Primary sulphide	129.1	2.80	0.38	486.5
	Inferred subtotal	**135.9**	**2.80**	**0.37**	**506.3**

6.1.3 Previous Mineral Reserve estimates and statements

DumpSolver Pty Ltd (DumpSolver) produced the 2012 Technical Report Mineral Reserve estimate for Sentinel (DumpSolver, May 2012). The estimate was subsequently updated by FQM for the 2015 Technical Report (FQM, May 2015) and is listed by classification in Table 6-3. The reported Mineral Reserve was based on an economic cut-off grade which accounted for longer-term copper metal and gold price projections of $3.00/lb ($6,615/t) and $1,200/oz, respectively. The Mineral Reserve inventory reflected the optimisation and phased pit designs that were current at the time, and a corresponding mining and processing production schedule.

Table 6-3 Sentinel Mineral Reserve statement, depleted for mining as at 31st May 2015, and based on a $3.00/lb Cu price

Mineral Reserve statement as at end of May 2015 at $3.00/lb Cu				
Classification	**Material**	**Tonnes (Mt)**	**TCu (%)**	**Cu metal (kt)**
Proven	Non-primary sulphide	80.9	0.44	357.4
Proven	Primary sulphide	738.3	0.51	3,772.6
	Proven subtotal	**819.3**	**0.50**	**4,130.0**
Probable	Non-primary sulphide	17.4	0.45	78.5
Probable	Primary sulphide	148.6	0.52	776.6
	Probable subtotal	**166.0**	**0.52**	**855.0**
	Proven and Probable subtotal	**985.3**	**0.51**	**4,985.0**
Probable	Stockpiles	0.5	0.58	3.1
	Proven and Probable total	**985.8**	**0.51**	**4,988.1**

Table 6-3 includes the additional Mineral Reserve held in surface stockpiles, as at May 2015. The Proven Mineral Reserve was marginally in excess of Measured Mineral Resource for reasons of differing cut-off grade criteria.

The depleted Mineral Reserve for the open pit, as at the end of 2018, was reported in the Company's Annual Information Form (AIF, FQM, March 2019) as a total of 811.3 Mt at an average grade of 0.50% TCu (Table 6-4). This statement was consistent for the Mineral Resource model basis and the metal prices relevant to the May 2015 statement.

Table 6-4 Sentinel Mineral Reserve statement, depleted for mining as at 31st December 2018, and based on a $3.00/lb Cu price

Mineral Reserve statement as at end of December 2018 at $3.00/lb Cu				
Classification	**Material**	**Tonnes (Mt)**	**TCu (%)**	**Cu metal (kt)**
Proven	Non-primary sulphide	50.7	0.44	225.0
Proven	Primary sulphide	618.5	0.50	3,099.9
	Proven subtotal	**669.2**	**0.50**	**3,324.9**
Probable	Non-primary sulphide	10.9	0.44	47.7
Probable	Primary sulphide	131.2	0.51	664.0
	Probable subtotal	**142.1**	**0.50**	**711.7**
	Proven and Probable subtotal	**811.3**	**0.50**	**4,036.6**
Proven	Stockpiles	0.2	0.54	0.8
Probable	Stockpiles	29.4	0.27	78.6
	Proven and Probable total	**840.8**	**0.49**	**4,116.0**

6.2 Enterprise

6.2.1 Prior ownership exploration and development work

AACCA first identified the presence of Ni, Cu and Co mineralization in the Kawako area (now known as Enterprise), in 1996. In 2008, the prospect was acquired by Kiwara PLC (Kiwara) and in 2011, FQM obtained the leases.

Exploration was completed by Kiwara in May 2008, including grab sampling of drill spoil from the surrounding area. Encouraging assays from this work led to a follow up geochemical soil sampling programme.

Follow up drilling was guided by the soil geochemistry survey data. This drilling consisted of two vertical diamond drillholes, approximately 850 m apart. During 2009, Kiwara performed a ground magnetic survey and a fixed loop ground electromagnetic survey of the Enterprise area, as well as additional diamond drilling.

6.2.2 Previous Mineral Resource estimates

Prior to ownership by the Company, there were no known Mineral Resource estimates for the Enterprise deposit. On completion of 2012 diamond drilling, FQM tasked CSA Global (UK) Ltd (CSA) to complete a Mineral Resource estimate for the Enterprise deposit. The estimate was the subject of the 2012 Technical Report and used the available geological data as at November 2012. The Mineral

Resource statement is presented in Table 6-5 and is reported at a 0.15% Ni cut-off and was inclusive of Mineral Reserves.

Table 6-5 2012 Mineral Resource statement for the Enterprise deposit, using a 0.15% Ni cut-off grade

Classification	Area	Tonnes (Mt)	Ni (%)	Ni metal (kt)
Measured	Main	2.7	1.51	41.0
Indicated	Main	34.3	1.08	372.0
	SW	3.1	0.60	18.0
Measured and Indicated total		**40.1**	**1.07**	**431.0**
Inferred	Main	2.5	0.92	23.0
	SW	4.6	0.58	27.0
Inferred total		**7.1**	**0.70**	**50.0**

During the first half of 2015, FQM completed an updated Mineral Resource estimate (Table 6-6) for the Enterprise deposit which included an additional 81 diamond drilled holes. This estimate was the subject of the 2015 Technical Report (FQM, May 2015) and included improved 3D geology modelling with post processed grade estimates into a SMU scale block size. These changes provided a better reflection of the grade and tonnages available at the scale of mining. The total Measured and Indicated Mineral Resource was 39.8 million tonnes at a Ni grade of 1.01%. Mineral Resources were inclusive of Mineral Reserves.

Table 6-6 Enterprise Mineral Resource statement as at 31st May 2015 using a 0.15% Ni cut-off grade

Classification	Material	Tonnes (Mt)	Density	Ni (%)	Ni metal (kt)
Measured	Non-primary sulphide				
Measured	Primary sulphide	5.1	2.90	1.56	79.9
Measured subtotal		**5.1**	**2.90**	**1.56**	**79.9**
Indicated	Non-primary sulphide	4.3	2.35	0.69	29.7
Indicated	Primary sulphide	30.3	2.90	0.96	290.7
Indicated subtotal		**34.6**	**2.83**	**0.92**	**320.4**
Measured and Indicated total		**39.8**	**2.84**	**1.01**	**400.4**
Inferred	Non-primary sulphide	5.3	2.41	0.47	25.1
Inferred	Primary sulphide	15.6	2.90	0.69	107.2
Inferred subtotal		**20.9**	**2.78**	**0.63**	**132.3**

6.2.3 Previous Mineral Reserve estimates and statements

DumpSolver Pty Ltd (DumpSolver) produced the 2012 Technical Report Mineral Reserve estimate for Enterprise (DumpSolver, December 2012). The estimate was subsequently updated by FQM for the 2015 Technical Report (FQM, May 2015) and is listed by classification in Table 6-7. The reported Mineral Reserve was based on an economic cut-off grade which accounted for a longer-term nickel metal projection of $7.50/lb ($16,535/t). The Mineral Reserve inventory reflected the optimisation and phased pit designs that were current at the time, and a corresponding mining and processing production schedule.

Table 6-7 Enterprise Mineral Reserve statement, as at 31st May 2015, and based on a $7.50/lb Ni price

Mineral Reserve statement as at end of May 2015 at $7.50/lb Ni				
Classification	**Material**	**Tonnes (Mt)**	**Ni (%)**	**Ni metal (kt)**
Proved	Non-primary sulphide			
Proved	Primary sulphide	5.3	1.43	76.2
	Proved subtotal	**5.3**	**1.43**	**76.2**
Probable	Non-primary sulphide	2.9	0.70	20.4
Probable	Primary sulphide	27.1	0.91	247.7
	Probable subtotal	**30.1**	**0.89**	**268.1**
	Proved and Probable total	**35.4**	**0.97**	**344.4**

The Mineral Reserve statement in Table 6-7 was reproduced in the Company's AIF for 2018 (FQM, March 2019).

6.3 Production from the property

Copper production commenced at Trident in 2015 from a copper processing circuit with a design capacity of 55 Mtpa. There has been no nickel production from the 4 Mtpa capacity nickel processing circuit.

From 2015 to December 2019, Trident copper production amounts to approximately 807 ktonnes of recovered copper (Table 6-8). Following the declaration of commercial production in November 2016, approximately 635 ktonnes of copper metal was recovered from 2017 to 2019.

Table 6-8 Copper production from the Trident Project, to date

Year	Ore (Mt)	Cu grade (%Cu)	Metal insitu (kt)	Metal rec'd (kt)	Recovery (%)
2015	14.3	0.56	79.9	33.0	41.3%
2016	36.4	0.57	207.3	139.6	67.3%
2017	42.1	0.53	221.7	190.9	86.1%
2018	48.7	0.50	245.0	223.7	91.3%
2019	49.4	0.50	245.9	220.1	89.5%
TOTAL	**190.9**	**0.52**	**999.9**	**807.3**	**80.7%**

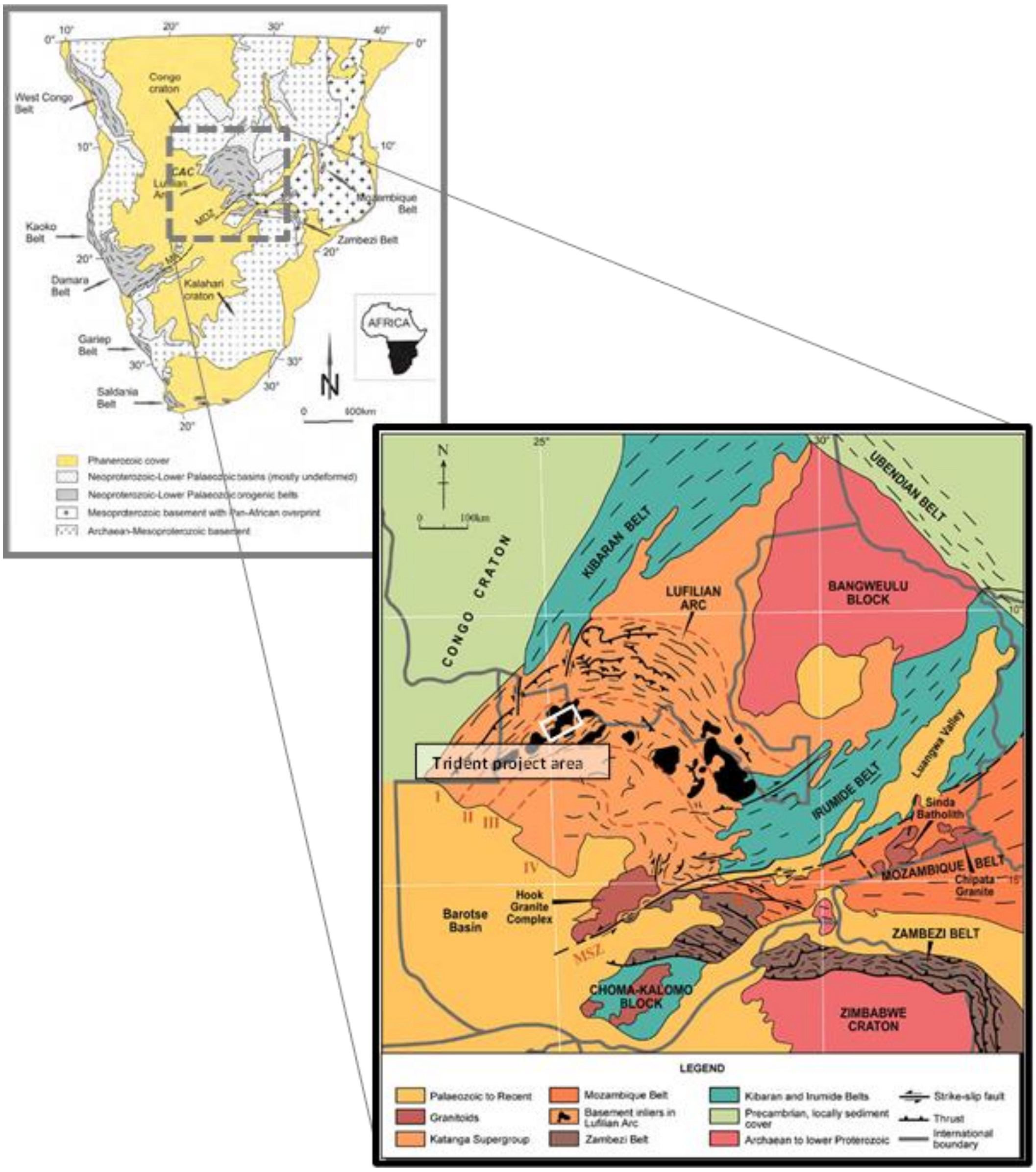
Copper concentrate was first smelted at Kansanshi in March 2015, and after a short ramp-up period, commercial production from the smelter (KCS) was achieved in July of the same year. The smelter processed 1.4 Mt of concentrate during 2018, supplied from the Kansanshi and Sentinel processing plants. A total of 347,000 tonnes of copper and 1.25 million tonnes of acid were produced from the smelter in 2018.

ITEM 7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Trident Project Area, including the Sentinel and Enterprise deposits, is located on the western end of the Lufilian Arc. The Lufilian Arc is a curvilinear structural belt formed during the Lufilian Orogeny (c.590-465Ma), and extends from northern Zambia, across the Katanga Province of the Democratic Republic of Congo, and into northeast Angola (Figure 7-1).

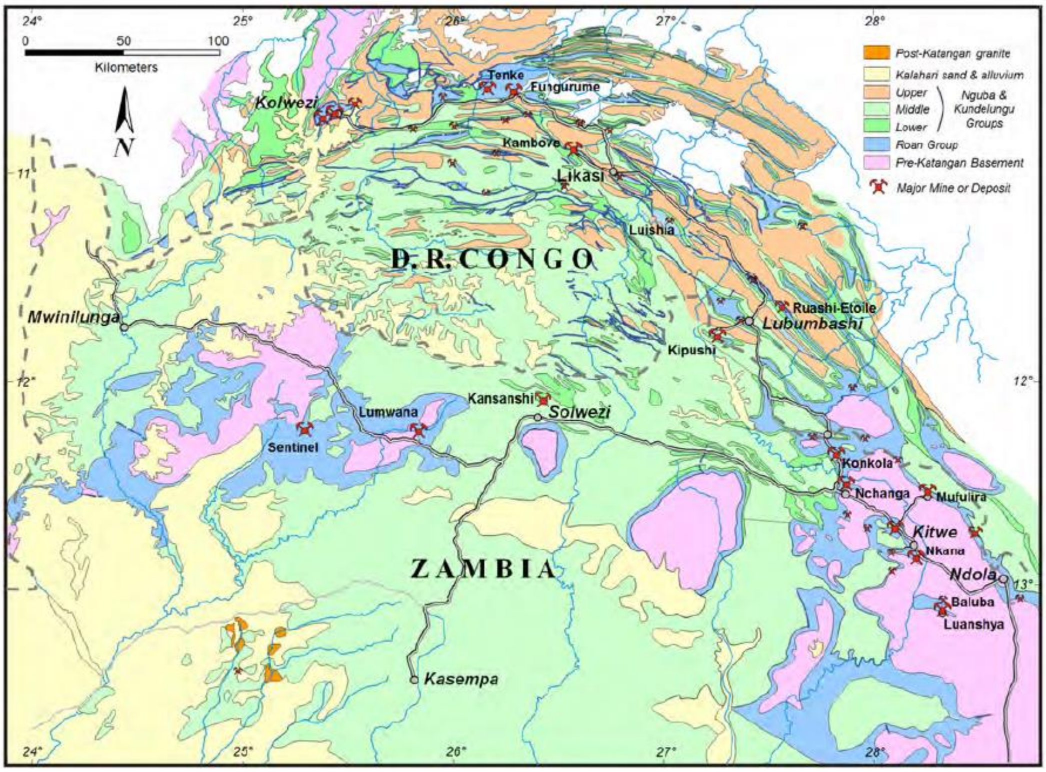
Figure 7-1 Tectonic setting of the Lufilian Arc and Trident Project area



Trident lies on the margins of the Mesoproterozoic Kapombo Dome, one of several antiformal basement inliers (Figure 7-2) in northwest Zambia, within a thick succession of Neoproterozoic sedimentary rocks of the Katanga Supergroup (c.880-550Ma).

Both basement and cover were deformed and metamorphosed during the Lufilian Orogeny. In Zambia, the metamorphic grade attained upper amphibolite-facies but has been locally retrogressed to greenschist facies. Peak metamorphism is recorded by the local development of talc-kyanite white schist within this 'Domes' region of northwest Zambia.

Figure 7-2 Simplified geology of Zambian and Congolese Copperbelt, showing the general distribution of Katanga sediments (and location of Sentinel and other major mines)



The basement rocks include fine to coarse grained biotite gneiss with local amphibolites, ultramafic and granite gneiss bodies, as well as more extensive schist units with varying proportions of phlogopite, muscovite and kyanite. Basement rocks within the Zambian Copperbelt have been interpreted to have undergone potassic alteration during the Katangan mineralizing event.

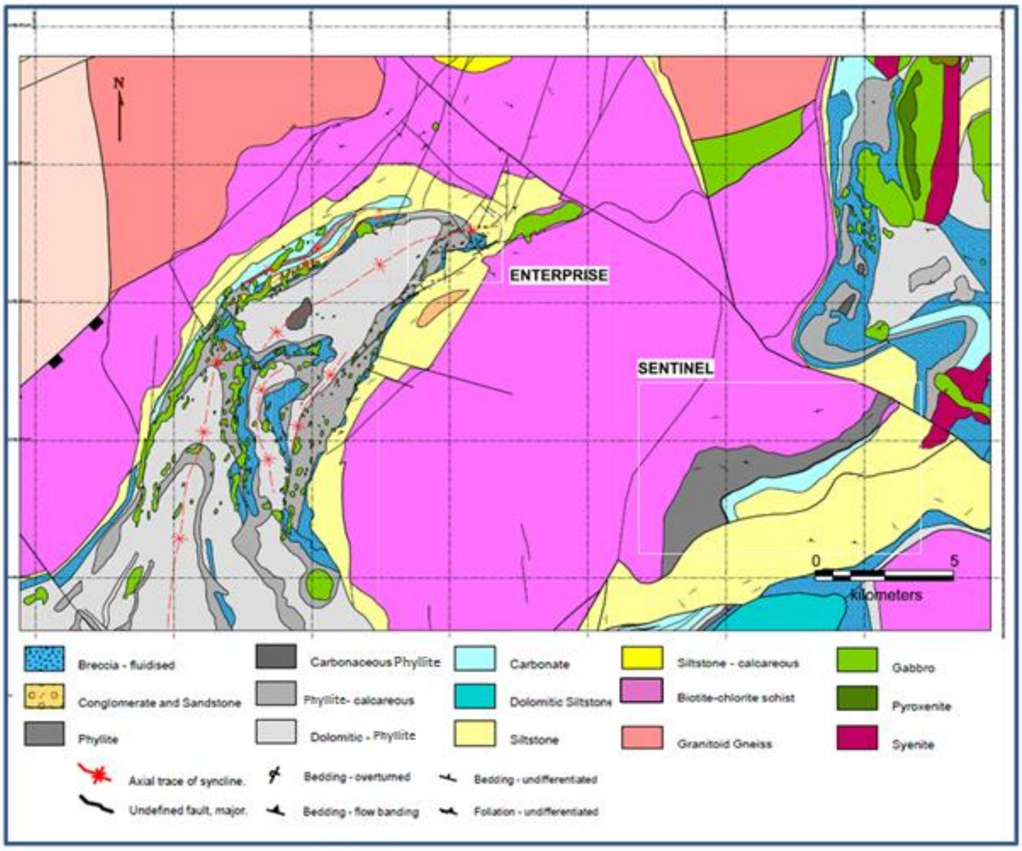
The Katangan metasedimentary rocks surrounding Kabompo Dome include a basal sequence of sandstones, siltstones, and conglomerates, grading up into a mixed siliciclastic-carbonate-evaporite sequence. These sediments have been extensively altered and modified to quartz-feldspar-phlogopite schists, dolomites, and talc-kyanite schists. The sequence is capped by a shaley diamictite (Grand Conglomerate), which is a basin-wide stratigraphic marker.

Folds in the Kabompo Dome region have two main geometries. Temporally separate, but similarly oriented upright folds occur in the basement and overlying Katangan rocks, and recumbent folds are restricted to the basal Roan siliciclastics and phlogopite-quartz horizon.

7.2 Sentinel local and property geology

Sentinel is located to the southeast of the Trident Project area (Figure 7-3). The deposit is hosted within structurally thickened, northwest dipping carbonaceous meta-pelitic rocks known as 'Kalumbila phyllite'.

Figure 7-3 Surface lithology and major structures for the Trident Project area

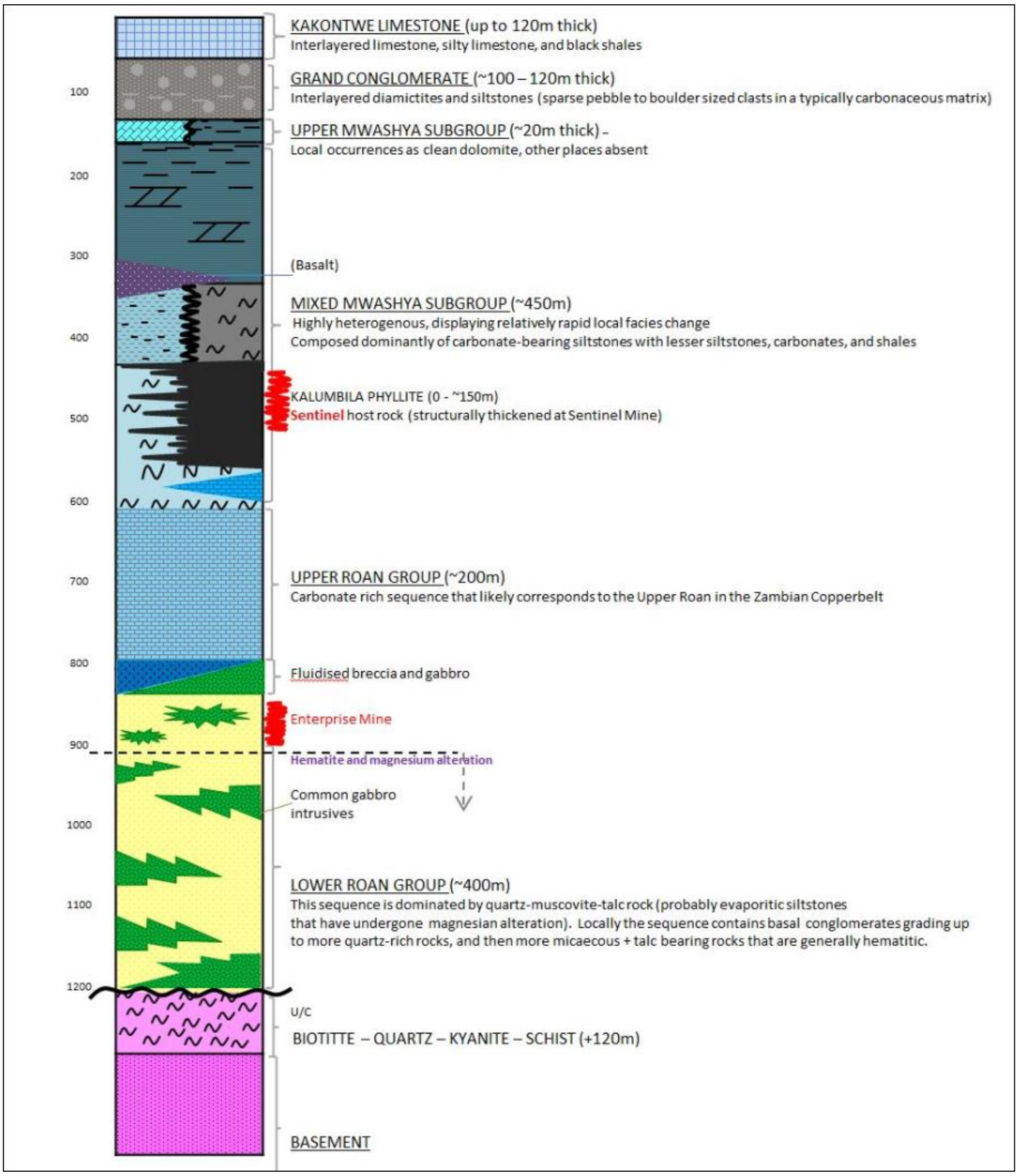


The geometry of the host phyllite is defined by recumbent, typically asymmetric, folds cut by late detachments. The phyllite is terminated to the northeast by the northwest-southeast trending Kalumbila Fault and to the south by a sub east-west cross-cutting structure, thought to be a detachment surface.

Compositionally, the Kalumbila phyllite is very fine grained, with quartz, muscovite, biotite, and iron-sulphides being the dominant minerals. Total organic carbon test-work has confirmed graphite content between 1-5%.

Recent observations and logging of drill core across the Trident Project area suggests a complete stratigraphic sequence may be present locally, from the base of the Katangan up through the lower Nguba Group (Figure 7-4).

Figure 7-4 Generalised stratigraphic section from Trident Project area



The hangingwall and footwall rocks are both quartz-feldspar-biotite schists, displaying strong petrographic similarities. Diamond drilled core from the area suggests the hangingwall schist north of Sentinel represents the Mwashya subgroup into Grand Conglomerate. The thin footwall schist and silt sandstone packages to the south were interpreted to represent the Upper Roan Group.

Overburden and regolith depths across the deposit are typically 0 to 5 m, with in-situ laterite and saprolitic layers underlying both woodlands and tall grassland.

7.2.1 Structural geology

The geometry of the Kalumbila phyllite at Sentinel has been recently interpreted (2018) as a synformal anticline where the different phyllite facies have been folded at a local scale, characterised by recumbent, typically asymmetric, folds with several detachments cutting lithology at slightly steeper angles than the fold limbs.

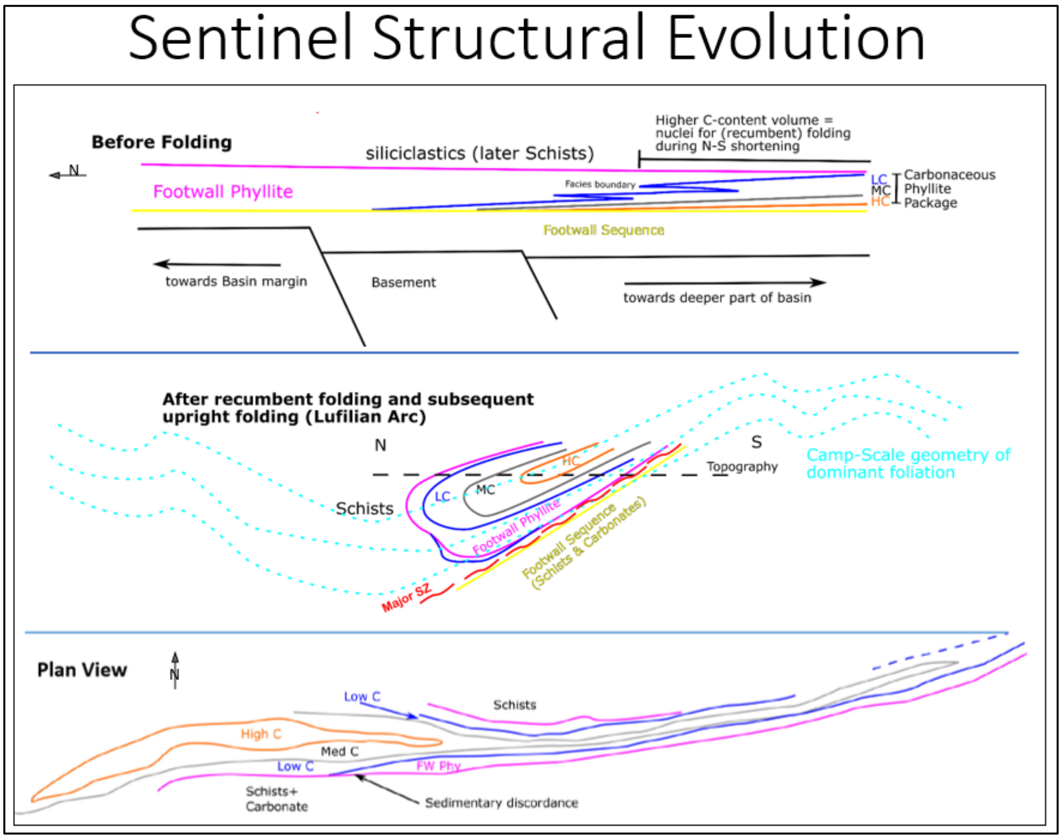
The S-shaped surface expression of the phyllite is indicative of non-cylindrical folding, likely resulting from differences in speeds of movement along the fold axis and across rheological changes during progressive deformation within the shear environment.

The dominant foliation dips towards north-northwest, at an average of 20-30 degrees, typically parallel to the fold axial plane of the pervasive folds. The recumbent folding, detachments, and dominant foliation are likely expressions of top-north directed shearing. The overall recumbent folding and subsequent late upright folding resulted in significant local thickening of the original phyllite stratigraphy at Sentinel (Figure 7-5 and Figure 7-6).

The updated structural and geological model is supported by structural logging of diamond cores, pit mapping data collected from exposed walls during the mining stages, and geochemical data studies from 12,435 core samples and 41,172 RC samples analysed for multi-element assays (ME-MS61 ALS method, 48 elements by four acid digestion and ICP-MS finish).

Model Earth consultants completed a 3D geology model using Leapfrog software after several (annual) site visits and in collaboration with the Sentinel mine geology team.

Figure 7-5 Schematic model outlining the structural evolution of the Sentinel deposit



The detachment horizons and dominant foliation have served as fluid conduits, with trace element geochemistry suggesting fluid flow direction is up-dip of dominant foliation.

The distribution of copper is closely related to the structural framework, with the main volume of chalcopyrite-bearing veinlets occurring parallel to bedding or to the fold axial planar foliation. Modelling the distribution of copper was therefore guided by these two structural trends, while also respecting their east-west trending intersection lineation as a main control on the generally east-west striking ore-body.

7.2.2 Alteration

Coarse centimetre scale, often euhedral, blebs of kyanite alteration are common throughout the phyllite host rock. Kyanite is generally associated with copper mineralization, though kyanite intensity does not correlate to copper grade.

Biotite, phlogopite, and sericite alteration is common, and typically concentrated along bedding and foliation planes. Metre-scale lenses of intense and pervasive sericite alteration occur sporadically along the contact between the phyllite and hangingwall schist. These intensely altered lenses are devoid of mineralization.

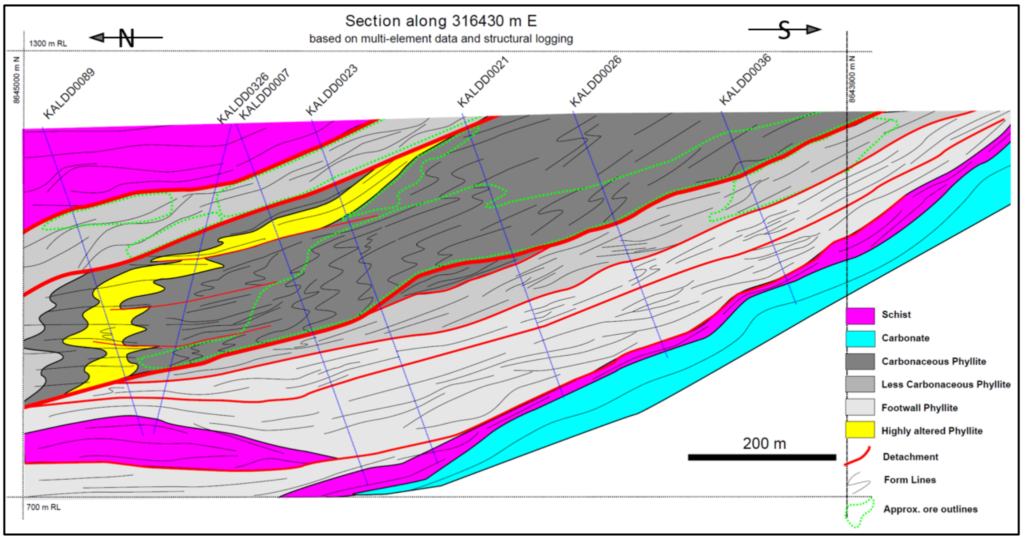
Round, mm to cm scale, cordierite poikiloblasts are scattered within the phyllite matrix. Carbonate alteration is common in the footwall phyllite, proximal to the footwall schist contact, and to a minor extent within the phyllite matrix.

7.2.3 Mineralization

Copper mineralization at Sentinel is limited to the strongly deformed phyllite unit, with rare low-grade mineralization extending only 1-2 metres into the hanging and footwall. The orebody strikes approximately east-west for 11 km and dips 20 to 30 degrees north, generally parallel to the dominant foliation (Figure 7-6).

The dominant copper bearing mineral is chalcopyrite which occurs within bedding/foliation parallel quartz-kyanite-carbonate mm-scale veinlets. Within folded zones, veinlets tend to be thicker (mm scale), blebby, and more irregular, and often contain a relatively higher proportion of chalcopyrite. Late sulphide-bearing cross-cutting veinlets and disseminated or blebby chalcopyrite are less common.

Figure 7-6 A cross section, looking east, of the Sentinel deposit showing detachment locations and folds as derived from logging with support from geochemical multi-element data



The oxidised horizon, up to approximately 70 m deep, contains non-primary sulphide Cu-minerals, predominantly chalcocite, and tarnished chalcopyrite. The top 5-15 metres from surface is mostly leached of copper, or contains mixed refractory copper and trace oxide minerals (Figure 7-6).

Nickel-cobalt mineralization exists in the form of cobalt-pentlandite, with trace amounts of vaesite. Apart from rare sporadic metre-scales lenses (likely related to structure) the Ni-Co mineralization occurs as a discrete horizon within the lowermost poorly copper mineralized 'footwall' phyllites. Ni-Co mineralization is best developed in the northeast extent of the deposit, proximal to the Kalumbila Fault.

7.3 Enterprise local and property geology

Enterprise is located in the northwest of the Trident Project area (Figure 7-3), and is hosted within a sequence of sedimentary units which form part of the Katangan System. The Katangan System is locally sub-divided into the formations: Wushingwi, Wamikumbi (which hosts the Enterprise mineralization), and Luigishi. The Wamikumbi Formation is described as being dominated by biotite and muscovite schists with subordinate amounts of marble, quartzite, graphitic phyllite and basic metavolcanics.

The lithologies (Figure 7-4) below the upper carbonate horizon have undergone significant modification by recrystallization during intense metasomatism. Sedimentary textures are largely destroyed. Widespread, intense talc and carbonate alteration suggests that the alteration fluids were composed of saline brines and is supported by the presence of hydrothermal vein and vein selvedge kyanite.

Enterprise is located on the southern limb of a gentle kilometre-scale syncline (Figure 7-3). It is hosted in Lower Roan rocks, and is along strike from a large gabbro that intrudes along a north-northeast structure. A soil sampling programme identified a single soil sample that yielded 1,250 ppm Ni and a linear Ni anomaly which corroborated the previous soil geochemistry conducted by Kiwara. Elevated Ni in soil in the same area was the reason for historic drilling by Anglo American, and followed up with further drilling by Kiwara prior to FQM.

The host sequence to nickel is strongly altered, and in many places original rock composition is unidentifiable. Multi-element analysis of historical drill samples in 2018 has allowed for more robust lithological and structural modelling. The dominant hosts to mineralization are a black shale, remnants of which remain locally recognisable, and the underlying siltstone package.

A strongly foliated quartz-phlogopite/biotite rock is located below the host sequence. This rock represents a major décollement between basement and Katangan rocks, and is essentially a transition zone that could be derived from either basement or Katangan.

7.3.1 Structural geology

The geometry of the shale and siltstone host units is characterised by early recumbent folding and structural thickening of the black shale unit. Shear deformation has been accommodated to different degrees within the host units, owing to rheological contrast between the softer shales and comparatively harder siltstones (and also underlying quartz-rich silts and overlying carbonates), allowing larger folds to be formed within the shale. Fold axes of recumbent folds trend approximately west-northwest to west-southwest with usually gentle plunges. The recumbent folding event is likely an expression of progressive top-north shearing related to a pervasive low-angle foliation.

A later upright folding event (striking west-southwest to east-northeast) refolded folds established during recumbent folding. Rheology changes from the siltstone units around the shale, resulted from intense talc alteration and allowed pronounced horizontal shortening during the later refolding event. This has created the overall lobate/cuspate shape that now characterises Enterprise Main deposit. Enterprise southwest deposit is less folded and likely further from main fluid conduits. Late brittle fault structures (possibly transfer faults) have also been observed, some of which cut and offset mineralization.

The structural and geological models are supported by structural logging of oriented diamond cores, pit mapping from pit exposures and geochemical studies from 9,281 two metre core composite samples analysed for multi-element assays (ME-MS61 ALS method, 48 elements by four acid digestion with ICP-MS finish).

The resulting models were created in Leapfrog software by Model Earth consultants together with regular site visits and collaboration with the Enterprise mine geology team.

7.3.2 Alteration

Regional alteration is not significantly different from alteration proximal to sulphide mineralization. The siliciclastic part of the host sequence is modified to quartz-talc-kyanite-magnesite. Black shales within the siliciclastic package are quartz-kyanite ± talc altered and contain sulphide mineralised veins. Some holes preserve the transition from black shale to a quartz-kyanite rock, proximal to quartz-kyanite-sulphide veins. Black shale in many places is difficult to recognise, as it has been locally transformed to a quartz-kyanite rock in mineralized zones. Layered specular hematite–talc rock is characteristic of the host sequence, and likely to represent an early stage of the alteration paragenesis. Hematite has been removed/destroyed where sulphide mineralization occurs.

7.3.3 Mineralization

Nickel sulphide minerals include vaesite (NiS_2), pentlandite $(Fe,Ni)_9S_8$, millerite (NiS), nickeliferous pyrite ($[Fe,Ni]S_2$), bravoite ($[Ni,Fe]S_2$) and carrollite ($Cu[Co,Ni]_2S_4$). Sulphide mineralization occurs within, or as an alteration halo to quartz-kyanite ± talc veins and vein breccias. Sulphides are concentrated within altered black shales and to lesser amounts in proximal siliciclastic rocks. Hydrothermal mineralization appears related to a series of deep seated structures having an extensional regime. The dominant control on the location of mineralization is a combination of structure, rheology and chemistry. Black shales host most mineralization and are a favourable rheology for brittle deformation, as well as being a reductant for oxidised hydrothermal fluids. Other siliciclastic units can be mineralized, but typically at lower concentrations than the black shale. Minor copper mineralization underlies nickel mineralization located 350m below surface. Copper grades of up to 2.80% have been intersected by drilling with intersepts of 5 m having 1.27% Cu (ENTDD0120) and 14 m having 1.72% Cu (ENTDD0019) as examples. Copper minerals include chalcopyrite, bornite and chalcocite.

ITEM 8 DEPOSIT TYPE

8.1 Sentinel

8.1.1 Mineral deposit type

The Sentinel deposit is a sediment-hosted stratiform copper deposit. Mineralization is predominantly primary sulphide copper occurring as foliation parallel sheet-like horizons which dip at 20-30^{o} north. The carbonaceous phyllite host is structurally deformed and characterised by recumbent, typically asymmetric folds with several later crosscutting detachments.

8.1.2 Guiding principles for exploration and modelling

The extent and geometry of the phyllite host is well defined by drilling, airborne EM and magnetic geophysical surveys, and soil/core geochemical sampling. Drillhole alignments were guided by the orientation of mineralized horizons, to ensure a high angle of intersection to mineralization.

Detailed drill core logging and supporting multi-element data allowed the identification and modelling of deposit-scale recumbent folds and detachments. The distribution of copper is related to this structural framework with most chalcopyrite veinlets occurring parallel to bedding and the axial planar foliation of folds. Detachments have served as fluid conduits. Mineralization is often truncated along the northern, down-dip extents of fold-noses which typically contain higher grade mineralization.

Modelling the distribution of copper was based on following these structural trends, while respecting their east-west intersection lineation as a control on the overall strike of the deposit.

8.2 Enterprise

8.2.1 Mineral deposit type

Enterprise is a hydrothermal nickel deposit with mineralization hosted in a sequence of shale and siltstone units. These units have been preferentially mineralized due to rheological and geochemical interactions with mineralizing fluids. Enterprise mineralization has an unusual lack of spatial control from mafic intrusives and the primary source of nickel remains unclear. Structural deformation (faulting and folding) was modelled from core logging, early pit mapping and multi-element data.

8.2.2 Guiding principles for exploration and modelling

The extent and geometry of the host sequence is well defined by drilling, airborne EM and magnetic geophysical surveys and soil/core geochemical sampling. Drilling directions were guided by the mineralization orientations in order to ensure a high angle of intersection to mineralization.

Detailed drill core logging, with supporting multi-element data, allowed for identification and modelling of deposit-scale lithology and structures. The distribution of nickel is controlled by structure, rheology and chemical criteria with black shales been favourable rheology for brittle deformation and a reductant for oxidised hydrothermal fluids. The primary source of the Ni remains unclear. Modelling of mineralized volumes was based upon structural (folds, faults, shears) and lithological trends.

ITEM 9 EXPLORATION

9.1 Historical Trident exploration activities

Historical exploration included comprehensive soil geochemical sampling programme and multiple geophysical surveys across the Trident Project area. Surface outcrop mapping was active during much of the exploration phase. The following geophysical surveys were completed during 2010:

- A LIDAR survey was flown by Fugro Maps (South Africa), providing the mine planning team with a high-resolution topographic elevation surface.
- A combined helicopter-borne magnetic and radiometric survey was undertaken, by New Resolution Geophysics, at 100 m line spacing, contributing to the identification of geological contacts and structures.
- Spectrum Air Ltd flew an electromagnetic (EM) survey at a line spacing of 200 m, assisting with the identification of geological contacts and characteristics across the Trident Project area.

Three section lines of Audio-Magnetic Tellurics (AMT) were undertaken by Geophysical Surveys and Systems, proximal to Sentinel deposit with the aim of delineating key geological structures. Much of the local property geology information, as described in Item 7, has been derived from drillhole logging data, these surveys and outcrop mapping. Drill planning was guided by the results from these exploration activities.

9.2 Recent Trident Project exploration activities

In addition, the following more recent activities were conducted across the respective mining leases:

- 15868-HQ-LML: In 2018, a geophysical survey for resistivity, magnetics, and electro-magnetics was conducted to aid the mapping of sub-surface structures proximal to the Sentinel deposit.
- 15869-HQ-LML: During 2017-2018, a multi-phase infill soil geochemistry survey was undertaken across areas proximal to Enterprise mine. The key aim was to improve classification of hydrothermal nickel geochemical footprint to better direct future exploration targeting. In 2018, phase 2 of a collaborative project with AMIRA International utilizing new technology in multi-spectral ground-penetrating radar was conducted. This was supplemented by additional resistivity, magnetic, and electro-magnetic surveys at a minor scale.
- 15870-HQ-LML: Exploration on this lease focused on the extension of Intrepid aggregate quarry, to support mining operations.
- 15871-HQ-LML: In 2018, geophysical surveys for resistivity, magnetics, and electro-magnetics were conducted to aid with the mapping of key structures.
- 15872-HQ-LML: During 2018, phase 1 of a collaborative project with AMIRA International to provide access to new technology in multi-spectral ground-penetrating radar was completed.

ITEM 10 DRILLING

10.1 Sentinel

10.1.1 Historical drilling

Drilling at Sentinel dates back to 1959. Mwinilunga Mines Ltd, a subsidiary of Roan Selection Trust Technical Services Ltd (RST), drilled 31 holes before 1964. Anglo American Corporation Central Africa (AACCA) completed three diamond drilled holes and 34 reverse circulation holes between 1993 and 2000, exploring for nickel mineralization at Sentinel. Kiwara PLC drilled 62 diamond holes and 19 reverse circulation holes during 2007-2010.

Historical holes drilled by RST and AACCA were deemed to be of sub-standard quality (limited database data and minimal QAQC) to support Mineral Resource estimates. In contrast, drill holes completed by Kiwara PLC, had good quality data and QAQC and were used in this Mineral Resource estimate.

10.1.2 Earlier drilling by FQM

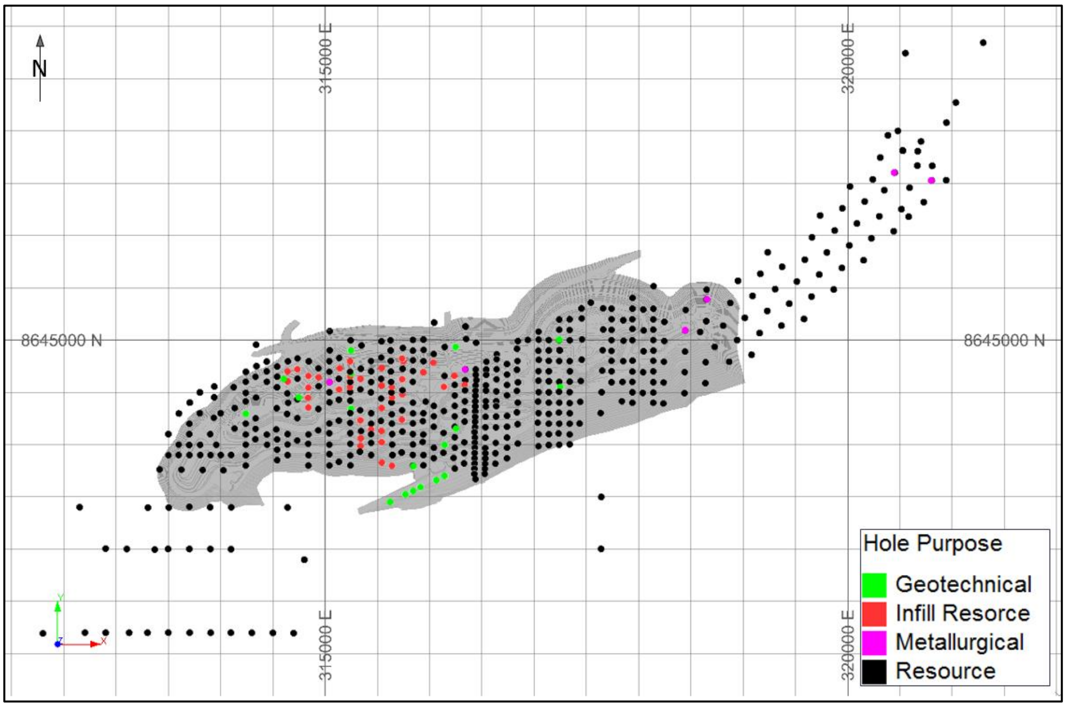
A total of 677 diamond drilled holes were drilled by FQM from start to 2015 across the Sentinel deposit area and used during estimation. Of these, 34 holes were drilled after the main drilling campaign as infill, aimed at better defining the extent and continuity of mineralization. 17 geotechnical holes were drilled to provide information and data for pit slope design and for in-pit crusher pocket design, whilst 5 metallurgical drill holes provided samples for metallurgical test-work (Figure 10-1).

All diamond holes were drilled with a standard core barrel using a triple tube method and were collared using PQ sized core barrels with a standard drill run length of 1.5 m. This was done to ensure maximum recoveries in weathered material and to reduce downhole deviations. Holes were typically cased to a HQ core diameter around 20-30 m deep and then further reduced to NQ at around 150 m depth to end of hole. Improved rock competency with depth allowed for the drill run length of both HQ and NQ to be increased to 3 m. No drilling utilised a core size less than NQ.

As a standard, drill hole core was logged for lithology, regolith, oxidation, alteration, mineralization and structure. Holes were also logged for geotechnical data, including core recovery and rock quality.

Samples were analysed for copper using a four acid digest and Atomic Absorbtion Spectroscopy (AAS). Samples reporting Cu above 0.1% were analysed for AsCu using a single acid digest and AAS. The nominal sample length was 1 m. All drill holes sampled and assayed were subject to comprehensive QAQC sampling and analysis.

Figure 10-1 Diamond drillhole collar plan across the Sentinel deposit area, holes used in the previous Mineral Resource estimate (May 2015)



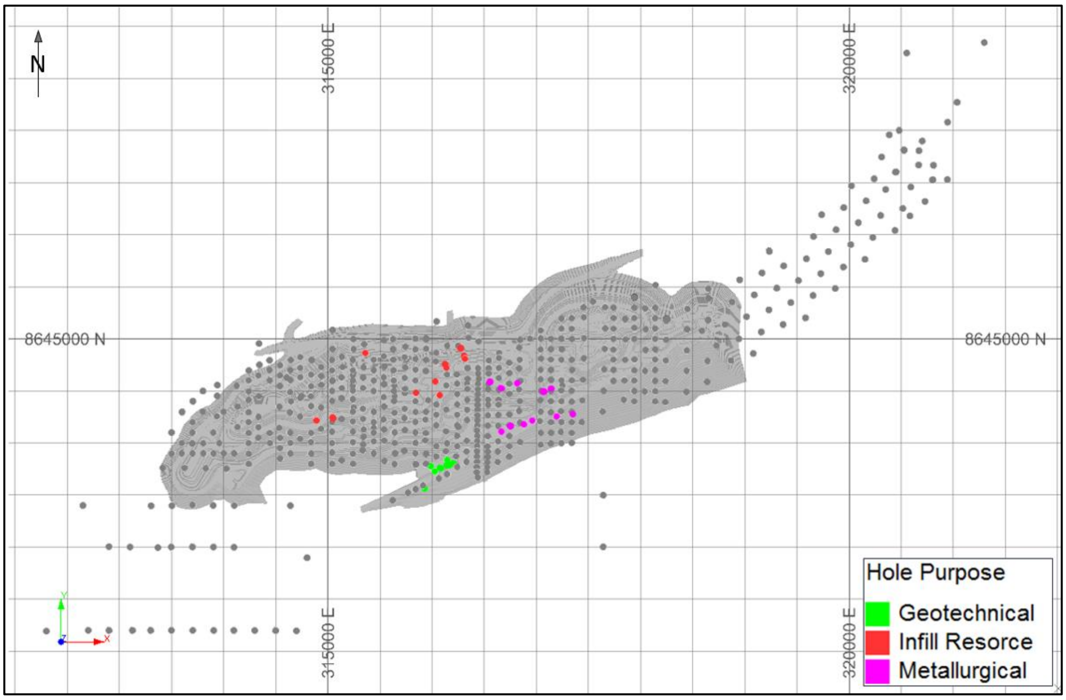
10.1.3 Recent drilling by FQM

10.1.3.1 Diamond drilling

Since the 2015 Technical Report (FQM, May 2015), additional resource development drilling at Sentinel includes 38 infill, geotechnical and metallurgical diamond drill holes focussed within the life of mine (LOM) area.

Of these 38 holes (Figure 10-2), 11 were drilled targeting the transitional mineralization on the eastern cutback of the pit, and provided samples for metallurgical test-work. An additional 11 holes were drilled as infill, aimed at better defining the extent and continuity of mineralization. Sixteen geotechnical holes were drilled to provide information and data for pit slope design and for in-pit crusher pocket design.

Figure 10-2 Plan view of the Sentinel deposit area highlighting the life of mine pit area and the diamond holes drilled since the previous estimate

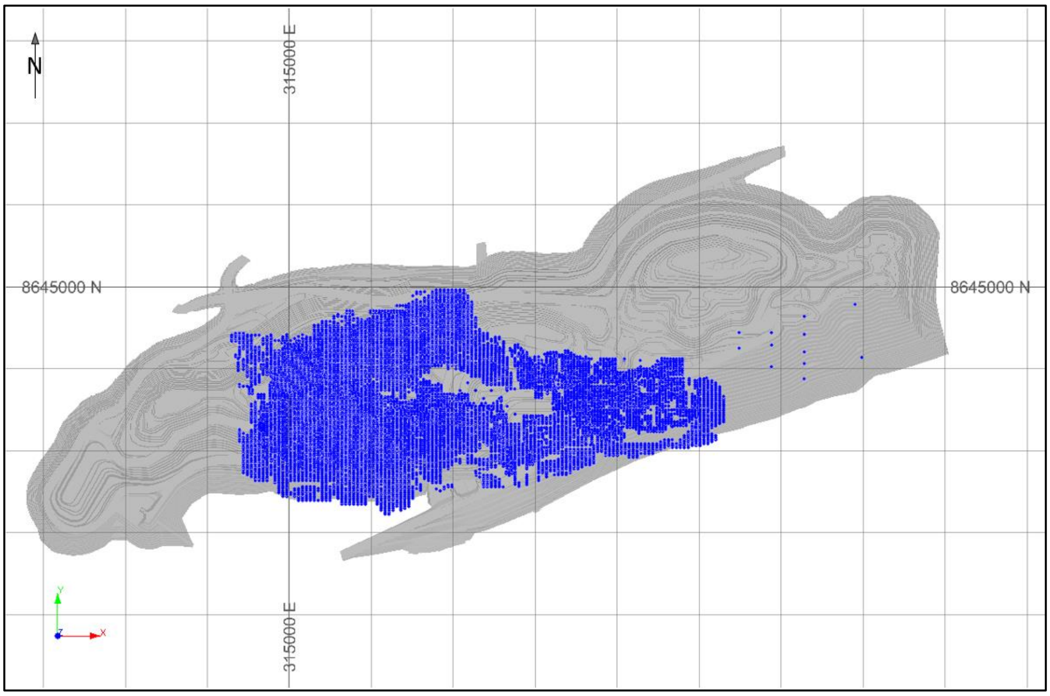


10.1.3.2 Grade control Reverse Circulation (RC) drilling

Grade control RC drilling commenced at Sentinel in May 2014 across the starter pit area, using a 24 m (north) by 12 m (south) grid, an azimuth of 180° and at dip angle of 70°. The average depth of holes was 54 m (49 m on vertical depth), or the equivalent of 4 x 12 m high mining benches. All drill holes were drilled with a 140 mm diameter drill bit and samples were collected at every 3 m interval.

RC drilling grid was reduced to 18 m (north) by 12 m (south) in September 2017 to improve definition of mineralization. During drilling and sampling, RC chips from each interval were collected, washed and stored in sample trays for logging. At the time of this estimate, a total of 17,553 RC holes (Figure 10-3) were completed with survey, logging and assay information and were included.

Figure 10-3 Plan view of the Sentinel deposit area highlighting the life of mine pit area and the Reverse Circulation holes drilled at Sentinel for grade control (blue)



10.1.4 Summary of drilling used in this Mineral Resource Estimate (MRE)

The Sentinel database includes 700 diamond holes including exploration, resource delineation, infill and geotechnical holes. From these, 64 holes were excluded for either being outside of the deposit or not having relevant information (geotechnical holes) or of low quality (historical exploration holes).

The following table (Table 10-1) provides a summary of drill holes used for this estimate:

Table 10-1 Summary of drill holes used in this Sentinel MRE

Company	Period	Type	# holes	# meters	# samples	# Cu assays
Kiwara	*2010-2013*	*DD*	*44*	*14,036*	*10,710*	*10,539*
FQM	2010-2013	DD	570	193,391	135,681	132,694
FQM	2017-2018	DD	22	3,547	3,551	3,549
FQM	2014-2019	RC	17,553	899,088	299,384	295,030
TOTAL			**18,189**	**1,110,062**	**449,326**	**441,812**

10.1.5 Drilling orientation

Sentinel mineralization has a general east-west strike and a shallow north-northwest dip. Diamond drilled holes were oriented at 70° dip towards 180°, maximising the angle of intersection to mineralization. Grade control RC drilling uses the same dip and azimuth as diamond drilling. Geotechnical drill holes were drilled at different inclinations and directions according to their planned targets (e.g. crusher pockets or pit wall design).

10.1.6 Collar surveys

Drill hole collars were surveyed by a registered surveyor using a differential GPS and the ARC 1950 (mean) 35S coordinate system. All surveyed drill hole collar coordinates were validated against the Lidar topographical survey with no deviations noted.

10.1.7 Downhole surveys

Downhole survey measurements were standard procedure on all diamond and RC drilled holes, using the Reflex EZ Shot, an electronic single shot (point) magnetic survey tool.

During 2011-2014 diamond drilling, surveys were taken at 50 m intervals in order to record downhole deviations. In areas where historical deviations were too high, an initial 25 m survey was taken to measure deviation at the start of hole. For 2017-2018 infill diamond drilling, surveys were taken at 20 m intervals.

Downhole surveys for RC drilling (grade control) were taken 5 m below collar and then every 15 m intervals. Where deviation was greater than 10 degrees, these surveys were identified for re-survey or re-drill, depending on the extent and details of deviations. In the case of RC holes, minor and irregular occurrences of poor ground conditions would prevent completing a downhole survey. Given the narrow range of variability observed across the high volume of downhole survey measurements, minor gaps in this data were not considered a risk to this estimate.

10.1.8 Core orientations

Drilling contractors were tasked with ensuring core trays were labelled with the correct drill hole identity and box number. At the end of each drill run, core blocks were inserted and marked with depths, approximate recovery and core loss or gain information. Where successful, the core was marked with orientation marks and relevant driller breaks.

All drilled core was orientated for each drill run by the drilling contractor using the Reflex ACT II RD Orientation tool. Orientations were aligned and marked up by a suitably trained FQM Geology Technician. These ‘ori marks’ were dependent on the confidence of the orientations, where a minimum of two orientation points must be within 10° to be considered. A solid continuous line was drawn using a permanent marker and represents high confidence orientations. Medium and no confidence were marked as dashed and dotted lines respectively. Core orientation marking was completed each day and monitored by the responsible geologist.

10.1.9 RQD and density measurements

Core recovery and Rock Quality Designation (RQD) was recorded from the core by suitably trained FQM Geology Technicians. RQD and core recovery data were used to support quality samples and geotechnical modelling. In addition, 30 cm long whole core samples were taken every 10 m for measuring specific gravity. All data was recorded manually and then recorded into the Sentinel Datashed database.

19,518 density measurements were taken providing a sufficient number of samples per key lithology and weathering domain for a reasonable density estimate.

Drill core recovery ranged from 76% in the weathered horizons to 98% in fresh rock. Average core recovery was 97%, with 92% of all core having a recovery greater than 90%. Grade control RC samples had an average mass recovery of 85%. The measured diamond core and RC mass recovery values were believed sufficient to minimise risks to sample quality.

10.1.10 Geological data

All drill holes (core and RC chips) were logged by qualified geologists for lithology, regolith, oxidation, alteration and mineralization.

Structural features in drill core, such as bedding, foliation, fold axial planes, joints and faults were measured using kenometers for both geological and geotechnical purposes. Data was captured using digital tablets and imported into the Sentinel database.

Whole core was photographed, both wet and dry, by a suitably trained FQM Geology Technician. These photos were stored on the site server and linked to the Sentinel database. Drilled core was logged for RQD and geology and was sampled for density and element analysis.

It is the opinion of the QP that diamond and RC drilling was reasonably oriented and drilled. Together with good data management, the resulting diamond and RC drilling data supports accurate 3D downhole sample positions. There were no known drilling factors that could materially affect the accuracy and reliability of this Mineral Resource Estimate.

10.2 Enterprise

10.2.1 Historical drilling

Kiwara PLC drilled 11 diamond holes across the Enterprise area in 2009. These holes intersected intermittent areas of nickel mineralization on a wide grid spacing. Quality of core and sampling were reviewed and considered acceptable to be included in this estimate.

10.2.2 Earlier drilling by FQM

FQM completed diamond drilling between 2010 and 2013. Holes commenced with a PQ diameter which was reduced to HQ and then NQ from approximately 200 m depths. All drilling used triple tube to minimise core loss in the shallower depths affected by weathering.

Holes were drilled at a dip of 60° to the south-east with a few steeper drilled holes. Drilling maximized the angle of intersection with mineralization.

The exploration drilling strategy at Enterprise targeted nickel mineralization intersected from the wider spaced Kiwara holes. Holes were infilled to a grid of 50 m on dip and 100 m along strike. Drilling covered the initial 2 km zone as defined by anomalous Ni and Cu values from soil sampling and aimed to connect the intermittent zones of mineralization.

In order to gain a full understanding of the lithostratigraphy of the area, the target depth of each hole was initially set as the upper contact of the biotite-quartz rock found at the base of the Enterprise sequence, even though this involved drilling through barren rock beneath the mineralized zone.

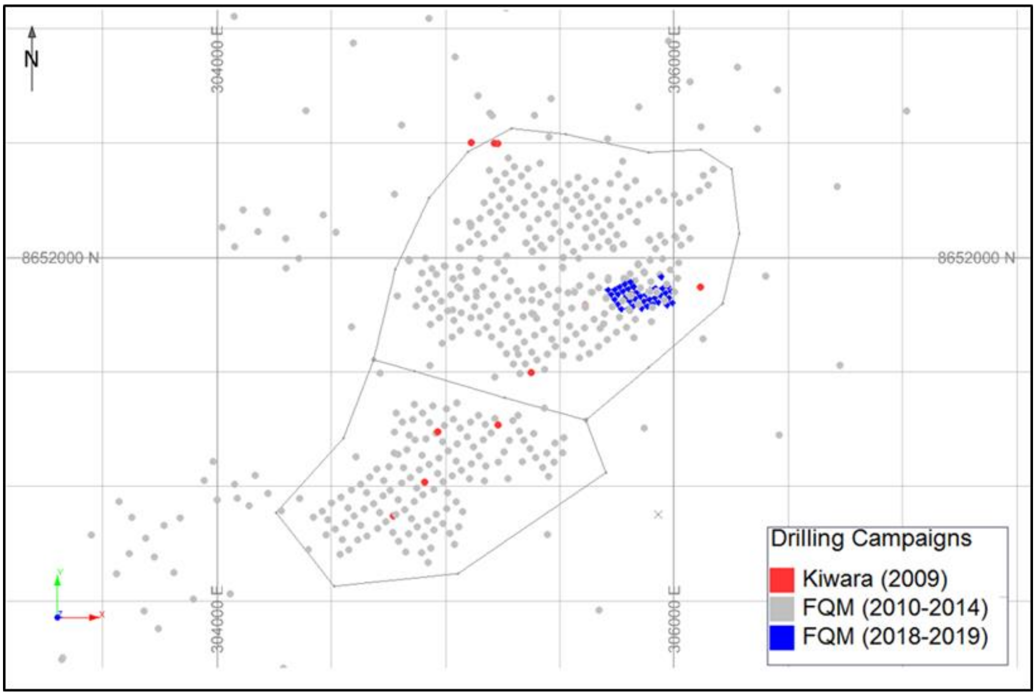
10.2.3 Recent drilling by FQM

During 2018 and 2019, FQM conducted drilling across a small area south of the starter pit. A 25 m grid was used to guide grade control grid spacing and to gain additional knowledge on deposit structure and lithology controls on mineralization.

Holes were drilled following the same guidelines as previous drilling and used a dip of 60° and azimuth of 145°. The hole length was determined by the depth of the contact with the quartzite rocks, ranging between 30 and 155 meters.

A plan view of the drill collar locations is shown in Figure 10-4.

Figure 10-4 Plan view of drill collars for the different drilling campaigns at Enterprise, showing mineralization limits of Enterprise Main (top polygon) and Enterprise SW (bottom polygon)



10.2.4 Summary of drilling used in this MRE

The following table (Table 10-2) provides a summary of the drill holes used for this Mineral Resource Estimate:

Table 10-2 Summary of drill holes used in this Enterprise MRE

Company	Period	Type	# holes	# meters	# samples	# Ni assays
Kiwara	*2009*	*DD*	*11*	*1,907*	*1,593*	*1,588*
FQM	2010-2014	DD	528	160,571	137,127	136,159
FQM	2018-2019	DD	41	3,758	3,761	3,761
TOTAL			**580**	**166,236**	**142,481**	**141,508**

10.2.5 Core recovery

Core recovery data has been reviewed by the QP. There are no drilling, sampling, or recovery factors (field errors) that were believed material to the 3D position, accuracy and quality of the logging and assay grades obtained from drill hole samples. Holes were drilled using PQ core diameter to intersect the near surface less consolidated material, with a change to HQ and then NQ core diameter at depth in more competent rock.

10.2.6 Collar surveys

Collar surveys for 2010-2014 diamond drilling were measured by Surtech International Limited. Historical collar surveys were originally in WGS84 and subsequently converted into ARC1950 (Zambia) and then into ARC1950 (Mean) with all coordinates shown as UTM/UPS. All drill hole collars have been resurveyed in ARC1950 (Mean) or converted into this grid system. Several survey reports which detail the methods and equipment used to determine the relative collar positions exist, including that of Surtech International.

Collar surveys for 2018-2019 diamond drilling campaign were measured by FQM accredited surveyors in ARC1950 grid system.

10.2.7 Down hole surveys

Down hole surveys were carried out by the drilling contractors using the Reflex EZ Shot instrument, a single shot (point), magnetic survey instrument. Standard requirements for the 2010-2014 drilling were to survey drill holes on 50 m intervals whilst the hole was being drilled in order to monitor down hole deviation as the hole progressed. Downhole survey measurements were taken at 25 m intervals for the 2018-2019 infill diamond campaign. A re-survey would be completed if the measured magnetic field at the time of the survey measurement was too high (> 10 degrees). Surveys were also flagged for investigation if their deviation in either dip or azimuth was greater than five degrees over 50 m.

10.2.8 Oriented core

For 2010-2014 drilling, HQ and NQ core was oriented at the end of each drill run, usually 3m lengths, unless ground conditions dictated otherwise. PQ core was not oriented as it was usually in the weathered, upper portions. Initially, orientations were determined from the ‘spear test’ technique. However, this method yielded unsatisfactory results and was abandoned for Reflex ACT and Reflex ACT II RD orientation instrument measurements.

During 2018-2019 drilling some measurements were taken from PQ core at shallow depths where rock competency permitted.

10.2.9 Core handling, density measurements and RQD determination

Drill core was transported from the drill site to the safe and secure core shed at the Enterprise exploration camp. Suitably trained field technicians measured and calculated recovery and Rock Quality Designation (RQD). Magnetic susceptibility measurements were obtained at the metre marks for the length of each hole (2010-2014). Specific gravity (SG) measurements were for 20 cm sample lengths every 10 m. Full core photography, both wet and dry, was captured by a suitably trained technician using a SLR digital camera. The core photos were renamed according to the drill hole identity and depth intervals.

10.2.10 Geological and structural logging of core

Drill core was logged for lithology, mineralization, alteration and structure by suitably qualified geologists using a standardised logging code system to ensure consistency. This data was collected and validated by the site database administrator. Structural measurements of bedding, foliation, fault planes, fold axes, lineations and vein orientations were obtained by measuring alpha, beta and gamma angles using kenometers.

ITEM 11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sentinel

11.1.1 Diamond core and RC chip logging

Drilled diamond core and RC chip samples were stored in the secure and safe core shed facilities at Sentinel.

All diamond holes were logged for lithology, regolith, oxidation, mineralization, alteration, and structure by suitably qualified geologists using a standardised logging code system to ensure consistency. Logging of the washed RC chips (in trays) followed the same practices, but excluding structural logging. Specific gravity of diamond core was measured every 10 m from approximately 20 cm sample lengths.

All data was recorded into MS Excel workbook templates with standardised libraries and subsequently imported, formatted and validated in the Maxwell DataShed site-based SQL database.

11.1.2 Diamond core and RC sample preparation

On completion of logging, recovery and RQD measurements, the geologist marked out diamond core to 1 m sample lengths according to key geology contacts. The full length of each hole was sampled. Un-cut core was photographed, wet and dry, prior to any sampling. A diamond saw was used to cut marked core in half, with one half bagged according to pre-defined sample numbers and sealed for quality and security. The soil and saprolite zones were split in half using a spade, with half the material sampled.

Recovery per 3 m RC chip sample retrieved from the drill rig cone-splitter was determined by weighing each sample. Each 3 m sample had an approximate split sample mass of 12 kg from the drilled 120 kg sample. The 12 kg samples were split at the drill rig into 3 kg samples using a two-tier riffle splitter. Samples were placed into porous calico bags. The full length of each RC hole was sampled. In the rare event that RC samples were retrieved wet, samples were separated from the main stream and laid out to dry prior to dispatch.

All bagged drill hole samples were assigned a batch number and packed into larger polyweave bags for transportation. Each polyweave bag detailed the contained samples and batch numbers. The polyweave bagged samples were transported to the ALS preparation laboratory facility at Kansanshi mine for sample preparation. Between 2010-2013, sample preparation followed this procedure:

- Samples were crushed to 90% passing 2 mm using a jaw crusher.
- Crushed sample was split through a single-tier splitter until a 500 g sample mass was retrieved.
- LM2 pulverisers were used to produce a pulp sample with 90% passing 106 microns.
- Approximately 250 g of pulp was retrieved.
- Samples were despatched via Mercury couriers or DHL to ALS Chemex Johannesburg for analysis.

Sample preparation subsequent to 2013 followed this procedure (for diamond core and chips):

- Samples were crushed to 90% passing 2 mm using a jaw crusher.
- Crushed material was split through a single-tier splitter until a 1 kg sample mass was retrieved.
- LM2 pulverisers were used to produce a pulp sample with 90% passing 106 microns.
- A single-tier splitter was used to retrieve an approximately 250 g pulp sample.
- A 50 g lab split was scooped into an envelope for the analytical lab analysis.
- On completion of laboratory analysis pulp rejects were returned to and stored on site.

It is the QP's opinions that samples have been collected, prepared and numbered using good standards and that the transportation and sample data management was completed in a safe and secure manner.

11.1.3 Diamond core and RC sample analysis

Diamond core sampling between 2010 and 2013 (up to KALDD0513) were assayed at ALS Chemex Johannesburg laboratory in South Africa. A 50 g pulp charge was exposed to a four acid digest followed by analysis using atomic absorption spectroscopy. Samples with more than 0.1% total copper were analysed for acid soluble copper in order to define proportions of soluble copper from oxide copper minerals. Key elements analysed were copper, cobalt and nickel.

During 2013, a small subset of samples (KALDD0514-KALDD0558) were analysed for copper and nickel at Intertek Genalysis laboratory in Ndola, Zambia. This method exposed the pulp sample to a three acid digest, followed by an AAS finish. The three and four acid assay results were investigated for bias with no evidence for either method dissolving more or less metal.

Recent (2017 to 2018) diamond core samples were analysed at the ALS Chemex Kansanshi laboratory, which achieved full accreditation in October 2017. Routine analysis included sequential leach copper, nickel, and cobalt using a four acid digest with AAS finish.

Between May 2014 and Jan 2017, all RC samples were analysed at the ALS Chemex Kansanshi laboratory. Whilst the lab was not officially accredited during this period, it was independently managed by ALS personnel according to ALS global standards and procedures. Routine analysis included total copper, acid soluble copper, cyanide soluble copper, nickel, and cobalt using a four acid digest with AAS finish.

From January 2017 to date, all RC samples were analysed at the ALS Chemex Kansanshi laboratory for sequential leach copper, nickel, and cobalt using four acid digest with AAS finish.

11.1.4 QAQC procedures

Diamond drilled core and RC chip samples were exposed to industry QAQC practices. Diamond core samples between 2010 and 2013 had a blank sample and a range of Certified Reference Material (CRM) samples each inserted every 40th sample. Field duplicates were completed as part of the QAQC procedures. Coarse crush duplicates were inserted every 28th sample and pulp duplicates were inserted every 46th sample. RC and diamond drilled samples from 2014 to date had a blank and several CRM samples inserted every 20th sample. Field, coarse crush and pulp duplicates inserted every 20th sample.

Returned blank sample values demonstrate that contamination was adequately controlled during sample preparation (Figure 11-6 and Figure 11-7). The inserted CRM's analysis highlighted acceptable primary laboratory accuracy, with only three samples having evidence for sample mislabelling. Most CRM assayed values were within two standard deviations of the certified values verifying accuracy of the primary laboratory

A summary of key QAQC graphs for 2017-2018 diamond drilling and 2018 RC drilling is presented in Figure 11-1 to Figure 11-11. 2018 data for RC drilling has been selected to demonstrate recent and average performance of the QAQC programmes over time.

The inserted coarse crush and pulp duplicates highlight good precision between pairs of samples with more than 90% of the returned sample pair values been within 10% of the original sample value (Figure 11-1 and Figure 11-2).

QAQC includes a comprehensive programme of umpire check analysis of pulp duplicates from both diamond and RC drill programmes. Umpire duplicate pulp samples were submitted to the accredited ALS laboratory in Johannesburg on a yearly or bi-annual basis. From 2014 to 2017, 3% of all core and RC chips samples drilled were submitted for umpire checks. For 2018 and 2019 the rate increased to 5%, with the latest batch submitted in April 2019.

The umpire duplicate results demonstrated good precision with more than 90% of sample pairs having values within 10% (Figure 11-3 and Figure 11-4). The results demonstrate repeatable and quality analyses of the material sampled and analysed at the primary laboratories.

Figure 11-1 HARD plot for diamond drilling copper duplicate samples from 2010-2018

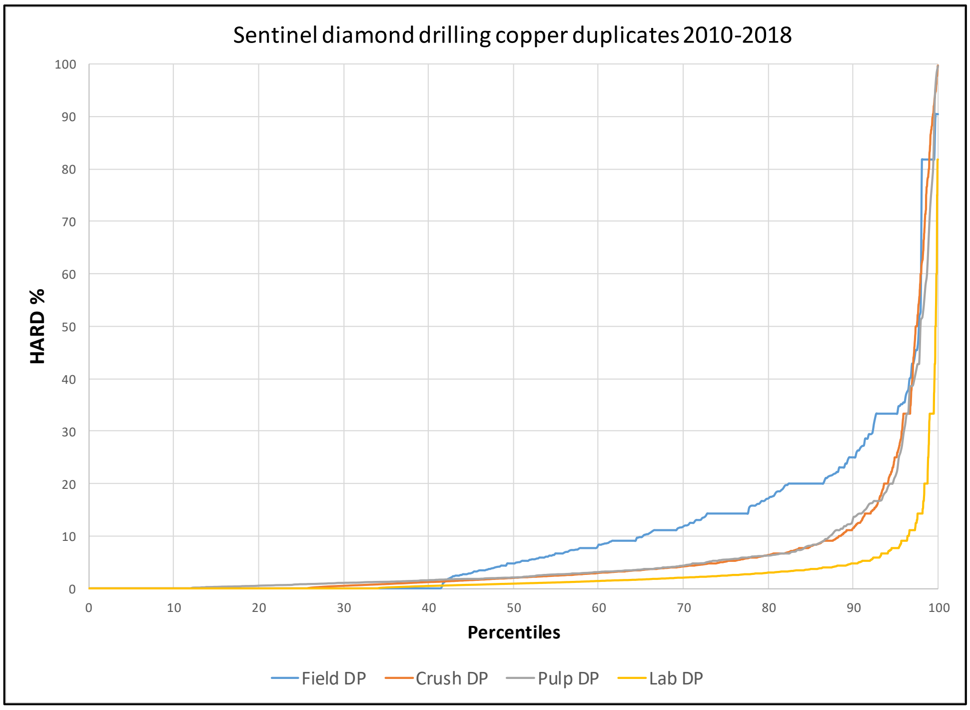


Figure 11-2 HARD plot for RC copper duplicate samples from 2014-2019

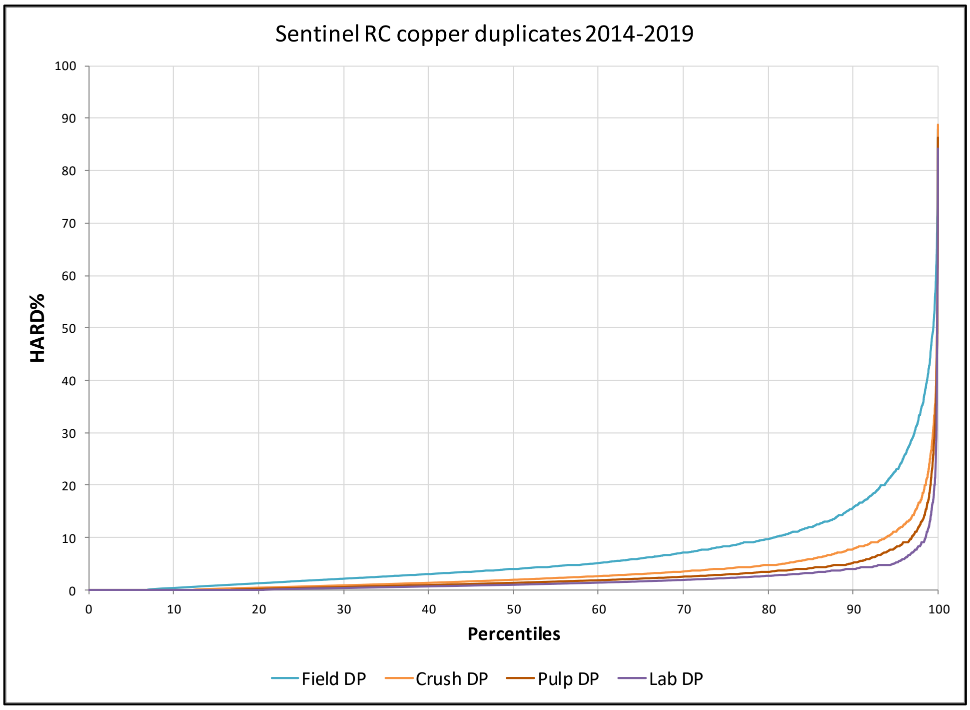


Figure 11-3 Scatter plot for Diamond drilling copper umpire checks

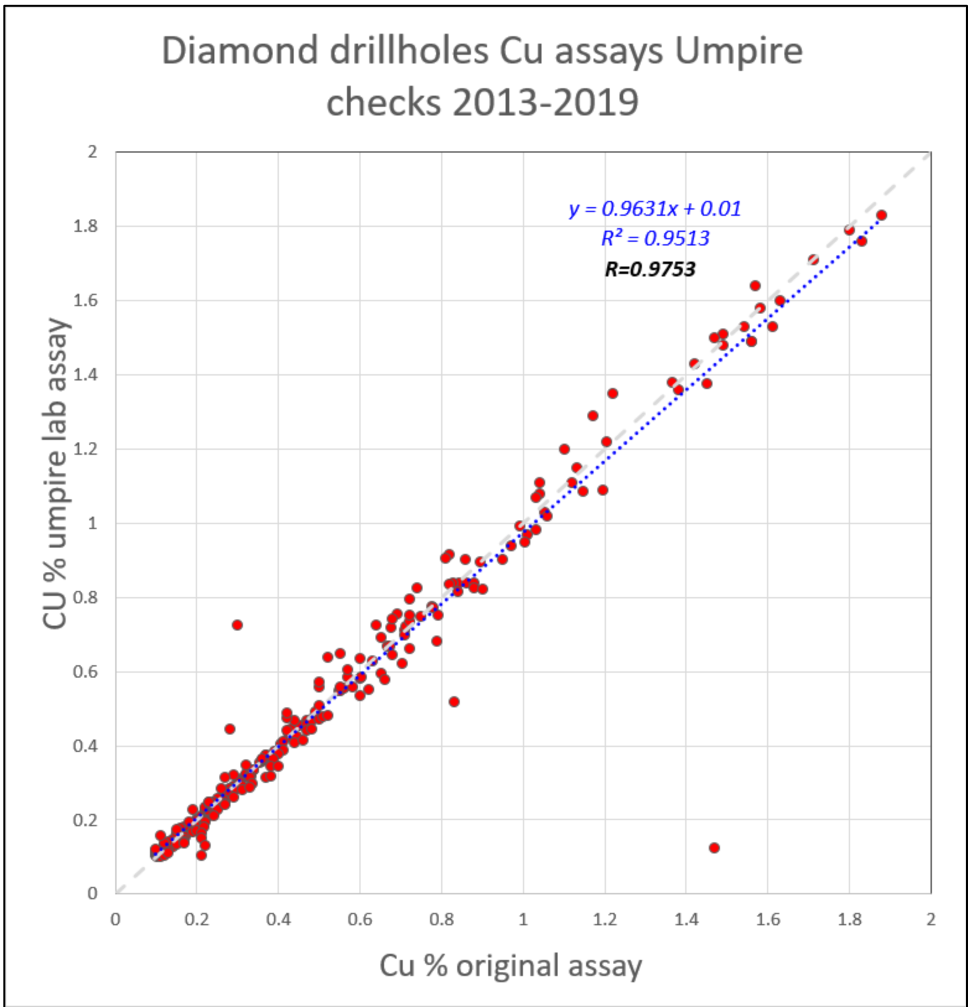


Figure 11-4 Scatter plot for RC copper assays umpire checks

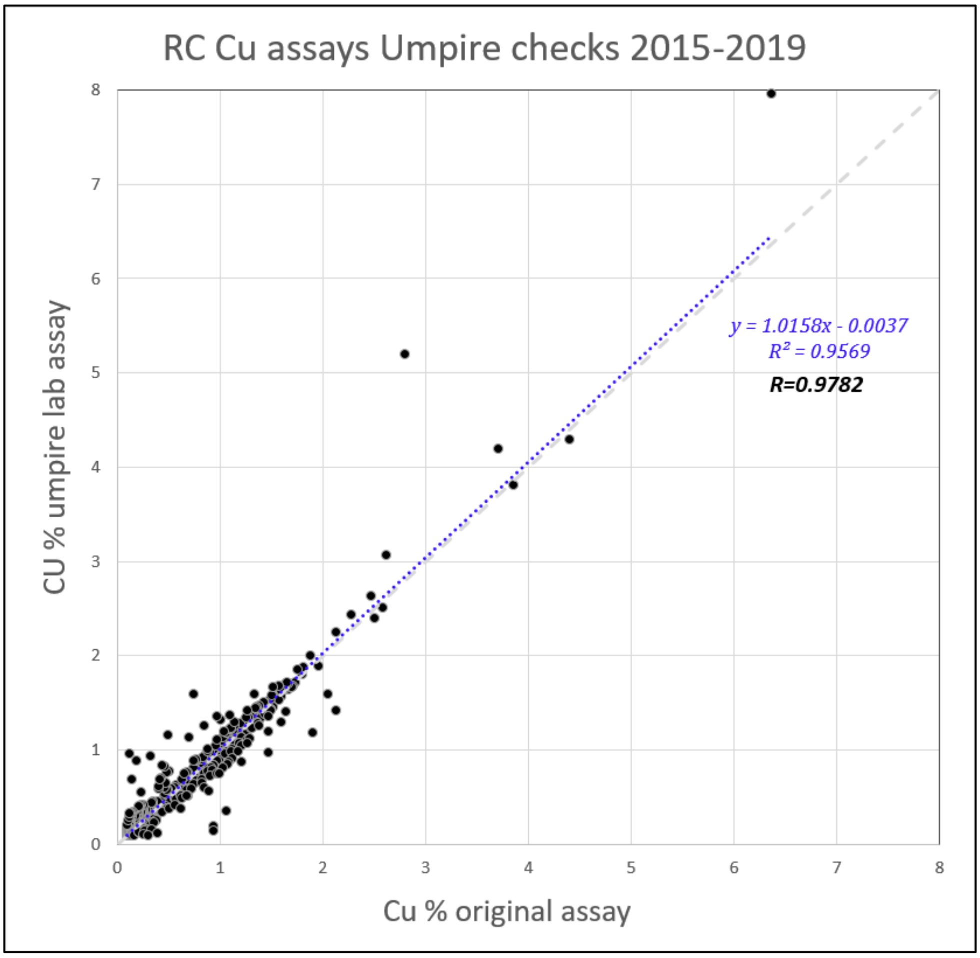


Figure 11-5 Sentinel CRMs control chart for 2017-2018 diamond drilling campaign

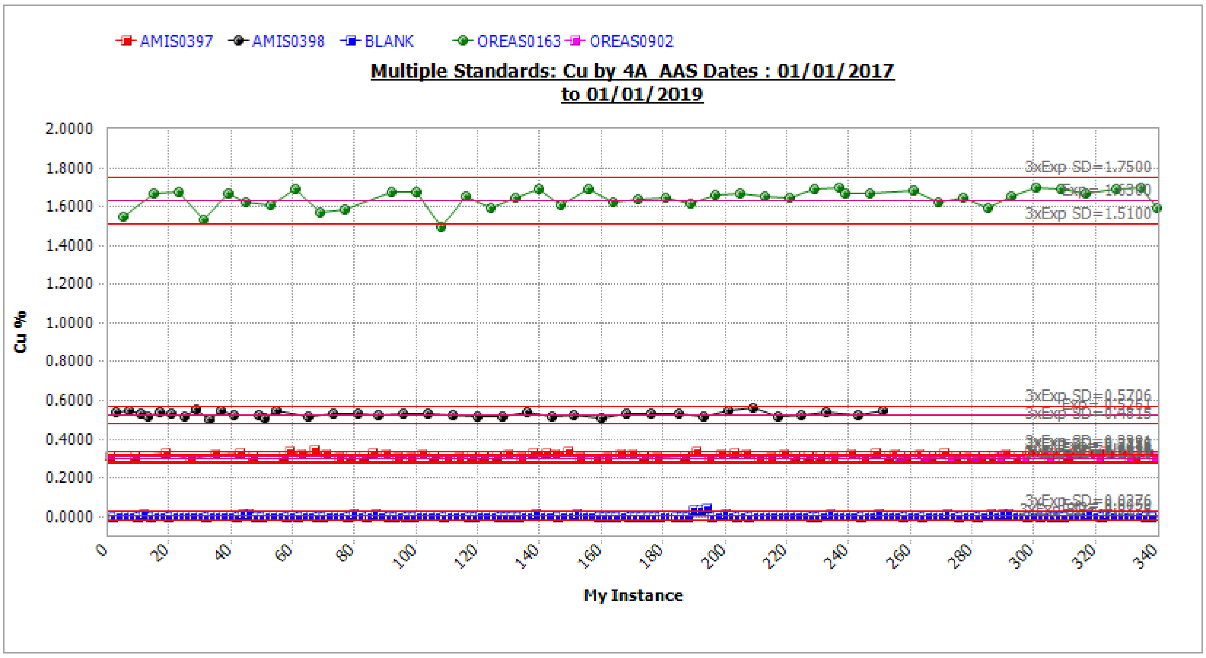


Figure 11-6 Blank samples control chart for 2017-2018 diamond drilling campaign

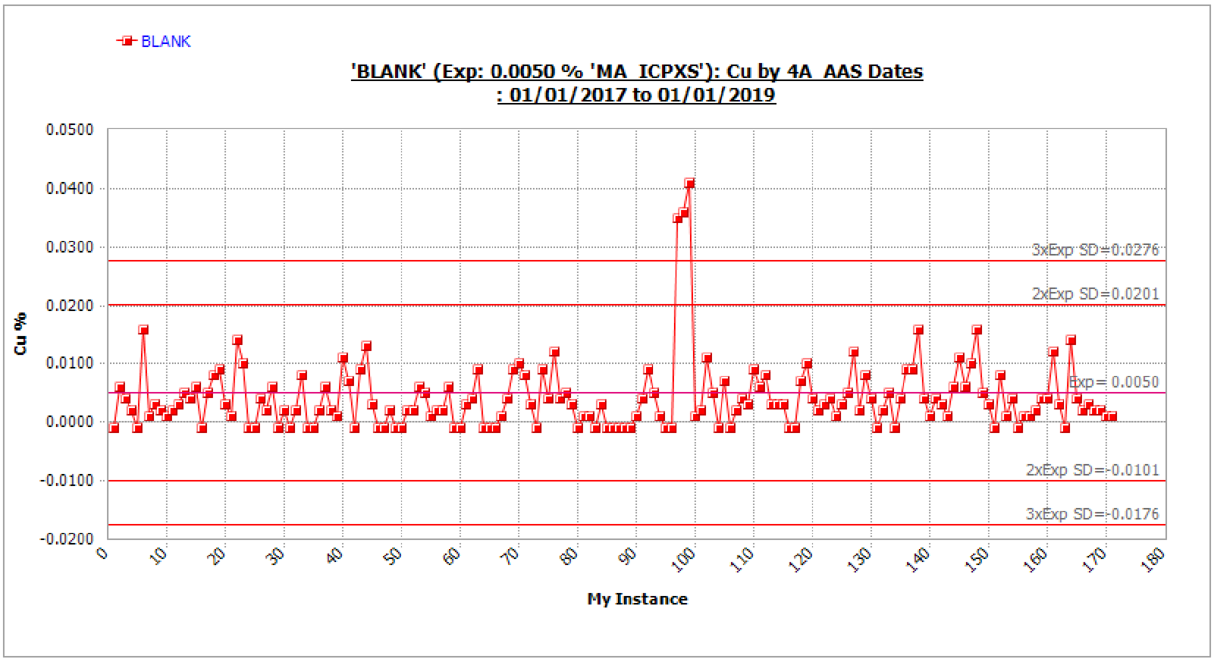


Figure 11-7 Sentinel RC Blank samples control chart for 2018

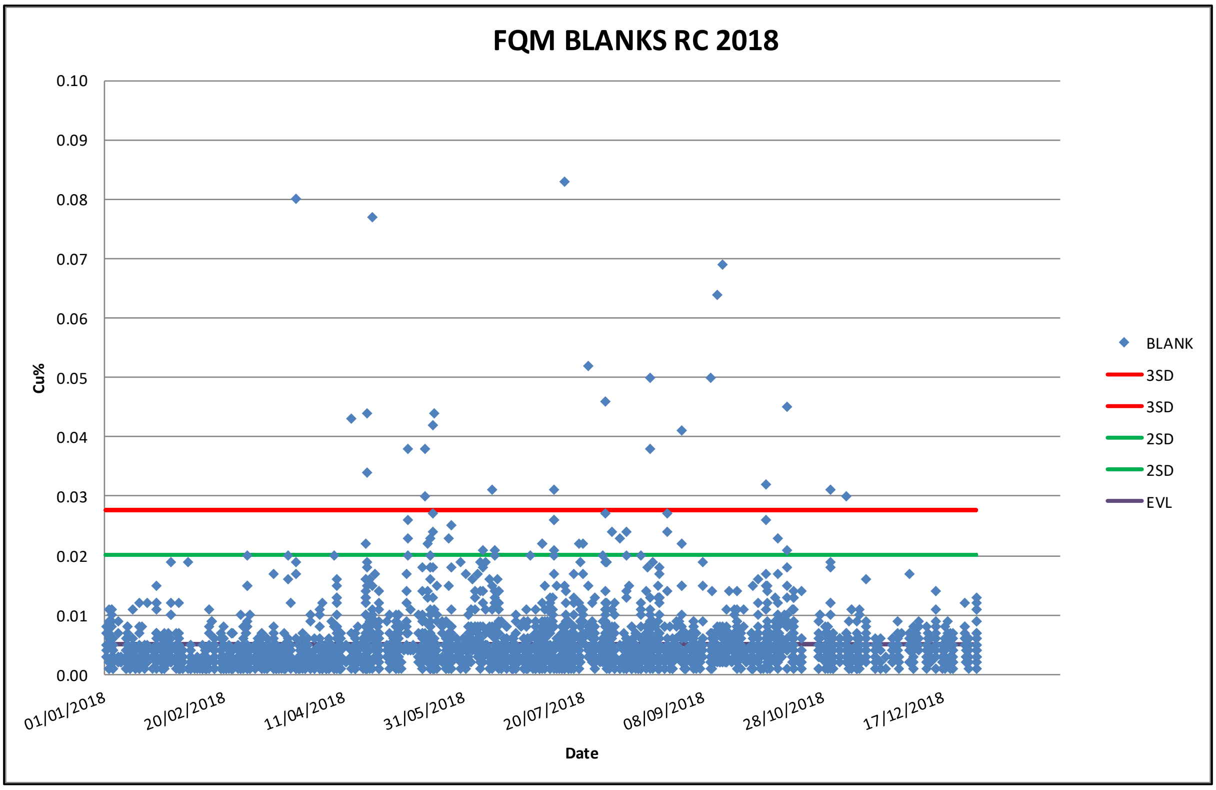


Figure 11-8 Sentinel CRM AMIS0397 Control chart for RC samples 2018

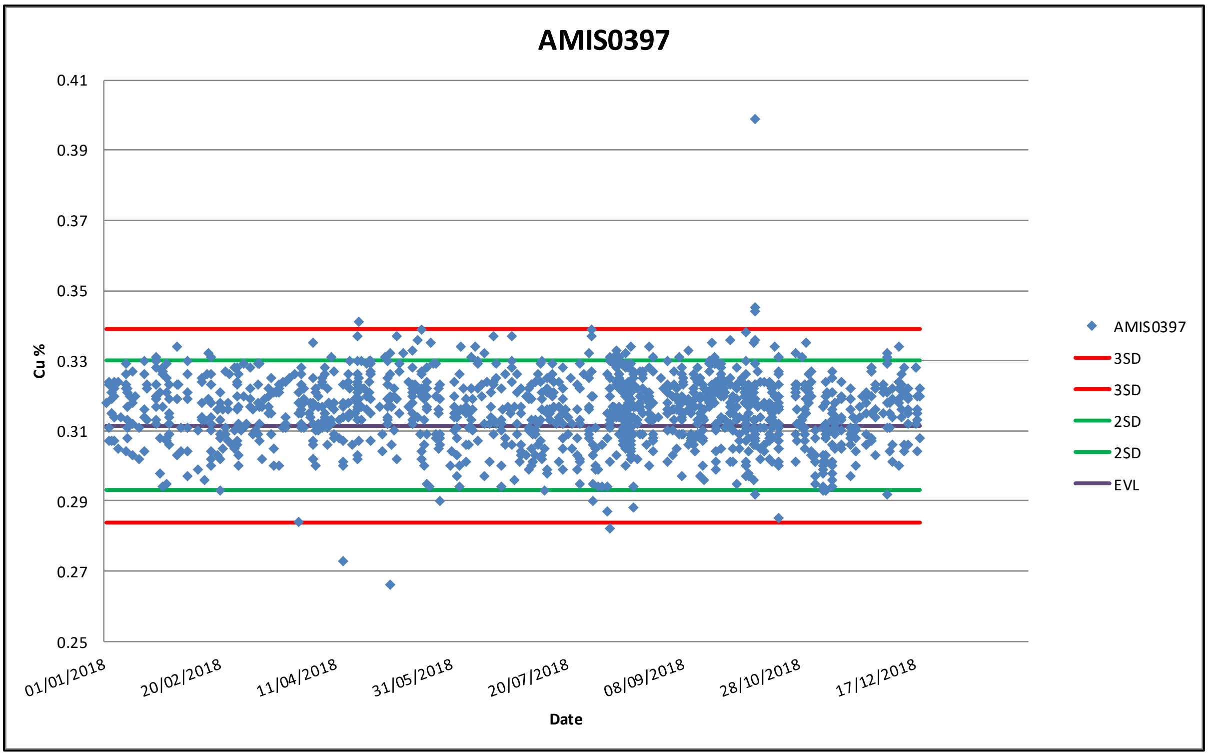


Figure 11-9 Sentinel CRM AMIS0398 Control chart for RC samples 2018

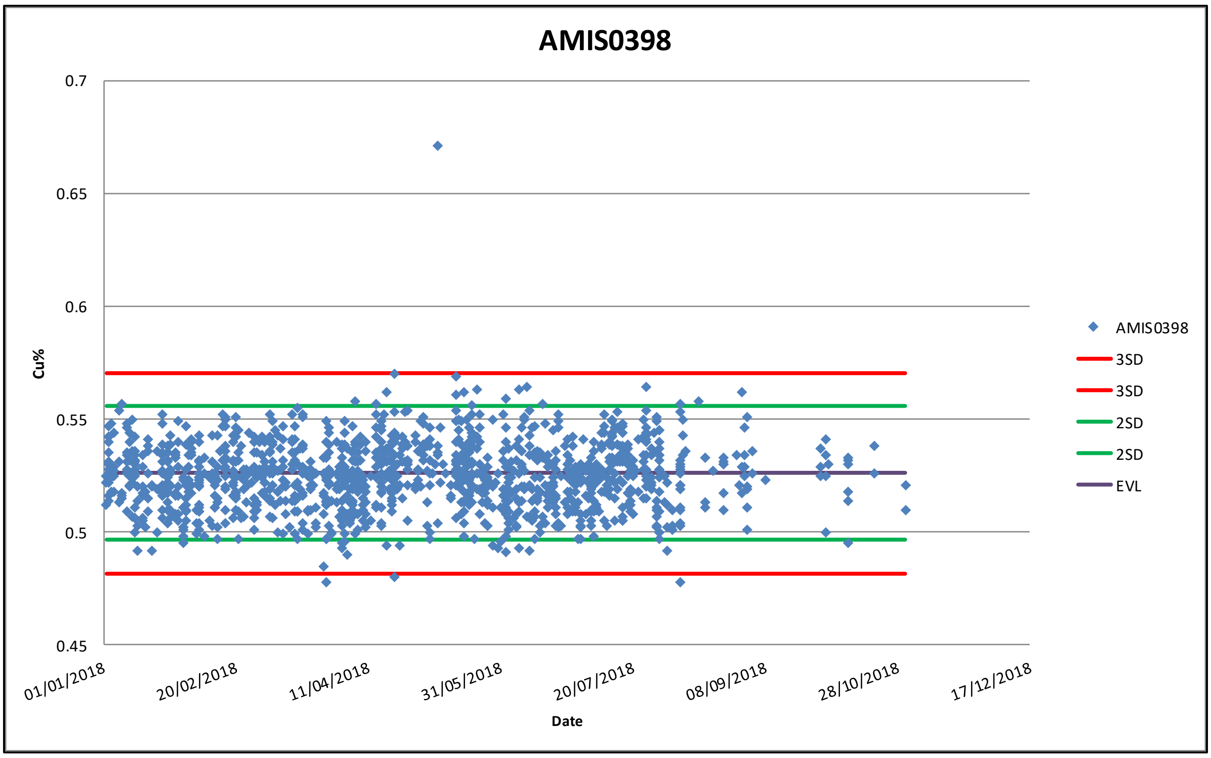


Figure 11-10 Sentinel CRM OREAS0163 Control chart for RC samples 2018

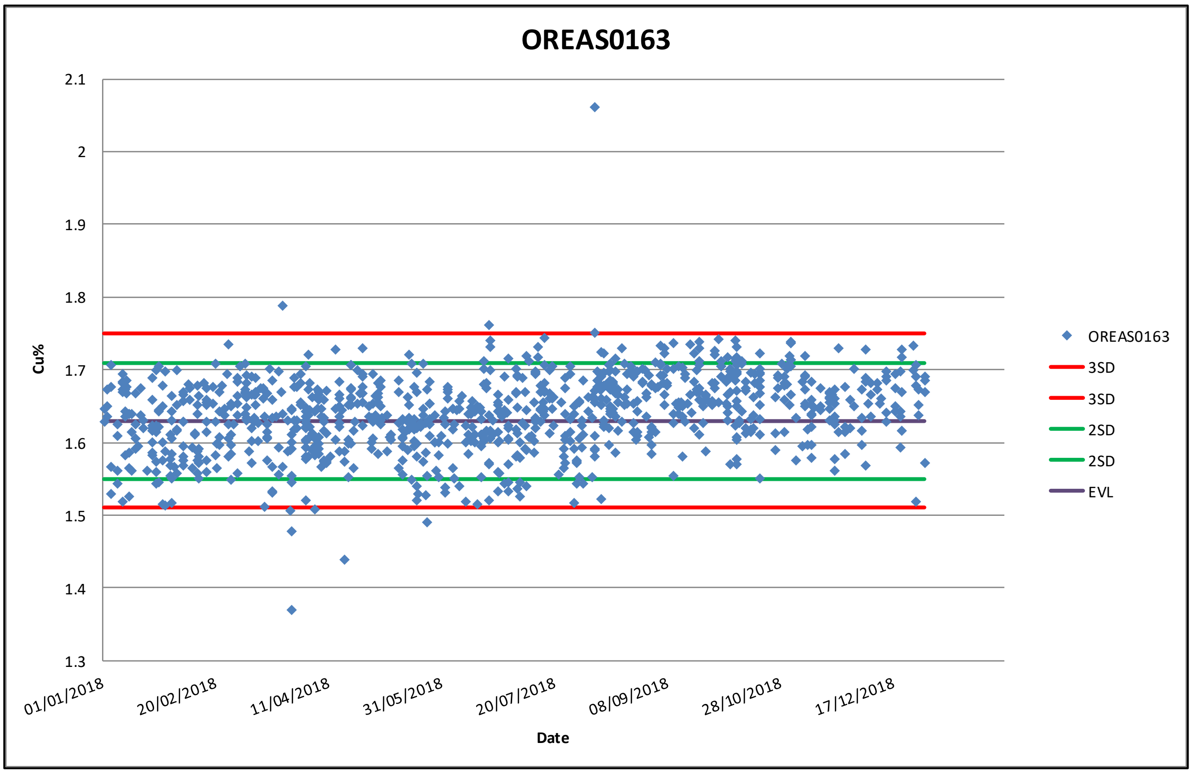
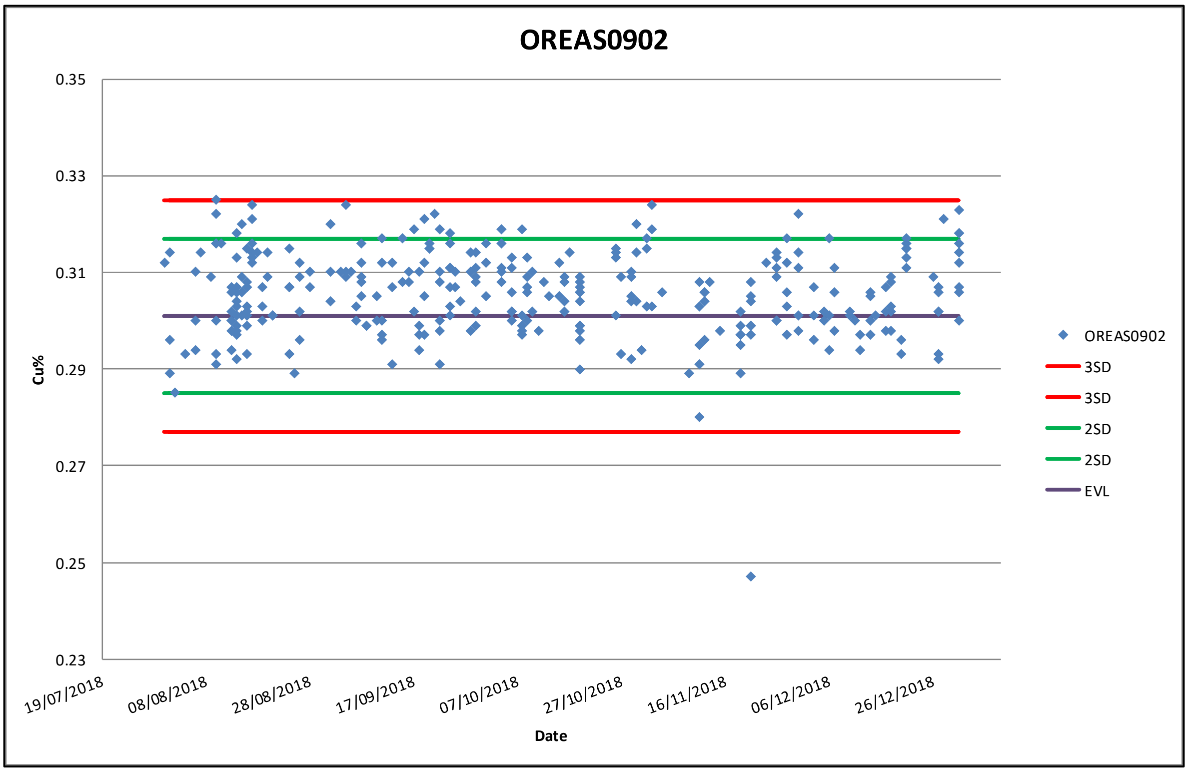


Figure 11-11 Sentinel CRM OREAS0902 Control chart for RC samples 2018



11.1.5 QP's opinion

Safe and secure sampling, sample data management and sample preparation have enabled consistent and repeatable sample analysis. Since November 2012, samples have been analysed using a four acid digest with an AAS finish by ALS laboratories at Kansanshi and Johannesburg. FQM's blank, CRM, duplicate and umpire sample inserts have assured representative and repeatable results. Blanks demonstrate that contamination was controlled during sample preparation. Duplicates have adequate precision with good repeatability. CRM samples demonstrate accurate results with limited evidence for analytical biases and sample mislabelling.

It is the QP's opinion that Sentinel sample preparation, analysis and security is adequate for use in this Mineral Resource Estimate.

11.2 Enterprise

11.2.1 Diamond core logging

Diamond drilled core was returned to the core shed at Enterprise exploration camp for safe and secure storage.

Core was logged for lithology, mineralization, alteration and structure by suitably qualified geologists using a standardised logging code system for ensuring consistent data recording. Structural data was converted from core measurements to dip/dip direction and dip/plunge using a conversion sheet which accounts for the variable orientation of the drill hole. Results together with their original α, β and γ measurements were stored in the Datashed database. Suitably trained field technicians calculate the core recovery and Rock Quality Designation (RQD). Specific gravity was measured every 10 m from approximately 20 cm sample lengths. Logged data was formatted and validated using the Maxwell DataShed site-based SQL database. Full core, both wet and dry, was photographed using a SLR digital camera to obtain high resolution digital photographs.

11.2.2 Diamond core sample preparation

From 2012, sampling stopped at the base of the quartz-biotite as it was proven to be completely barren. Holes were sampled on the basis of lithology and observed mineralization. Standard sampling lengths were 1 m. No sub-metre samples were taken over higher mineralized zones by FQM, as practiced by Kiwara.

The soil and saprolite horizons were sampled from a split of the core tray material using a spade. Competent core was cut in half using a diamond saw. Cutting and sampling of half core was completed in such a way so as to preserve the orientation line to ensure that the left hand half of the core was consistently sampled. The resulting half core samples were believed representative of the mineralization with limited to no bias from preferential drill orientations.

Cut half core samples were placed into plastic sample bags with a sample ticket. The batch number, hole identity and sample numbers were marked on the bag with a permanent marker. Bags were stapled closed and samples per hole were packed into polyweave sacks labelled with the batch number for transportation to the preparation laboratory.

Sample preparation of diamond drilled core (2011 to 2013) used the following procedure:

- Sample preparation was carried out at the Intertek Genalysis laboratory in Ndola, Zambia.
- Samples were dried at 100°C +/- 20°C
- The drill core was crushed to sub-10 mm through a jaw crusher.

- All samples above 3 kg were riffle split to obtain a sample weight between 0.6 kg to 1.2 kg.
- The split sample was pulverised using the Essa LM2 pulveriser so that more than 85% of the material passed 75micron.
- Approximately 150 g of pulp was packed into a pulp envelope and sent for analyses.

The infill diamond drill programme carried out between 2018-2019 followed the below procedure:

- Samples were prepared by crushing the core or chips to 90% passing 2 mm using a jaw crusher.
- Crushed material was split through a single-tier splitter until a 1 kg sample mass was retrieved.
- LM2 pulverisers were used to produce a pulp sample with 90% passing 106 microns.
- The pulverised samples were split through a single-tier splitter until approximately 250 g of pulp was retrieved.

The 250 g pulp sample was submitted to the analytical lab for analysis. All coarse and pulp rejects from the sample preparation process were returned to site.

Transportation and handling of samples was carried out by a representative of FQM or its designated contractors. It is the QP’s opinion that samples have been adequately prepared and secured from their source to destination.

11.2.3 Diamond core sample analysis

2011 to 2013 diamond drilled samples were assayed by Intertek Genalysis in Ndola, Zambia using a 25 g charge exposed to a three acid digest and followed by an AAS finish. Key elements analysed were nickel and copper. Given that the Genalysis laboratory in Ndola was not accredited, ten percent of the 2011 and 2012 samples (combination of pulp and coarse crush duplicates) were dispatched to ALS Chemex Perth (accredited) for umpire check assaying where a four acid digest followed by an ICP-OES finish was used. Results of the umpire checks (Figure 11-12) confirmed that the primary laboratory analytical accuracy was adequate for using these samples in this Mineral Resource estimate update.

For the 2018 to 2019 infill diamond drill programme, samples were assayed by ALS Chemex Kansanshi accredited laboratory, Zambia using a 50 g charge and subjected to a four acid digest with an AAS finish.

11.2.4 QAQC procedures

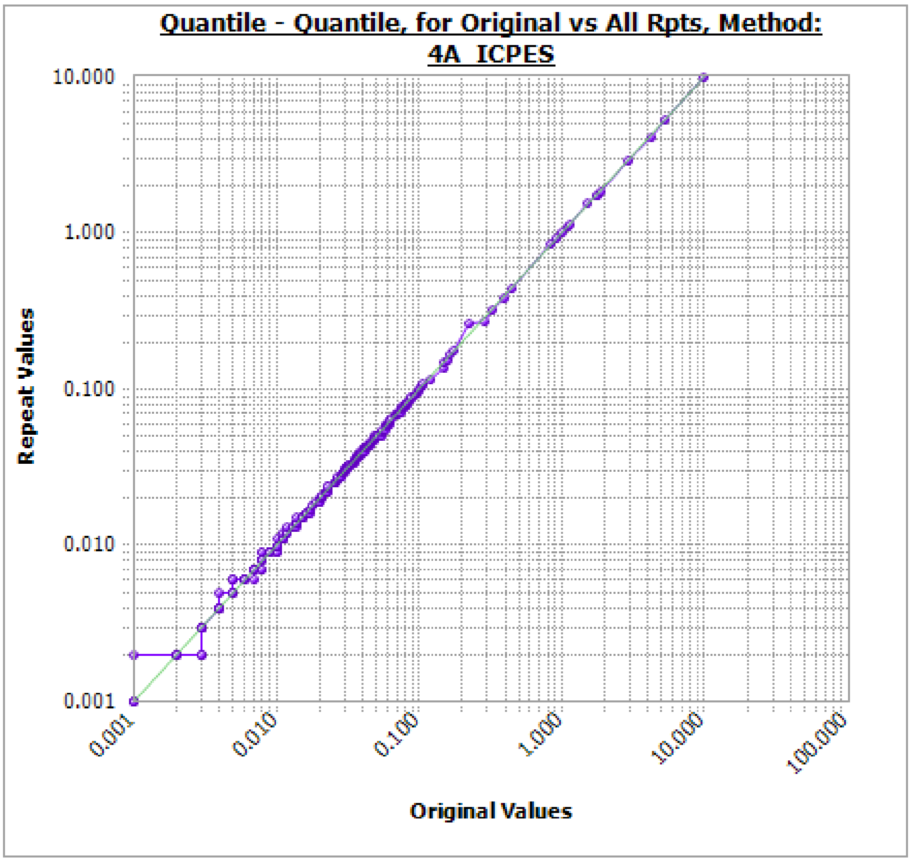
11.2.4.1 2011 to 2013 diamond drill programme

CRM’s, field duplicates, and blank samples were inserted at a rate of 1 in 50 during the core sampling programme. Returned CRM analysis demonstrate that the laboratory had acceptable accuracy with most values within three standard deviations of the certified value (Figure 11-15 and Figure 11-16).

There was evidence for some mislabelling of CRM sample inserts. Mislabelling of CRM sample inserts was rectified by using a coloured sticker system which reduced the degree of mislabelling. Internal laboratory QAQC was completed using laboratory standards at a rate of 1 in 25 samples and their own blanks at a rate of 1 in 20.

FQM randomly selected and submitted 10% of samples to ALS Chemex in Perth, Australia as an umpire analysis (Figure 11-12). 64% of these check assays were from original sample pulps and 36% were re-split from bulk reject samples.

Figure 11-12 Q-Q plot for umpire check assays conducted at ALS Chemex Perth



Results of the field duplicates demonstrate adequate repeatability and precision (Figure 11-13).

Inserted blank samples demonstrate that contamination was contained during sample preparation (Figure 11-14).

11.2.4.2 2018-2019 diamond drill programme

CRM's, blank samples, field duplicates, coarse crush duplicates, and pulp duplicates were inserted every 20th sample.

The inserted CRM's demonstrate the laboratory has acceptable analytical accuracy, with very few results that suggest mislabelling (Figure 11-16). Most CRM assayed values were within two standard deviations of the certified values verifying accuracy of the primary laboratory except for AMIS0171 which reported blank values. The CRM AMIS0330 showed lower values than expected, while the rest of CRMs and duplicates performed correctly. The performance of this standard was under investigation with the analytical laboratory and the CRM manufacturer at the time of this report. Blank insert sample values showed isolated instances of contamination, whereas overall performance demonstrate that contamination was adequately controlled during sample preparation (Figure 11-14).

Coarse crush duplicates and pulp duplicates for both drilling campaigns have high precision with more than 90% of sample pairs been within 10% of the original sample value (Figure 11-13).

All QAQC data was stored in the Datashed system with QAQC'R software been used for analysis of QAQC sample results.

Figure 11-13 MAPD plot for all Enterprise duplicates

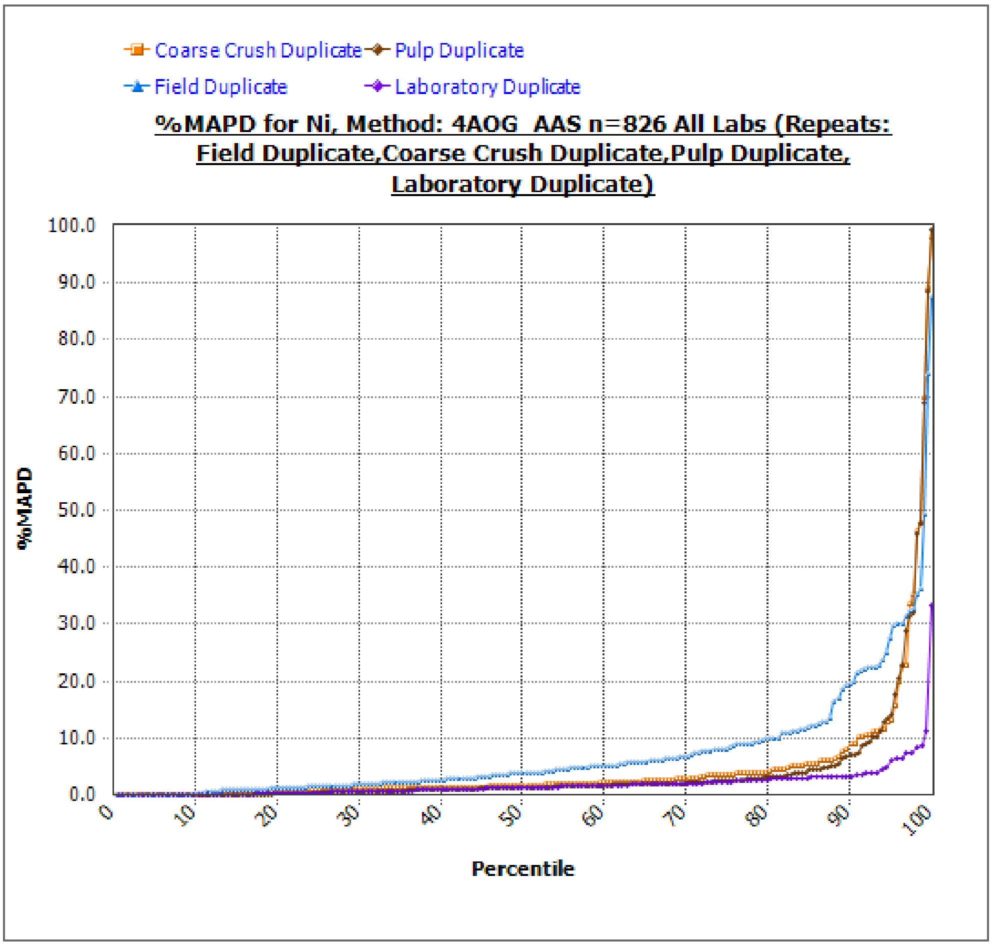


Figure 11-14 Blank assays control chart (2013-2019)

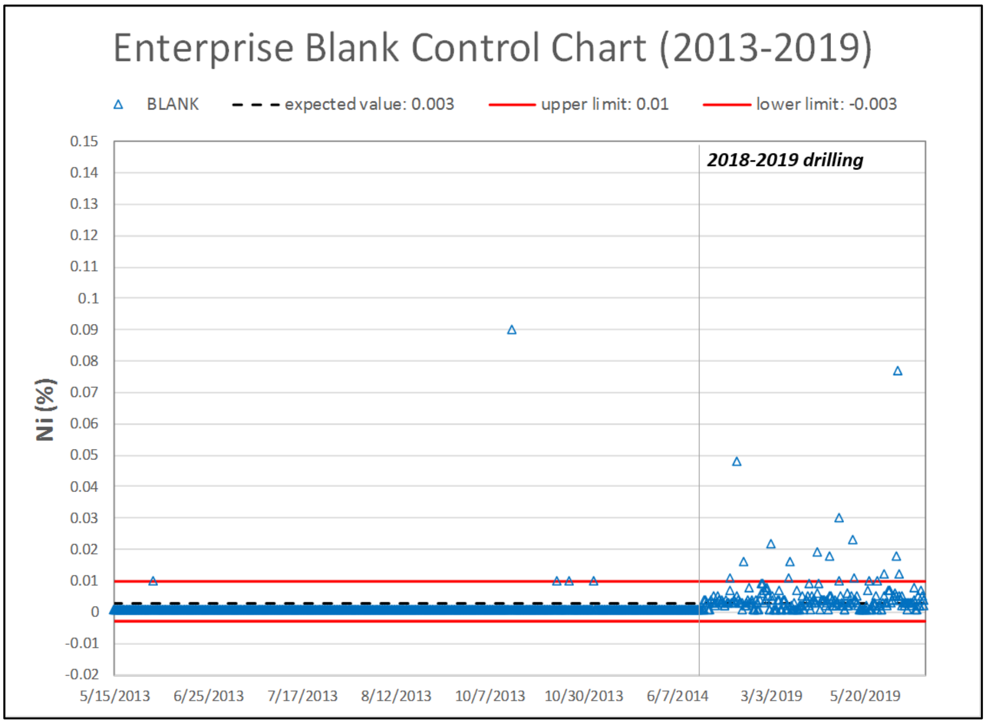


Figure 11-15 Enterprise CRMs control chart summary for 2010-2013 drilling campaign

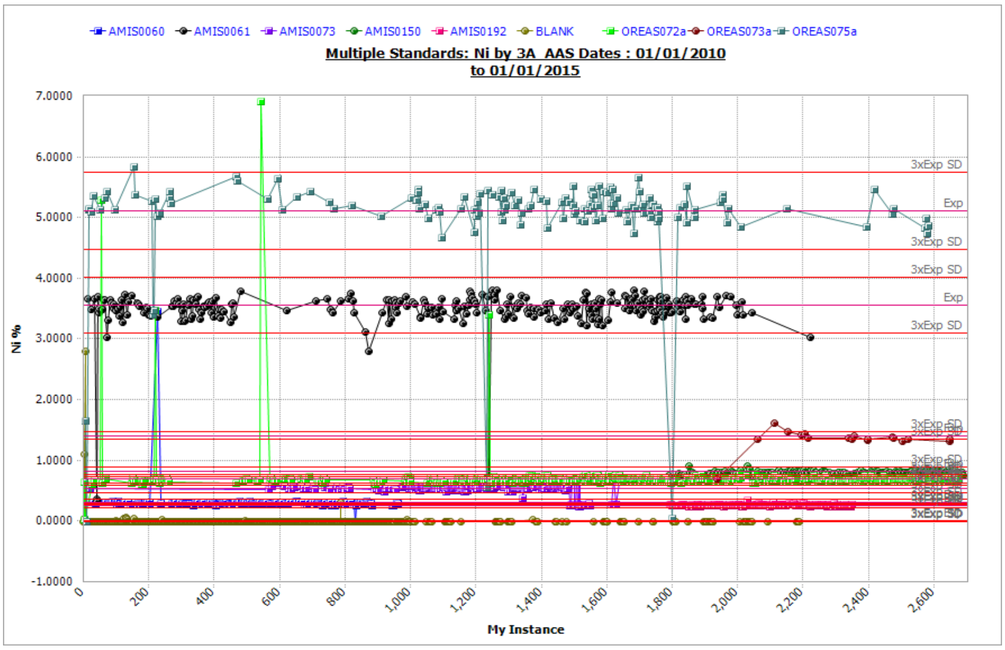
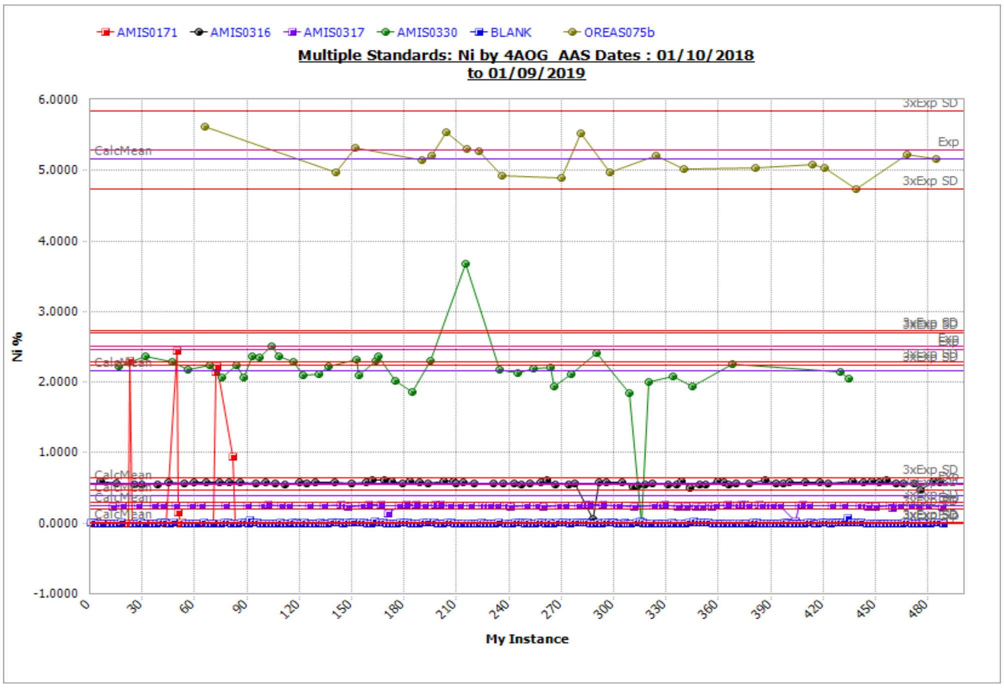


Figure 11-16 Enterprise CRMs control chart for 2018-2019 drilling campaign



11.2.5 QP's opinion

It is the QP's opinion that sample preparation, assays and analytical results have been satisfactory. Sample preparation and laboratory analysis were believed to have adequate controls. Enterprise assay results demonstrated acceptable QAQC with contamination contained and accurate and precise assay values. Sample values were believed to be representative of the prevailing mineralization and therefore adequate to be used in this Mineral Resource estimate.

ITEM 12 DATA VERIFICATION

12.1 Sentinel

David Gray (QP) has visited the Sentinel site on several occasions over the last six years, with the most recent visit in August 2019. Carmelo Gomez has worked at Sentinel since March 2016. Both Mr. Gray and Mr. Gomez have gained good familiarity and confidence in the available diamond and RC drilling data together with their sampling and data management processes as well as the geology models and mineralization as exposed in the pit.

The following verifications were completed:

- Diamond drillhole collar coordinates were verified through visual checks against the Lidar surface topography and were verified in the field against the database records with a handheld GPS.
- Database sample data was visually inspected against the corresponding samples taken from remaining stored drill core.
- Grade control RC drilling data was subjected to daily, weekly and monthly data checks and validations, by qualified geologists and under the supervision of the QP.
- Database validations included:
    - checks for outlier values of relevant data fields
    - overlaps and duplicates in sample and logged data
    - that database assay data reflects original assay certificates
    - checks on the relative magnitudes of downhole survey data.
- QAQC data was investigated together with the processes used for analyses and was verified as robust for assuring assay accuracy and precision with evidence for control on contamination.
- ALS Chemex Kansanshi lab facilities were recently audited by Carmelo Gomez and historically by qualified exploration geologists on a quarterly basis. No major deviations or flaws were noted.
- In-pit observations have served to verify the prevailing geology and its association with the different styles of mineralization as per logged data and 3D geology models.

12.1.1 QP's opinion

Drill hole sampling and geology logging data were verified from visual observation of remaining drill core and chips, core sampling markings, field inspections to the RC rigs and sample storage areas. Mineralization and geology as modelled for this resource estimate, was verified from mapped data and visual observations of in-pit exposures. The QP believes that the available data used for the Sentinel Mineral Resource estimate is of adequate quality to represent its mineralization.

12.2 Enterprise

David Gray (QP) has visited the site of Enterprise deposit on several occasions over the last 6 years, with his most recent visit in August 2019. During these visits Mr. Gray has gained confidence in the available data, geology and prevailing mineralization. Carmelo Gomez has visited the Enterprise deposit site on several occasions over the last three years, with special attention during the latest drilling campaign in 2018 to 2019. During this time, Mr Gomez has gained confidence in the available data, geology and prevailing mineralization. The following verifications were completed:

- Diamond drillhole collar coordinates were verified through visual checks against the Lidar surface topography and were verified in the field against the database records with a handheld GPS.
- Database sample data was visually inspected against the corresponding samples taken from remaining stored drill core.
  - Database validations included:
  - checks for outlier values of relevant data fields
  - overlaps and duplicates in sample and logged data
  - that assay data in the database reflects the data in the original assay certificates
  - checks on the relative magnitudes of downhole survey data.
- QAQC data was investigated together with the processes used for analyses and was verified as robust for assuring assay accuracy and precision with evidence for control of contamination.
- ALS Chemex Kansanshi lab facilities were recently audited by Mr. Gomez and historically by the qualified exploration geologists on a quarterly basis. No major deviations or flaws were noted.
- Mapping of limited in-pit exposure thus far supports 3D geology models.

12.2.1 QP's opinion

Review of the QAQC results, data management procedure and database security supports quality data representative of in-situ mineralization. In addition, the QP, David Gray with Carmelo Gomez have completed several phases of visual inspections of drill core, drill collars and outcropping geology. The QPs are confident that the data and information available are appropriate and of a suitable standard for estimation of the Enterprise deposit Mineral Resource.

ITEM 13 MINERAL PROCESSING AND METALLURGICAL TESTING

The following information has been recompiled from the 2015 Technical Report (FQM, May 2015) and updated by Andrew Briggs (QP) in November 2019.

13.1 Sentinel

From the initial understanding of the mineralogy of the Sentinel deposit, the processing flowsheet and metallurgical designs showed that a typical copper concentrator flowsheet, treating predominantly chalcopyrite ores similar to those encountered elsewhere on the Zambian Copperbelt, would be suitable. The final flowsheet therefore comprised:

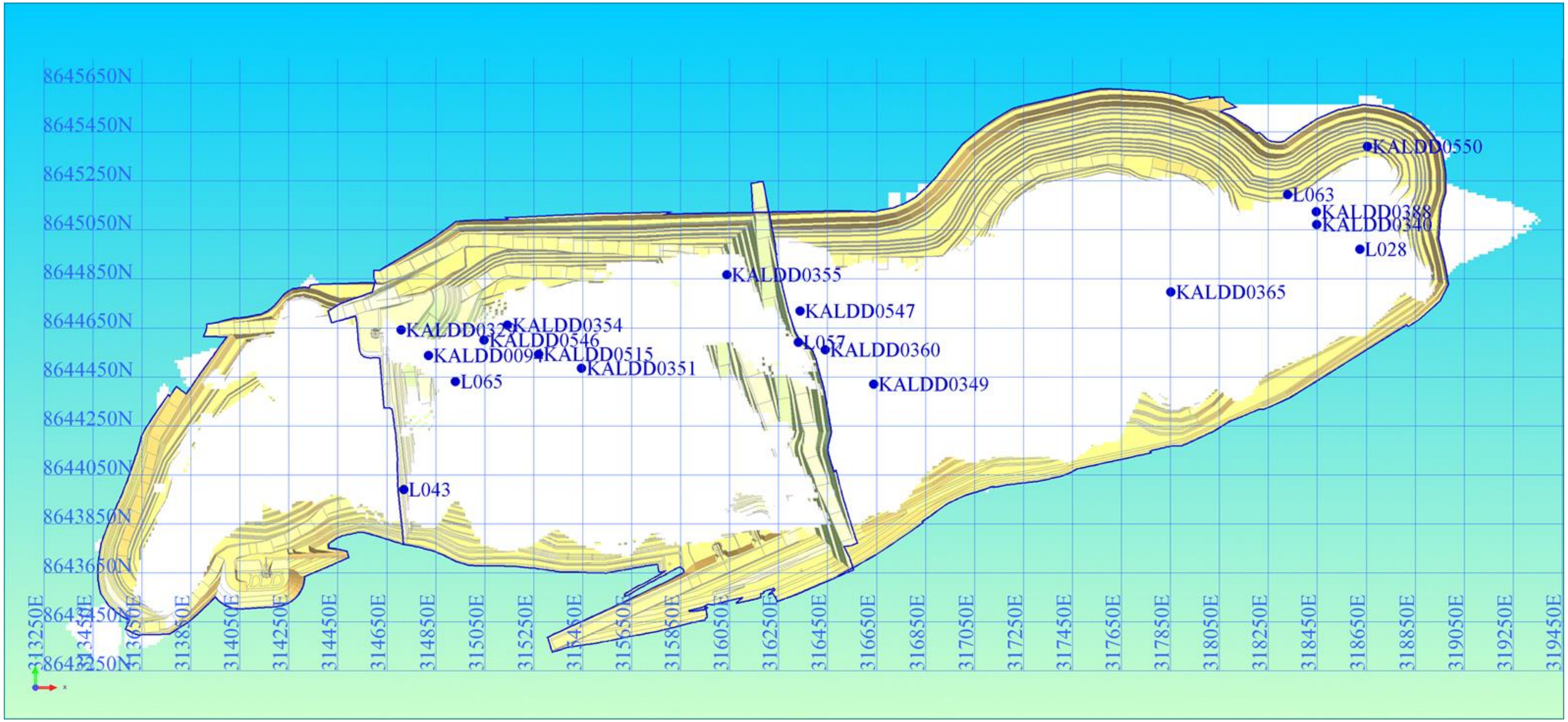
- crushing, conveying and ore stockpiling (making use of in-pit crushing and conveying)
- secondary crushing
- crushed ore stockpile reclaim and milling in a SABC (SAG/Ball/Pebble crushing) circuit
- flash flotation on a bleed stream from cyclone underflow
- rougher, scavenger and cleaner flotation
- concentrate handling
- tailings disposal
- reagent mixing, storage and distribution
- water and power supply
- other services

Prior to detail design of the processing facilities, metallurgical test work was carried out in three phases:

1. The first phase of the test work was conducted at the Company's Kansanshi operation and was scoping in nature.
2. The second phase of the test work was conducted on whole core recovered from holes drilled in six locations across the orebody, ie holes KALDD0329, KALDD0354, KALDD0360, KALDD0349, KALDD 0365 and KALDD0340, where shown on Figure 13-1. Holes KALDD0329 and KALDD0354 were located within the initial phase mining limits. Each hole was subdivided into four sampling intervals by depth, providing 24 samples in all for comminution and flotation test work to define the variability of performance with depth (varying between 50 and 290 m below surface) and location in the ore body. This work was performed by SGS Lakefield in Perth, beginning in May 2011, and continuing through to 2014.
3. A third phase of test work was completed in April 2013, on four samples of material obtained from the area of the pit where mining was expected to commence, ie from drillholes KALDD0094, KALDD351, KALDD0355 and KALDD0515, where shown on Figure 13-1. This work by JK Tech[1] was to confirm comminution parameters for the first material to be treated in the plant during ramp up.

[1] JK Tech is a specialist technical laboratory associated with the University of Queensland in Australia.

Figure 13-1 Sentinel metallurgical test work, drillhole sample locations



Since start-up, routine test work has been carried out and is ongoing, using Base Metal Laboratories (BML) in Kamloops, British Columbia, Canada. This test work has assisted in understanding recovery issues associated with non-primary sulphide ore in the near surface mining horizons (ie, ore type OTB), and to optimise current operations with respect to reagent types and addition rates.

13.1.1 Mineralogy

Early studies indicated that mineraliZation was almost all sulphide in nature with oxide derivatives generally insignificant and depth limited to 10 m to 20 m below surface, in the weathering profile. However, in practice, the weathering profile was more varied than anticipated and the first year's plant feed comprised mainly chalcocite mineralization, with some oxide and native copper, plus tarnished chalcopyrite. There are extensive zones of pyrite, particularly in the hanging wall, and pyrrhotite and carbon occur throughout the ore body.

Primary sulphide mineralization appears to be relatively late (ie, syn-metamorphic/deformation) with relatively coarse chalcopyrite intergrown with pyrite and/or pyrrhotite in blebs, overgrowths and small veinlets and stringers that can locally be seen to cut foliation and porphyroblasts.

The copper sulphides are focussed in a relatively carbon-rich zone in the centre of the phyllite hosted orebody. The host phyllite is very fine grained, with quartz, muscovite, biotite and iron sulphides present. Lesser nickel mineralization appears to be mostly limited to the north-eastern end of the deposit, outside of the current ultimate pit extents.

13.1.2 Metallurgical test work summary

13.1.2.1 Test work conclusions

The flotation test work programmes provided the following design parameters:

- primary grind of 80% passing 212 µm
- starch addition 300 to 500 g/t
- SIBX (sodium isobutyl xanthate) as a collector at 60 g/t
- pH in roughers of 7 – 8 (natural)
- pH in cleaners of 11 – 12
- rougher/scavenger flotation time 20 minutes
- primary sulphide recovery of 92%
- concentrate grade of 24%
- non-primary sulphide recovery of 70%

At the coarse primary grind of 80% passing 212 µm, liberation of chalcopyrite particles was virtually complete. Evaluation of concentrate regrind showed no benefit and as such was not included in the final flowsheet. Test work to reduce the requirement for high starch consumptions to depress gangue was not effective. Pre-flotation of easily floatable gangue was tested but did not reduce starch addition rates nor improve concentrate grades, and hence this step was not included in the final flowsheet for the processing of primary sulphides.

The grinding test work provided the following design parameters:

- rod mill work index (RWi) 12.3 to 16.1 kwh/t
- ball mill work index (BWi) 12.6 to 17.1 kwh/t
- A*b parameter of 47.3 to 39.8

Milling throughput rates for the various samples were modelled by Orway Mineral Consultants (OMC) using the actual mills purchased for Sentinel. Rates were evaluated for a number of different scenarios for optimising mill throughput for the tougher ores. It was concluded that maximum throughput could be achieved if the quantity of plus 130 mm in the mill feed could be minimised through secondary crushing, and if the SAG mill pebbles were crushed before return to the mill. This led to the inclusion of partial secondary crushing and pebble crushing circuits within the Sentinel comminution flowsheet.

13.1.2.2 Variability tests

Batch flotation work was carried out on all samples of primary ore (four per hole) at the optimised conditions, to provide an indication of variability of flotation response.

The results indicated that in general, the flotation of primary ore at Sentinel provides high copper recoveries into relatively high grade concentrates. Ten samples gave 90% Cu recovery in the first roughing stage, and eight samples produced a first stage concentrate with a grade of greater than 25% Cu.

Only five samples had a recovery of less than 95% Cu after six stages of roughing-scavenging in the laboratory, with the poorest recovery being 91.5% from a sample with a head grade of 0.75% Cu.

Recovery efficiencies and concentrate grades do not appear to be related to sample head grades; thus mass pull and concentrate grades do have a strong correlation. The lowest grade sample tested still provided 95.8% recovery after six rougher-scavenger stages, and produced a 20% Cu concentrate from the first stage, albeit at very low mass pulls.

The conclusion from these tests is that whilst the Sentinel primary ore does show some variability in terms of flotation response and final concentrate grade, the range of results for primary ores is restricted. However, the flotation response for the non-primary ores is not as consistent, recovery is lower, and the degree of variability decreases with depth. These ores showed typical recoveries in the range of 60% to 70% with increased variability in concentrate grade. These non-primary ores constitute a smaller proportion of the ore to be treated and with blending, their impact will be limited.

The comminution test work showed a general increase in hardness with depth, ranging from an A*b parameter of around 47 down to 50 m, and 40 below 50 m depth.

13.1.3 Operational improvements

After the initial commissioning period, several enhancements were made to the circuit to improve operations. Changes to the reagents used in the plant were supported by test work completed by BML in Canada.

The presence of elevated carbon levels in the final concentrate resulted in a carbon removal circuit being included in the flowsheet. This circuit comprised desliming cyclones which removed the minus

10 µm carbon particles in a slimes fraction, with the cyclone underflow containing the majority of the copper species with a carbon content of less than 1% C.

In practice, however, unacceptable copper losses in the slimes fraction were experienced, and no method could be found to recover the copper without also recovering the carbon. The concentrate deslime circuit was therefore abandoned, and two flotation columns were installed as an extra stage of cleaning. This was done to improve concentrate grades and provide a means of washing entrained carbon from the concentrate.

A Jameson cell was also added to the circuit. This cell receives rougher concentrate from the first two rougher cells in each train (coarse fast floating chalcopyrite) and produces a final grade concentrate that is fed to the columns for carbon removal. The Jameson cell takes the load off the conventional cleaner cells, which can then be used more effectively on the finer, slower floating chalcopyrite recovered from the rougher scavenger flotation cells.

Starch was originally added to the circuit to depress the easily floatable gangue, and to reduce the quantity of material that would otherwise require treatment in cleaner flotation. It became clear, however, that the starch also had a depressing effect on pyrite and on chalcopyrite. In particular, it was found that the primary loss of copper in the rougher and cleaner scavenger tails was in the minus 20 µm size fraction, which normally has slow flotation characteristics, but the starch was depressing the kinetics even further.

Despite the original test work indicating that CMC (carboxymethylcellulose) was ineffective as a depressing agent at Sentinel, a specific CMC was found that worked well, and starch addition was replaced by this reagent. Similarly, process development work has resulted in a small lime addition to the mill, and on occasions addition of sodium hydrosulphide (NaHS) helps stabilise the flotation operation.

These changes have led to recoveries of 91% to 92% and improved rejection of pyrite from the final concentrates, and grades of plus 26% Cu.

13.1.4 Potential for deleterious elements

A full elemental analysis was provided by SGS for concentrate samples produced in locked cycle test work. These analyses indicated low levels of deleterious elements, listed as follows, and therefore were not expected to attract any treatment penalties:

- Arsenic, As 4 ppm
- Chloride, Cl typically 100 to 150 ppm
- Fluoride, F approx. 250 ppm
- Lead, Pb between 40 and 170 ppm
- Antimony, Sb 0.4 ppm
- Selenium, Se 3 to 6 ppm
- Thorium, Th 5 to 15 ppm
- Tin, Sn 40 to 100 ppm
- Strontium, Sr 4 to 20 ppm
- Uranium, U 6 to 20 ppm

- Zinc, Zn 200 to 900 ppm
- Nickel, Ni 400 to 500 ppm
- Cobalt, Co 400 to 900 ppm

Whilst concentrates produced at Sentinel have not been analysed for these elements, no treatment penalties have been incurred on concentrate sales to date.

13.1.5 Design metallurgical parameters

Based on all of the previous test work, and the operating results for the last two years, the following metallurgical parameters are recommended for mine planning and operating cost estimation purposes:

- mill throughput of 55 Mtpa (but able to be expanded to 62 Mtpa)
- primary grind of 80% passing 212 µm
- CMC addition 40 to 60 g/t
- Dithiocarbamates/dithiophosphinate collectors at 10 to 16 g/t
- pH in roughers of 7 – 8 (natural)
- pH in cleaners of 10 – 12
- concentrate grade = 26.5% Cu

Comprehensive models for sulphide recovery have been developed, based on the fraction of cyanide soluble copper in the feed, and the ratios of acid soluble copper to cyanide soluble copper and insoluble copper:

OTA RECOVERY = *PLNTREC=91.23205+(5.22*FINCUPCT)-(24.45*ASCUR)-(0.695/FINCUPCT) (where ASCUR = Acid Soluble Cu ratio, FINCUPCT=%TCu)*

OTB RECOVERY = *PLNTREC=1.483*(72.7-(9.46*0.88*CNCUR)-(7.42*logn(0.88*CNCUR)))-44 (where CNCUR = Cyanide Soluble Cu ratio)*

These are first pass formulae for which there are a series of calculation adjustments made depending on the ASCu content. Top and bottom cuts are also applied.

Use of these models in the mine plan give an average recovery of copper for high grade primary (OTA feed type) and for low grade primary (WTA type feed) ores of 91.4% and 87.7%, respectively. Despite the original variability test work indicating non-primary recoveries in the range of 60% to 70%, the performance based recovery models now indicate 79.5% and 73.1% for the non-primary high grade (OTB feed type) and low grade (WTB feed type) ores, respectively.

13.1.6 Plant expansion to 62 Mtpa

To date, the two milling trains have not run at their maximum annual capacity due to ore supply constraints in the mine whilst expanding into multiple mining areas and horizons. In the first months of 2019, the plant had been running at average rates equivalent to 50.5 Mtpa with 100 hours per month across the two milling circuits attributed to ‘Stockpile Low’. There is also some capacity loss in the milling circuits when the feed rates are reduced to manage the stockpile levels, rather than the mills being stopped completely. In terms of downtime across the three primary crushers, ‘No Ore Supply’ represents about 450 hours per month or 20% of the available run time.

Mill throughput in 2018 was 48.7 Mt, with a mill running time of 81.5%. This translates to 54.6 Mtpa at the design run time (RT) of 91.3%, and 56.3 Mtpa at the targeted run time of 94%. Figures for the first 10 months of 2019 are 40.5 Mt at 84.1% RT, giving a throughput rate of 52.8 Mtpa at 91.3% RT, and 54.4 Mtpa at 94% RT.

There have, however, been short periods of high mill throughput rates on relatively soft ore. In the second half of February 2019 the plant ran at rates up to the equivalent of 60 Mtpa for a ten day period.

Initial comminution test work conducted on samples taken from six drill holes across the entire Sentinel orebody indicated that some of the ore was very hard and that the target 55 Mtpa throughput rates would not be achieved without some amount of secondary crushing. Subsequent test work and modelling using eight samples representative of the first three years of production indicated at least 60% of the feed should be secondary crushed to achieve the target throughput (Table 13-1).

Table 13-1 Modelled Sentinel production rates, inclusive of secondary crushing

Sample	Interval		Throughput (Mtpa) with Secondary Crushing (% of feed)			
	From (m)	To (m)	50%	60%	80%	100%
KALC 001	25	48	62.6	79.2	84.6	89.3
KALC 002	60	100	37.5	47.5	50.7	53.5
KALC 003	40	63	44.6	56.5	60.3	63.6
KALC 004	70	93	37.6	47.6	49.4	49.6
KALC 005	Combined		43.8	55.5	59.3	62.5
KALC 006	55	95	30.3	38.3	40.9	43.2
KALC 007	60	100	52.6	66.5	71.1	70.5
KALC 008	100	140	50.5	57.5	57.5	57.5
		Average	**44.9**	**56.1**	**59.2**	**61.3**

Current throughputs are being achieved with three primary in-pit crushers, and a single secondary crusher. Table 13-1 indicates that additional secondary crushing would be required to enable 62 Mtpa to be achieved. A fourth primary crusher has been purchased, and is to be installed during 2021. This crusher will eliminate any bottlenecks as the pit develops in depth and towards the east.

A second secondary crusher has been installed, but to date the circuit has been operated with one crusher in standby mode. Conveyor capacities have been upgraded to allow both crushers to be run together as the throughput and ore hardness warrants. The feed to secondary crushing is unscalped. Designs have been produced to screen the secondary crusher feed to improve the efficiency of this circuit. This circuit would be installed if required to meet the higher mill throughput.

In general, the ore treated to date does not appear to be as hard as expected. This could be the result of optimised blasting, or the use of 140 mm balls in the SAG mill. Pebble generation rates are currently low, and mill throughput could be increased further by the installation of bigger openings in the discharge grates, generating more pebbles for pebble crushing.

With these options, the throughput target of 62 Mtpa appears to be achievable. Flotation capacity is considered adequate for 62 Mtpa through the rougher flotation, because these throughputs have been achieved for short periods in the past. A programme of debottlenecking the circuit has been undertaken, involving several pump upgrades. An additional two flotation columns will be installed to upgrade the cleaner circuit to handle the higher copper flows, and to further improve final concentrate grades.

13.2 Enterprise

13.2.1 Mineralogy

Nickel mineralization occurs as secondary sulphides to a depth of 40 m to 80 m, and below that as primary sulphides. Nickel sulphide mineralization includes vaesite, millerite, nickeliferous pyrite, bravoite and carrollite, and generally occurs as coarse-grained aggregates associated with irregular quartz-kyanite and pyrite. Rapid tarnishing occurs when sulphide surfaces are exposed, which could result in soluble Ni losses if surface stockpiling was contemplated. Secondary supergene nickel minerals include garnierite. Minor nickel mineralization is contained within silicates including chlorite and talc. Mineralization style is variable and found in veins, parallel to foliation and as disseminations. The majority of high grade nickel mineralization is hosted within the quartz-kyanite shale unit with lower grades found within the surrounding Mg-altered siltstone. Mg-alteration minerals include talc, magnesite and phlogopite.

13.2.2 Preliminary test work programme

Preliminary test work on initial samples from Enterprise was conducted at the Company's metallurgical laboratories at Kansanshi, between June and August 2010. The test work was conducted on a quarter core 10 kg sample of high grade (5% Ni), taken from drill hole KW1 at a depth of 63.8 m to 78 m. The location of this sample hole is in the Enterprise South West Pit area as shown in Figure 13-2.

Figure 13-3 is a cross section showing metallurgical sample holes within the initial mining area.

13.2.2.1 Scoping test work results and conclusions

Flotation test work was performed at grind sizes of 80% passing 190, 140, 110, 80, and 60 µm. After six stages (fourteen minutes in total) of nickel flotation, nickel recoveries were over 91%, with a concentrate grade of between 16% and 18% Ni. It should be emphasised that the high recovery and good concentrate grades were achieved from high grade samples containing about 5% Ni, which is not representative of the ore body.

This scoping test work demonstrated that the nickel minerals at Enterprise could be recovered into a high grade nickel concentrate with high recoveries. The work also identified the presence of talc in the ore, and the need to pre-float the talc prior to nickel recovery. Specific conclusions from the scoping test work were:

- nickel losses to the talc pre-float are grind dependant, ie losses increase as the grind size coarsens and may be due to poor liberation of the nickel minerals from the talc
- nickel losses in the nickel float are also grind dependant, ie losses increase at finer grinds, possibly due to slurry viscosity effects and slower kinetics for fines flotation
- the combination of the above effects indicated an optimum grind size of between 140 and 110 µm

13.2.2.2 Initial comminution work

Two samples were sent to JKTech Pty Ltd in April 2011 for drop weight test work, in order to characterise ore breakage and provide indications of the grinding circuit requirements for treating

Enterprise ore. The two samples, one shallow and one deep, resulted in A*b values of 64.1 (moderately soft ore) and 33.7 (hard ore) respectively.

Figure 13-2 Enterprise metallurgical test work, drillhole sample locations

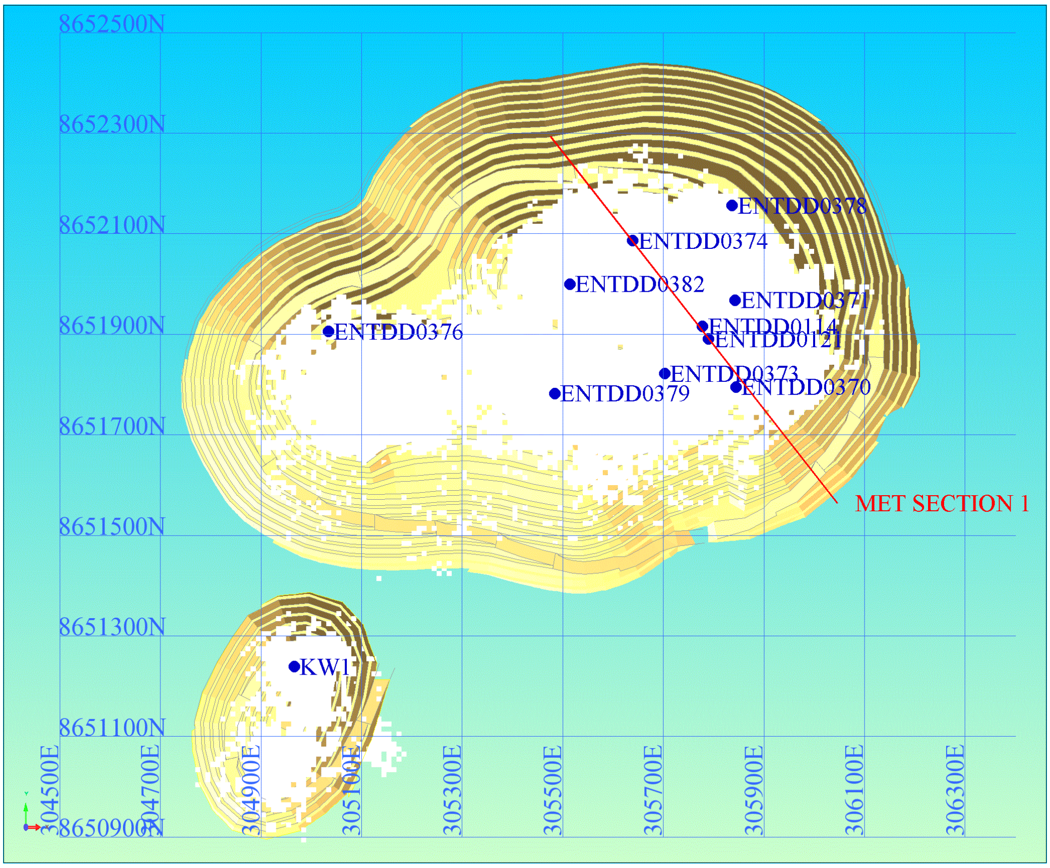
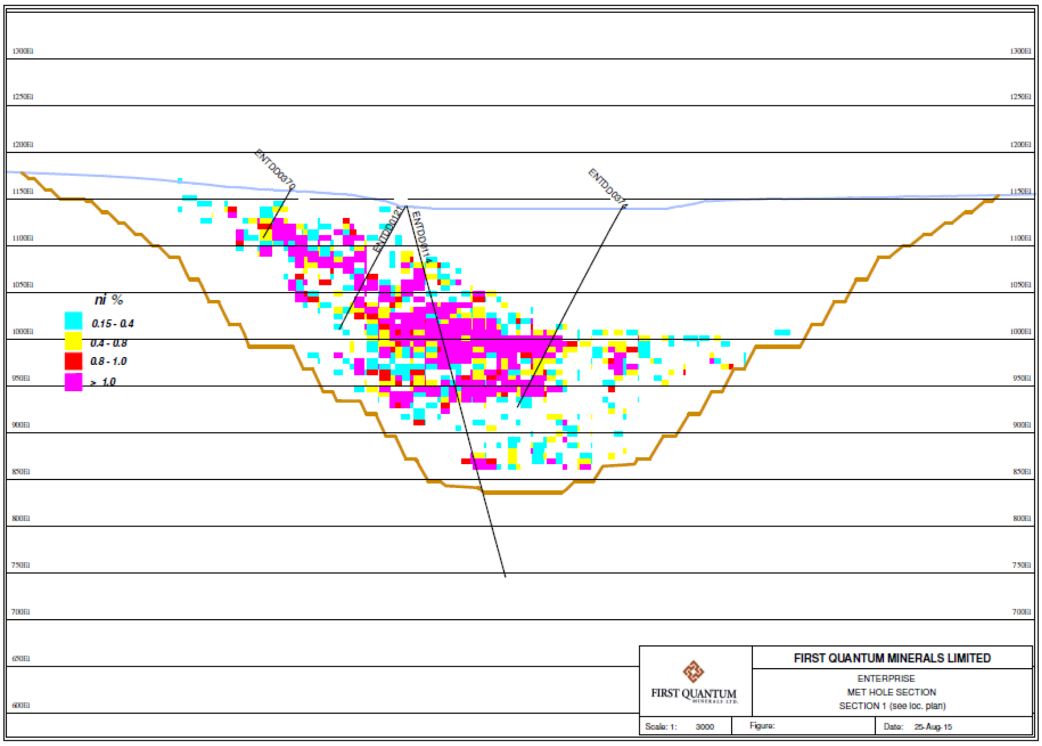


Figure 13-3 Enterprise metallurgical test work, cross-section through stage 1 and 2 pit



13.2.3 Second test work programme

The second test work programme was performed on samples from two metallurgical holes which were drilled in late 2011, from an area in the centre of the deposit. These holes were ENTDD0114, and ENTDD0121, where shown on Figure 13-2. The test work was carried out by SGS Lakefield (Perth) in 2012.

13.2.3.1 Comminution test work

The samples selected for comminution test work were from the top and bottom of the sampled intervals. These samples were composited into shallow and deep samples for both holes. The majority of these samples contained significant talc, carbonate and mica alteration and will therefore represent the soft Mg-enriched domains rather than the hard quartz-kyanite domains. Further samples and test work are proposed to assess the comminution of the harder domains.

The test work provided the following results:

- rod mill work index (RWi) 9.5 to 11.7 kwh/t
- ball mill work index (BWi) 11.1 to 17.5 kwh/t
- A*b parameter of 54 to 69 (indicating moderately soft ore)

13.2.3.2 Flotation test work

Flotation tests were performed under conditions similar to those used during the initial test work, using MIBC (methyl isobutyl carbinol) as the frother and SIBX (sodium isobutyl xanthate) as the collector.

The results indicate that a concentrate of about 17% Ni can be obtained, with almost 92% recovery using a pH of 9 in the cleaning circuit, with either no regrind, or regrind to 70 µm only. Running the cleaner circuit at a higher pH or a fine regrind size of 80% passing 53 or 45 µm results in increased losses of Ni to the cleaner tails.

As with the first test work programme, the samples tested were high grade samples with nickel grades generally between 1.74% and 4.4% Ni, and with two samples showing grades of 6.6% and 8.2% Ni. The high recoveries and concentrate grades should therefore be viewed with caution.

13.2.3.3 Variability test work

Batch pre-float and rougher flotation work was carried out on all mineralised samples from the two holes, for nine samples in all, to provide an indication of variability of flotation response. The mineralogy (from the geological logs) has been included in a table of test results, to try and link the minerals to the flotation response (refer to Table 13-2).

The results presented in this table are quite variable:

- Test number 159 showed high nickel losses to the pre-float concentrate, but still produced a high grade rougher concentrate. The sample had a high head grade of 8.23% Ni, but still produced a low rougher tail. The pre-float and rougher flotation cons contained 97.7% of the nickel in the feed.

Table 13-2 Enterprise test work, variability in flotation response

Test	Sample depth m	Mineralogy	Grades, % Ni		Pre Float Cons			Rougher Cons 1			Rougher Cons 6 (Cum)		
			Head (Calc)	Flot Tails	Grade % Ni	Recovery, % Ni	Recovery, % % Ni	Grade % Ni	Recovery, % % Ni	Recovery, % Mass	Grade % Ni	Recovery, % Ni	Recovery, % Mass
	ENT 0114												
98	111-118	Vaesite, Ni-Pyrite	1.76	0.04	4.58	3.0	1.17	17.30	92.5	9.42	14.00	95.0	12.0
159	129-144	Millerite, Ni-Pyrite	8.23	0.09	48.60	47.5	8.05	18.60	50.2	22.20	17.10	51.7	24.9
160	151-158	Millerite, Vaesite	3.29	0.15	0.73	1.4	6.09	18.80	91.9	16.10	15.30	95.3	20.6
161	163-166	Vaesite, Ni-Pyrite	3.98	0.23	0.52	1.7	12.90	16.80	82.6	19.60	15.10	94.7	25.1
162	168-174	Vaesite, Millerite, Ni-Pyrite	4.43	0.21	2.71	5.3	8.67	18.10	65.8	16.10	15.20	91.7	26.7
163	198-216	Ni-Pyrite, bit Millerite	3.02	0.19	0.41	0.9	6.41	19.40	82.9	12.90	14.20	94.6	20.1
164	282-291	Vaesite	1.74	0.13	0.53	1.9	6.27	15.80	88.4	9.75	14.00	92.1	11.5
	ENT 0121												
165	72-76	All 3 samples Ni-Pyrite with minor Vaesite	0.56	0.05	0.26	9.4	20.00	2.99	81.7	15.20	2.57	85.1	18.4
166	78-85		6.58	0.30	1.53	1.7	7.30	18.20	80.4	29.00	17.00	95.7	36.9
167	92-98		3.51	0.27	0.65	3.0	16.10	6.08	38.5	22.30	9.39	93.2	34.9

- Samples from ENTDD0121 did not respond to flotation as well as those from ENTDD0114 – except for Test 166, where the head grade was high (6.58% Ni).
- For samples from ENTDD0114, higher flotation tails appear to coincide with vaesite/Ni-pyrite mineralogy, though the high losses also coincide with high pre-float cons. Tests on samples from ENTDD0121 also show a similar relationship of Ni losses vs pre-float cons (note: vaesite is a minor mineral within sample ENTDD0121).

The majority of samples exhibited high mass pulls, despite the talc pre-float.

13.2.4 Third test work programme

Five master composites and one other sample were prepared from drilling samples taken in late 2012 (Table 13-3). Samples were selected spatially across the deposit, representing the three main mineralised lithologies (siltstone, shale and meta-carbonate) and the associated dominant Ni minerals (vaesite, Ni-pyrite and millerite). Samples were all whole HQ diameter core excluding sample MC-5, which was PQ diameter. The drillhole sample locations are shown on Figure 13-2.

Table 13-3 Master Composite criteria and sampling for third test work programme

Master Composite	Hole ENTDD #	Hole ENTDD twin	From (m)	To (m)	Lithology	Alteration	Mineralogy	Head assays Ni (%)	MgO (%)	Fe (%)	S (%)
1	0056	0371	86	101	siltstone	talc feldspar kyanite	vaesite, chalcop., pyrite, Ni-pyrite	0.71	29.30	3.23	2.04
2	0067	0378	124	134	shale	mica, talc silica kyanite	vaesite, chalcop., pyrite, Ni-pyrite	1.01	8.30	6.66	6.56
3	0115	0379	142	160	siltstone	talc, musc., hematite, carbonate	vaesite, Ni-pyrite garnierite	0.55	32.00	0.75	0.65
4	0120	0374	212	223	MCB	talc, dolomite, muscovite, phlogopite	millerite, chalcop., vaesite, Ni-pyrite	4.23	29.50	4.36	5.24
5	0041	0370	34	49	shale siltstone	FeOx, talc kyanite, silica	vaesite, garnierite, Mo, Ni-pyrite, fuchsite	1.67	23.70	6.29	1.98
Var 1	0042	0376	191	203	MBC, siltstone	mica, talc, silica, chlorite	Ni-pyrite, vaesite	0.30	33.40	1.25	0.49

All samples were submitted in 2013 to SGS Lakefield Perth, for bench-scale flotation tests, with further investigation into grind size, addition of guar and the primary cleaning of rougher concentrates.

13.2.4.1 Flotation test work

Findings from the flotation test work performed on the five Master Composites can be summarised as follows:

- Nickel loss/recovery to the pre float concentrate was variable, ranging between 2 to 6% for MC 2 and 3, about 10% for MC 1 and 3 and as high as 39.5% for MC 5. For MC 5, this led to poor Ni recoveries in rougher flotation.
- MC 2 (shale) is low in MgO and achieved high Ni recoveries (~ 90%), but low concentrate grades (~ 5% Ni). The mineralogy consists of pyrite and Ni pyrite so the iron (Fe) and S levels in the concentrate were high.

- MC 3 (siltstone) is a low Ni, Fe, S but high Mg composite. High S and Ni recoveries, but low Fe recovery to concentrates was achieved.
- The addition of guar in rougher flotation (at up to 500 g/t) had a significant impact in suppressing the flotation of magnesium minerals.

13.2.4.2 Comminution test work

Each of the composite samples in Table 13-3 were sent for full comminution test work. Test work was conducted at the SGS Lakefield lab in February 2013, excluding the JK drop weight work which was done by JKTech Pty Ltd.

The test work provided the following results:

- rod mill work index (RWi) 7.0 to 14.8 kwh/t
- ball mill work index (BWi) 9.7 to 16.8 kwh/t
- A*b parameter of 52 to 116 (indicating a relatively soft ore)

The toughest sample with high rod and ball work indices was Composite 2, which was a siltstone lithology with kyanite alteration. These results were used in the sizing of the comminution circuit.

13.2.5 Recent test work programme

In August 2019, four coarse rejects composite samples from prior Enterprise drilling campaigns (labelled Shale 1 of 2, Shale 2 of 2, Siltstone and Lith Carbonates) were subject to flotation test work in the Sentinel laboratory. This work was completed to generate concentrate for a mineralogical study for evaluation of concentrate treatment routes, and to evaluate the response of the Ni minerals to flotation given the possibility of aging whilst in storage. Test protocol was derived from the original test work performed on Enterprise samples, with changes to the depressant and xanthate used based on Sentinel operating experience.

A summary of the results is presented in Table 13-4 and Table 13-5.

Table 13-4 2019 test work programme, rougher test results

Composite name	Head grade (%Ni)	Rougher recoveries (%)					Rougher conc. grade (%)				Prefloat conc. recoveries (%)					Prefloat conc. grade (%)			
		MP	Ni	Cu	MgO	C	Ni	Cu	MgO	C	MP	Ni	Cu	MgO	C	Ni	Cu	MgO	C
Siltstone	1.95	21.17	15.91	17.79	25.18	16.91	1.47	0.08	11.76	0.27	14.86	12.57	11.42	18.74	7.74	1.65	0.08	12.46	0.18
Shale 1 of 2	3.55	34.23	90.80	78.99	58.98	46.66	9.43	0.10	12.09	1.35	7.03	1.11	6.48	18.44	2.48	0.56	0.04	18.39	0.35
Shale 2 of 2	2.08	26.26	73.01	78.21	31.58	20.98	5.79	0.11	11.87	0.65	15.31	1.98	7.21	29.52	0.94	0.27	0.01	19.03	0.05
Lith. carbonates	2.55	24.73	90.71	71.27	21.19	10.01	9.34	0.07	26.69	2.23	7.94	0.56	2.41	7.36	0.45	0.18	0.07	28.87	0.31

Table 13-5 2019 test work programme, cleaner test results

Composite name	Cleaner recoveries (%)				Final conc. grade (%)			
	Ni	Cu	MgO	C	Ni	Cu	MgO	C
Siltstone	83.64	43.45	43.70	24.06	17.37	0.81	8.09	5.01
Shale 1 of 2	69.25	74.87	21.84	18.79	13.88	0.51	9.87	0.30
Shale 2 of 2	26.78	11.07	18.83	7.61	19.50	0.48	6.75	0.48
Lith. carbonates	85.18	66.77	13.71	10.01	52.86	0.01	11.68	0.29

Head grades varied from 1.95% to 3.55% Ni; much higher than the 1% Ni expected in the plant feed.

The primary nickel mineralization in the samples was vaesite and it responded well to flotation under the un-optimised float conditions. The floatability of the nickel mineralization was fair to good given the amount of time the samples had been in storage.

As can be seen from the tabled results, recoveries and concentrate grades are variable, depending on the mineralogy and rock type, with recoveries ranging from 70% to 85% except for one of the shale samples, which exhibited very poor recoveries to final concentrate, albeit at a high concentrate grade of 19.5% Ni. A significant amount of nickel was recovered from this sample in the pre-float concentrate, with the talc.

The lith. carbonate sample performed extremely well, giving exceptional concentrate grades at 85% recovery.

Talc was high in the concentrate and this requires more work on effective depression since MgO is typically a penalty element in nickel smelters.

13.2.6 Representivity of metallurgical samples

Due to the volume of high grade samples that have been sent for flotation and grindability test work it is recommended that further test work includes a quantity of lower grade samples. Thus far, a good range of the mineralised lithologies and alteration styles have been represented in the test work but only for the higher grades. Furthermore, little work has been performed on near surface material in the area of the proposed starter pit, and in view of this, lower recoveries in the order of 60% could be expected for the non-primary ore. Whilst the non-primary ore constitutes a smaller proportion of the total ore to be treated, further test work is warranted.

For the comminution work, samples of only quartz-kyanite material with variable grades should be tested to ascertain the minimum A*b value (i.e. the toughest rocks) expected from the Enterprise deposit.

13.2.7 Further test work

A proposed additional test work programme has been defined for new metallurgical samples obtained during recent (2018/2019) resource and geotechnical drilling. This test work is being undertaken by BML in Canada, and whilst commenced in late 2019 the results will not be available until late 2020.

The testing is to be performed on eight composites intended to represent the mine feed during the initial starting phase (oxidised shale and siltstone) and material encountered later in the mining sequence. The material later in the mining sequence will have less oxidation (ie, it will be fresh shale and siltstone).

The parameters under investigation will be:

- primary grind size (32 rougher tests)
- investigate the effect of mill media type (mild vs high chrome; 16 tests)
- investigate specific talc/carbon depressants (CMC, guar gum, Dextrin; 40 rougher tests)
- investigate alternative collectors (SEX, DTP; 16 tests)
- Investigate alternative depressants/dispersants (SMBS, Calgon, TETA, etc, 32 rougher tests)

- cleaner tests with regrind and depressants (40 cleaner tests)
- contingency of eight rougher tests and eight cleaner tests to allow for interaction between parameters and to replicate tests
- locked cycle tests for each composite using the optimised scheme
- eight concentrate leaching tests to reduce MgO of the final concentrate

13.2.8 Potential for deleterious elements

The flotation test work tracked recoveries of Cu, Fe, S, and Mg. Platinum and palladium recoveries were not evaluated because the head grade of these species was very low, ie in the ppb range.

Copper was present in all samples in very low concentrations of typically 0.03% and was generally recovered to the rougher concentrate; but still at low values of 0.10% to 0.15%.

Mg levels in concentrates were typically between 5% and 12% Mg. More work is required on reducing this number, and future flotation work will investigate depressants such as guar, desliming and leaching to remove MgO.

Recoveries of other species of interest were:

- cobalt levels in rougher concentrate were about 1,000 ppm
- lead levels in concentrates were typically less than 100 ppm
- arsenic levels reached 200 ppm in one concentrate, but were typically lower than 50 ppm

13.2.9 Design metallurgical parameters

Based on the test work to date, the following metallurgical parameters are recommended for mine planning purposes:

- mill throughput of 4 Mtpa (ie, 500 tph)
- primary grind of 80% passing 150 µm
- primary sulphide recovery = 85%
- non-primary sulphide recovery = 60% (on the basis of minimal test work)
- concentrate grade = 14% to 16% Ni

ITEM 14 MINERAL RESOURCE ESTIMATES

14.1 Sentinel

14.1.1 Introduction

Mineral Resource estimates were completed for copper (Cu) and acid soluble copper (ASCu) which were interpolated into a 3-Dimensional (3D) geology block model using ordinary kriging and commercially available software i.e. Datamine Studio RM version 1.5.62.0 and Snowden Supervisor version 8.11. Estimates were completed in October 2019 by Carmelo Gomez and the site geologists of FQM under the supervision of David Gray (QP), FQM Group Mine and Resource Geologist. The project limits and coordinates were based upon the ARC1950 (Mean) 35s UTM coordinate system.

Estimates used an updated database which includes all drill hole sample assay results and the interpretation of a geological model that relates to the spatial distribution of copper mineralization. Interpolation parameters were based upon the geology, styles of mineralization, 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, confidence in the panel grade estimate and the reasonable prospects for eventual economic extraction. Reporting was guided by the Australasian JORC Code (JORC, 2012) and the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014).

The Sentinel Mineral Resource estimate was depleted for material mined as at 31st December 2019 and was reported using a 0.13% copper cut-off grade.

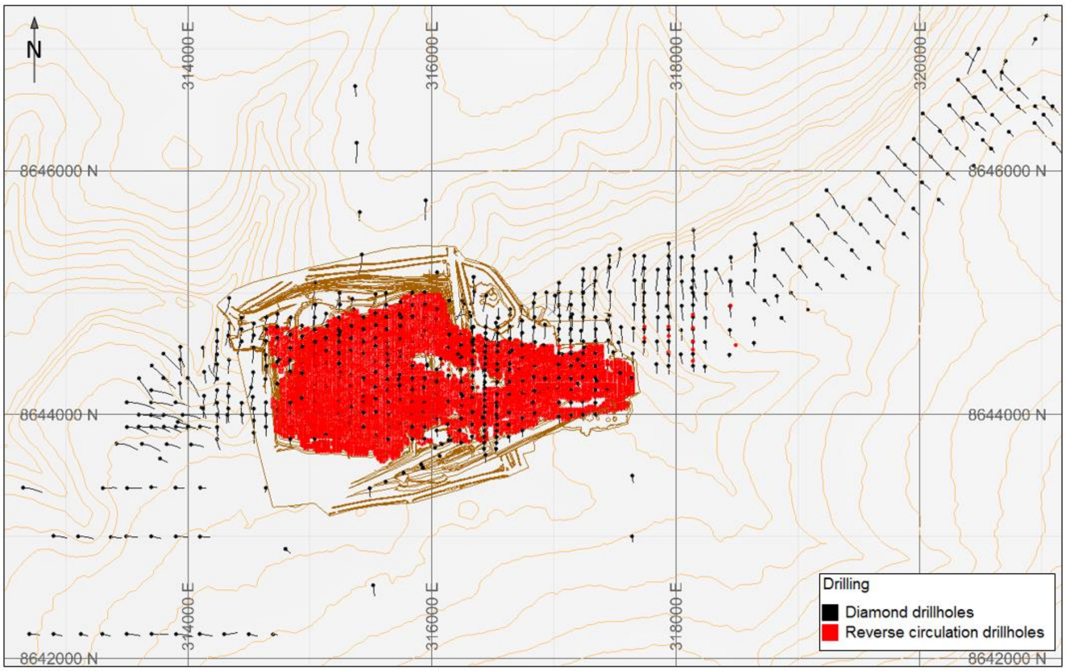
14.1.2 Available data

The upper limits of the 3D block model were defined by a pre-mining detailed topographic surface. The surveyed pit floor (31st Dec 2019) was used to define the upper limit of unmined Mineral Resources (Figure 14-1).

Only holes drilled by Kiwara PLC (2007 to 2010) and FQM (2010 to current date) were used in this Mineral Resource estimate. Historical holes drilled by RST and AA were deemed to be of sub-standard quality with missing data to support robust Mineral Resource estimates. In total, 636 diamond drilled holes for 210,974 m and 149,942 assayed samples and 17,553 reverse circulation drilled holes for 899,088 meters and 295,030 assayed samples were used in this estimate (Figure 14-1).

Diamond core samples from 2010 to 2013 drilling campaigns were analysed for copper using a four acid digest and AAS. Samples reporting Cu above 0.1% were analysed for AsCu using a single acid digest and AAS. The nominal sample length was 1 m. QAQC for the selected drillhole sample assay data was deemed to have acceptable levels of precision and accuracy for use in this Mineral Resource estimate. Diamond hole samples from the 2017 to 2018 campaign were analysed for total copper and sequential leachable copper.

Figure 14-1 A plan view of the Sentinel operations topography as at 31st Dec 2019 with diamond and reverse circulation drilled holes



Additionally, as a standard, drillhole core was logged for lithology, regolith, oxidation, alteration, mineralization and structure. Holes were also logged for geotechnical data, including core recovery and rock quality.

Reverse circulation samples were analysed for total copper, acid soluble copper and cyanide soluble copper since the start of drilling in 2014. All RC samples from August 2017 were analysed for total copper and sequential leachable copper.

Sample assay data, logged data, surveyed collars and downhole survey data were combined to generate a 3D drillhole trace using Datamine's standard de-surveying process. Prior to de-surveying, a series of data validations were completed in order to ensure representative data. Validations included:

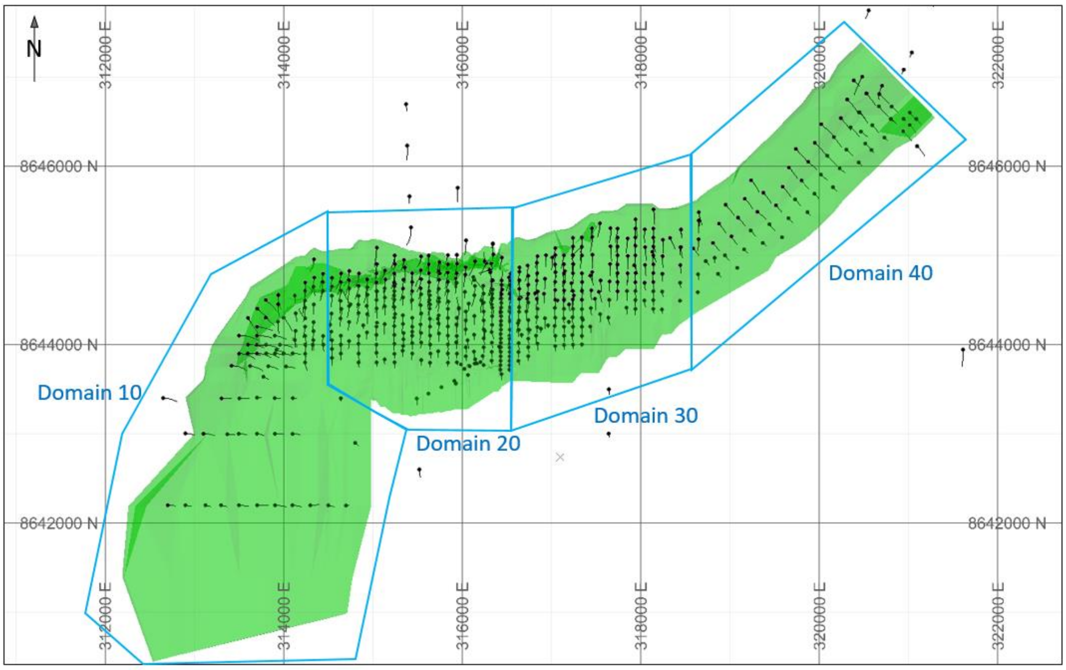
- visual checks of collar coordinates against topography
- investigation for overlaps, duplication and gaps in the logging and sampling data.
- assay values were checked for excessively high values.
- downhole survey data was assessed for excessive deviations.

De-surveyed drillhole data was used for visual and statistical analysis, geology modelling, spatial analysis of mineralized domains and estimation thereof.

14.1.3 Geological and mineralization models

The Sentinel deposit strikes approximately east west for about 11 km and, on plan, has a sinusoidal shape. Mineralized horizons dip 20 to 30^{o} to the north (Figure 14-2). The deposit is truncated in the northeast by the northwest trending Kalumbila fault and in the southwest by a series of smaller east west faults. The deposit was subdivided into four geologically distinct domains (Figure 14-2) with different mineralization characteristics.

Figure 14-2 A plan view illustrating the extents and sinusoidal shape of the Sentinel deposit mineralization host rock (green) together with diamond drillhole distribution and the four key geology domains



The dominant primary sulphide (chalcopyrite) copper mineralization is hosted by a structurally deformed and metamorphosed phyllite which has a strong foliation overprinting older remnant folding and sedimentary layering. Mineralization tends to be sub parallel to the foliation and occurs as overlying sheets at the deposit scale. Coarser grained sulphides, often with higher copper grades and thicker widths of mineralization, tend to be associated with the more deformed (folded) and altered lithology's. Higher grade copper zones are often surrounded by medium to lower grade mineralization (Figure 14-4).

3D geology modelling (Figure 14-4) was completed using vertical sections for the main rock types: carbonaceous phyllite and high carbonaceous phyllite; for weathering: base of saprolite and base of saprock; and for oxidation: base of refractory material and base of supergene alteration (secondary copper sulphides).

Weathering and oxidation ranges from a few metres below surface and is seldom deeper than 70 m. Depths are variable and dependent upon local lithology and prevailing structures (Figure 14-4). The uppermost saprolite horizon (<5m thick) is well-developed and is barren of copper mineralization (Figure 14-4). An underlying hematised and calcium poor saprock zone is similarly leached of copper. The underlying oxidised zone is defined by the base of partial oxidation where fresh (un-tarnished) chalcopyrite is visible and where carbonate minerals are not dissolved. The oxidised zone, also described as non-primary sulphide zone, is comprised mostly of tarnished chalcopyrite and some secondary chalcocite. These contact horizons were defined as sub horizontal surfaces with the aid of logging and sample assay data.

Copper mineralization volumes of the previous estimate were defined by manually digitized wireframes. However, this approach results in subjective volumes having irregular sharp sectional artefacts.

Copper mineralization volumes for this estimate were generated using probabilistic methods (categorical indicators), on a 6m x 6m x 6m block model. Three copper grades domains were chosen from visual inspection of the drill hole sample values as well as from review of the statistical distribution of sample copper values (Figure 14-3).

Figure 14-3 A log scale histogram of all sample copper percent values illustrating natural inflections for background waste, with very low grades (<0.20% Cu, light blue), low to medium grade mineralization (0.2-0.6% Cu, yellow) and high grade mineralization (>0.6% Cu, red)

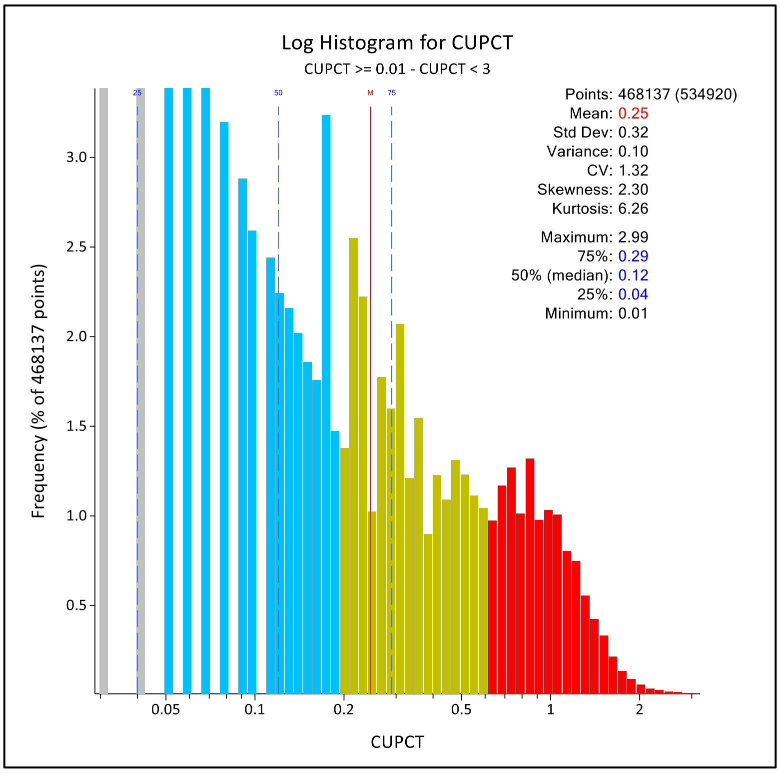
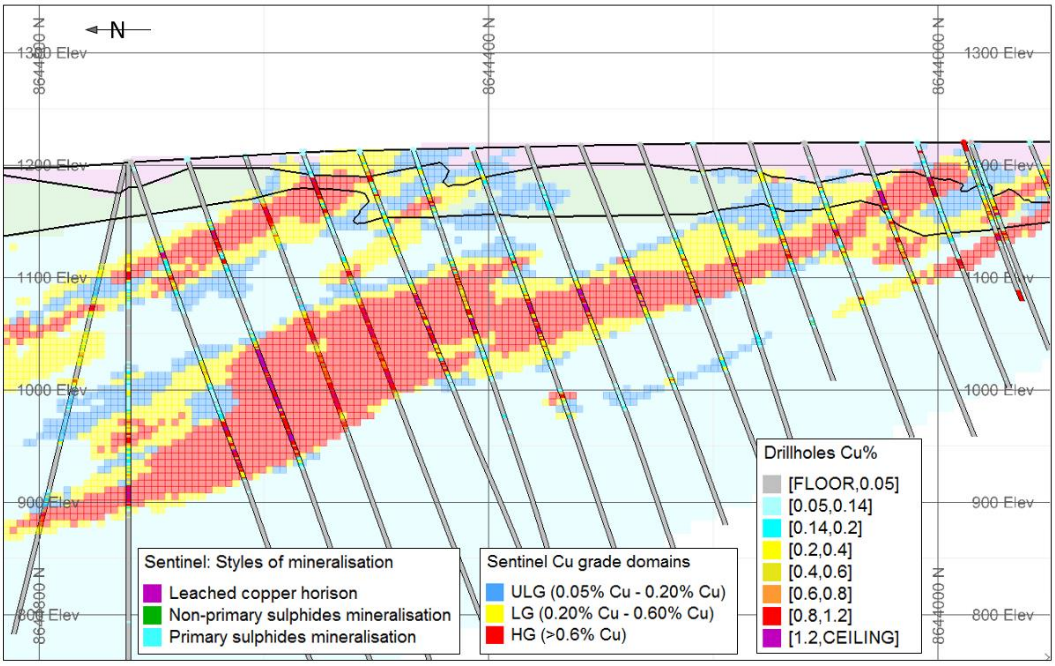
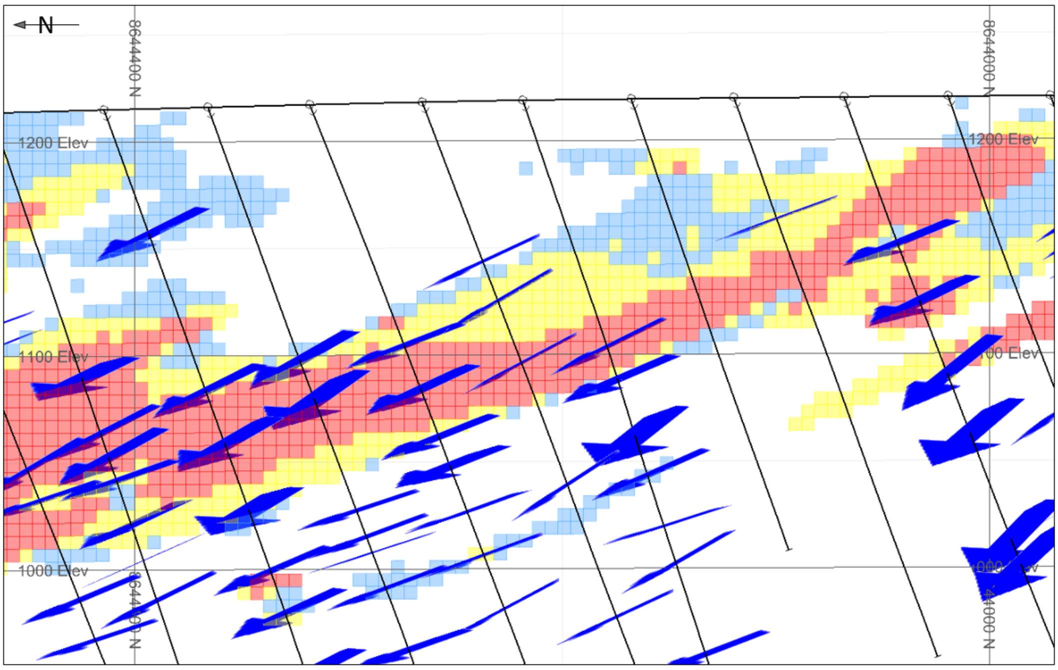


Figure 14-4 A vertical north south section illustrating copper grade domains (block cells) in relation to drillhole sample grades and the respective styles of mineralization (background colour)



The search orientations for estimating the mineralised volumes using categorical indicators were guided by dynamic anisotropy (DA) vectors. The DA vectors were assigned to each block according to the rock foliation data (Figure 14-5). The DA vectors capture foliation orientation changes as per the host rock folding.

Figure 14-5 Cross section showing copper mineralised grade volumes and the foliation orientation logged in the diamond drillhole core



Mineralized volumes, structural zones, weathering and oxidation surfaces define volumes of similar geology. These volumes/zones were numbered (Table 14-1) and combined to provide domains of mineralization for coding 3D drillhole samples. Coded drillhole samples were used for spatial analysis and selection of estimation parameters used during estimation.

Table 14-1 Domain numbers used for coding during estimation

Style of mineralisation	
300	Leached copper horison
200	Non-primary sulphides
100	Primary sulphides
Geological domain	
10	
20	
30	
40	
Copper grade domain	
1	Low grade (0.2% Cu - 0.6% Cu)
2	High grade (>0.6% Cu)
3	Ultra Low grade (0.05% Cu - 0.20% Cu)
0	Waste (<0.05% Cu)

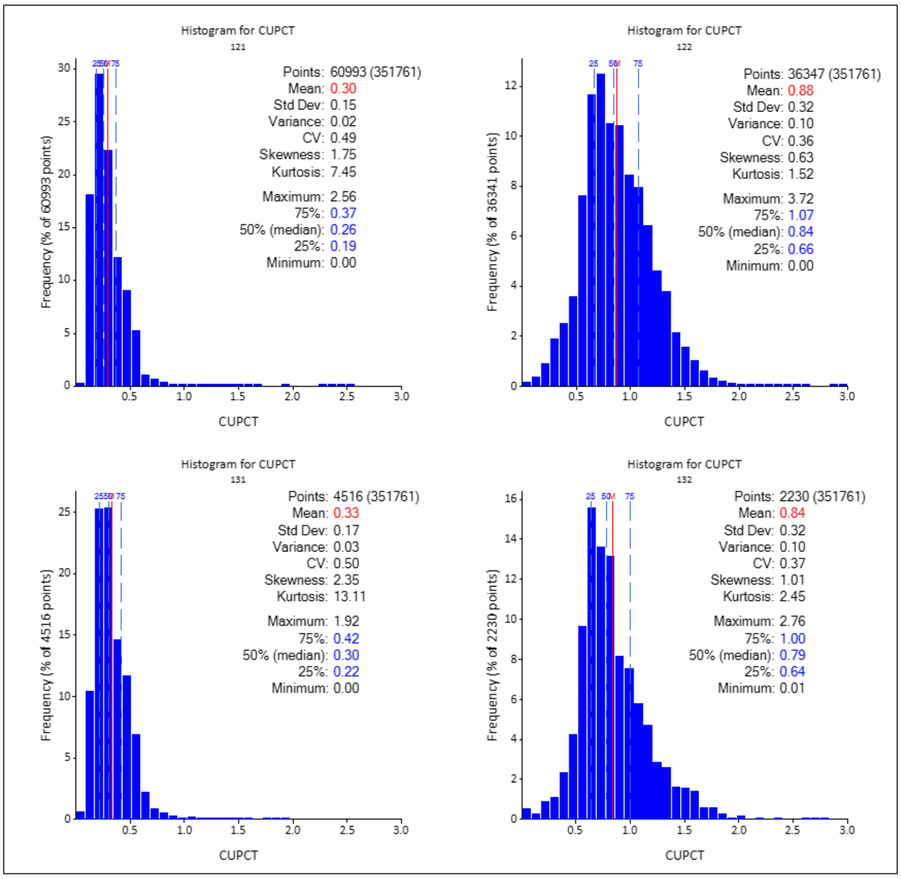
14.1.4 Sample compositing

De-surveyed drillhole samples were composited (combined) down the hole so as to reduce the effect of varying sample lengths on grade values and to support robust statistical and spatial analysis. Three meter downhole composite lengths were used to honour the 6 m smallest mining unit (SMU) bench height and to provide equal support from diamond and RC samples. The compositing routine combines consecutive samples down-the-hole so as not to cross domain boundaries and to ensure minimal metal loss.

14.1.5 Statistics

Statistical analysis of composite data per domain was completed using Snowden Supervisor software. Data distributions were investigated for mixed populations and excessive variability using histograms, log probability plots and descriptive statistics. Statistical analysis (Figure 14-6) highlight that selected domains were well defined with minimal mixing. Coefficients of variation (CV) were below 1, reducing the need for top cutting and that excessively high sample values would not distort block estimates in areas of low support. No top cutting was required for copper, acid soluble copper, cyanide soluble copper or residual copper (sequential leach) sample grades. Limited domain mixing and low data variability supported reasonable spatial analysis (variography) which in turn supports the use of ordinary kriging estimation.

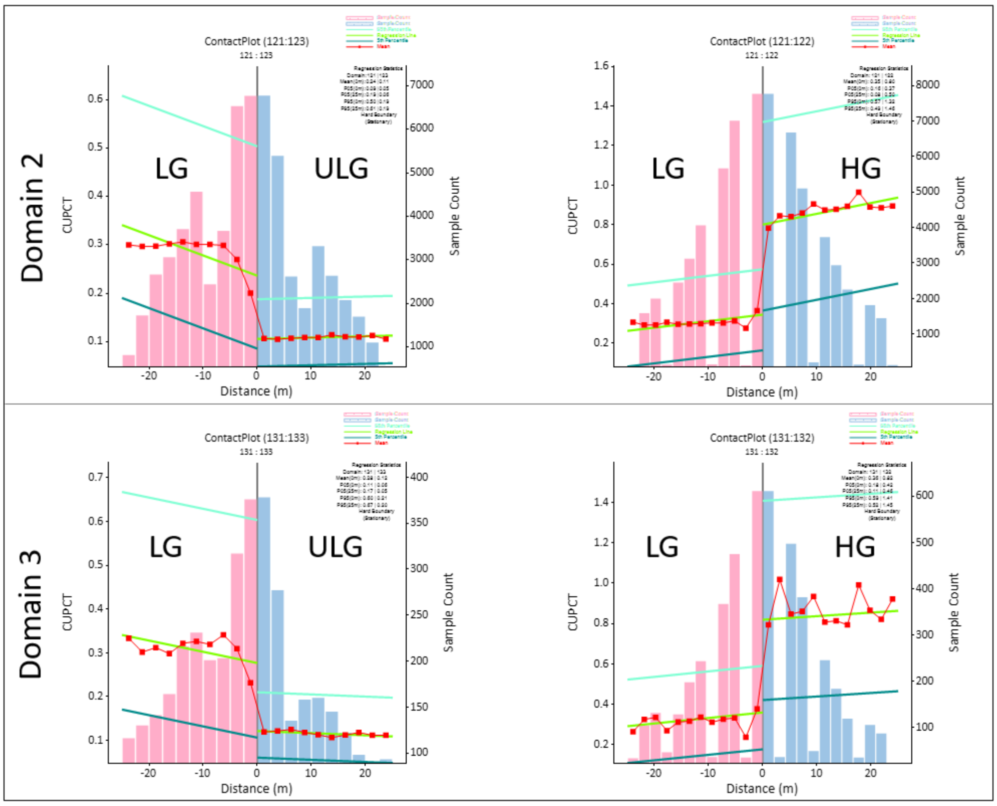
Figure 14-6 Histogram distributions for the main copper domains at Sentinel. (Top: geological domain 2; Bottom: geological domain 3. Left: low grade; Right: high grade)



14.1.6 Boundary analysis

Contact profiles were generated to evaluate copper grade changes across domain boundaries. Sharp grade changes across waste to low grade and low to high grade domains (Figure 14-7) existed. Hard boundaries were employed across other domain contacts in order to limit grade smoothing. Soft boundaries were used between oxide and sulphide domains for copper.

Figure 14-7 Boundary analysis between low and ultra-low, and low and high grade domains for geological domains 2 and 3



14.1.7 Spatial analysis

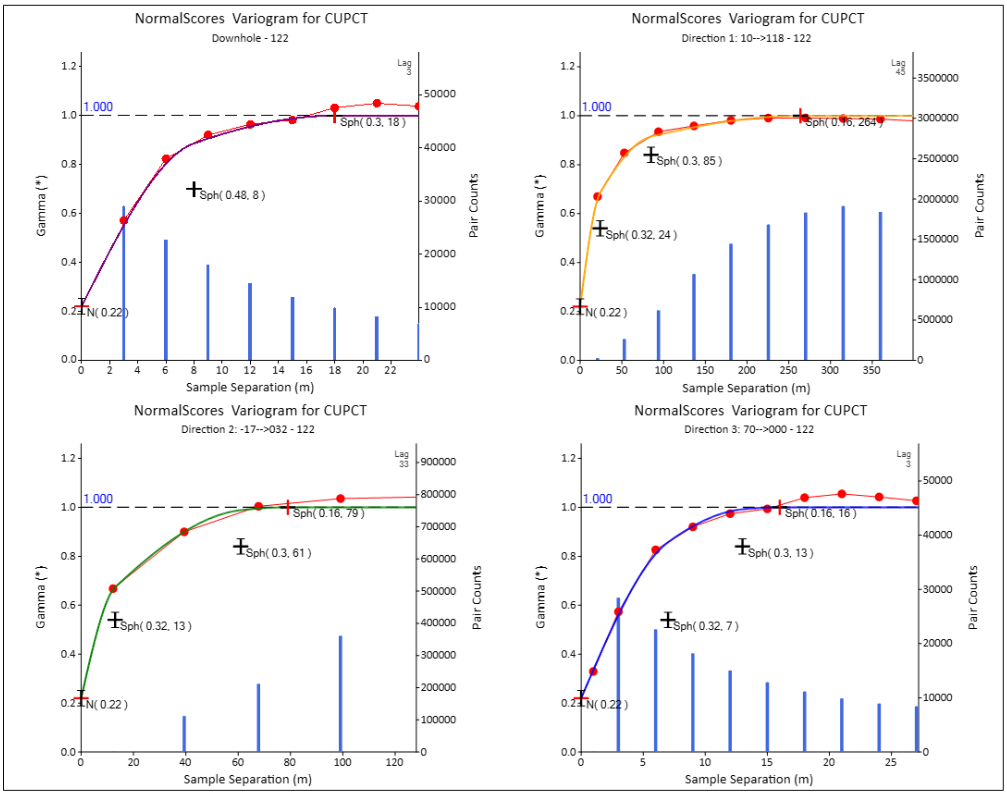
3D continuity of domain sample grades was modelled using spatial analysis and variography. The following method was applied:

- 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

- variograms were standardised to the population variance per domain in order to facilitate post-processing of the copper panel estimates to SMU estimates
- variogram models were used to guide search parameters and complete ordinary kriging estimation.

Variograms had nugget values of around 0.2 with well defined ranges of continuity, clear anisotropy and orientations. The dominant ranges of continuity (150 m to 170 m) were in the approximate north-west direction. An example of a high grade copper domain's variography (domain 121) is shown in Figure 14-8. Variogram models for fault block one and four, which have much wider drill grid spacing, had longer ranges of continuity and were not as easily defined. As a result, large volumes of these domains were deemed of lower confidence.

Figure 14-8 Variography for copper domain 122, a major high grade domain.



14.1.8 Kriging neighbourhood analysis

A kriging neighbourhood analysis (KNA) was undertaken to determine the optimal block size, sample selection ellipse dimensions, and the minimum and maximum number of samples to be used during grade estimation. KNA was completed using Snowden Supervisor software, which uses the modelled variograms and a series of estimates detailing the kriging efficiency and slope of regression values.

A block size of 60 mE by 36 mN by 12 mRL/bench height was selected as having optimal kriging efficiency and regression slope values. Similarly, a minimum of 12 samples and maximum of 28 samples were selected for supporting an optimal estimate into the 60 m by 36 m by 12 m parent block. Varying search ellipses were assessed with selected dimensions been similar to variogram ranges.

14.1.9 Block model

An empty 3D Datamine block model was prepared by using the optimal parent block size of 60 mE by 36 mN by 12 mRL, as per the kriging neighbourhood analysis. Block model origins were set at 312,500 mE, 864,2000 mN and 500 m elevation. The model was extended by 9,660 mE, 6400 mN and 900 m vertically. Blocks were sub-celled by ten times in the easting direction, six times in the northing direction and twice in the vertical dimension, in order to improve filling accuracy of wireframe volumes. Sub-celling only occurs along geology boundaries. The sub-celled block dimension is 6 mE by 6 mN by 6 mRL and is aligned with the 6 mE by 6 mN by 12 mRL smallest mining unit (SMU). The empty block model uses the same domain coding as the coded 3D sample data. The upper limits of the block model were constrained to the topography DTM surface as per the original Lidar survey.

14.1.10 Density assignments in the block model

In-situ dry density measurements from 19,518 samples have good deposit coverage. Sample density values were measured using weight of samples in air and then in water. There were sufficient numbers of samples per domain to minimise variability in measurements from moisture or pore spaces. Weathering had the strongest influence on density values, followed by rock type. Density data was analysed statistically, outlier values were removed, and density was estimated per weathering domain. Areas with insufficient samples for estimation were assigned mean values. The mean density values per weathered horizon is tabled in Table 14-2.

Table 14-2 Average density data

Weathering	Rock	Mean density
Saprolite	Phyllite/Schist	1.90
Saprock	Phyllite	2.67
	Schist	2.70
Fresh	Phyllite	2.79
	Schist	2.77

14.1.11 Ordinary kriging estimation

Domain copper, acid soluble copper, cyanide soluble copper and residual copper grades were estimated into the parent blocks using ordinary kriging (OK). OK was deemed an appropriate estimation technique of the Sentinel domains of mineralization owing to the near normal distribution of sample copper values and the limited evidence for domain grade population mixing. Kriging estimation parameters were based upon variography, KNA, geological continuity and drill grid spacing. Parent block estimates used a discretization of 6 (X points) by 3 (Y points) by 2 (Z points) so as to better represent block dimensions.

Most blocks were estimated within the first search ellipse. Blocks in areas with wider spaced samples required a second search ellipse, double the first. A third search pass was required for estimating blocks in areas remote of regular grids of close spaced data.

Post processing of parent block estimates was completed in order to provide estimates at the scale of mining i.e. the SMU. Localised uniform conditioned (LUC) copper estimates were determined per parent block's sub cell by using the estimated panel grade. Uniform conditioning provided the proportions of parent block sub cells above a range of cut-off grades. LUC then determines a grade per sub cell within the parent block, while maintaining metal content of the parent. The LUC model was re-blocked to the SMU block dimension of 6 mE by 6 mN by 12 mRL. LUC improves the representation of grade and tonnages that may be expected during mining. The LUC sub cells, re-blocked to the SMU block dimension, honours the mining bench height and presents a degree of mixing (dilution and loss) between grades and material types. This information is of value to mining and processing particularly for the transitional areas.

Post processed LUC grades were validated as follows:

- visual comparisons of drillholes, OK panel grades and LUC block grades
- confirming that metal correlations from LUC blocks compare to those of the drillhole data
- checking that contained copper metal, at zero cut-off grade, was the same for panel and LUC models.

14.1.12 Waste model

An estimate of waste grades was completed in order to capture erratic mineralization not included during estimation of mineralized domains. Grades were estimated into waste volumes using a single pass search ellipse (as per variography ranges and anisotropy) in order to avoid grade smoothing.

14.1.13 Model validation

Copper and acid soluble copper grade estimates reflect input data and were considered locally representative of mineralization. Northing, easting and vertical swath plot slices were used to validate block estimates against input data (Figure 14-9, Figure 14-10 and Figure 14-9). In addition, block estimates were visually validated with input data along cross sections as well as by comparing the mean values per domain.

Figure 14-9 An example of Swath plot validation slices in northing, easting and vertical orientations

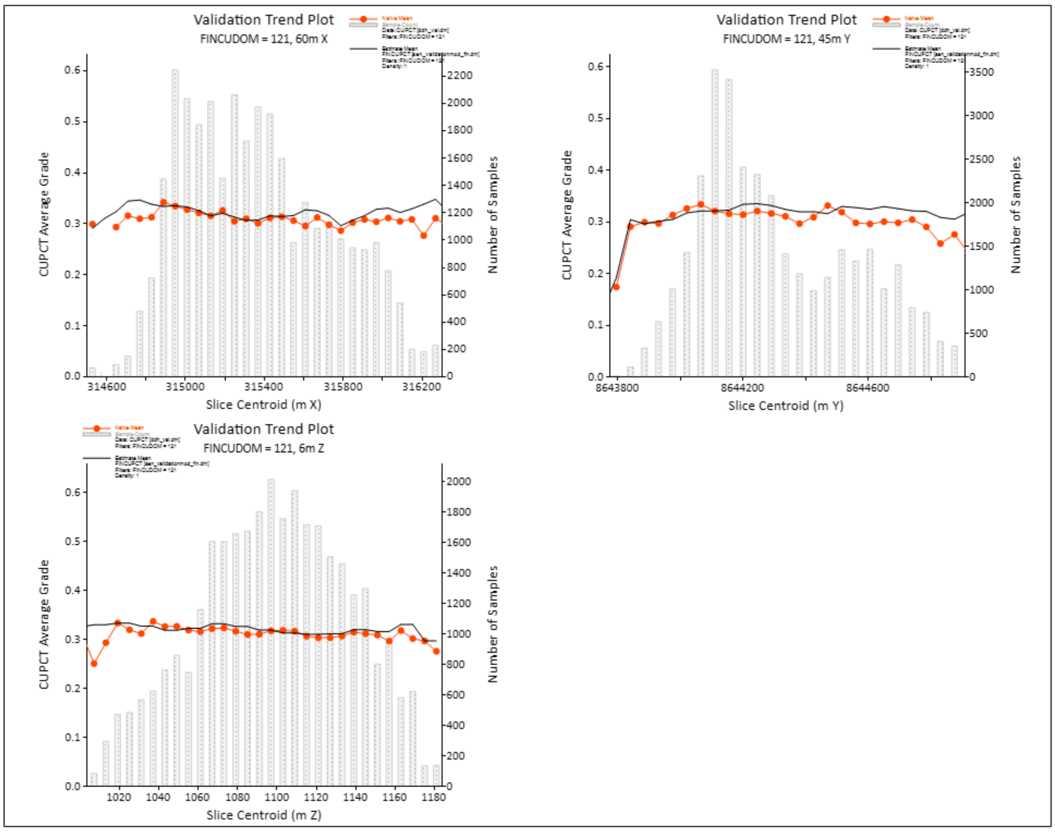
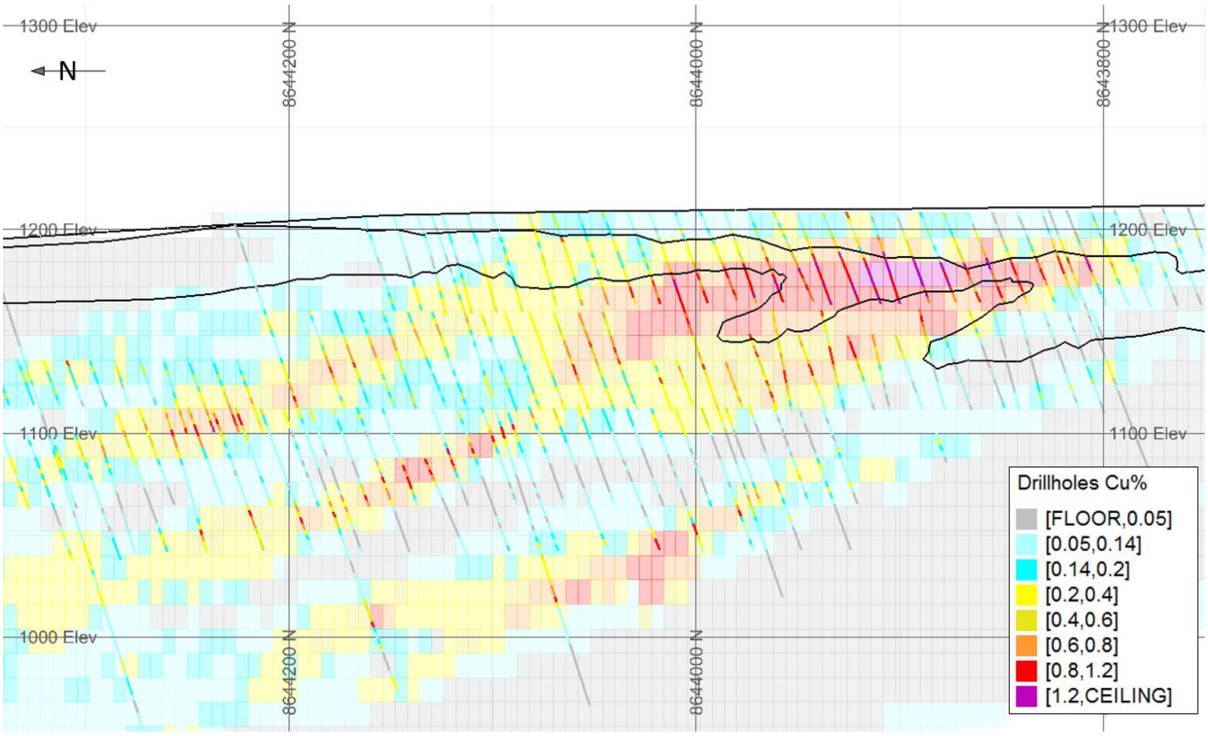


Figure 14-10 A vertical cross section showing visual validation of copper grade estimates and drillholes



14.1.14 Sentinel Mineral Resource classification and reporting

Block model estimates were classified and reported in accordance with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014). The Mineral Resource statement (Table 14-3) is inclusive of Mineral Reserves. The Mineral Resource classification was guided by a life of mine pit shell. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-3 The Sentinel Mineral Resource statement depleted as at 31st December 2019 and using a 0.13% copper cut-off. Mineral Resources are inclusive of Mineral Reserves

Classification	Material	Tonnes (Mt)	Density	TCu (%)	Cu metal (kt)
Measured	Non-primary sulphide	78.9	2.71	0.36	287.16
Measured	Primary sulphide	578.8	2.79	0.49	2,852.77
	Measured subtotal	**657.7**	**2.78**	**0.48**	**3,139.94**
Indicated	Non-primary sulphide	26.0	2.77	0.30	76.82
Indicated	Primary sulphide	277.0	2.81	0.42	1,163.38
	Indicated subtotal	**303.0**	**2.81**	**0.41**	**1,240.20**
	Measured and Indicated total	**960.7**	**2.79**	**0.46**	**4,380.14**
Inferred	Non-primary sulphide	5.3	2.73	0.27	14.05
Inferred	Primary sulphide	57.0	2.80	0.37	212.41
	Inferred subtotal	**62.3**	**2.80**	**0.36**	**226.46**

Classification was guided by confidence in drillhole data, geological continuity, the quality of kriged estimates and the reasonable prospects for eventual economic extraction. Geological confidence was supported by logging data having good coverage from sufficiently closely spaced diamond drilled holes and a closer grid of reverse circulation holes. There was a good understanding of prevailing geology controls at regional and local scales, as per in-pit exposures and pit mapping. Confidence in kriged estimates was supported by adequate validations, reasonable drill grid spacings and good sample data quality.

Measured Resources comprise 68% of the total Measured and Indicated Mineral Resources. Inferred Mineral Resources had adequate data to imply geological and grade continuity. In addition, the Mineral Reserve cut-off grade was considered when classifying Mineral Resources.

There were no known factors related to environmental, permitting, legal, title, taxation, socioeconomic, marketing, or political issues that were believed to materially affect the estimate of Sentinel Mineral Resources.

14.1.15 Grade tonnage curves

The grade tonnage curve for Measured and Indicated estimates, depleted as at December 2019, is graphed in Figure 14-11 and tabled in Table 14-4. Changes at lower cut off grades incur significant tonnage variations. At 0.13% Cu cut off, tonnes are 960.7 Mt with 0.46% Cu grade.

Figure 14-11 Grade tonnage curves for the measured and indicated sulphide and non-primary sulphide Mineral Resources as at 31st December 2019

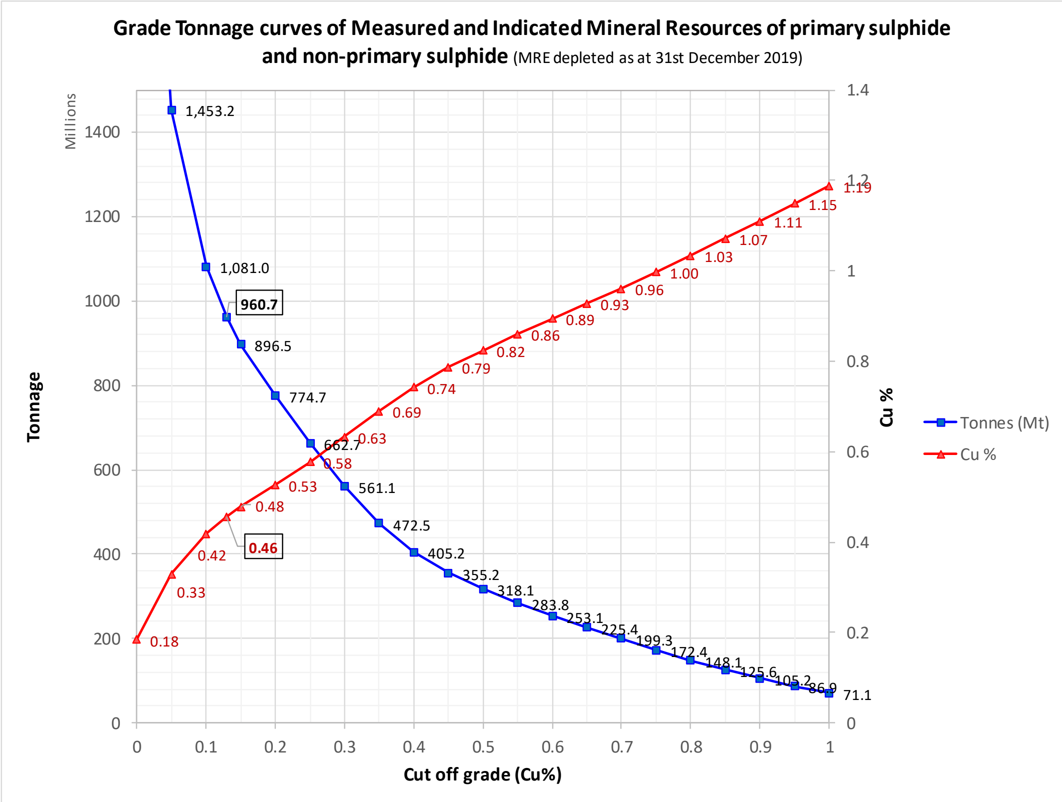


Table 14-4 Grade/tonnage curve for Sentinel Measured and Indicated Resources, depleted as at 31st December 2019

Cu % cut-off	Volume (Mm^3)	Density (t/m^3)	Tonnes (Mt)	Cu (%)
0.00	979.7	2.78	2720.9	0.18
0.05	521.6	2.79	1453.2	0.33
0.10	387.6	2.79	1081.0	0.42
0.13	**344.4**	**2.79**	**960.7**	**0.46**
0.15	321.4	2.79	896.5	0.48
0.20	277.7	2.79	774.7	0.53
0.25	237.5	2.79	662.7	0.58
0.30	201.1	2.79	561.1	0.63
0.35	169.4	2.79	472.5	0.69
0.40	145.3	2.79	405.2	0.74
0.45	127.4	2.79	355.2	0.79
0.50	114.1	2.79	318.1	0.82
0.55	101.8	2.79	283.8	0.86
0.60	90.7	2.79	253.1	0.89
0.65	80.8	2.79	225.4	0.93
0.70	71.4	2.79	199.3	0.96
0.75	61.8	2.79	172.4	1.00
0.80	53.0	2.79	148.1	1.03
0.85	45.0	2.79	125.6	1.07
0.90	37.6	2.79	105.2	1.11
0.95	31.1	2.79	86.9	1.15
1.00	25.4	2.79	71.1	1.19

14.1.16 Comparisons with previous estimate

This Sentinel estimate reflects additional diamond drilling data, grade control drilling (RC drillholes), improved geological understanding from drill core logging, pit mapping, geochemical and structural models, inclusion of sequential copper leach analysis and reconciliation data.

A comparison with the previous estimate, using a 0.2% copper cut-off grade, may be summarised as follows:

- Measured and Indicated Mineral Resource tonnes decreased by 7.8%, with a 0.7% increase in grade and net 7.2% decrease in copper metal. Reduced tonnes were due to tighter controls on mineralized volumes from data added during 2015 to 2019. Reduced tonnes were aligned with reconciliation data which had estimates using wider spaced diamond drill data overstating tonnes when compared to estimates using close spaced data.
- Inferred Mineral Resources were guided by this study's resulting pit shells. The narrow, less continuous and lower grade eastern extents were deemed to no longer be economically viable. 114 million tonnes of Inferred Mineral Resources were removed resulting in a 0.37% to 0.45% increase in copper grade for the remaining inferred material (using a 0.2% Cu cutoff).

Table 14-5 The previous (May 2015) Mineral Resource statement for the Sentinel deposit, using a 0.2% copper cut-off and depleted as at 31st December 2019

Classification	Material	Tonnes (Mt)	Density	TCu (%)	Cu metal (kt)
Measured	Non-primary sulphide	43.2	2.76	0.45	194.63
Measured	Primary sulphide	538.6	2.80	0.56	3,042.28
Measured subtotal		**581.8**	**2.80**	**0.56**	**3,236.91**
Indicated	Non-primary sulphide	12.3	2.76	0.43	52.75
Indicated	Primary sulphide	246.4	2.80	0.45	1,103.75
Indicated subtotal		**258.8**	**2.80**	**0.45**	**1,156.50**
Measured and Indicated total		**840.6**	**2.80**	**0.52**	**4,393.41**
Inferred	Non-primary sulphide	6.9	2.78	0.29	19.79
Inferred	Primary sulphide	150.6	2.80	0.38	568.65
Inferred subtotal		**157.5**	**2.80**	**0.37**	**588.44**

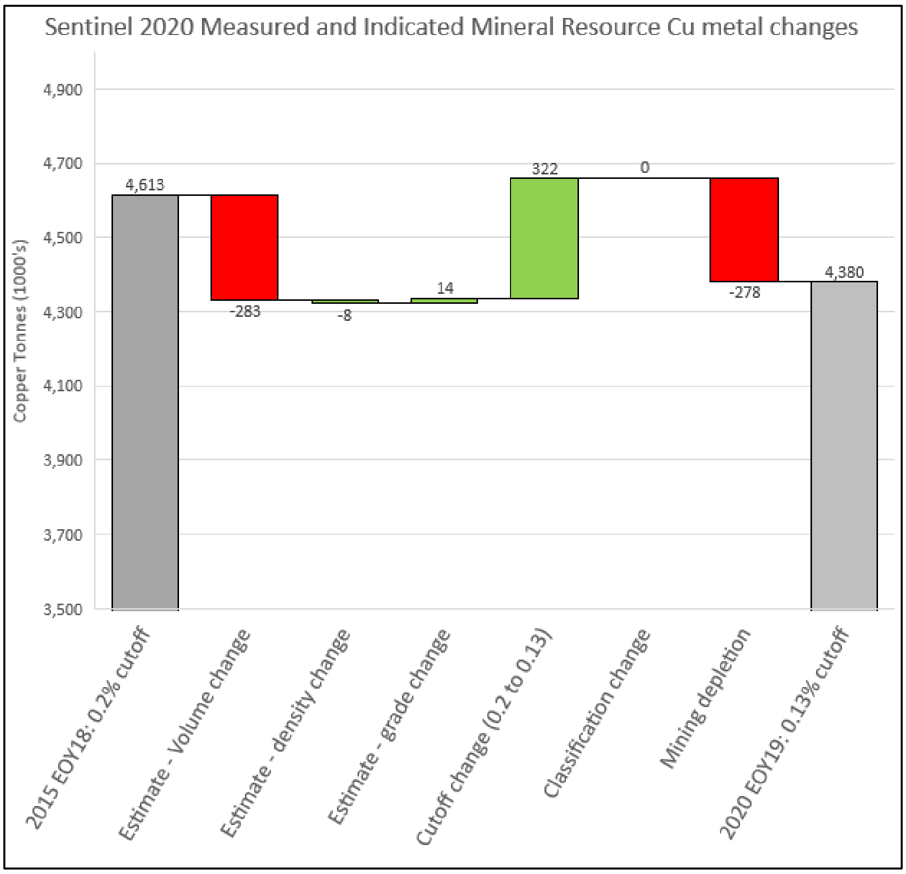
Table 14-6 2019 Mineral Resource statement for the Sentinel deposit, using a 0.2% copper cut-off and depleted as at 31st December 2019

Classification	Material	Tonnes (Mt)	Density	TCu (%)	Cu metal (kt)
Measured	Non-primary sulphide	56.4	2.71	0.45	251.09
Measured	Primary sulphide	474.2	2.79	0.57	2,682.60
Measured subtotal		**530.6**	**2.78**	**0.55**	**2,933.69**
Indicated	Non-primary sulphide	18.4	2.77	0.35	64.41
Indicated	Primary sulphide	225.8	2.81	0.48	1,080.61
Indicated subtotal		**244.2**	**2.81**	**0.47**	**1,145.02**
Measured and Indicated total		**774.7**	**2.79**	**0.53**	**4,078.71**
Inferred	Non-primary sulphide	3.6	2.73	0.32	11.45
Inferred	Primary sulphide	40.0	2.80	0.46	185.23
Inferred subtotal		**43.6**	**2.80**	**0.45**	**196.67**

The addition of reverse circulation drillholes with sequential copper leach analysis has improved this estimates ability to define the position of oxidised, refractory and non-primary sulphide mineralization volumes.

In context of Measured and Indicated Mineral Resources available for conversion into Mineral Reserves, copper metal changes were calculated between the May 2015 estimate and this estimate. Metal changes are presented in the waterfall chart below (Figure 14-12). The reduction in mineralised volumes is off-set by the change in copper cutoff grade. Excluding mining depletion, there is a net metal gain to this estimate.

Figure 14-12 A waterfall chart illustrating copper metal changes between the May 2015 estimate and this estimate



It is the opinion of the QP that the resulting changes to this Mineral Resource statement reflects the confidence in the underlying added data and that the estimates are believed representative of the prevailing mineralization.

14.2 Enterprise

14.2.1 Introduction

Mineral Resource estimates were completed for nickel (Ni) and copper (Cu) which were interpolated into a 3-Dimensional (3D) geology block model using ordinary kriging and commercially available software i.e. Datamine Studio RM version 1.5.62.0 and Snowden Supervisor version 8.11. Estimates were completed in October 2019 by Carmelo Gomez and the site geologists of FQM under the supervision of David Gray (QP), FQM Group Mine and Resource Geologist. The project limits and coordinates were based upon the ARC1950 (Mean) 35s UTM coordinate system.

Estimates used an updated database and were classified according to geological continuity, QAQC, density data, drillhole grid spacing, grade continuity, confidence in the panel grade estimate and the reasonable prospects for eventual economic extraction. Reporting was guided by the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014).

The Enterprise Mineral Resource estimate was depleted for minor material stripped as at 31st December 2019 and was reported using a 0.15% nickel cut-off grade. The Mineral Resource classification was guided by a life of mine pit shell. The estimate has updated the Enterprise Main area due to the addition of data and geological knowledge. The Enterprise South West Resource remains unchanged as per the 2015 Technical Report (FQM, May 2015).

14.2.2 Available data

The Enterprise Main deposit estimate was updated using drilling completed since the last estimate. Added drilling comprised 41 short diamond holes from 2018 to 2019 drilled at a close grid of 25 m by 25 m in the area of the starter pit at Enterprise Main area (Figure 14-13).

In addition, FQM undertook a multi element assaying campaign on drill cores for Enterprise Main area for a total of 9,281 composites of 2 metre lengths (Figure 14-14) in order to increase knowledge on geology, geochemistry, structure, lithologies, alterations and nickel mineral species. A total of 580 diamond drilled holes were available for 166,236 meters and 141,508 nickel assayed samples.

Data was formatted and loaded into Datamine from exported DataShed database tables. Data included collar, survey, assay, geology, minerals, alteration, structure, and density. The files were subject to validation steps checking for missing holes, sample overlaps, sample duplicates and ensuring all holes have survey information. There were no validation issues identified for this data.

Figure 14-13 2018-2019 drill data (blue) and mineralization limits (black strings)

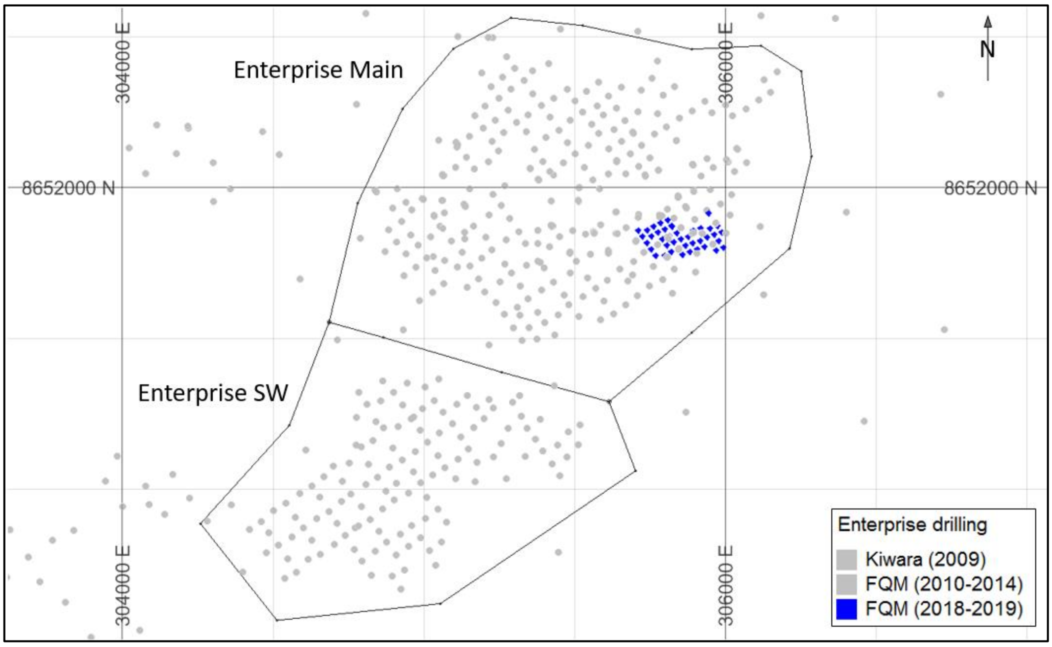
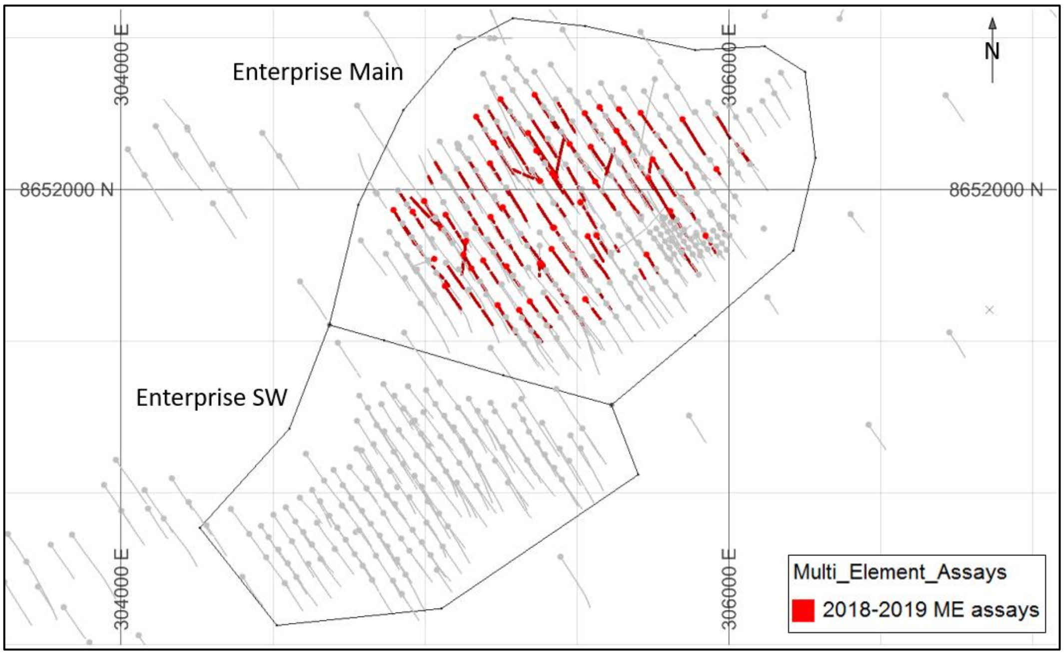


Figure 14-14 Location of samples assayed for multi element in 2018-2019 (red)



14.2.3 Geological and mineralization model

The updated Main deposit estimate relied upon an interpretation of mineralization influenced by folding and faulting. The geology model was completed by site and exploration geologists and external consultants (Model Earth Pty Ltd) during 2018-2019. The model was based upon data from available logging and structural data from drill cores, field mapping on the limited exposed areas of the starter pit and the additional geochemical data and models. In developing the geological and structural models, Model Earth consultants collaborated with the site based geology team over time and gained good familiarity with the data, deposit, and it's regional context.

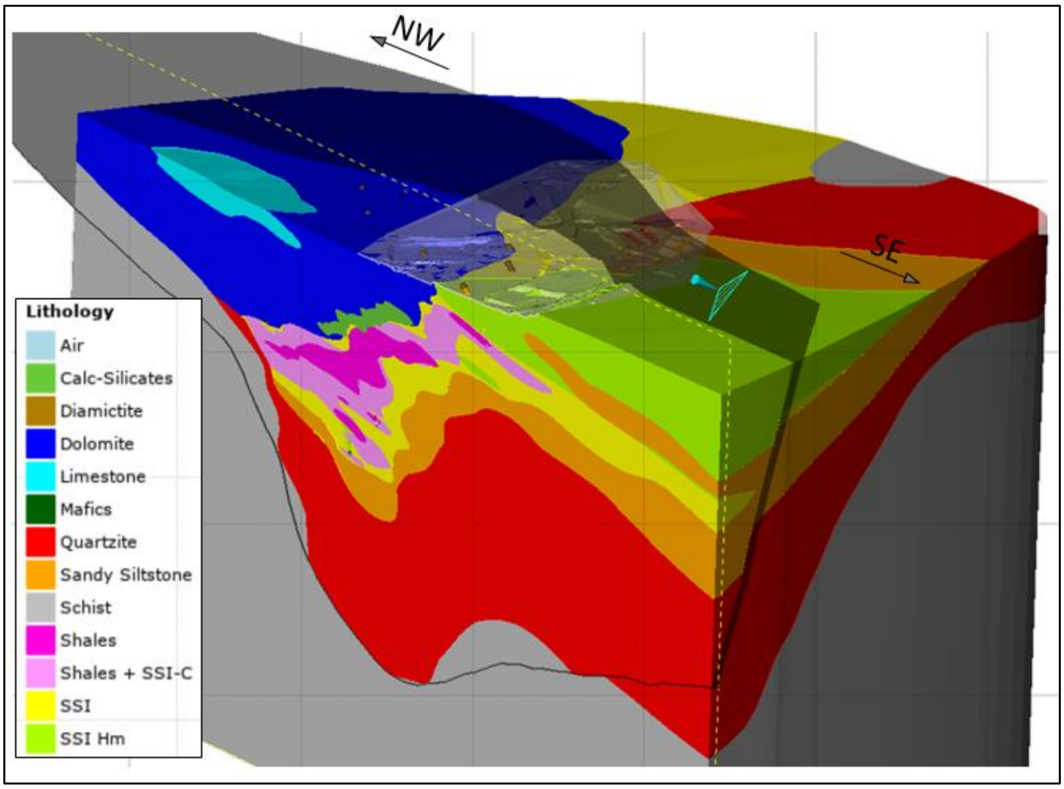
The Enterprise deposit was divided into the SouthWest (SW) and Main deposits, as shown in Figure 14-13. The Main deposit was updated for this Mineral Resource estimate. There was no new information for Enterprise SW deposit.

14.2.3.1 Enterprise Main geology and mineralization

Weathering and oxidation profiles of the Main deposit were reviewed and updated using Datamine and vertical sections, by Keitumetse Tshoganetso (FQM's Enterprise project geologist).

Geochemical analysis of sample multi-element assays, structural information from logged core, surface mapping and regional context were used to define the geological model for Main deposit. Recumbent folds and a series of transfer faults were defined according to the structural and geochemical data available. Key lithological units were included in this model. Leapfrog and ioGAS software were used to analyze and model the different features (Figure 14-15).

Figure 14-15 Oblique view of the Enterprise Main deposit geological model showing the largest transfer fault (semi-transparent black) and the recumbent fold architecture



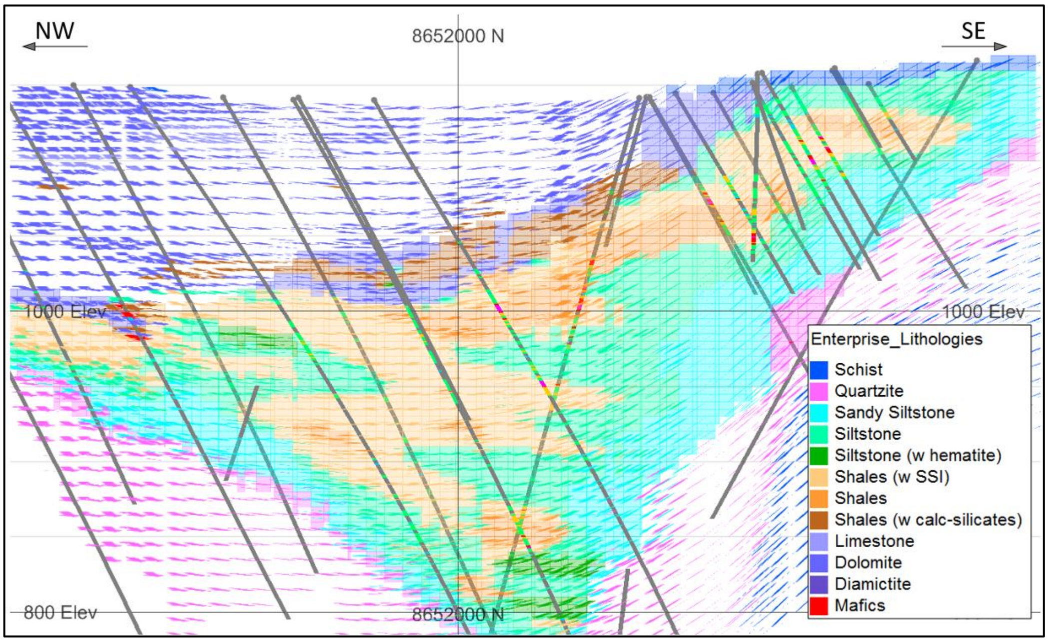
The resulting geology model wireframes were imported into Datamine where lithological units were grouped to allow for consistency and continuity in the estimates (Table 14-7).

Table 14-7 Wireframes for lithological units and Rock / Rock group fields

Wireframe (Litho)	ROCK	ROCKGROUP
LF-Diamictite	50300	50000
LF-Dolomite	50200	
LF-Limestone	50100	
LF-Calc-silicates	40300	40000
LF-Shales	40200	
LF-Shales-SSI	40100	
LF-SSI_HM	30300	30000
LF-SSI	30200	
LF-SandySiltstone	30100	
LF-Quartzite	20000	20000
LF-Schist	10000	10000
LF-Mafics	90000	90000

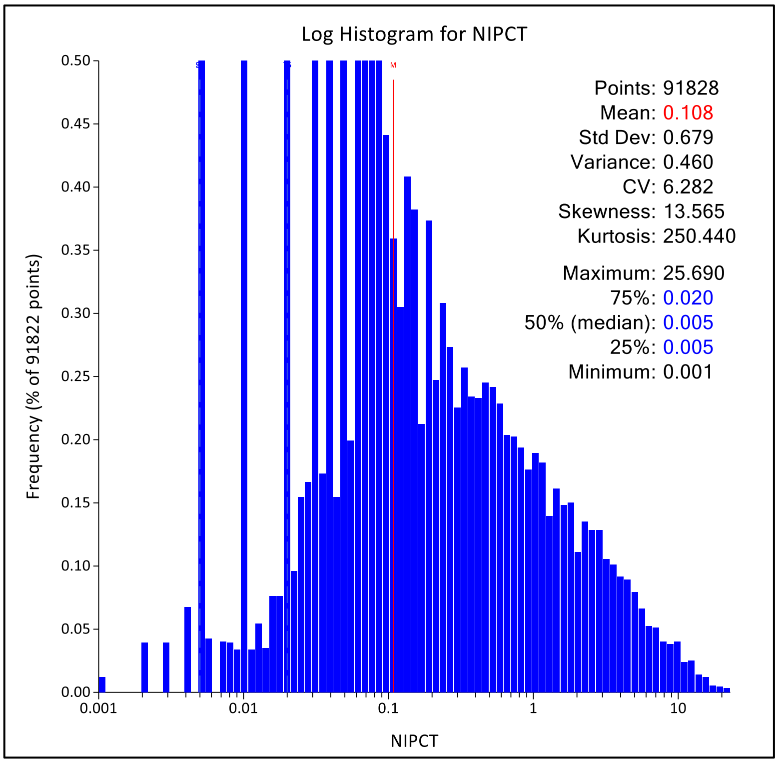
The lithology wireframes and folds were used to determine Dynamic anisotropy vectors for guiding the estimation sample selection routine.

Figure 14-16 NW-SE cross section across Enterprise Main showing lithologies and dynamic anisotropy vectors



Mineralization domains were defined by using categorical indicators, as manually digitized wireframes were subjective, leading to undesired levels of mixed populations. Two main nickel domains were defined from the statistical analysis of the sample assay values. An ultra-low grade domain having nickel values below 0.17%, and a low to high grade domain for values above 0.17% Ni. Histogram of the nickel dataset is show in Figure 14-17.

Figure 14-17 Histogram for Enterprise Main. 0.17% Ni was selected as threshold between ultra low grade mineralization (background) and low-high grade mineralization

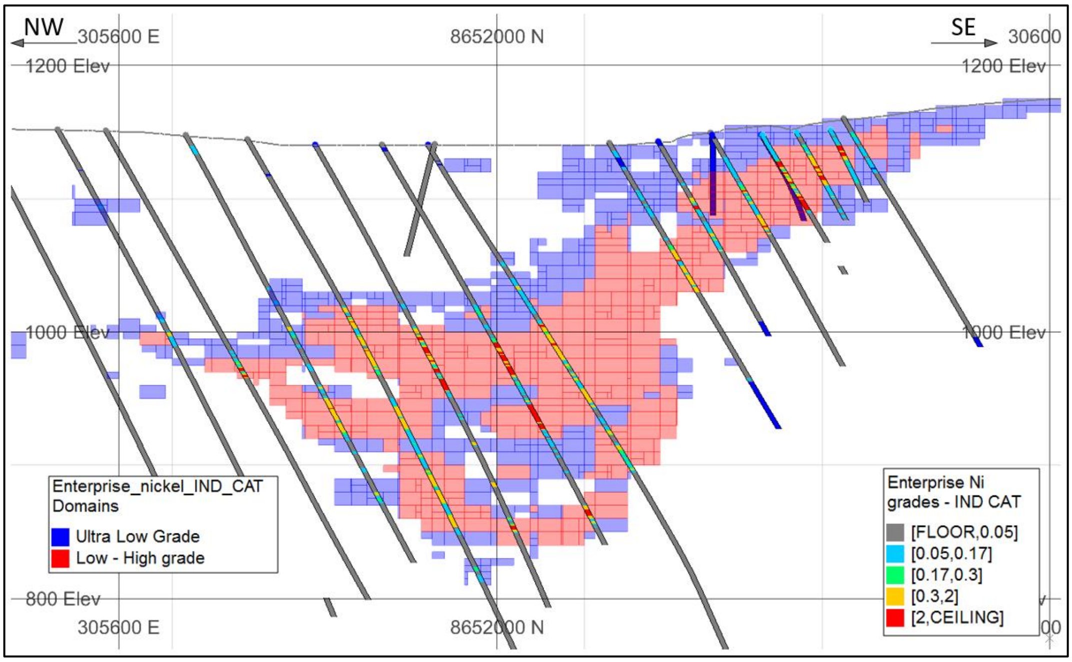


Samples were coded per domain. Statistical and spatial analysis of domained data ensured selection of an optimal threshold value for the categorical indicator estimate. Probability values were estimated for each block using ordinary kriging with dynamic anisotropy orienting the search ellipses. The categorical indicators estimated probability values were analysed for population inflections. The inflections were identified (Table 14-8) from the distribution of the probability values which in turn were visually validated against input data to ensure reasonable mineralization volumes (Figure 14-18).

Table 14-8 Categorical Indicator thresholds for probabilities on nickel mineralization domains

Indicator Category	cut-off
Ultra Low Grade (ULG)	0.10
Low-High grade (LG)	0.20

Figure 14-18 NW-SE Enterprise Main cross section showing nickel grades in the drill holes and nickel mineralization domains (categorical indicators)



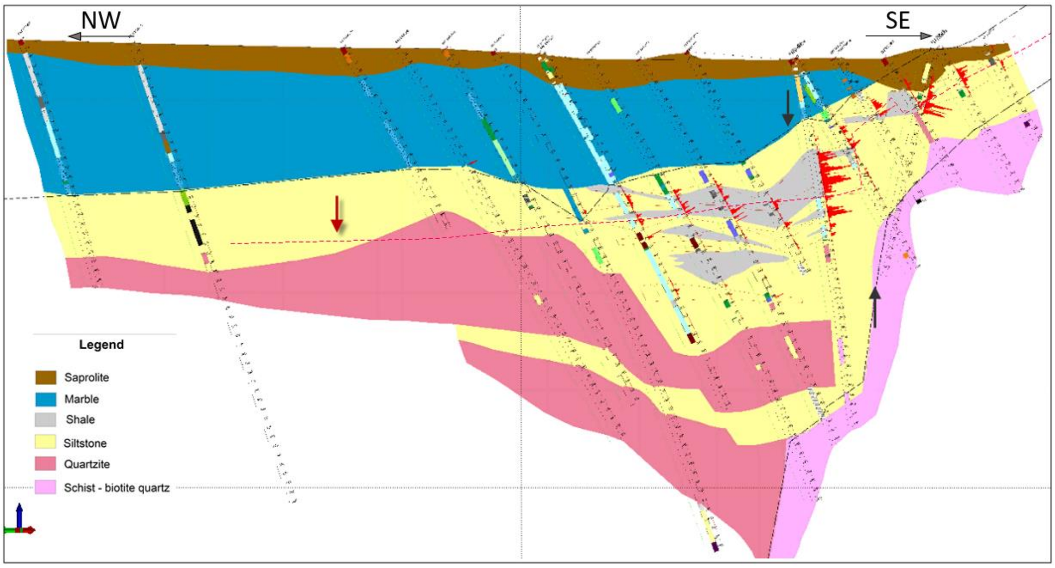
14.2.3.2 Enterprise SW geology and mineralization

Weathering and oxidation surfaces were derived using vertical section string interpretations of logged data. Strings per section were linked to form wireframe surfaces used to code drillhole sample data and to constrain estimates. Stratigraphic volumes were derived from logging data using vertical sections and string envelope interpretations. The stratigraphy string envelopes were linked to form stratigraphic volumes. The weathering and oxidation surfaces were used with the stratigraphic volumes to determine and assign density values. The stratigraphic volumes were not used to constrain estimation.

A single surface was defined for the zone of mineralization and was used to guide estimation. A centreline was digitised through the mineralization which was used as a plane to un-fault and un-fold the geological model prior to spatial analysis and estimation. A geological vertical section is displayed in Figure 14-19.

Grade estimates were estimated into the 3D block model containing the key geology domains. Estimates were completed using geostatistical applications from commercial mining software, Datamine Studio 3, Surpac and Isatis.

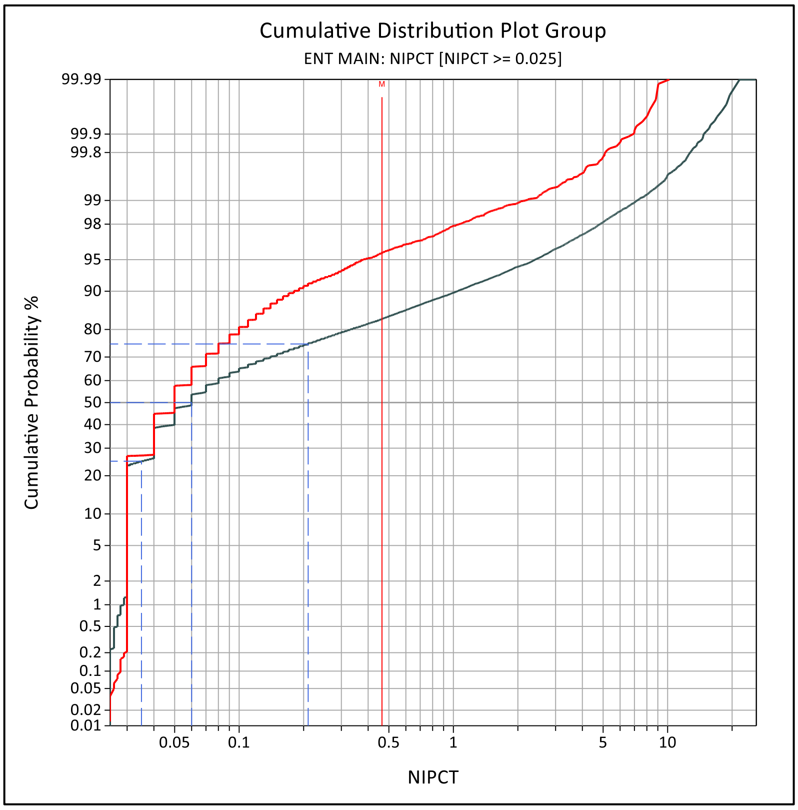
Figure 14-19 FQM vertical section looking south-east, showing hanging wall (marble, blue) and footwall (schists, pink) and centre line (dashed red line/red arrow)



14.2.4 Data preparation

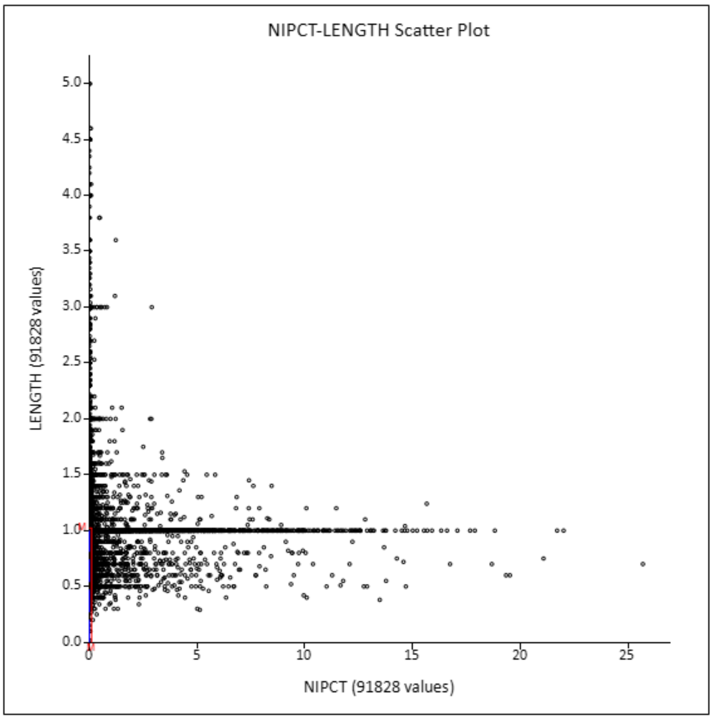
Sample assay data was coded by deposit and according to the wireframes for mineralization, oxidation, weathering and rock groups. Samples were coded for the nickel mineralization domains according to the volumes derived from the categorical indicator block model. Classical statistics were reviewed for all domains to ensure minimal domain mixing and in order to establish a suitable estimation method. Main deposit has a higher nickel grade tenor when compared to SW (Figure 14-20).

Figure 14-20 Nickel grade distributions for Enterprise Main (black) and Enterprise SW (red)



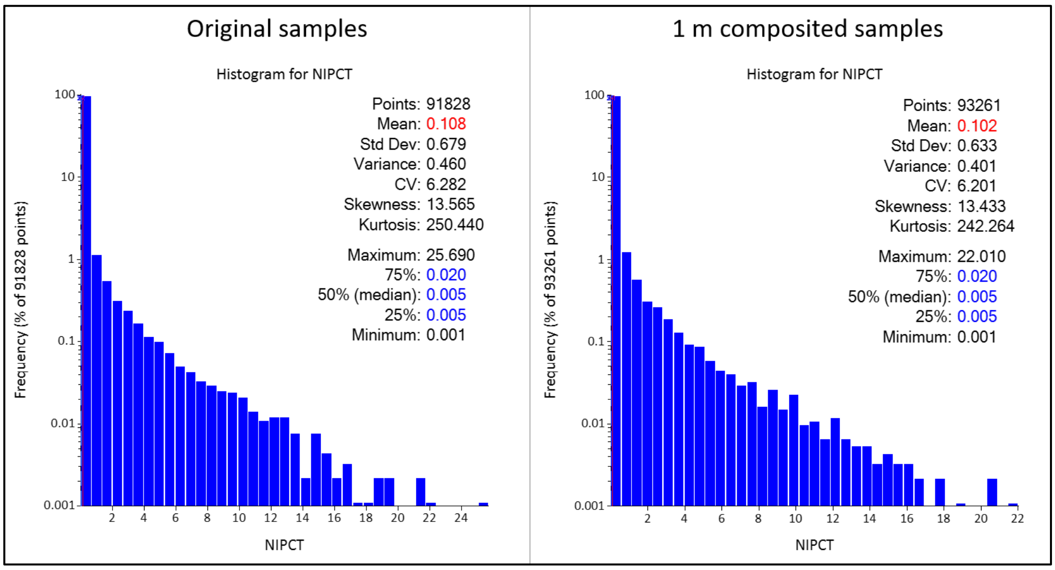
Original drill core sample length versus nickel grade highlights limited to no risk of any high grade values coming from small sample intervals (Figure 14-21).

Figure 14-21 Enterprise samples showing nickel values vs sample length



Drill hole sample data was composited to the dominant 1 m sample length. Mean grades and coefficients of variation were marginally affected by the compositing process (Figure 14-22).

Figure 14-22 Original samples and 1 meter composited samples for Enterprise Main.



14.2.5 Top and bottom cuts

Prior to estimation, histograms and coefficients of variation of estimation domains were reviewed to determine the presence of high grade samples within the sample population. Top cuts would be used in order to restrict the influence of the identified high grade samples that may be real but not representative of the local grade distribution.

For the Main ultra low grade nickel domain (NICAT=1), a bottom cut of 0.011% nickel was applied to remove the lower tail of samples which were affected by lower detection limits of the analytical process. A top cut of 1.1% Ni (99.8% distribution) was applied to the same domain. Less than 0.2% of the sample population was affected (i.e. only 65 samples), however, the coefficient of variation reduced from 4.567 to 1.99 (Figure 14-23) improving domain variability for estimation. No top cut was required for the high grade domain (NICAT=2), and no top cuts were applied in the previous estimate of the SW deposit.

Figure 14-23 Top cut applied to ultra low grade nickel domain (NICAT=1) at 1.1% nickel (99.8% distribution)

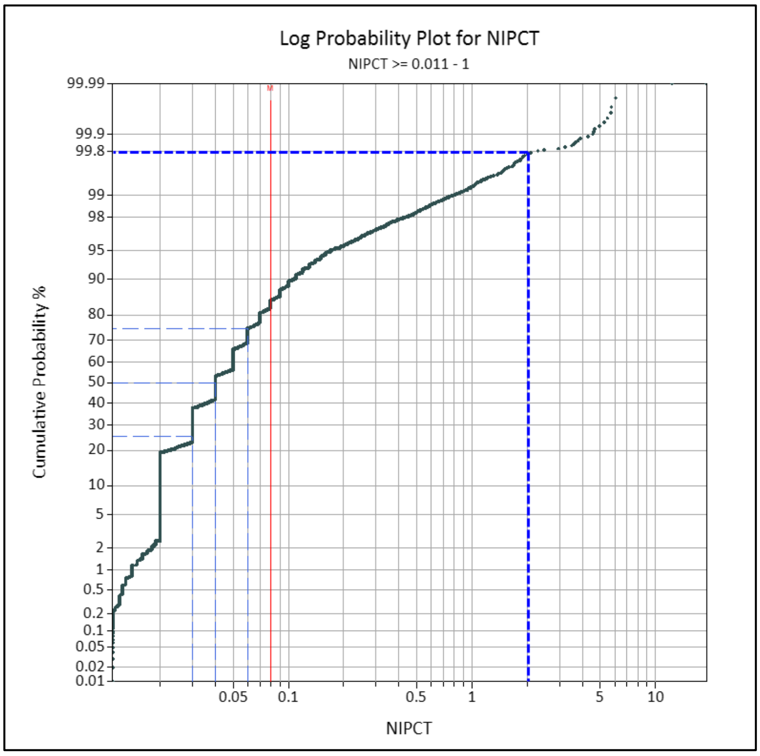
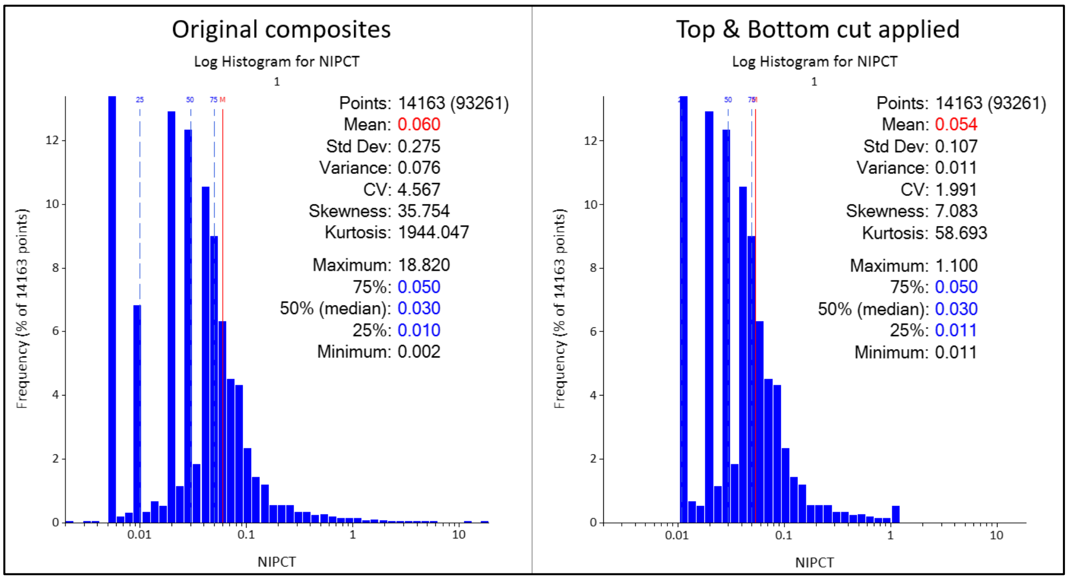


Figure 14-24 Ultra low grade domain (NICAT=1) composites before (left) and after (right) bottom and top cuts applied



14.2.6 Spatial analysis

Variography was completed on 1 m composites in Supervisor software for Ni and Cu per domain. Directional variogram analysis used a total sill value normalized to the domain sample population variance. Nugget variances were derived from downhole variograms at a lag value of the composite interval length. Variography suggests that grade continuity at Main is different to SW.

The nugget at Main for ultra low grades (<0.17% Ni) was 16%, with approximately 77% of the variability linked to the short scale structure in the first 39 m; nugget for low grades (>=0.17% Ni) was 10% with 51% of the variability in the first 37 m. In contrast, the variability at SW has a nugget of 10%, and approximately 79% of the variability in the first 55 m.

Variogram models for Ni and Cu had different orientations. Copper mineralization exists at depths below the known Ni mineralization and is understood to be independent of Ni mineralization.

The Ni variogram at Main is oblique to the strike of the mineralization and is aligned with major folding, which may contribute to the controls on higher grades.

14.2.6.1 Enterprise Main

Variogram models and resulting search parameters used for estimation of Main deposit are summarised in Table 14-9 to Table 14-20.

Table 14-9 Summarised variogram models for nickel indicator categories estimate

Domain	Grade (Cat. Indicator)	Zone Control	Variogram rotation angles			Nugget	Spherical model 1 ranges				Spherical model 2 ranges				Spherical model 3 ranges			
			Z	X	Z		X	Y	Z	Sill	X	Y	Z	Sill	X	Y	Z	Sill
NICAT	1 (ULG)	ROCKGRP	-25	20	-105	0.18	14	29	5	0.4	108	58	20	0.24	178	251	54	0.18
	2 (LG-HG)		-35	20	-105	0.26	24	8	5	0.49	61	75	34	0.25	-	-	-	-

Table 14-10 Summarised search ellipses for nickel indicator categories estimates

Domain	Grade (Cat. Indicator)	Search axis rotation			First Pass search radius			Second Pass radius multiplier	Third Pass radius multiplier
		Z	X	Z	X	Y	Z		
NICAT	1 (ULG)	using dynamic anisotropy vectors			178	251	54	2	3
	2 (LG-HG)				61	81	32	2	3

Table 14-11 Summarised search parameters for nickel indicator categories estimates

Domain	Grade (Cat. Indicator)	Search Pass	Min # of comp.	Max # of comp.	Search ellipse	Max # comp. per hole
NICAT	1 (ULG) 2 (LG-HG)	First Pass	8	20	initial	-
		Second Pass	8	20	2 x the initial	
		Third Pass	8	20	3 x the initial	

Table 14-12 Summarised variogram models for nickel grade estimates

Domain (NIDOM)	Grade	Variogram rotation angles			Nugget	Spherical model 1 ranges				Spherical model 2 ranges				Spherical model 3 ranges			
		Z	X	Z		X	Y	Z	Sill	X	Y	Z	Sill	X	Y	Z	Sill
1 (ULG = W_NIDOM)	Nickel	-40	30	0	0.16	39	13	4	0.61	82	69	11	0.23	-	-	-	-
2 (LG-HG = O_NIDOM)	Nickel	-20	25	0	0.10	37	23	4	0.41	94	116	8	0.27	262	142	23	0.22

Table 14-13 Summarised search ellipses for nickel grade estimates

Domain (NIDOM)	Grade	Search axis rotation			First Pass search radius			Second Pass radius multiplier	Third Pass radius multiplier
		Z	X	Z	X	Y	Z		
1 (ULG = W_NIDOM)	Nickel	using dynamic anisotropy vectors			65	55	8	1.2	1.5
2 (LG-HG = O_NIDOM)	Nickel				65	55	4	1.2	1.5

Table 14-14 Summarised search parameters for nickel grade estimates

Domain (NIDOM)	Grade	Search Pass	Min # of comp.	Max # of comp.	Search ellipse	Max # comp. per hole
1 (ULG = W_NIDOM)	Nickel	First Pass	6	12	initial	-
		Second Pass	6	12	1.2 x the initial	
		Third Pass	6	12	1.5 x the initial	

Domain (NIDOM)	Grade	Search Pass	Min # of comp.	Max # of comp.	Search ellipse	Max # comp. per hole
2 (LG-HG = O_NIDOM)	Nickel	First Pass	4	8	initial	-
		Second Pass	6	12	1.2 x the initial	
		Third Pass	8	20	1.5 x the initial	

Table 14-15 Summarised variogram model for copper indicator category estimate

Domain (CUCAT)	Grade (Cat. Indicator)	Variogram rotation angles			Nugget	Spherical model 1 ranges				Spherical model 2 ranges				Spherical model 3 ranges			
		Z	X	Z		X	Y	Z	Sill	X	Y	Z	Sill	X	Y	Z	Sill
1 (Cu > 0.08)	CUCAT=1	-25	15	-90	0.46	29	51	12	0.37	252	274	163	0.17	-	-	-	-

Table 14-16 Summarised search ellipse for copper indicator category estimate

Domain (CUCAT)	Grade	Search axis rotation			First Pass search radius			Second Pass radius multiplier	Third Pass radius multiplier
		Z	X	Z	X	Y	Z		
1 (Cu > 0.08)	Copper	using dynamic anisotropy vectors			100	100	20	1.5	3

Table 14-17 Summarised search parameters for copper indicator category estimate

Domain (CUCAT)	Grade	Search Pass	Min # of comp.	Max # of comp.	Search ellipse	Max # comp. per hole
1 (Cu > 0.08)	Copper	First Pass	4	14	initial	6
		Second Pass	4	14	1.5 x the initial	
		Third Pass	4	14	3 x the initial	

Figure 14-25 Variography for nickel low-high grade domain (NIDOM=1)

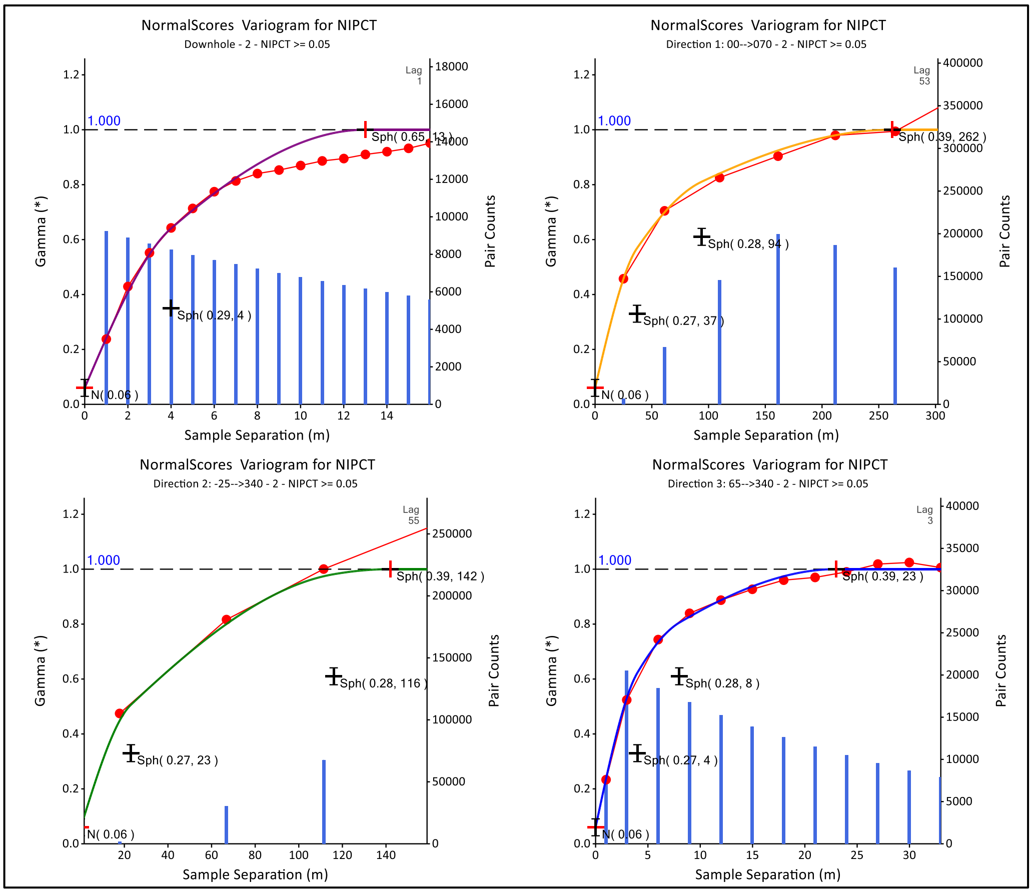


Table 14-18 Summarized variogram model for copper grade estimate

Domain (CUDOM)	Grade	Variogram			Nugget	Spherical model 1				Spherical model 2				Spherical model 3			
		Z	X	Z		X	Y	Z	Sill	X	Y	Z	Sill	X	Y	Z	Sill
1	Copper	-25	15	-90	0.46	29	51	12	0.37	252	274	163	0.17	-	-	-	-

Table 14-19 Summarized search ellipse for copper grade estimate

Domain (CUDOM)	Grade	Search axis rotation			First Pass search			Second Pass radius	Third Pass radius
		Z	X	Z	X	Y	Z		
1	Copper	using dynamic anisotropy vectors			70	70	20	1.5	3

Table 14-20 Summarized search parameters for copper grade estimate

Domain (CUDOM)	Grade	Search Pass	Min # of comp.	Max # of comp.	Search ellipse	Max # comp. per hole
1	Copper	First Pass	4	14	initial	6
		Second Pass	4	14	1.5 x the initial	
		Third Pass	4	14	3 x the initial	

14.2.6.2 Enterprise SW

Variogram models and resulting search parameters used for estimation of SW (as in May 2015) are summarized in Table 14-21 to Table 14-23.

Table 14-21 Variogram parameters for nickel

Area	Nugget	Structure 1				Structure 2			
		Sill	Range 1	Range 2	Range 3	Sill	Range 1	Range 2	Range 3
Enterprise SW	0.1	0.79	55	56	17	0.12	283	275	40

Table 14-22 Variogram parameters for copper

Area	Nugget	Structure 1				Structure 2			
		Sill	Range 1	Range 2	Range 3	Sill	Range 1	Range 2	Range 3
Enterprise SW	0.12	0.82	31	78	10	0.06	138	107	19

Table 14-23 Variogram rotations (Isatis Mathematician format)

Area	Variable	Z	Y	X
Enterprise SW	Ni	-30	0	-20
	Cu	-70	0	30

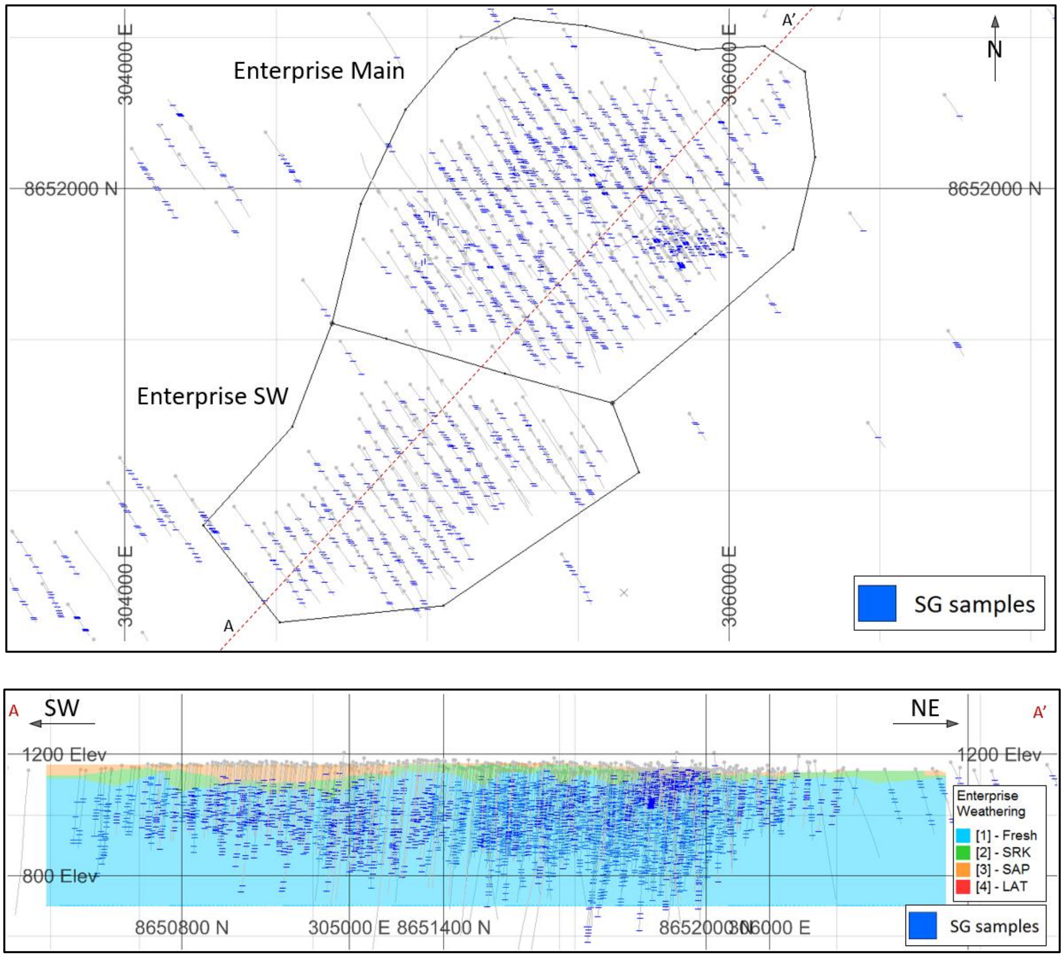
14.2.7 Density data analysis

A total of 9,386 in-situ dry density measurements were available with good coverage across the deposit (Figure 14-26). Sample density values were measured using the traditional Archimedes method of weighing samples in air and then in water.

Most of Main deposit is comprised of fresh material with low variability in density. SW has saprolite, saprock and fresh material. The majority (96%) of density data collected at SW was in fresh material. There were insufficient samples to provide a confident average density value for saprolite and saprock for SW. Densities were supplemented with data from adjacent deposits within the Copperbelt and assigned to the respective materials (Table 14-25).

For 2018 to 2019 drilling, density measures were cross-checked with a second methodology using a precision digital caliper to calculate the volume of the core, and thus calculating the dry density value after weighting the dried sample. Both methods reported within 0.5%, suggesting that the digital caliper method may be used in competent core samples to reduce time and provide more measurements.

Figure 14-26 Specific gravity samples distribution across Enterprise



Density data was analysed statistically. Weathering has the strongest influence on density values, followed by rock type. Average density values per weathered horizon and rock type were determined and assigned to each block as presented in Table 14-24. There were sufficient samples to determine a robust mean in-situ dry bulk density.

Figure 14-27 Histogram for density data in Enterprise

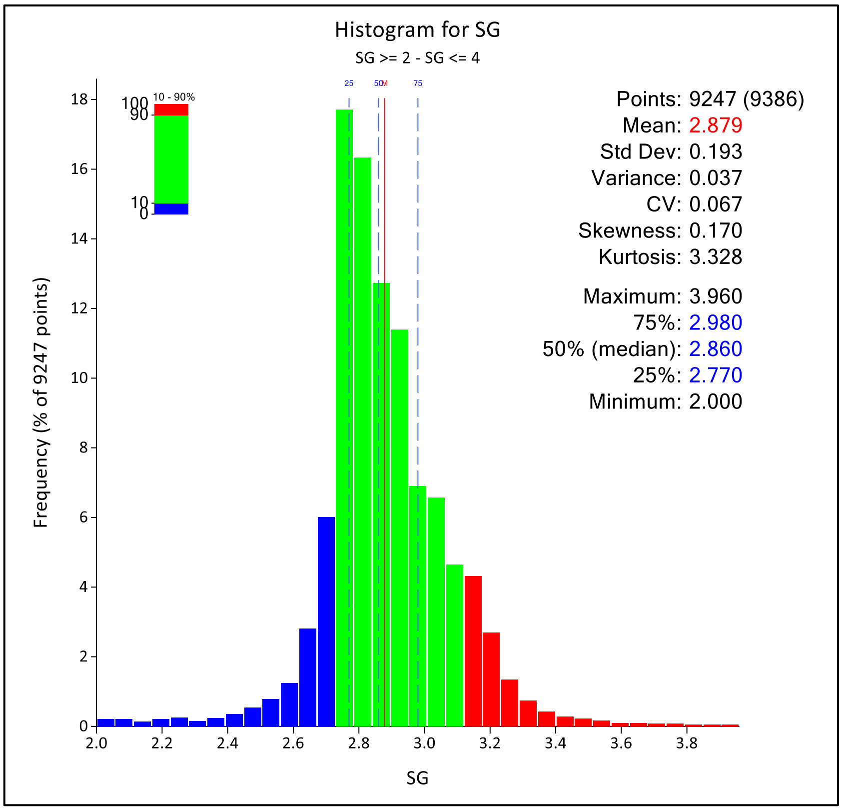


Table 14-24 Density values for Enterprise

Weathering	Rock type	Density (t/m^3)
Saprolite	-	1.80
Saprock		2.70
Fresh		2.90
Fresh	Schist	2.79
	Quartzite	2.80
	Siltstone	2.91
	Shales	2.86
	Carbonates	2.79
Saprock	Siltstone	2.22
	Shales	2.14
	Carbonates	2.58
Saprolite	Siltstone	2.00
	Carbonates	2.15

Average density values were applied per weathering horizon for Main and SW deposits. Specific values per lithology were assigned for Main to better reflect density variation.

Table 14-25 Published density values for neighbouring deposits

Deposit	Material	Average Density	Density of material relative to Fresh
Kinsevere	0-35m	1.87	0.81
	35-70m	1.98	0.86
	70-110m	2.05	0.89
	Fresh	2.30	1.00
Mashamba East	Oxide	2.20	
	Mixed	2.40	
Tilwezembe Mine	Oxide_Dol	1.96	0.87
	Oxide_Breccia	1.81	0.81
	Oxide_TillArg	1.98	0.91
	Sulphide_Dol	2.26	
	Sulphide_Breccia	2.24	
	Sulphide_TillArg	2.18	
Kananga	Oxide_Upper ore	1.80	0.86
	Oxide_Middle ore	1.80	0.90
	Oxide_Lower ore	2.00	0.95
	Sulphide_Upper ore	2.10	
	Sulphide_Middle ore	2.00	
	Sulphide_Lower ore	2.10	
Kipoi Copper Project	Oxide_Silt	1.72-2.48	0.75-0.94
	Transitional_Silt	2.50	0.94
	Fresh_Silt	2.65	
Kansanshi	Saprolite_Average	1.87	0.70
	Saprock_Average	2.41	0.91
	Sulphide_Average	2.66	
Enterprise	**Saprolite**	**1.80**	**0.62**
	Saprock	**2.70**	**0.93**
	Sulphide (Samples)	**2.90**	

14.2.8 Block model construction

Datamine was used to build an empty 20 m by 20 m by 10 m parent block model with sub cells of 10 m by 10m by 5 m. Parent block size was determined from kriging neighbourhood analysis (KNA) using Snowden Supervisor software. KNA enabled the selection of an optimal block size, search parameters (ellipse dimensions), and number of samples to be used for grade estimates.

The selected block size was close to the drill grid spacing of 50 m by 50 m and is compatible with the proposed bench height of 5 m with the SMU size of 10 m by 10 m.

The block model was coded using topography, weathering, oxidation, lithology and mineralization domains. The block model for Main was populated with dynamic anisotropy vectors calculated from lithologies and main folding structure, whereas for SW the block model and composites were un-faulted/un-folded in Datamine using the centre line plane of the mineralization zone. Parent block model sizes for SW remained unchanged at 40 m by 40 m by 5 m.

Block model extents are presented in Table 14-26**.**

Table 14-26 Enterprise block model grids used in this MRE.

Model setting		Parent (2019)	Parent Enterprise SW (2013)	SMU
X Origin		303500 mE	303500 mE	303500 mE
Y Origin		8649840 mN	8649840 mN	8649840 mN
Z Origin		700 mRL	700 mRL	700 mRL
Parent cell size	X	20 m	40 m	10 m
	Y	20 m	40 m	10 m
	Z	10 m	5 m	5 m
Minimum cell size	X	10 m	10 m	10 m
	Y	10 m	10 m	10 m
	Z	5 m	5 m	5 m
Number of parent blocks	X	165	83	330
	Y	148	74	296
	Z	60	120	120

14.2.9 Grade estimation

14.2.9.1 Main deposit

Domain nickel grades were estimated into the 20 m x 20 m x 10 m parent block model using Datamine's ordinary kriging. Top-cut composited samples were used for the low grade domain. No top-cut was required for the low-to-high grade domain. Search ellipse orientations were oriented by the dynamic anisotropy vectors assigned to each block. Ordinary kriging estimation parameters were based upon KNA, variography and geological continuity. A pseudo hard boundary of one sample each side was used between ultra low grade and low-high grade domains. Most blocks were estimated within the first search pass with only few blocks using the second pass with an ellipse double of the first one. Estimates of parent blocks used a discretization of 3 by 3 by 3 (X, Y, Z).

Post processing of parent block estimates was completed in order to provide estimates at the scale of mining i.e. the SMU. Localised uniform conditioned (LUC) copper estimates were determined per parent block's sub cell by using the estimated panel grade. Uniform conditioning provided the proportions of parent block sub cells above a range of cut-off grades. LUC then determines a grade per sub cell within the parent block, while maintaining metal content of the parent. The LUC model was re-blocked to the SMU block dimension of 10 mE by 10 mN by 5 mRL. LUC improves the representation of grade and tonnages that may be expected during mining.

Post processed LUC grades were validated as follows:

- visual comparisons of drillholes, OK panel grades and LUC block grades
- confirming that metal correlations from LUC blocks compare to those of the drillhole data
- checking that contained copper metal, at zero cut-off grade, was the same for panel and LUC models.

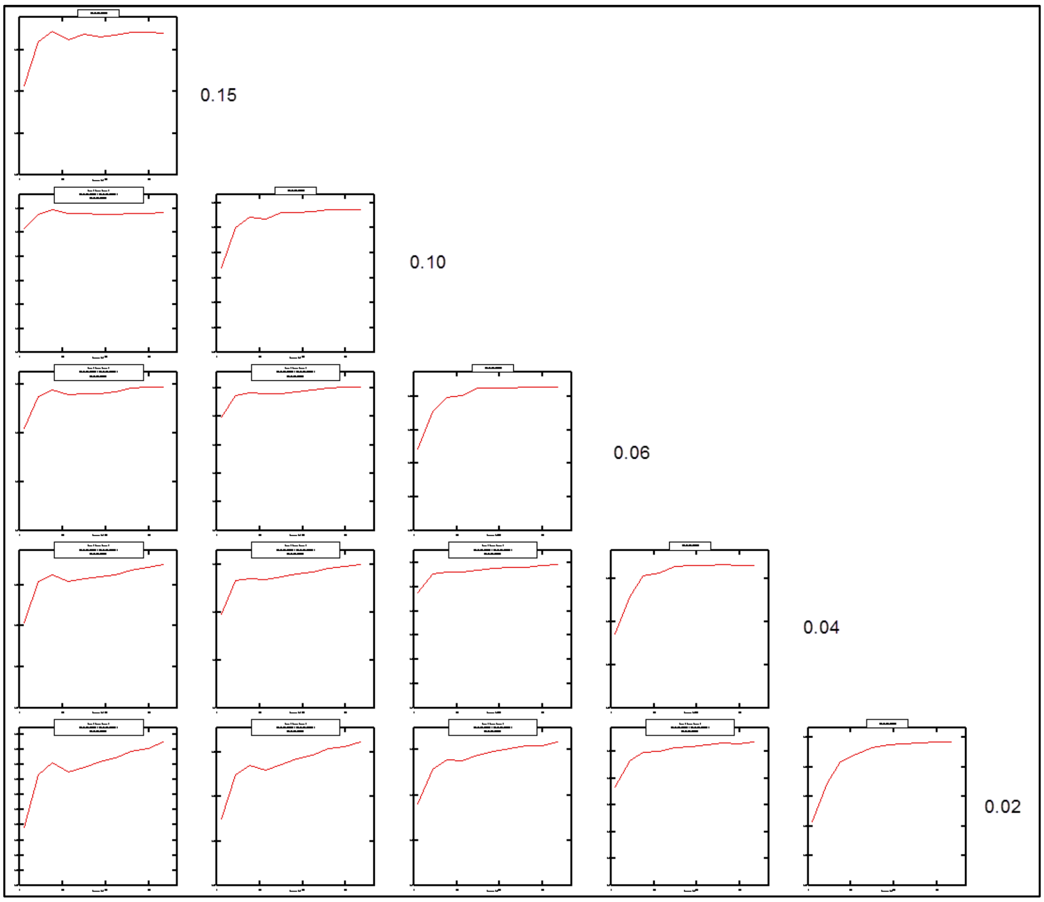
14.2.9.2 SW deposit

Isatis software was used to estimate grade using UC. UC is a recoverable estimation method used to estimate the grade tonnage relationship for defined cut-off grades and selective mining units (SMU).

The analysis of the ratio of the cross variogram to simple variogram was completed for a range of Ni cut offs (Figure 14-28). The structured behaviour of these variograms supports the assumption that grade classes are not independent of each other and have the characteristics of a Diffusion Model mineralization. This supported the choice of UC for grade estimation.

Grades were estimated into the 40 m by 40 m by 5 m block model using ordinary kriging (OK). The final block model was localised to SMU sized blocks using Localised Uniform Conditioning (LUC).

Figure 14-28 Ratio Cross Variogram/Simple Variogram for Ni showing diffuse behaviour of grades



Search parameters derived from the variography and neighbourhood analysis used for the OK grade estimates are presented in Table 14-27 and Table 14-28.

Table 14-27 Sample search criteria. Discretisation used was 3x3x5

<table>
<tr><th rowspan="2">Area</th><th rowspan="2">Pass</th><th rowspan="2">Variable</th><th colspan="3">Search Ranges</th><th rowspan="2">Min Samples</th><th rowspan="2">Sectors</th><th rowspan="2">Max samples per sector</th><th rowspan="2">Min samples per drill hole</th></tr>
<tr><th>X</th><th>Y</th><th>Z</th></tr>
<tr><td rowspan="6">Enterprise SW</td><td>1</td><td rowspan="3">Ni</td><td>100</td><td>80</td><td>7</td><td rowspan="2">18</td><td rowspan="6">4</td><td rowspan="2">10</td><td rowspan="6">6</td></tr>
<tr><td>2</td><td>140</td><td>110</td><td>10</td></tr>
<tr><td>3</td><td>140</td><td>110</td><td>10</td><td>8</td><td>5</td></tr>
<tr><td>1</td><td rowspan="3">Cu</td><td>100</td><td>80</td><td>7</td><td rowspan="2">18</td><td rowspan="2">10</td></tr>
<tr><td>2</td><td>140</td><td>110</td><td>10</td></tr>
<tr><td>3</td><td>140</td><td>110</td><td>10</td><td>8</td><td>5</td></tr>
</table>

Table 14-28 Search ellipse rotations (Isatis Mathematician)

Area	Rotation		
	Z	Y	X
Enterprise SW	-30	-20	0

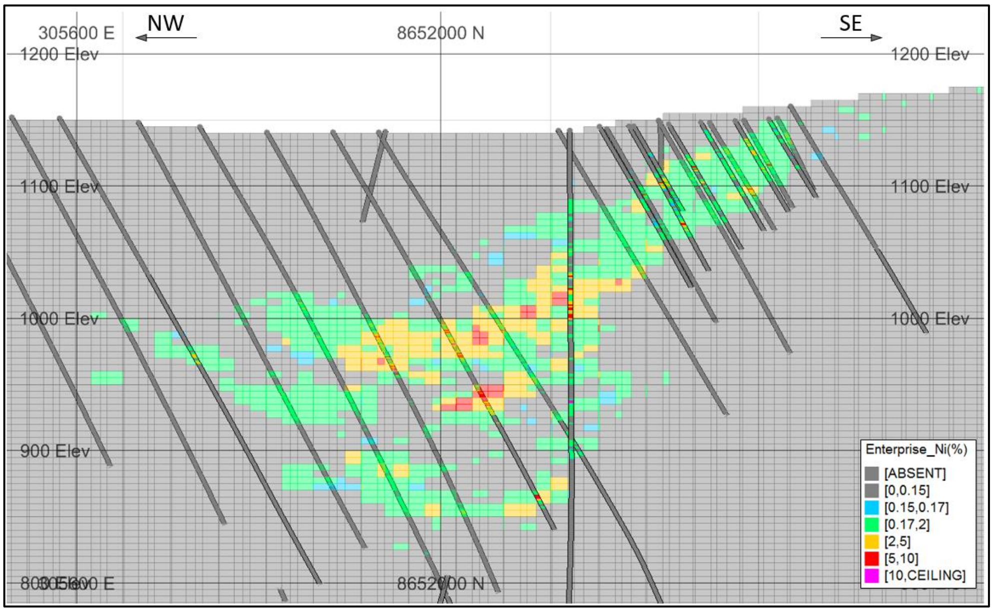
14.2.10 Model validation

A series of validation steps were completed to ensure block grade estimates represent prevailing geology and input sample data:

- Validation of sample grade and block grade in 2D cross sections (Figure 14-29).
- Compared respective domain mean sample data grades with mean estimated grades of the block model.
- Swath plots along northings, eastings and RL (Figure 14-30).

Visual validation suggests the grade tenor of the input data is represented in the block model estimates. Swath plot validations demonstrate estimates compare with input data, particularly where sufficient data informed block estimates.

Figure 14-29 Vertical Sections through Enterprise Main (top) and Enterprise SW (bottom) comparing composites and model



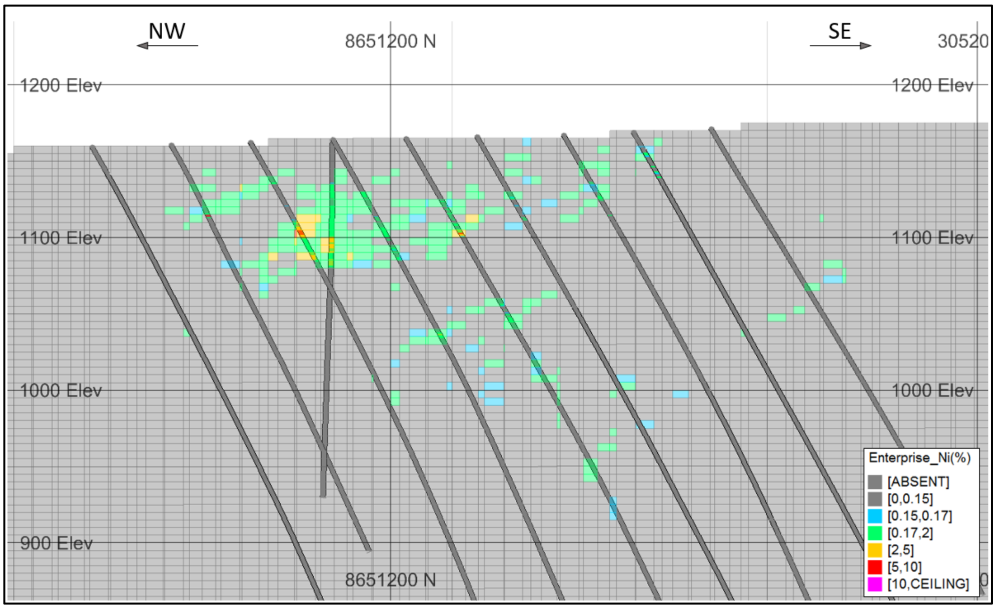
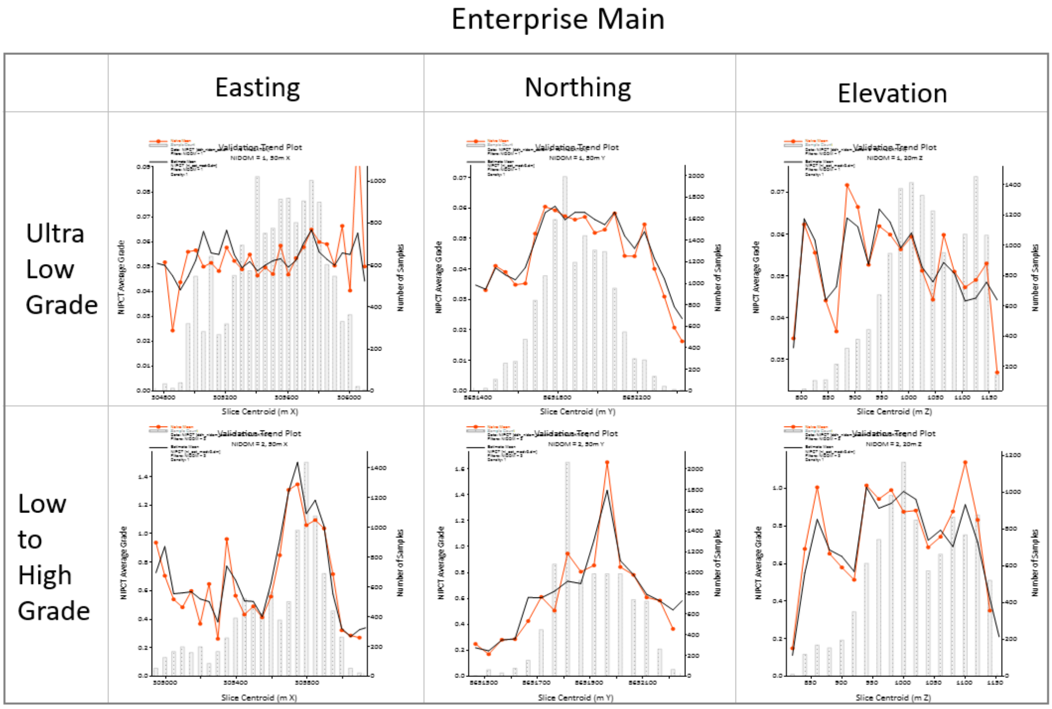


Figure 14-30 Swath plots for Enterprise Main Ni grade domain for Ultra Low Grade (top) and for Low to High Grade domains (bottom), at 50 m Northing, 50 m Easting and 20 m elevation increments



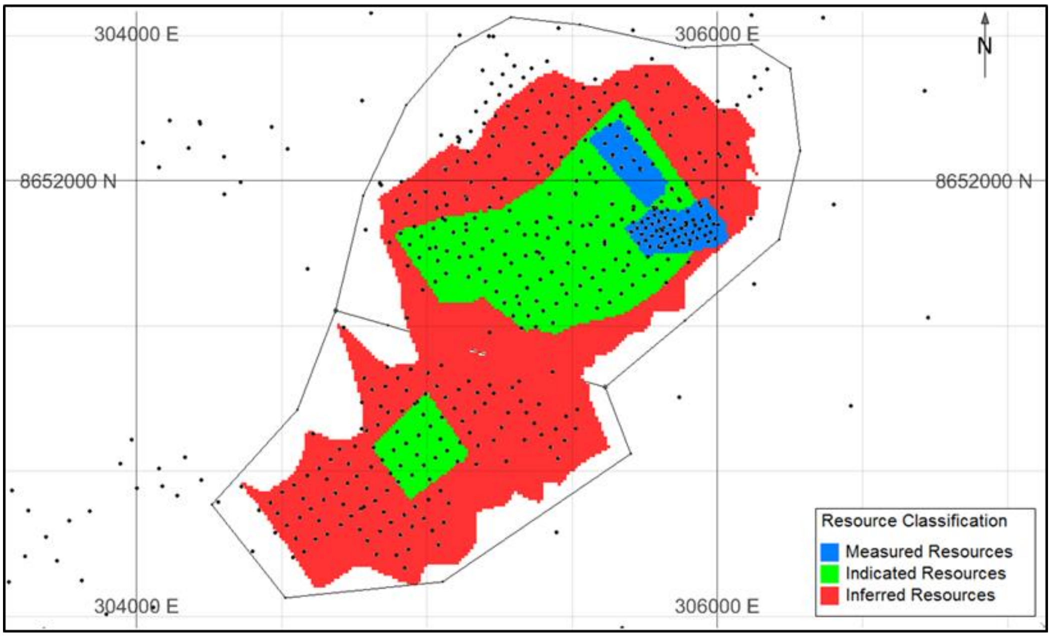
Validation confirmed that the block model reflects the tenor of composite grades, and can be considered as a reliable representation of the in-situ mineralization.

14.2.11 Mineral Resource classification

The Mineral Resource estimate was classified (Figure 14-31) as Measured, Indicated and Inferred in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM, 2014). Classification was based upon: verification of tenement title; review of drilling, sampling, assaying and geology; assessment of the reliability of the geological model; appropriate in-situ dry bulk density for estimation of tonnage; OK variance statistics and appropriate comparison of composites with estimated block grades. The OK slope of regression was an indicator of relative confidence of the grade estimate and was combined with geological confidence and sample spacing to guide classification assignments. Specifically, the following criteria were considered by the QP:

- Measured Mineral Resources wireframes were constructed to delineate estimated blocks within a drill spacing of 50 m by 50 m and had an average slope of regression >= 0.7.
- Indicated Mineral Resources wireframes were constructed to delineate estimated blocks with slope of regression >= 0.5 at Main and that were not already classified as Measured Resources. A small volume of fresh material was classified as Indicated at SW. There is potential for upgrade of Inferred Resources once further bulk density analysis has been completed for saprolite and saprock.
- Inferred Mineral Resource wireframes were constructed to delineate the remaining estimated blocks within the mineralized domains where geological and grade continuity persisted with reasonable estimate confidence, and where the location and character of blocks have a reasonable potential for future economical extraction.

Figure 14-31 Plan view of the Enterprise drillholes and block model coloured by classification



14.2.12 Mineral Resource estimate statement

The Mineral Resource estimate for Enterprise is presented in Table 14-29 and is reported using a 0.15% nickel cut-off grade. The Mineral Resource statement is inclusive of Mineral Reserves and was guided by a life of mine pit shell. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-29 Enterprise Nickel Deposit Mineral Resource statement as at 31st December 2019 and using a 0.15% nickel cut-off grade. Mineral Resources are inclusive of Mineral Reserves

Classification	Material	Tonnes (Mt)	Density (t/m^3)	Ni (%)	Ni metal (kt)
Measured	Non-primary sulphide	3.7	2.56	1.07	39.3
	Primary sulphide	5.5	2.87	1.60	88.1
Measured subtotal		**9.2**	**2.75**	**1.38**	**127.4**
Indicated	Non-primary sulphide	2.7	2.71	0.53	14.4
	Primary sulphide	25.7	2.87	0.95	244.3
Indicated subtotal		**28.5**	**2.86**	**0.91**	**258.7**
Measured and Indicated total		**37.7**	**2.83**	**1.03**	**386.1**
Inferred	Non-primary sulphide	1.1	2.71	0.60	6.7
	Primary sulphide	8.2	2.87	0.73	59.8
Inferred subtotal		**9.3**	**2.85**	**0.71**	**66.5**

To the best knowledge of the QP, the stated Mineral Resources were not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues that prevent this resource from reasonable prospects for economic extraction.

Grade and tonnage tabulations of the Measured and Indicated material at Enterprise are presented in a grade tonnage chart shown in Figure 14-32 and Table 14-30.

Figure 14-32 Grade/tonnage curve for Enterprise Measured and Indicated Resources, depleted as at 31st December 2019

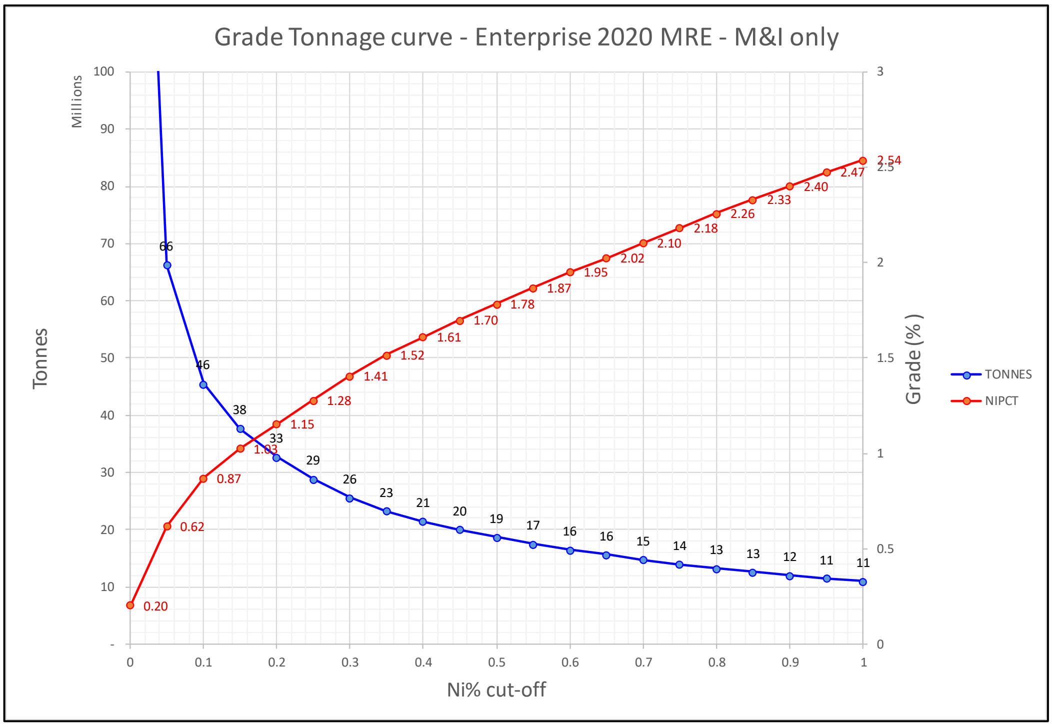


Table 14-30 Grade tonnage curve for Enterprise, for Measured and Indicated only, depleted as at 31st December 2019

Ni % cut-off	Volume (Mm^3)	Density (t/m^3)	Tonnes (Mt)	Ni (%)
0.00	75.3	2.67	209.7	0.20
0.05	23.6	2.60	66.2	0.62
0.10	16.1	2.61	45.5	0.87
0.15	**13.3**	**2.63**	**37.7**	**1.03**
0.20	11.6	2.64	32.7	1.15
0.25	10.2	2.65	28.9	1.28
0.30	9.0	2.66	25.6	1.41
0.35	8.2	2.67	23.2	1.52
0.40	7.6	2.67	21.4	1.61
0.45	7.1	2.67	20.0	1.70
0.50	6.6	2.67	18.7	1.78
0.55	6.2	2.68	17.5	1.87
0.60	5.8	2.68	16.4	1.95
0.65	5.5	2.68	15.6	2.02
0.70	5.2	2.68	14.7	2.10
0.75	4.9	2.68	13.9	2.18
0.80	4.7	2.68	13.2	2.26
0.85	4.4	2.68	12.6	2.33
0.90	4.2	2.68	12.0	2.40
0.95	4.0	2.68	11.4	2.47
1.00	3.9	2.68	11.0	2.54

14.2.13 Comparison with previous Mineral Resource Estimate

Comparison with the previous Mineral Resource estimate is shown in Table 14-31, Table 14-32 and Table 14-33.

Table 14-31 Previous (May 2015) Mineral Resource statement for the Enterprise deposit, using a 0.15% nickel cut-off and depleted as at 31st December 2019

Classification	Material	Tonnes (Mt)	Density (t/m ^{3})	Ni (%)	Ni metal (kt)
Measured	Non-primary sulphide				
	Primary sulphide	5.1	2.90	1.56	80.1
	Measured subtotal	**5.1**	**2.90**	**1.56**	**80.1**
Indicated	Non-primary sulphide	2.2	2.72	0.72	15.6
	Primary sulphide	30.8	2.89	0.96	294.7
	Indicated subtotal	**33.0**	**2.88**	**0.94**	**310.4**
	Measured and Indicated total	**38.1**	**2.88**	**1.02**	**390.5**
Inferred	Non-primary sulphide	2.9	2.71	0.43	12.7
	Primary sulphide	16.4	2.88	0.69	113.0
	Inferred subtotal	**19.4**	**2.86**	**0.65**	**125.7**

Table 14-32 2019 Mineral Resource statement for the Enterprise deposit, using a 0.15% nickel cut-off and depleted as at 31st December 2019

Classification	Material	Tonnes (Mt)	Density (t/m ^{3})	Ni (%)	Ni metal (kt)
Measured	Non-primary sulphide	3.7	2.56	1.07	39.3
	Primary sulphide	5.5	2.87	1.60	88.1
	Measured subtotal	**9.2**	**2.75**	**1.38**	**127.4**
Indicated	Non-primary sulphide	2.7	2.71	0.53	14.4
	Primary sulphide	25.7	2.87	0.95	244.3
	Indicated subtotal	**28.5**	**2.86**	**0.91**	**258.7**
	Measured and Indicated total	**37.7**	**2.83**	**1.03**	**386.1**
Inferred	Non-primary sulphide	1.1	2.71	0.60	6.7
	Primary sulphide	8.2	2.87	0.73	59.8
	Inferred subtotal	**9.3**	**2.85**	**0.71**	**66.5**

Table 14-33 Percentage difference between the 2019 MRE and the 2015 MRE using a 0.15% nickel cut-off and depleted as at 31st December 2019

Classification	Tonnes (Mt)	Ni (%)	Ni metal (kt)
Measured	79%	-11%	59%
Indicated	-14%	-3%	-17%
Measured and Indicated	**-1%**	**0%**	**-1%**
Inferred	-52%	10%	-47%

Tonnes of combined Measured and Indicated resources decreased by 1% due to the improved controls on the grade estimates. Nickel grade remains similar, with metal decreasing by 1% due to the decrease in tonnage.

Inferred Mineral Resource tonnes were assessed against the Mineral Reserve cut-off grade (Item 15.4). The deeper, less continuous and lower grade material was deemed no longer economically viable. 10.1 million tonnes of Inferred Mineral Resources were removed resulting in a 10% increase in nickel grade for the remaining inferred material.

This Mineral Resource estimate uses a much improved geological model to guide and control grades estimates. Mineralization dynamic anisotropy vectors guide estimates and so honour the orientation of mineralization. Additional saprolite and saprock density measurements are required across Enterprise.

It is the opinion of the QP that the resulting changes to this Mineral Resource statement reflects the confidence in the underlying added data and that the estimates are believed representative of the prevailing mineralization.

ITEM 15 MINERAL RESERVE ESTIMATE

Detailed technical information provided under this item relates specifically to the Sentinel and Enterprise Mineral Reserve estimates completed for this Technical Report and based on the Mineral Resource models and estimates as reported in Item 14.

As part of the estimation process, the original pit optimisation work completed for the 2015 Technical Report (FQM, May 2015) was reviewed and updated. Along with more recent detailed pit designs completed by FQM personnel, this review and design work was overseen and supervised by Michael Lawlor (QP) of FQM.

To conform with NI 43-101 standards, the Mineral Reserve estimate is derived from Measured and Indicated Resources only. The Measured and Indicated Mineral Resource estimates as listed in Table Table 14-3 and Table 14-29 are reported inclusive of the associated Mineral Reserve.

15.1 Introduction

Mine planning and Mineral Reserve estimation/reporting in the 2015 Technical Report (FQM, May 2015) addressed the planned continuation of mine development since commencement in 2013, and following the establishment of a starter pit and near-surface pockets excavated for the initial siting of IPCs.

The initial starter pit and mining phase at Sentinel is now over 200 m deep and three IPCs are in operation. Whilst the ultimate design limits of the Sentinel Pit have not changed significantly since 2015, the internal phase layouts and proposed future crusher positions have changed. The development and sequencing of these phases have an impact on the required annual material movement rates and the current proposal to expand the processing capacity from 55 Mtpa to 62 Mtpa.

Other than an update to the Mineral Resource model basis, there are few changes to the Enterprise development plan and concepts as described in the 2015 Technical Report.

15.2 Methodology

The conversion of the Mineral Resource estimate to a Mineral Reserve estimate followed a conventional approach, commencing with open pit optimisation techniques incorporating economic parameters and other “modifying” factors.

The ultimate (optimal) pit outlines (shells) were used to create practical and detailed open pit designs accounting for the siting of IPCs, and the inclusion of batters, berms and haul roads.

These pit designs then provided the bench-by-bench ore and waste mining inventories for the detailed production schedule that demonstrates viable open pit mining. This schedule, which in turn provides the physical basis for cash flow modelling, is described in Item 16.

In the case of Sentinel, the optimisation, mine design and production scheduling were completed in mid-Q4 2019, reflective of mining depletion to the end of October 2019. The production schedule was subsequently updated to reflect mining depletion to year-end. Consequently, the effective date of the Sentinel Mineral Reserve estimate is 31st December 2019.

15.3 Sentinel

15.3.1 Mine planning model

A regularised block model to 6 m x 6 m x 12 m (x, y ,z dimensions) size was provided for pit optimisation and mine planning work. The modelling process as described in Item 14 can be considered as providing a suitable allowance for “planned dilution”.

For the sake of time and efficiency in dealing with an otherwise large model, the regularised model was reblocked to 48 m x 24 m x 12 m within the optimisation software, but without compromising the definition of the original model.

15.3.2 Pit optimisation

Conventional Whittle Four-X software was used to determine the optimal pit shell for the Sentinel deposit. Optimisations, including sensitivity analyses, were completed on a maximum undiscounted cashflow basis, and with recoveries to copper metal in concentrate determined from variable process recovery relationships.

The 2015 Technical Report (FQM, May 2015) described an optimisation analysis process leading to the adoption at that time of an elevated 0.2% Cu cut-off grade. This was appropriate in 2015 when seeking to improve the initial plant feed grade and recovery profile, and thereby consign a portion of the mineralised waste grade (ie, the WTA and WTB ore types) material to long term stockpile. This approach led to a similarly elevated cut-off grade strategy for the life of mine production schedule, the upshot of which was that about 90.5 Mt of marginal grade ore was scheduled as feed to the plant (in the last eight years of mining) and about 147 Mt of the same grade material was consigned to an unreclaimed long term stockpile.

The cut-off grade strategy has since been revised and a marginal cut-off grade optimisation analysis has been carried through into the production scheduling process.

15.3.2.1 Pit slope parameters

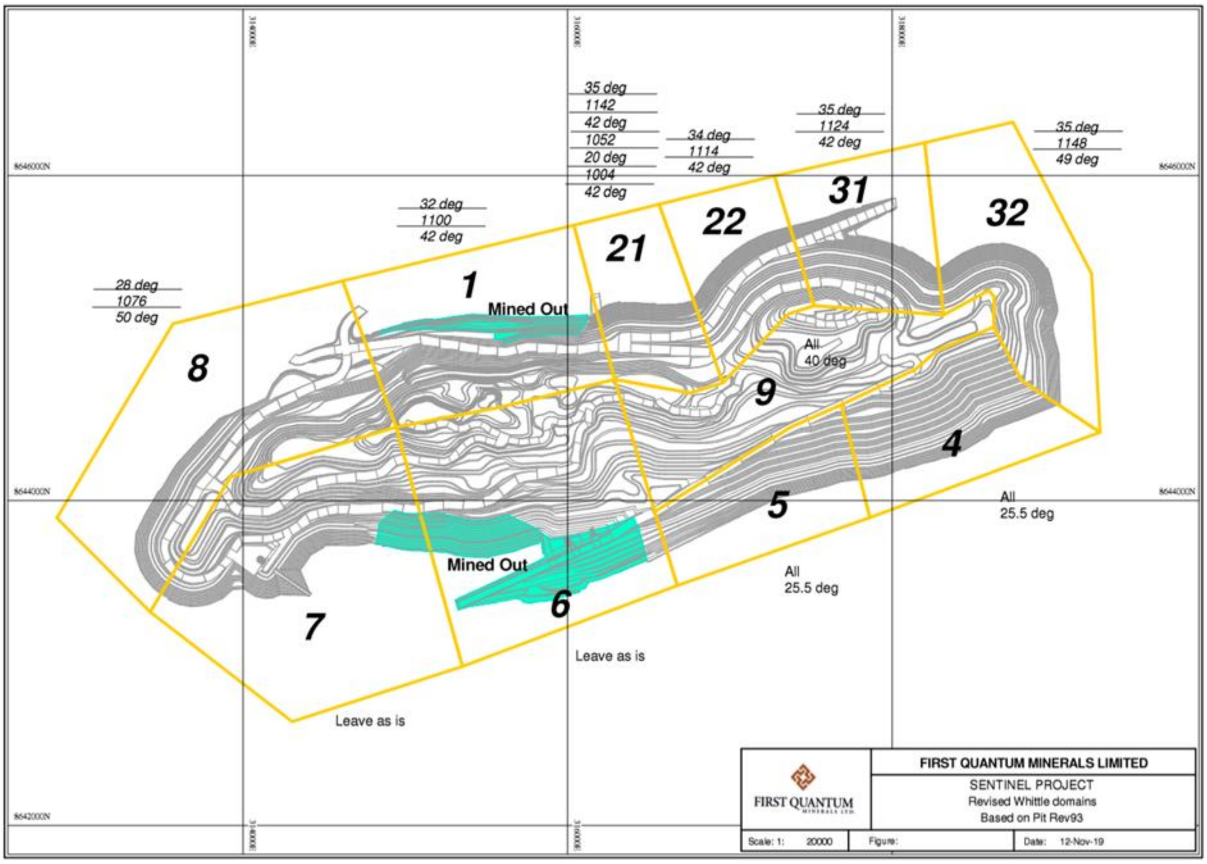
Subsequent to the 2015 Technical Report (FQM, May 2015), pit slope design parameters for Sentinel were geotechnically reassessed in late 2015, and again in 2016 (XStract, 2015 and June 2016). To reflect these geotechnical assessments, the overall slope angles shown in Figure 15-1 were adopted for the pit optimisation; the geotechnical design sectors are shown relative to an interim ultimate pit design.

Geotechnical information in relation to the updated slope design parameters is outlined in Item 16.1.2.

15.3.2.2 Metal price

A long term average copper price of US$3.00/lb (US$6,614/t) was used for the optimisation. This is marginally less than the median long term consensus forecast figure as tabled in Item 22.1.

Figure 15-1 Sentinel overall slope parameters for pit optimisation, Q4 2019



15.3.2.3 Metal recoveries

The two broad ore types differentiated at Sentinel are:

- primary sulphide ore (OTA), and
- non-primary sulphide ore (OTB)

There are also mineralised waste types, defined by a copper grade between the marginal cut-off grade and 0.2%Cu, as follows:

- primary sulphide mineralised waste (WTA), and
- non-primary sulphide mineralised waste (WTB)

For these particular ore types, metal recovery figures in the pit optimisation process reflect performance based equations developed by the site metallurgical team and carried in the mine planning model, yielding overall weighted averages as follows:

- OTA and WTA = 91.4% and 87.7%, respectively; average for the two ore types = 91.2%
- OTB and WTB = 79.5% and 73.1%, respectively; average for the two ore types = 79.0%

The basis for these equations and average recovery figures is described in Item 13.

15.3.2.4 Mining costs

Variable mining costs comprising load and haul unit costs, on a bench-by-bench basis, were determined from a haulage optimisation study (Item 16.1) taking account of longer term trolley-assisted haulage routes and shortened ore hauls to in-pit crushers. This approach has been a significant improvement on the estimation of the mining unit costs used for the 2015 Technical Report optimisation (FQM, May 2015). Item 21 provides more information on the following updated relationships adopted for assigning incremental ore and waste mining costs in the pit optimisation process:

- incremental waste load and haul costs (bcm) = -0.0065 x RL + 10.548
- incremental ore load and haul costs (bcm) = -0.0029 x RL + 6.8718

From these relationships, the overall average load and haul costs are as follows:

- average waste mining cost = $3.41/bcm ($1.26/t)
- average ore mining cost = $3.83/bcm ($1.37/t)

A fixed cost of $1.65/bcm was added in the optimisation for drill and blast, irrespective of mining bench level.

15.3.2.5 Operating costs

Since the Project will be mill constrained, the process operating costs are the sum of the fixed and variable costs. The costs as used in the optimisation described in the 2015 Technical Report (FQM, May 2015) were reviewed and updated as follows:

- fixed operating costs (including general and administration costs (G&A)) = $1.87/t processed
- variable operating costs = $3.98/t processed
- total operating costs = $5.85/t processed

More information on these cost estimates and their derivation is provided in Item 21.

15.3.2.6 Metal costs

Metal cost assumptions were as set out in Table 15-1. The applicable copper metal royalty rate was increased from 6% to 7.5%.

Table 15-1 Sentinel metal costs for pit optimisation, Q4 2019

Transport, Smelting and Refining Charges	Units	$/unit
Costs associated with KCS smelting		
Concentrate freight cost	$/t con.	27.29
KCS smelter cost	$/t con.	82.50
Anode transport cost	$/t metal	183.64
Refining charge	c/lb Cu	6.08
Costs associated with smelting elsewhere		
Concentrate freight cost	$/t con.	130.99
Smelter cost	$/t con.	80.72
Refining charge	c/lb Cu	8.09

Concentrate from Sentinel will be road hauled to the Kansanshi smelter (KCS), the capacity of which is to be expanded to 1.6 Mtpa concentrate received. Product from the Kansanshi concentrator will have KCS smelting preference, therefore the balance of the smelter capacity will be taken up by concentrate from Sentinel. Sentinel concentrate that exceeds KCS capacity (estimated to be 37% of the total concentrate produced at Sentinel) is intended to be road hauled to other smelters in the Zambian Copperbelt.

The net effect of the costs listed in Table 15-1 is a proportional average metal cost of $0.60/lb Cu ($1,319/t Cu), inclusive of the 7.5% royalty charge. This represents a negligible change from the 2015 Technical Report (FQM, May 2015) equivalent estimates.

Further information on the metal cost estimates is provided in Item 21.

15.3.2.7 Mining dilution and recovery factors

For consistency with the production scheduling assumptions adopted for the 2015 Technical Report (FQM, May 2015), and to rectify a deficiency in the optimisation carried out at that time, an unplanned mining dilution factor of 103% and a mining recovery factor of 97% were included in the optimisation update. As per the conclusions drawn in Item 16.1, there was considered to be no basis for changing this allowance after reviewing production reconciliation data.

Despite this conclusion, the sensitivity to unplanned mining dilution was nevertheless tested in the optimisation sensitivity analyses described below.

15.3.2.8 Optimisation inputs summary

Table 15-2 lists updated pit optimisation parameters applicable after adopting the mining, processing, G&A and metal costs as per the information and tables above. The inputs from the 2015 Technical Report are also listed in Table 15-2 for comparison.

15.3.2.9 Marginal cut off grades

Whittle optimisation software uses the following simplified formula to calculate the marginal cut-off grade as listed in Table 15-3. The cut-off grades reported in the 2015 Technical Report are listed in Table 15-3 for comparison.

Marginal COG = (PROCOST x MINDIL) ÷ (NR), where …

PROCOST is the sum of the processing cost plus the ore mining cost differential,

MINDIL is the unplanned mining dilution factor, and

NR is the net return (ie, the metal price less the metal costs)

Table 15-2 Sentinel pit optimisation inputs, Q4 2019

SENTINEL OPTIMISATION	Units	2015 INPUTS		Q4 2019 INPUTS			
PROCESS		Non-Primary	Primary	Non-Primary	Primary	Non-Primary	Primary
Ore types		OTB, WTB	OTA, WTA	OTB, WTB	OTA, WTA	OTB, WTB	OTA, WTA
Smelter		**Kansanshi**	**Kansanshi**	**Kansanshi**	**Kansanshi**	**Copperbelt**	**Copperbelt**
Optimisation Metal Price							
Copper Price	$/lb	3.00	3.00	3.00	3.00	3.00	3.00
Copper Price	$/tonne	6,614	6,614	6,614	6,614	6,614	6,614
Mining Parameters							
Mining Recovery	Factor	1.00	1.00	0.97	0.97	0.97	0.97
Dilution Factor	Factor	1.00	1.00	1.03	1.03	1.03	1.03
Throughput Rates							
Milling Circuit	Mtpa	55.00	55.00	62.00	62.00	62.00	62.00
Metal Recovery Factors				Average	Average	Average	Average
TotCu to Concentrate Recovery (Linear Calc)	%	70.0%	92.0%	79.1%	91.2%	79.1%	91.2%
Mining Costs				Average	Average	Average	Average
overall average waste cost	$/t	1.84	1.84	1.87	1.87	1.87	1.87
overall average ore cost	$/t	1.77	1.77	1.97	1.97	1.97	1.97
overall average waste+ore cost	$/t	1.82	1.82	1.90	1.90	1.90	1.90
Treatment and G&A Costs							
Variable Treatment							
Sub-total variable	$/t process	5.50	5.50	3.98	3.98	3.98	3.98
Fixed Costs							
Sub-total fixed	$/t process	0.60	0.60	1.87	1.87	1.87	1.87
Total Treatment	**$/t process**	**6.10**	**6.10**	**5.85**	**5.85**	**5.85**	**5.85**
Metal Costs Copper Concentrate							
Royalty rate	%	6.0%	6.0%	7.5%	7.5%	7.5%	7.5%
Concentrate Grade	% Cu	24%	24%	26.5%	26.5%	26.5%	26.5%
Moisture Content	%	10%	10%	10%	10%	10%	10%
Realisation/freight							
Concentrate Transport (wet)	$/t conc	25.00	25.00	24.56	24.56	117.89	117.89
Concentrate Transport (dry)	$/t conc	27.78	27.78	27.29	27.29	130.99	130.99
subtotal	**$/t Cu**	**115.74**	**115.74**	**102.97**	**102.97**	**494.32**	**494.32**
subtotal	**$/lb Cu**	**0.05**	**0.05**	**0.05**	**0.05**	**0.22**	**0.22**
Kansanshi Smelter Treatment							
Treatment Cost (wet)	$/t conc	63.70	63.70	74.25	74.25		
Treatment Cost (dry)	$/t conc	70.78	70.78	82.50	82.50		
subtotal	**$/t Cu**	**294.91**	**294.91**	**311.32**	**311.32**	**0.00**	**0.00**
subtotal	**$/lb Cu**	**0.13**	**0.13**	**0.14**	**0.14**	**0.00**	**0.00**
Concentrate Shipped to Copper Belt							
Treatment Cost (wet)	$/t conc					72.65	72.65
Treatment Cost (dry)	$/t conc					80.72	80.72
subtotal	**$/t Cu**			**0.00**	**0.00**	**304.59**	**304.59**
subtotal	**$/lb Cu**			**0.00**	**0.00**	**0.14**	**0.14**
Smelter deductions (payability&conc losses)							
	$/t Cu	**296.30**	**296.30**				
subtotal	**$/lb Cu**	**0.13**	**0.13**				
Anode Transport Costs							
Transport (dry)	$/t metal			183.64	183.64		
subtotal	**$/t Cu**			183.64	183.64		
subtotal	**$/lb Cu**			**0.08**	**0.08**	**0.00**	**0.00**
Refining Charges							
	c/lb	7.00	7.00	6.08	6.08	8.09	8.09
subtotal	**$/t Cu**	**154.32**	**154.32**	**134.00**	**134.00**	**178.33**	**178.33**
subtotal	**$/lb Cu**	**0.07**	**0.07**	**0.06**	**0.06**	**0.08**	**0.08**
Royalties x% Gross							
	$/t Cu	**396.83**	**396.83**	**496.04**	**496.04**	**496.04**	**496.04**
subtotal	**$/lb Cu**	**0.18**	**0.18**	**0.23**	**0.23**	**0.23**	**0.23**
Total Concentrate Cu Metal Cost							
Concentrate proprtion to smelters	**%**	100%		63%		37%	
	$/t Cu	**1,258.10**	**1,258.10**	**1,227.97**	**1,227.97**	**1,473.28**	**1,473.28**
	$/10kg Cu	**12.58**	**12.58**	**12.28**	**12.28**	**14.73**	**14.73**
Average Concentrate Cu Metal Cost							
Concentrate Cu Metal Cost	**$/t Cu**			**1,319.32**		**1,319.32**	
	$/10kg Cu			**13.19**		**13.19**	

Table 15-3 Sentinel marginal cut-off grades, Q4 2019

SENTINEL OPTIMISATION	Units	2015 CUT-OFF GRADES		2019 CUT-OFF GRADES			
PROCESS		Non-Primary	Primary	Non-Primary	Primary	Non-Primary	Primary
Ore types		OTB, WTB	OTA, WTA	OTB, WTB	OTA, WTA	OTB, WTB	OTA, WTA
Smelter		**Kansanshi**	**Kansanshi**	**Kansanshi**	**Kansanshi**	**Copperbelt**	**Copperbelt**
Marginal Cut-Off Grade Calculation							
Grade Attribute		**TCu**	**TCu**	**TCu**	**TCu**	**TCu**	**TCu**
Process Cost	$/t mill	6.10	6.10	5.85	5.85	5.85	5.85
Dilution Factor		1.00	1.00	1.03	1.03	1.03	1.03
Net Return (Metal price - Total metal cost)	$/10kg	53.56	53.56	53.86	53.86	51.41	51.41
Recovery	%	70.0%	92.0%	79.1%	91.2%	79.1%	91.2%
Marginal Cut-Off Grade Direct Feed	**% Cu**	**0.16**	**0.12**	**0.14**	**0.12**	**0.15**	**0.13**
Average Marginal Cut-Off Grade Calculation							
Net Return (Metal price - Total metal cost)	$/10kg			52.95	52.95	52.95	52.95
Marginal Cut-Off Grade Direct Feed	**% Cu**			**0.14**	**0.12**	**0.14**	**0.12**

15.3.2.10 Optimisation results

Figure 15-2 shows the graphical results of the base case pit optimisation, whilst Table 15-4 lists the complete inventory of shell sizes and corresponding undiscounted cashflows (excluding Project capital). As was done in 2015, the optimal ultimate pit shell (highlighted yellow in Table 15-4) was selected on a maximum cashflow basis.

Figure 15-2 Sentinel pit optimisation results, Q4 2019

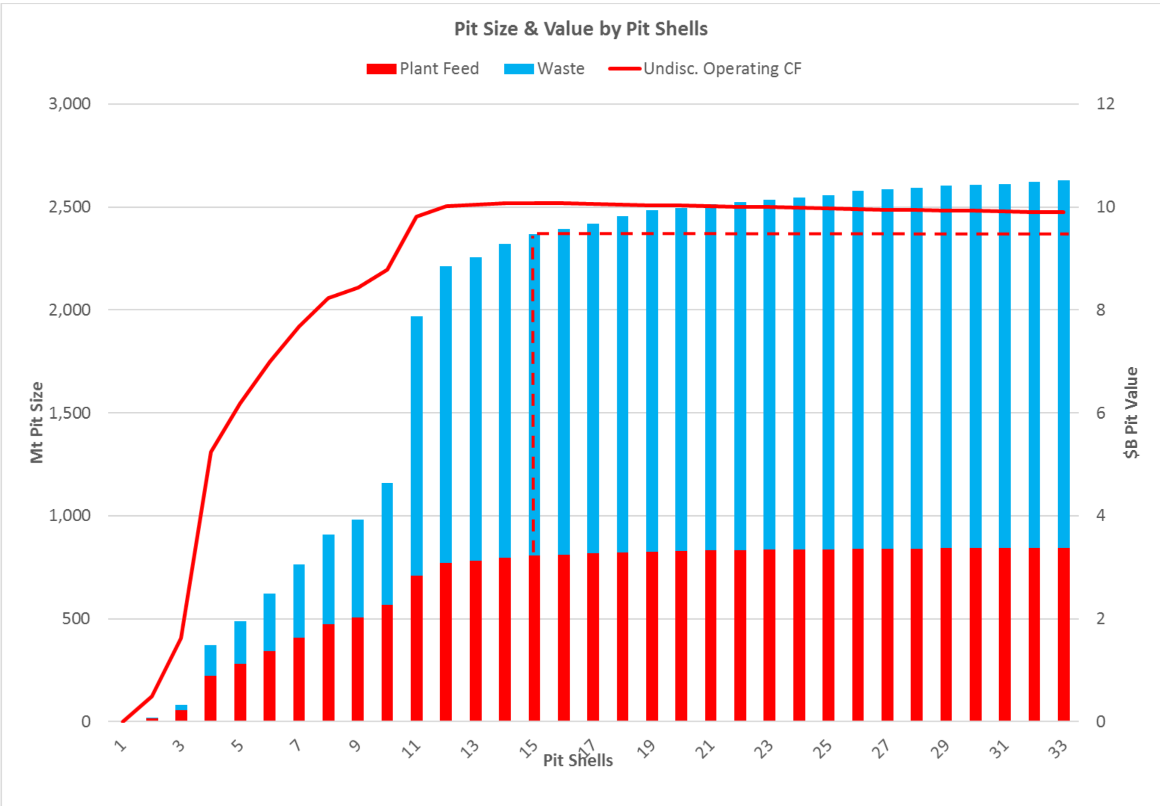


Table 15-4 Sentinel pit optimisation shell sizes and cashflow, Q4 2019

Pit Number	Revenue Factor	Pit Size (Mt)	Waste (Mt)	Strip Ratio	Plant Feed (Mt)	Plant Feed (%Cu)	OTA Feed (Mt)	OTB Feed (Mt)	Recovered Cu (kt)	Undisc. Cashflow ($B)
1	0.30	0.0	0.0	0.3	0.0	1.37	0.0	0.0	0.0	0.00
2	0.35	2.7	0.2	0.1	2.5	0.93	2.2	0.3	21.7	0.10
3	0.40	73.6	20.1	0.4	53.5	0.75	51.1	2.4	365.8	1.48
4	0.45	358.0	132.5	0.6	225.6	0.65	201.3	24.3	1,330.7	5.03
5	0.50	442.7	170.4	0.6	272.4	0.63	241.9	30.5	1,555.5	5.78
6	0.55	602.7	255.8	0.7	347.0	0.60	309.1	37.8	1,883.6	6.78
7	0.60	717.9	312.1	0.8	405.8	0.57	359.1	46.7	2,099.0	7.35
8	0.65	875.5	390.8	0.8	484.6	0.54	424.4	60.2	2,360.8	7.98
9	0.70	965.7	438.1	0.8	527.6	0.52	460.9	66.7	2,491.5	8.25
10	0.75	1,090.1	510.3	0.9	579.8	0.51	507.4	72.4	2,642.6	8.50
11	0.80	1,933.8	1,182.7	1.6	751.1	0.49	666.7	84.4	3,336.5	9.57
12	0.85	2,116.1	1,316.7	1.6	799.4	0.48	713.5	85.9	3,488.6	9.74
13	0.90	2,252.8	1,406.1	1.7	846.7	0.47	759.3	87.4	3,610.4	9.84
14	0.95	2,314.2	1,452.9	1.7	861.3	0.47	773.6	87.6	3,653.2	9.87
15	1.00	2,368.3	1,492.5	1.7	875.8	0.47	786.9	88.8	3,690.2	9.87
16	1.05	2,403.4	1,517.1	1.7	886.3	0.46	796.8	89.4	3,714.1	9.87
17	1.10	2,436.6	1,541.5	1.7	895.1	0.46	805.2	90.0	3,734.2	9.86
18	1.15	2,468.9	1,567.1	1.7	901.8	0.46	811.6	90.2	3,750.9	9.84
19	1.20	2,493.5	1,586.2	1.7	907.3	0.46	817.0	90.3	3,763.3	9.83
20	1.25	2,523.7	1,609.8	1.8	913.9	0.46	823.3	90.6	3,777.4	9.81
21	1.30	2,536.2	1,619.1	1.8	917.2	0.46	826.4	90.8	3,783.3	9.79
22	1.35	2,548.8	1,629.7	1.8	919.2	0.46	828.2	90.9	3,788.1	9.78
23	1.40	2,562.0	1,640.6	1.8	921.5	0.46	830.4	91.1	3,793.0	9.77
24	1.45	2,574.5	1,650.8	1.8	923.7	0.46	832.6	91.1	3,797.4	9.75
25	1.50	2,598.7	1,671.7	1.8	927.0	0.46	835.7	91.3	3,804.9	9.72
26	1.55	2,610.4	1,681.6	1.8	928.8	0.45	837.4	91.3	3,808.4	9.71
27	1.60	2,620.8	1,690.7	1.8	930.1	0.45	838.6	91.4	3,811.2	9.70
28	1.65	2,630.2	1,699.1	1.8	931.1	0.45	839.6	91.5	3,813.5	9.68
29	1.70	2,641.9	1,709.7	1.8	932.2	0.45	840.6	91.6	3,816.3	9.67
30	1.75	2,645.9	1,713.3	1.8	932.6	0.45	841.0	91.6	3,817.1	9.66
31	1.80	2,653.7	1,720.2	1.8	933.5	0.45	841.9	91.6	3,818.9	9.65
32	1.85	2,666.8	1,732.2	1.9	934.6	0.45	842.9	91.7	3,821.5	9.63
33	1.90	2,673.6	1,738.3	1.9	935.3	0.45	843.5	91.8	3,822.7	9.62

Figure 15-3 and Figure 15-4 show waterfall chart comparisons between the selected optimal shell and the corresponding shell from the 2015 Technical Report (FQM, May 2015)[2]. These charts are in respect of total plant feed (ore) and recovered copper, respectively, and serve to indicate the impact of updates to the Mineral Resource model, mining and processing costs, process recovery, mining dilution and metal cost.

[2] The 2015 shell is from the equivalent marginal cut-off grade optimisation.

Figure 15-3 Optimisation waterfall chart, ore comparison between 2015 and 2019 pit shells

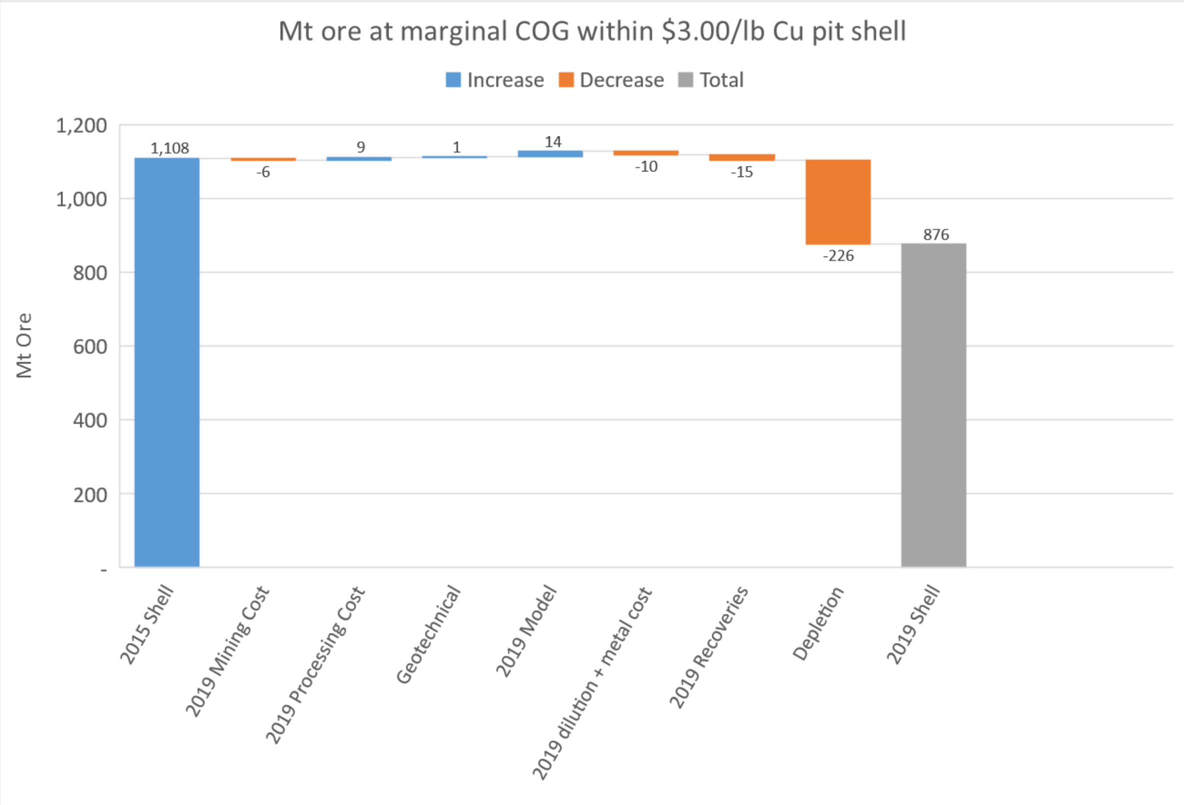
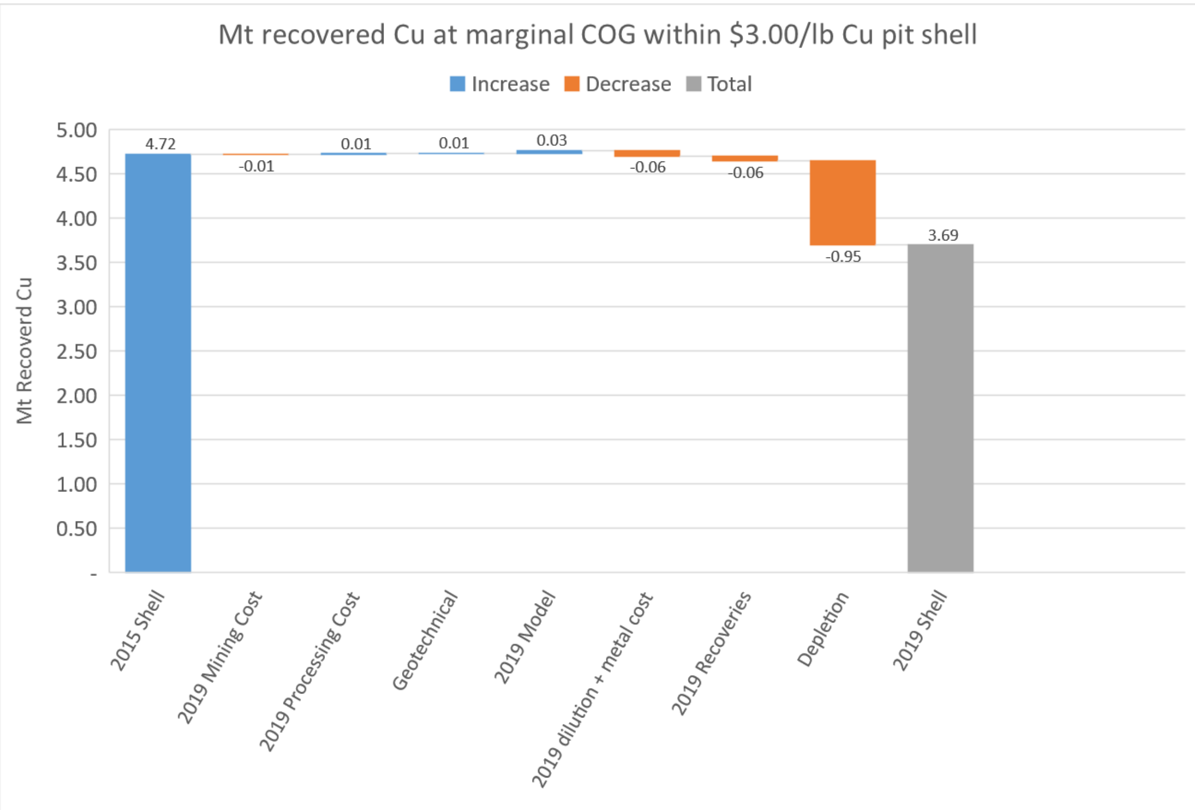


Figure 15-4 Optimisation waterfall chart, recovered metal comparison between 2015 and 2019 pit shells



15.3.2.11 Optimisation sensitivity analyses

Several optimisation sensitivity analyses (Table 15-5) were carried out in order to separately test the impact of:

- higher/lower mining costs
- higher/lower processing plus G&A costs
- higher/lower unplanned mining dilution allowance factors

These optimisation variables are typically less sensitive than those directly related to gross revenue, ie, plant feed grade, metal price and processing recovery. The magnitude of the cashflow impact of any of these three, when individually varied, is essentially the same. Whilst the sensitivity of these has been previously tested (FQM, May 2015), it is considered that:

- varying the modelled copper grades is unwarranted considering the rigour of the estimation process described in Item 14
- the copper metal price is based on a conservative median long term consensus projection
- variable processing recovery projections now reflect the best available information, based on plant performance to date

Accompanying the cost and mining dilution sensitivity analysis results in Table 15-5, Figure 15-5 and Figure 15-6 shows comparison 'spider' charts on the relative impacts of these sensitivity variables to plant feed and pit value, respectively[3].

Processing (plus G&A) cost changes have more of an effect on the optimal plant feed inventory, relative to mining cost changes. Processing and mining cost changes, on the other hand, have a similar level of effect on pit value since they both reduce or increase the net revenue to comparable extents. Whilst adding to the plant feed inventory, an additional 2% of unplanned mining dilution has a -4% impact on undiscounted cashflow.

Table 15-5 Sentinel optimisation sensitivity analysis results, Q4 2019

Pit Number	Revenue Factor	Pit Size (Mt)	Waste (Mt)	SR	Plant Feed (Mt)	Plant Feed by Ore Type			Recovered Cu (Mt)	Undisc. Cashflow ($B)	Comments
						(%Cu)	OTA (Mt)	OTB (Mt)			
Base Case Optimisation - depleted for May 2015 to October 2019											
15	1	2,368.3	1,492.5	1.7	**875.8**	0.47	786.9	88.8	**3.7**	**9.87**	Base, depleted
Sensitivity Case Optimisations - not depleted for May 2015 to October 2019											
14	1	2,975.5	1,874.2	1.7	**1,101.3**	0.47	957.5	143.7	**4.6**	**12.46**	Base, not depleted
14	1	2,945.2	1,850.8	1.7	**1,094.4**	0.47	951.7	142.7	**4.6**	**12.18**	+5% mining costs
14	1	2,921.9	1,832.9	1.7	**1,089.0**	0.47	946.5	142.4	**4.6**	**11.90**	+10% mining costs
14	1	2,985.7	1,881.2	1.7	**1,104.5**	0.47	960.5	144.0	**4.6**	**12.74**	-5% mining costs
14	1	3,013.3	1,903.4	1.7	**1,110.0**	0.47	965.7	144.2	**4.7**	**13.02**	-10% mining costs
14	1	2,949.0	1,873.7	1.7	**1,075.3**	0.48	936.7	138.6	**4.6**	**12.14**	+5% process and G&A costs
14	1	2,935.1	1,883.1	1.8	**1,052.0**	0.48	917.8	134.2	**4.6**	**11.83**	+10% process and G&A costs
14	1	2,983.0	1,854.7	1.6	**1,128.3**	0.46	979.8	148.6	**4.7**	**12.79**	-5% process and G&A costs
14	1	2,994.1	1,836.1	1.6	**1,158.0**	0.45	1,004.1	153.8	**4.7**	**13.12**	-10% process and G&A costs
14	1	2,978.7	1,871.7	1.7	**1,107.1**	0.47	962.4	144.7	**4.7**	**12.71**	2% unplanned mining dilution
14	1	2,960.6	1,866.8	1.7	**1,093.8**	0.47	951.1	142.7	**4.6**	**12.21**	4% unplanned mining dilution
14	1	2,948.6	1,862.8	1.7	**1,085.8**	0.46	944.8	140.9	**4.5**	**11.96**	5% unplanned mining dilution

[3] Pit shell 15 is the base case (selected shell) in optimisations inclusive of mining depletion. Pit shell 14 is the base case in sensitivity analysis optimisations exclusive of mining depletion

Figure 15-5 Optimisation sensitivity analysis 'spider' chart in relation to plant feed

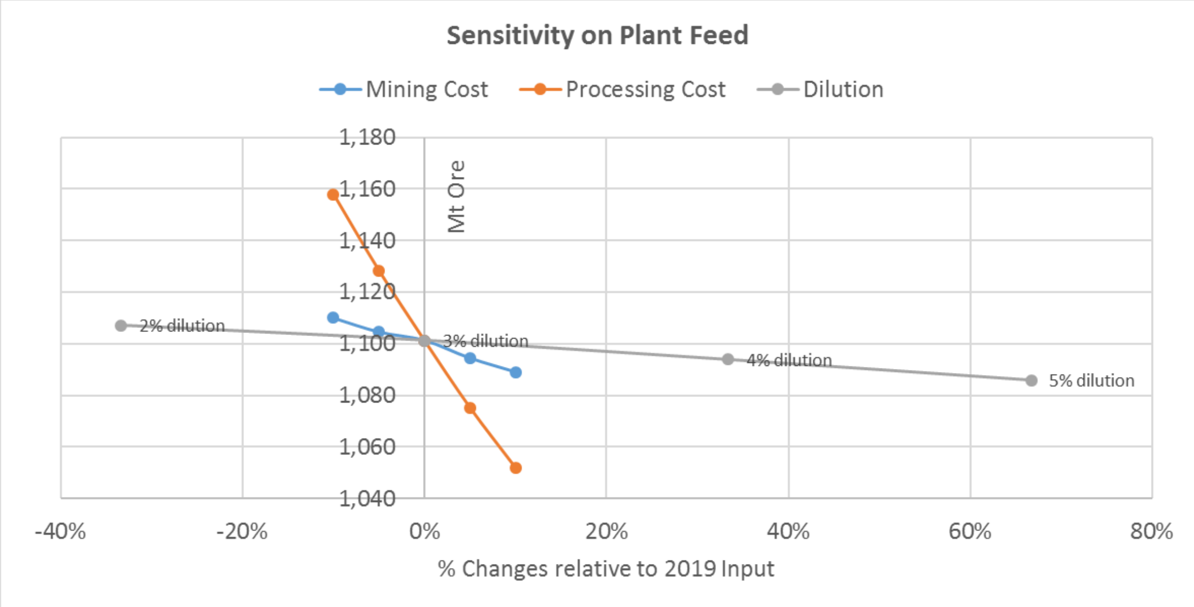
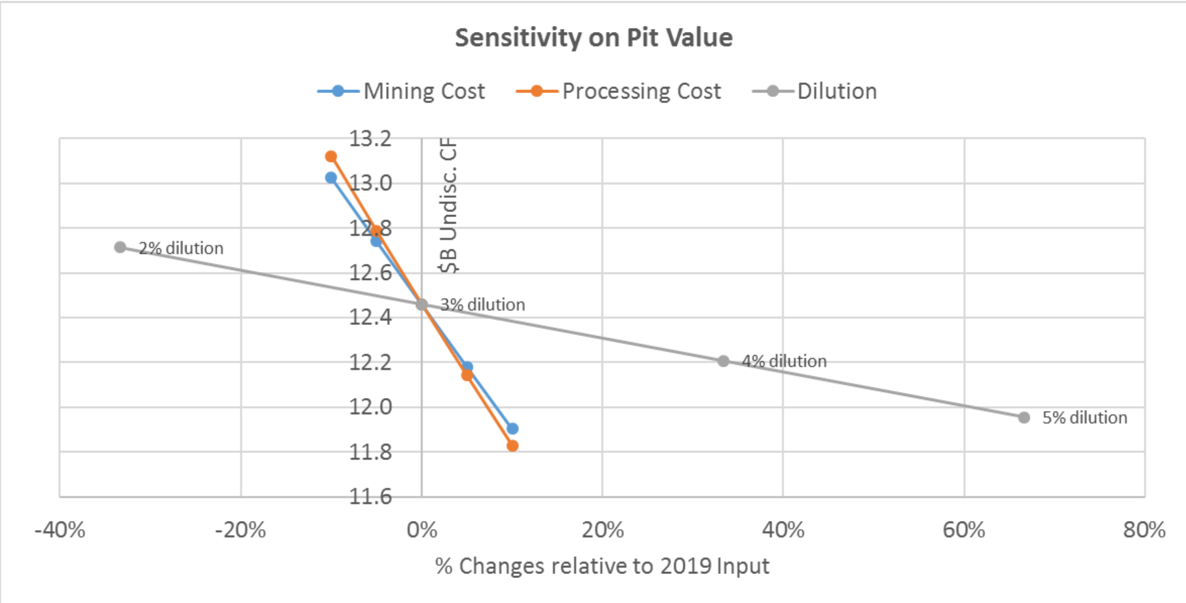


Figure 15-6 Optimisation sensitivity analysis 'spider' chart in relation to pit value



As a post-script to the economic analysis described in Item 22.2.4, two final sensitivity analyses were conducted to separately test the impact of reducing the ore mining cost to match that as revised for cashflow modelling (Item 22), and then increasing the average processing cost of $5.85/t (variable plus fixed plus G&A costs) to $6.02/t to account for revised Project cost estimates between the time of the optimisation and the cashflow modelling (ie, a 3% increase in the unit cost).

Table 15-6 shows the results of these separate analyses with respect to the base case optimisation. The separate impact of these is considered to be within the tolerance of converting the optimal pit shell into a detailed pit design.

Table 15-6 Sentinel optimisation sensitivity analysis results, Q1 2020

Pit Number	Revenue Factor	Pit Size (Mt)	Waste (Mt)	SR	Plant Feed (Mt)	Plant Feed by Ore Type (%Cu)	OTA (Mt)	OTB (Mt)	Recovered Cu (Mt)	Undisc. Cashflow ($B)	Comments
Base Case Optimisation - depleted for May 2015 to October 2019											
15	**1**	**2,368.3**	**1,492.5**	**1.7**	**875.8**	**0.47**	**786.9**	**88.8**	**3.69**	**9.87**	**Base, depleted**
15	1	2,373.2	1,482.1	1.7	**891.1**	0.46	798.1	93.0	**3.71**	**10.11**	$1.66/t ore mining cost
14	1	2,353.4	1,491.1	1.7	**862.3**	0.47	775.1	87.2	**3.67**	**9.72**	$6.02/t processing cost (Opex+GA)
Delta		100%	99%	98%	102%		101%	105%	100%	102%	
Delta		99%	100%	101%	98%		98%	98%	99%	98%	

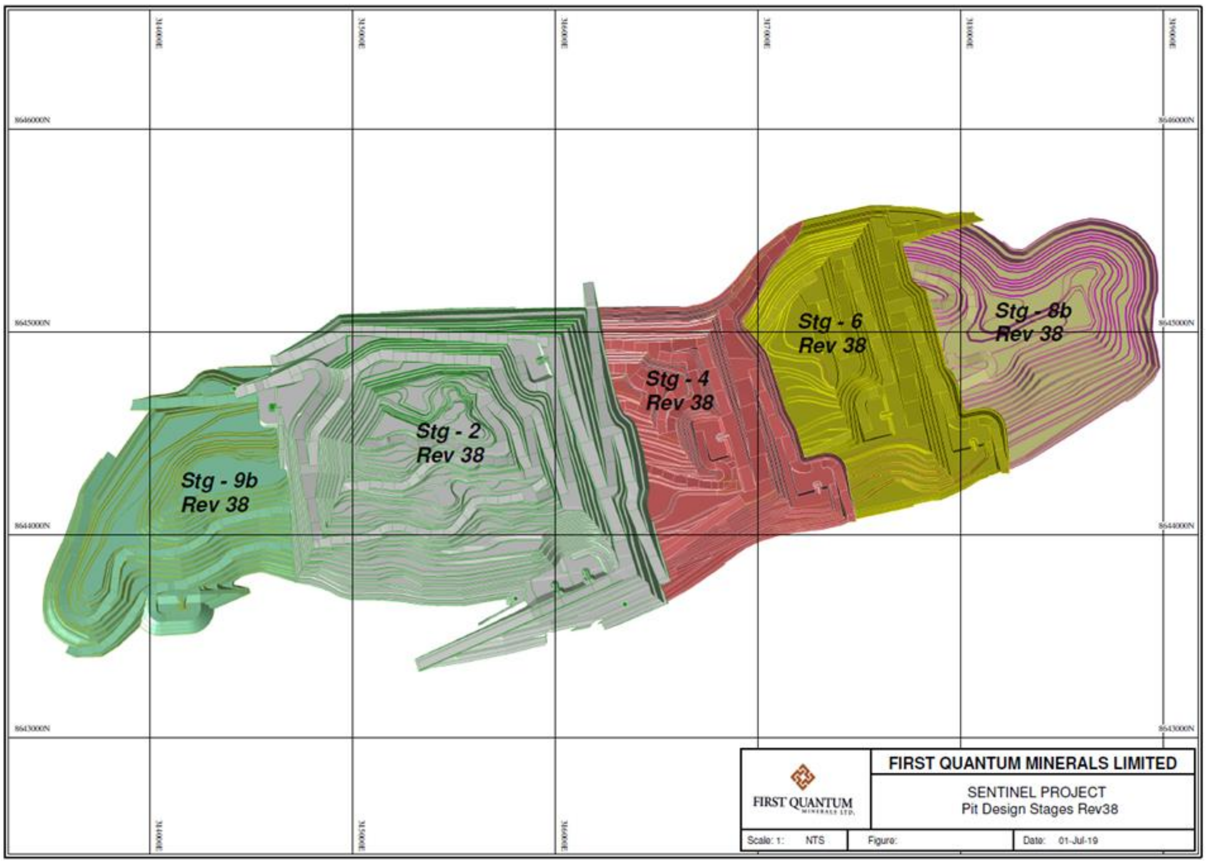
15.3.3 Detailed pit designs

Following the pit optimisation, a series of revised phase (or stage) pit designs were developed starting with the selected ultimate pit shell 15 (Figure 15-2 and Table 15-4).

15.3.3.1 Revised phase design concept

The 2015 Technical Report (FQM, May 2015), described an open pit design reflecting a layout of five intermediate phases, progressing to the ultimate limits as shown in Figure 15-7. The ultimate design was 5.4 km long, 1.5 km wide and 375 m deep. In Figure 15-7, Phase 3 overlies Phase 4, Phase 5 overlies Phase 6, and Phase 7 overlies Phase 8. The along strike dimensions and the number of pit phases had been determined from numerous production schedule iterations. These did not adhere strictly to lesser shells evident from the pit optimisation process, but rather reflected what was considered to be an orderly progression of phased or staged mining, suited to the deployment of ultra-class mining equipment, whilst satisfying desired processing targets and with an ability to accommodate a logical layout for in-pit crushing and conveying.

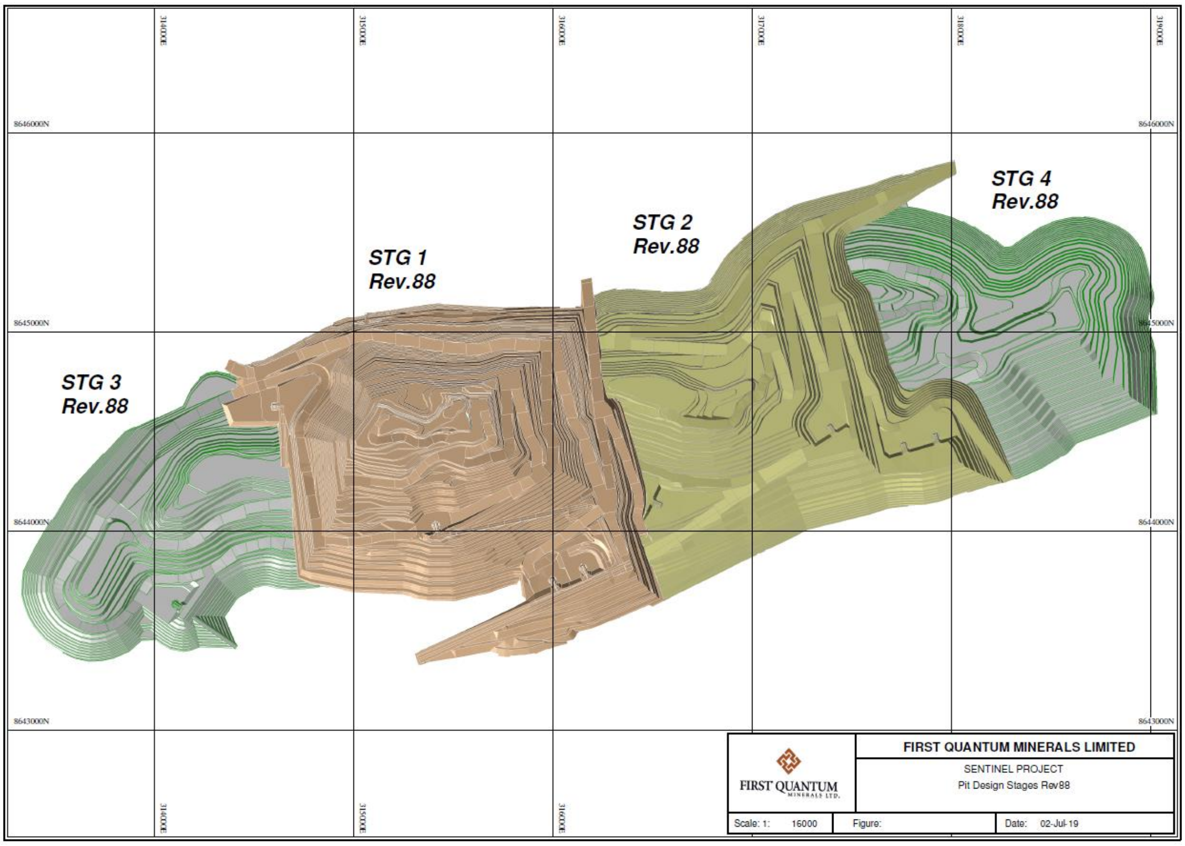
Figure 15-7 Sentinel Pit design phases, 2015 Technical Report



Technical analyses were completed in 2018 to reassess the Sentinel pit phases as to whether there should be fewer or more than five phases. The analyses involved strategic level designs, production schedules and cashflow models to assess relative ore supply from fewer or more than five phases, whilst also considering adequate working space for ultra-class equipment, in addition to total material movement and primary equipment requirements.

The most favourable economic outcome was for a four-phase pit layout as shown in Figure 15-8. This revised phase layout has since become the basis for Sentinel long term planning.

Figure 15-8 Sentinel Pit design phases, Q4 2019



15.3.3.2 Geotechnical design parameters

Table 15-7 lists the pit slope parameters used for detailed pit design.

The design height between pit berms is 24 m whilst blast blocks are typically 12 m high[4]. Mining flitches are generally 12 m high, reducing to 6 m high in certain areas as a means to minimise mining dilution.

Further geotechnical information in relation to pit slope design parameters is outlined in Item 16.1.2

[4] Depending on future operational performance, the bench height in the fresh rock ultimate phase walls may be reduced to 12 m, but without changing the inter-ramp slope angle.

Table 15-7 Sentinel pit slope design parameters, updated June 2016

Domain	Depth Range (m)	Wall orientation	Bench height (m)	Batter angle (degrees)	Min. berm width (m)	IRA (C to C)
Weathered	0 to 60~70	South facing wall Sectors 1-2-3	6	67.0	6	35.0
Weathered	0 to 30~40	South facing wall Sector 8	6	37.2	4	26.0
MPHB	> 60	South facing wall Sector 1-2-3-8	12	34.0	6	27.0
MPH	> 60	South facing wall Sector 1-2-3-8	24	75.0	10	55.6
MSC	> 60	South facing wall Sector 1-2-3-8	24	75.0	10	55.6
Weathered	0 to 35~45	North facing wall Sectors 4-5	6	37.5	4	27.0
Weathered	0 to 35~45	North facing wall Sectors 6-7	6	37.5	6	24.0
MPHB	> 45	North facing wall Sectors 4-5-6-7	12	34.0	6	27.0
MPH	> 45	North facing wall Sectors 4-5-6-7	24	37.6	16	27.0
MSC	> 45	North facing wall Sectors 4-5-6-7	24	75.0	10	55.6

15.3.3.3 Other design and planning parameters

Drawing on the information included under Item 16.1, general design parameters for haul road layouts were as follows:

- minimum haul road width = 32.5 m
- haul road width to cater for trolley-assist = 50 m
- maximum haul ramp gradient = 1: 10 (approximately 6^{o})

15.3.3.4 Phase and ultimate pit designs

Figure 15-9 shows the starting point for the design (and production scheduling) process. This figure shows the extent of mining as at the end of October 2019. An in-pit crusher (IPC3A) is shown where currently positioned in the north west of Phase 1. From this position, ore is conveyed ex-pit to a surface transfer bin and thence onwards to the crushed ore stockpile at the plant. Also shown on Figure 15-9 are the two in-pit crushers (IPC1A and 2A) in the south of Phase 1 and a conveyor ramp leading out of the pit towards the surface transfer bin.

Figure 15-9 also shows the current extent of waste dumping on either side of the pit, and the location of mineralized waste (WTA) and nickel mineralization stockpiles. The nickel mineralization has been encountered during mining but cannot be viably processed.

Figure 15-10 shows the updated ultimate pit, designed around the selected optimal pit shell 15. Figure Figure 15-11 and Figure 15-12 show the updated Phase 1 and Phase 2 pit designs, respectively. These figures shows the location of IPCs in each phase and the location of proposed surface conveyors.

15.3.3.5 In-pit crusher relocations

Engineering and planning around IPC locations has been a continuously on-going process since 2012. Many design iterations have been produced with each one attempting to optimise and account for numerous technical considerations. Item 16.1.2 describes the latest proposed crusher relocation concepts, relative to those as described and presented in the 2015 Technical Report.

The effective use of IPCs relies on the minimisation of ore stockpiles. The design and sequencing of the Sentinel mining phases, however, ensures that there is generally always one crusher located near surface. This enables some stockpiling/reclaim capability with a short haul across to a nearby crusher.

Figure 15-13 shows the updated design ultimate pit, including proposed longer-term relocation positions for the IPCs and inpit plus surface conveyor extensions. The sequencing and timeframe for the crusher relocations is described in Item 16.1.3. These positions satisfy the current Mineral Reserves planning and production requirements but may be modified in future strategic planning work.

By way of explaining the IPC labelling convention:

- there are four crushers, ie IPC1, IPC2, IPC3 and IPC4
- the initial location of each of these is designated as position A, eg IPC1A
- subsequent relocations are designated as positions B to C to D

Figure 15-9 Sentinel Pit at Q4 2019

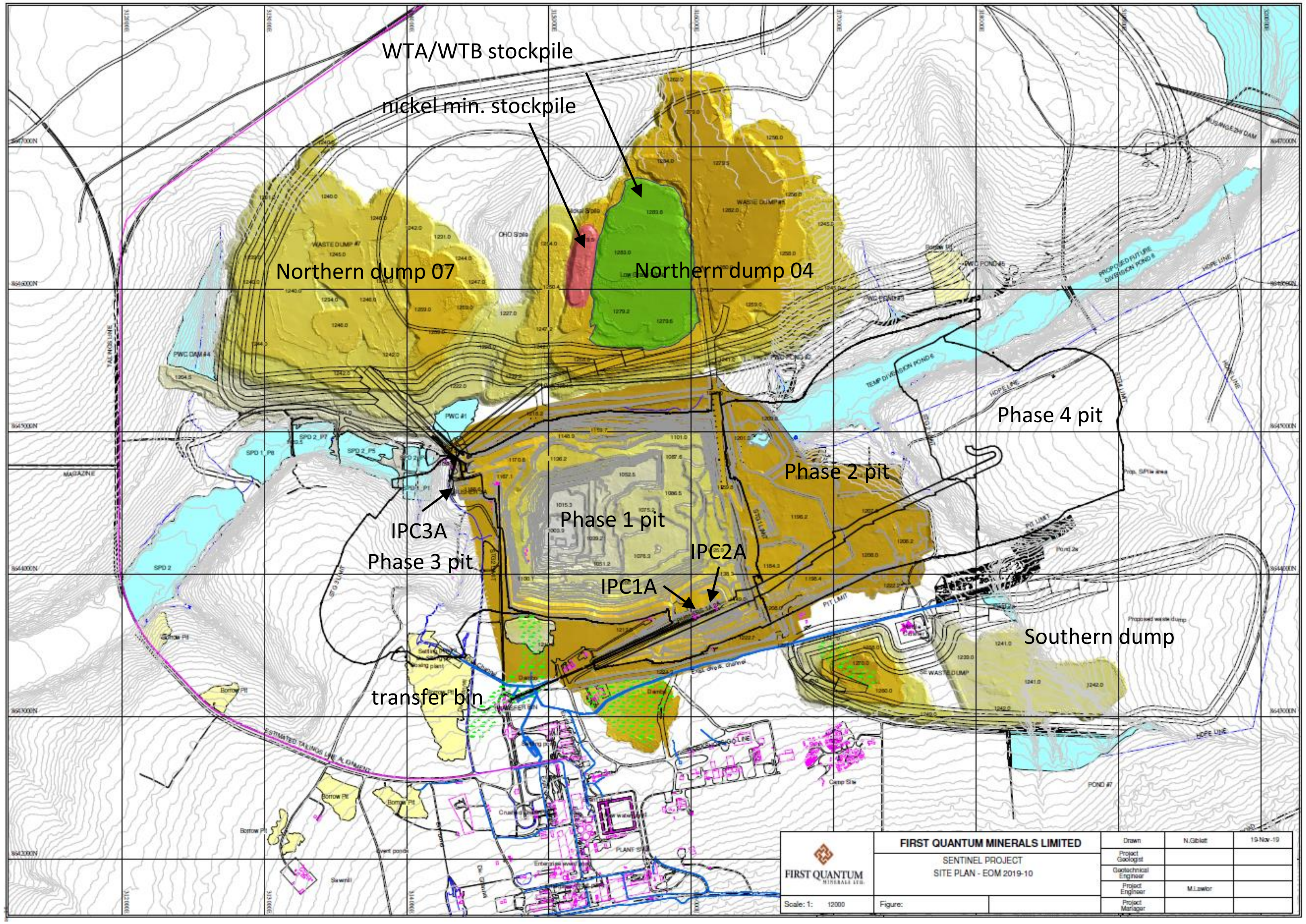


Figure 15-10 Sentinel ultimate pit design

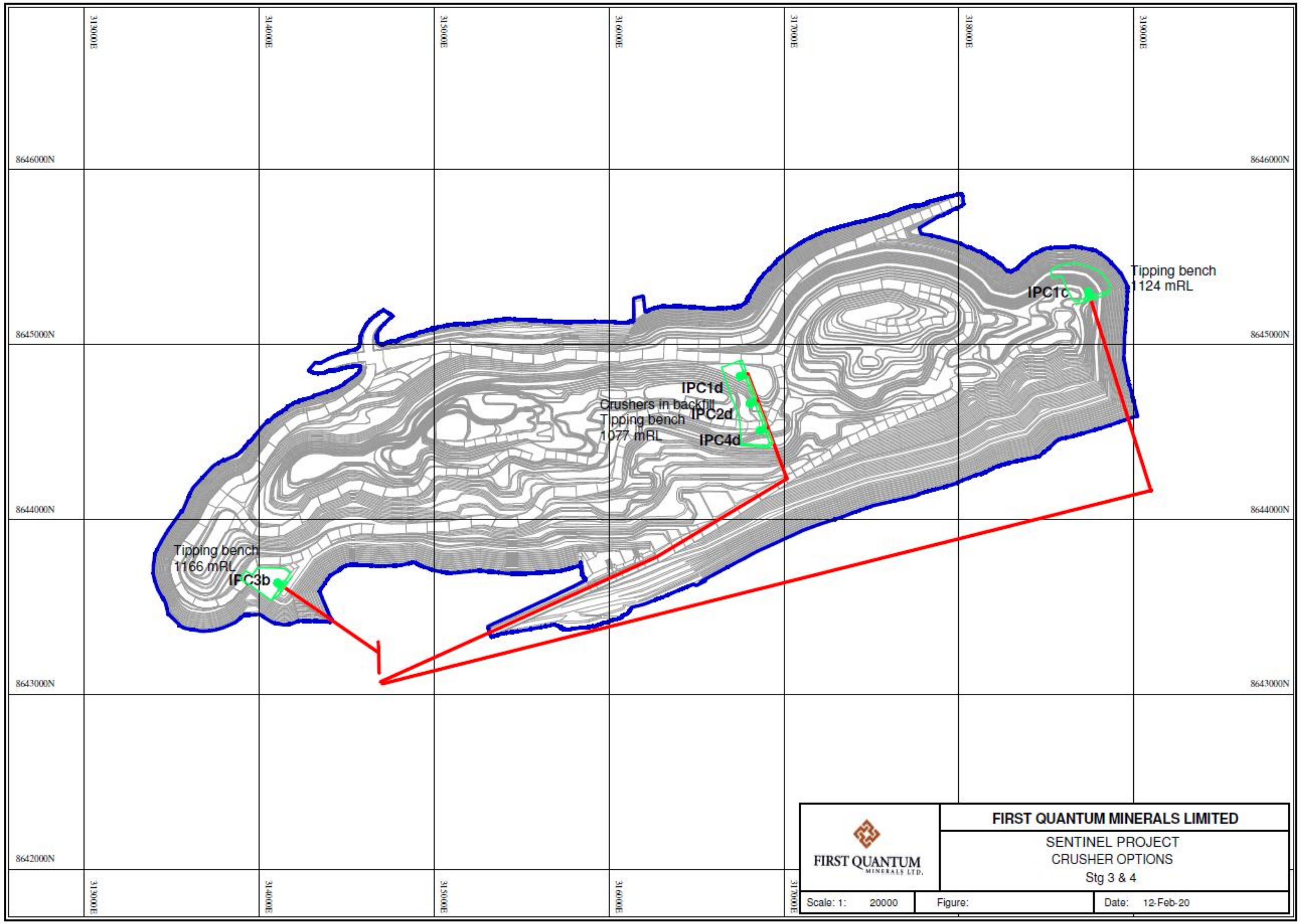


Figure 15-11 Sentinel Phase 1 pit design and IPC positions

Figure 15-12 Sentinel Phase 2 pit design and IPC positions

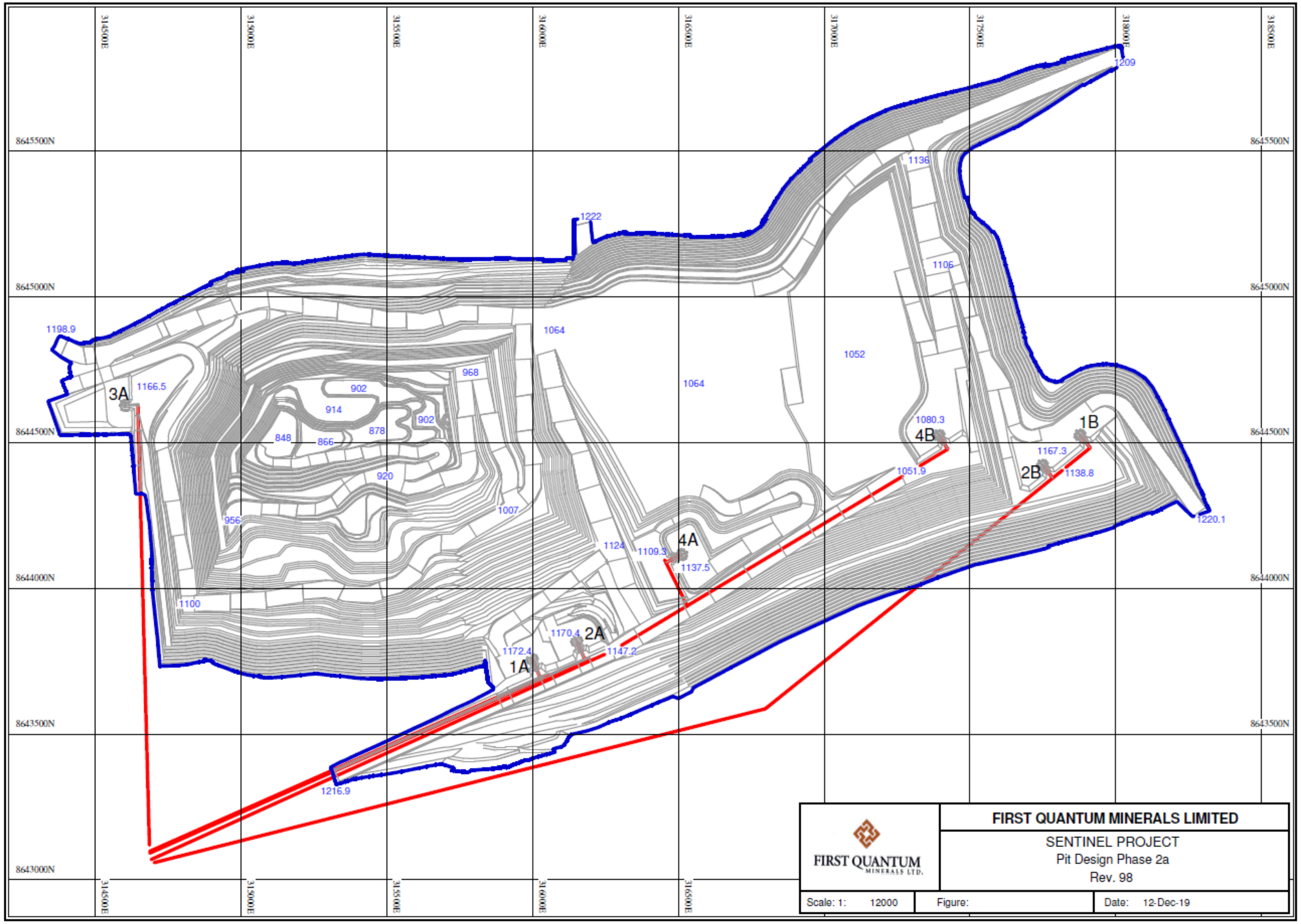
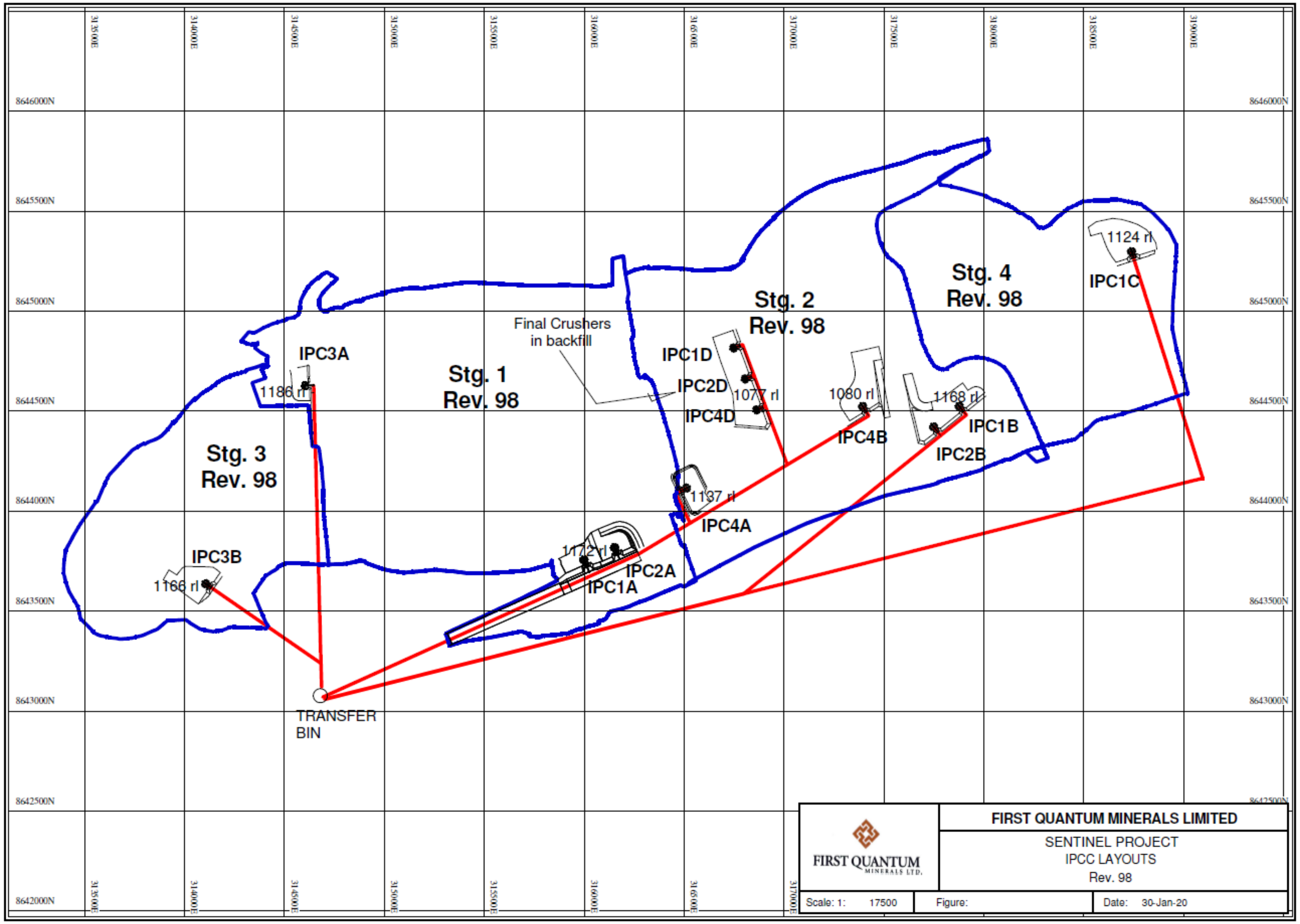


Figure 15-13 Sentinel in pit crusher relocation positions



15.3.3.6 Validation between pit shell and design pit

Table 15-8 shows a comparison validation between the selected shell 15 optimal pit inventory (the base optimisation) and the inventory for the corresponding detailed ultimate pit design (both reported at marginal cut off-grade). The inventories are depleted to the end of October 2019 and account for mining dilution and mining recovery. The comparison shows an acceptable overall variance.

To verify the conclusions made in Item 15.3.2.11 in relation to the impact of changed mining and processing costs (ie, between the time of optimisation and cashflow modelling), Table 15-8 also shows a comparison between optimal shells at revised mining and processing costs, and the designed ultimate pit. The differences in optimisation results are considered to be within a tolerable variance to the ultimate pit design.

Table 15-8 Validation between ultimate design pit and selected optimal pit shell, reflective of mining depletion to October 2019

Pit shell/design phase	OTA & WTA		OTB & WTB		TOTAL ORE		INSITU METAL	WASTE	TOTAL PIT
	Mtonnes	%Cu	Mtonnes	%Cu	Mtonnes	%Cu	ktonnes Cu	Mtonnes	Mtonnes
base optimisation					**875.8**	**0.47**	**4,089.9**	**1,492.5**	**2,368.3**
revised mining costs					**891.1**	**0.46**	**4,099.1**	**1,482.0**	**2,373.1**
revised process costs					**862.3**	**0.47**	**4,061.3**	**1,491.1**	**2,353.4**
Phase 3	154.3	0.35	24.5	0.33	178.8	0.35	624.6	245.2	424.0
Phase 1	153.4	0.51	0.1	0.30	153.5	0.51	775.5	125.7	279.2
Phase 1 - 2 wall	61.8	0.54	4.5	0.42	66.3	0.53	350.8	37.5	103.8
Phase 2	218.4	0.54	42.9	0.41	261.3	0.52	1,363.1	522.6	783.9
Phase 2 - 4 wall	119.7	0.43	4.4	0.37	124.1	0.43	534.7	172.1	296.2
Phase 4	63.2	0.47	11.1	0.37	74.3	0.46	341.4	541.5	615.8
Ultim. Design	**770.8**	**0.47**	**87.5**	**0.38**	**858.3**	**0.46**	**3,990.0**	**1,644.6**	**2,502.9**
base vs design					**98.0%**		**97.6%**	**110.2%**	**105.7%**
rev mining vs design					**96.3%**		**97.3%**	**111.0%**	**105.5%**
rev process vs design					**99.5%**		**98.2%**	**110.3%**	**106.4%**

15.3.3.7 Comparison between 2015 and 2019 ultimate pit designs

Figure 15-14 shows comparison plans of the 2015 (Revision 38) and the Q4 2019 (Revision 98) ultimate pit designs. There are a number of detail changes to the design that have arisen over the intervening years, namely:

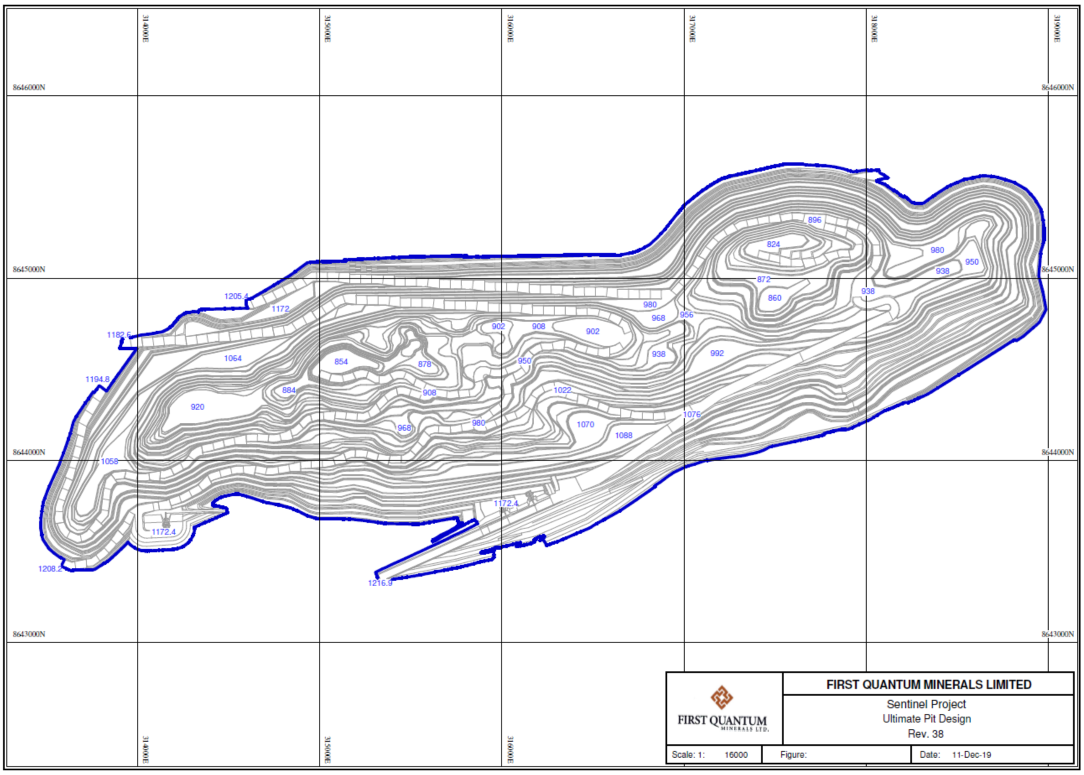
- the south wall overall slope angle in Phases 2 and 4 has been flattened in accordance with changed geotechnical design parameters; the overall slope is flatter than was previously adopted and is now conformable with the detachment zones (and foliation) (refer to Item 16.1.2)
- the Phase 1 and 2 north wall now has a single haul ramp, the orientation of which has been adjusted to compensate for localised instability on the upper slope
- a new waste haul ramp has been designed for the north wall of Phases 2 and 4
- the ramp layout for Phase 3 has been completely redesigned
- the future IPC3B crusher layout in Phase 3 has been reoriented

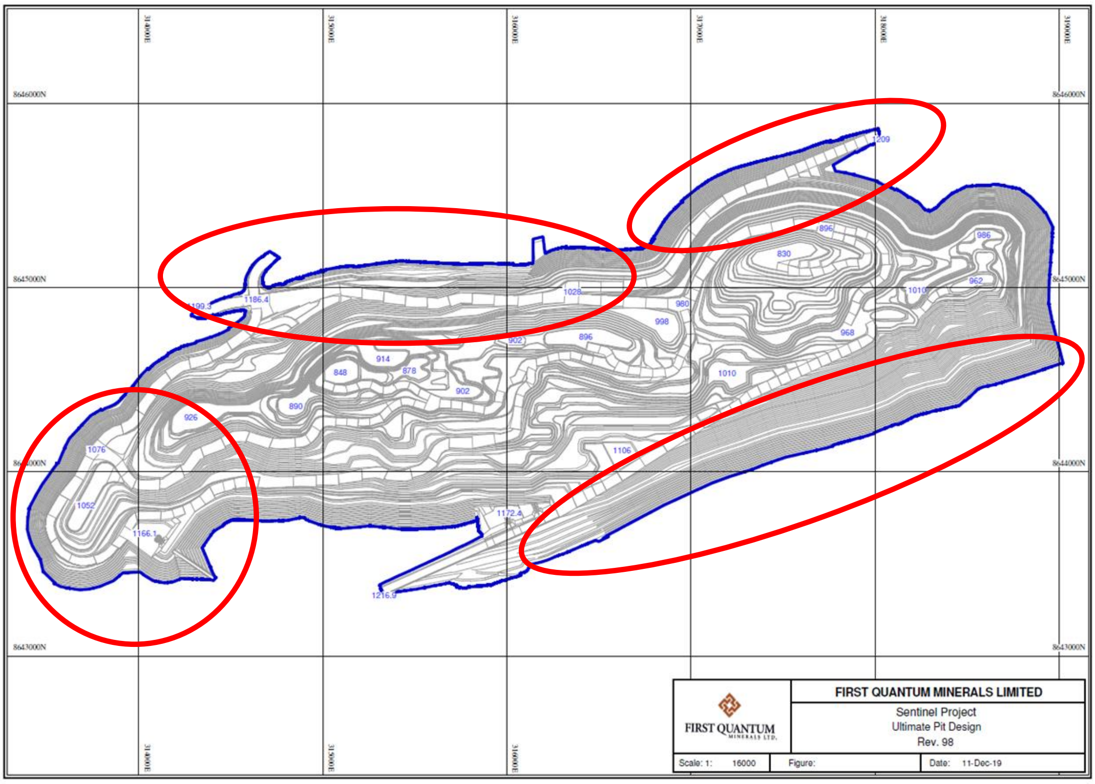
Relative to the current Mineral Resource model and reflecting depletion to the end of October 2019 (and before mining dilution and mining recovery adjustments), the overall net effect of these changes is a minor reduction in the ore and waste inventories (Table 15-9).

Table 15-9 Comparison between ultimate pit designs

Pit Design	Ore (Mt)	Ore (%TCu)	Waste (Mt)	Total Pit (Mt)
2015 ultimate	879.8	0.47	1,646.9	2,526.7
2019 ultimate	858.3	0.48	1,644.6	2,502.9
delta	**97.6%**	**101.3%**	**99.9%**	**99.1%**

Figure 15-14 Comparison between 2015 and 2019 ultimate pit designs





15.3.4 Mineral Reserve statement

As at the end of October 2019 (and consistent with optimisation, mine design and production scheduling work progress in Q4 2019), the Proven and Probable Mineral Reserve for Sentinel, inclusive of stockpiles, was 886.2 Mt at an average grade of 0.46% TCu. The Mineral Reserve within the designed pit was estimated as 858.3 Mt at an average grade of 0.46% TCu, and the accumulated stockpiles amounted to 27.9 Mt at an average grade of 0.24% TCu[5].

Table 15-10 lists the modified Mineral Reserve statement, as depleted to the end of December 2019. A breakdown by material type and classification is provided and the additional Mineral Reserve held in surface mineralised waste stockpiles is included in this table.

The reported Mineral Reserve is based on an economic cut-off grade which accounts for a longer-term copper metal projection of $3.00/lb ($6,614/t). The inventory reflects the phased pit designs and the mining production schedule described in Item 16.1.3.

Table 15-10 Sentinel Pit Mineral Reserve statement, depleted for mining as at 31st December 2019, and based on a $3.00/lb Cu price

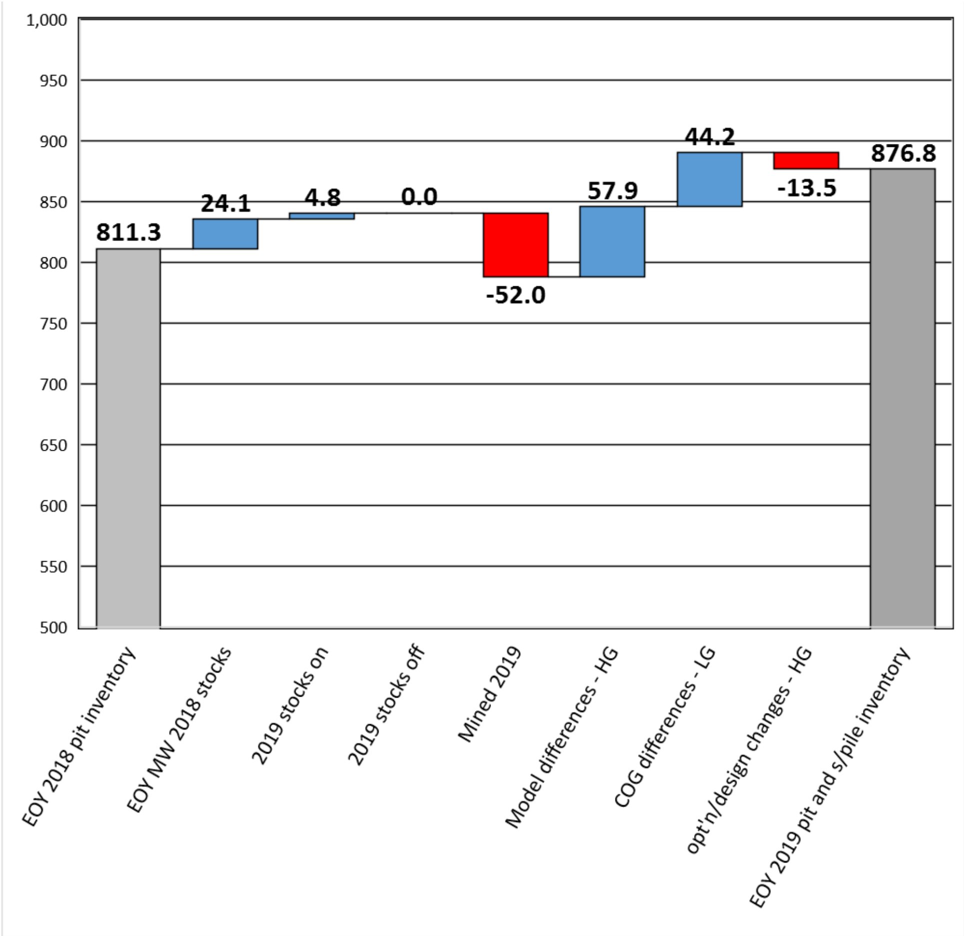
Mineral Reserve statement as at end of December 2019 at $3.00/lb Cu				
Classification	**Material**	**Tonnes (Mt)**	**TCu (%)**	**Cu metal (kt)**
Proven	Non-primary sulphide	63.7	0.40	254.0
Proven	Primary sulphide	539.1	0.49	2,663.9
	Proven subtotal	**602.8**	**0.48**	**2,917.9**
Probable	Non-primary sulphide	20.9	0.31	65.1
Probable	Primary sulphide	224.3	0.43	960.3
	Probable subtotal	**245.2**	**0.42**	**1,025.3**
	Proven and Probable subtotal	**847.9**	**0.47**	**3,943.3**
Probable	Stockpiles	28.9	0.24	69.1
	Proven and Probable total	**876.8**	**0.46**	**4,012.4**

The AIF reported Mineral Reserve for the depleted pit, as at the end of 2018 (FQM, March 2019), was 811.3 Mt at an average grade of 0.50% TCu (Table 6-4). A reconciliation of the changes to the Mineral Reserve inventory to the end of 2019 is shown as a waterfall chart in Figure 15-15. Explanatory comments are as follows:

- 811.3 Mt is the end of 2018 depleted inventory for the open pit
- 24.1 Mt is the primary ore mineralised waste stockpile inventory (WTA) as at the end of 2018, to which was added 4.8 Mt during 2019
- The mining depletion during 2019 is estimated as 56.8 Mt, inclusive of the 4.8 Mt that went to stockpile (52 Mt shown as ore processed)
- 57.9 Mt is the inventory increase reflective of the Mineral Resource model changes
- based on the new model, 44.2 Mt is the additional mineralised waste (WTA and WTB between the marginal cut off grade and 0.2% TCu) that was otherwise excluded from the 2015 Mineral Reserve plant feed strategy and perpetuated through to the 2018 statement
- 13.5 Mt is the inventory change reflecting modifications to the geotechnical design criteria and to phase and ultimate design changes made since 2015

[5] The stockpile inventory is for the primary ore mineralised waste (WTA) stockpiles only.

Figure 15-15 Waterfall comparison between 2018 and 2019 Mineral Reserve statements



15.4 Enterprise

15.4.1 Mine planning model

A regularised block model to 10 m x 10 m x 5 m (x, y, z dimensions) size was provided for pit optimisation and mine planning work. The modelling process as described in Item 14 can be considered as providing a suitable allowance for "planned dilution".

The regularised model was reblocked to 25 m x 25 m x 5 m for the pre-strip pit within the optimisation software, but without compromising the definition of the original model.

15.4.2 Pit optimisation

Conventional Whittle Four-X software was used to determine an optimal pit shell for the Enterprise deposit. Optimisations were completed on a maximum undiscounted cashflow basis, and with recoveries to nickel metal in concentrate determined from fixed process recovery values.

15.4.2.1 Pit slope design parameters

Subsequent to the 2015 Technical Report (FQM, May 2015), pit slope design parameters for Enterprise were geotechnically reassessed in 2018 through 2019. The updated inter-ramp (IRA) angles in Table 15-11 were translated into the required overall slope angles for optimisation by considering the layout and positioning of haulroads in a preliminary ultimate pit design.

Table 15-11 Enterprise pit slope design parameters for pit optimisation, October 2019

Domain	Depth Range (m)	Wall orientation	Bench height (m)	Batter angle (degrees)	Min. berm width (m)	IRA (C to C)
Domain 1 Saprolite south wall	Surface to 1126 mRL	All	6	30	6	20.1
Domain 1 Saprolite north wall	Surface to 1120 mRL	All	6	30	6	20.1
Domain 5 Saprock	1126 mRL to 1084 mRL	All	10	55	6	37.6
Domain 2, 3 & 4 Saprock	1126 mRL to 1110 mRL	All	10	55	6	37.6
Domain 4	1110 mRL to	NE (0° to 90°)	20	75	10	52.5
Fresh rock	1016 mRL	Elsewhere (90° to 360°)	20	70	10	49.2
Domain 2 Fresh rock	1110 mRL to pit floor	All	20	65	10	46.0
Domain 3 Fresh rock	1110 mRL to pit floor	All	20	75	10	52.5
Domain 5 & 6	1084 mRL to	NE (0° to 90°)	20	70	10	49.2
Fresh rock	pit floor	Elsewhere (90° to 360°)	20	65	10	46.0

Geotechnical information in relation to these updated slope design parameters is outlined in Item 16.2.2.

15.4.2.2 Metal price

The nickel metal price adopted for pit optimisation was US$7.50/lb (US$16,535/t). This is consistent with the long term consensus forecast figure as tabled in Item 22.1.

15.4.2.3 Metal recoveries

Metal recovery figures adopted for pit optimisation reflect the metallurgical recommendations in Item 13.2.9, as follows:

- Mixed ore, or saprock ore (also referred to as non-primary sulphide ore) = 60%
- Sulphide ore, or fresh ore (also referred to as primary sulphide ore) = 85%

15.4.2.4 Mining costs

Variable mining costs comprising load and haul costs, on a bench-by-bench basis, were determined from haulage simulations taking account of material movements within the initial stage pits. This was an update on the varying mining costs adopted for the optimisation described in the 2015 Technical Report (FQM, May 2015). Item 21 provides more information on the incremental relationship adopted for ore and waste mining costs in the pit optimisation process, as follows:

- incremental waste load and haul costs (bcm) = -0.0155 x RL + 20.621
- incremental ore load and haul costs (bcm) = -0.0134 x RL + 17.451

A fixed cost of $2.45/bcm was added in the optimisation for drill and blast, irrespective of mining bench level.

From these relationships, the overall average total mining costs inclusive of drill, blast, load and haul are as follows:

- average waste mining cost = $6.41/bcm ($2.39/t)
- average ore mining cost = $6.26/bcm ($2.26/t)

15.4.2.5 Operating costs

Unlike Sentinel, Enterprise is considered not to be mill constrained, hence the process operating costs should include only the variable costs and those fixed costs associated with the processing plant nickel circuit. G&A costs would be carried by the existing Sentinel operations.

In relation to processing plant cost inputs, the estimates outlined in the 2015 Technical Report (FQM, May 2015) have been updated as follows:

- variable processing costs = $6.21/t processed
- fixed processing costs = $2.25/t processed
- reclaim and haulage costs = $1.50/t processed
- total operating costs = $9.96/t processed

Details of these cost estimates and their derivation are outlined in Item 21.

15.4.2.6 Metal costs

In addition to 5% (gross) royalties, metal costs for the Enterprise product would comprise concentrate transport charges, treatment and refining charges, plus smelter deductions. This was the case considered in the 2015 Technical Report (FQM, May 2015). However, as advised by the Company’s metals marketing group, concentrate treatment charges are now typically simplified through the adoption of a single payability percentage which accounts for the separate cost items.

On this basis, the metal costs included in the Enterprise optimisation are as follows:

- overall treatment payability = 75%
- concentrate transport charge = $0.85/lb
- overall treatment and refining charges = $2.03/lb
- 5% royalty = $0.38/lb
- Total nickel metal cost = $3.25/lb

Details of these cost estimates and their derivation are outlined in Item 21.

15.4.2.7 Mining dilution and recovery factors

Pit optimisation assumptions included unplanned mining dilution and mining recovery factors of 105% and 95%, respectively. This is consistent with the assumptions in the 2015 Technical Report (FQM, May 2015). The dilution is assumed to be at a nil diluent grade.

15.4.2.8 Optimisation inputs summary

Table 15-12 summarises the pit optimisation input parameters applicable after adopting the mining, processing and metal costs as per the information and tables above. The inputs from the 2015 Technical Report are shown for comparison.

15.4.2.9 Marginal cut off grades

Based on the above parameters, the calculated marginal cut-off grade is as listed in Table 15-13. The cut-off grades reported in the 2015 Technical Report are also listed for comparison.

15.4.2.10 Optimisation results

Figure 15-16 shows the graphical results of the base case pit optimisation, whilst Table 15-14 lists the complete inventory of shell sizes and corresponding cashflows. As was done in 2015, the optimal ultimate pit shell was selected on a maximum net return (undiscounted) basis.

To note from Figure 15-16 is the marked jump in waste mining tonnes in going from shell 10 to the selected shell 11, for little difference in undiscounted cashflow. Shell 11 (at revenue factor 1) honours the notional project life and extents of the phased open pit as designed and scheduled into Reserves for the 2015 Technical Report (FQM, May 2015).

Without compromising the cashflow basis of selecting an optimal shell, pit shell 11 was chosen on the basis that, given no commitment to a timeframe for ore mining and processing:

a. production can commence from a starter pit and a second phase pit, defined within lesser shells up to number 10, for a period of approximately three to four years, and
b. whilst economic at $7.50/lb, progress into a third and fourth phase, with an associated elevated waste strip ratio, can be reassessed in the future in terms of the then prevailing nickel price and process recovery projections.

Another aspect to note from this current optimisation is that the revenue factor 1 pit shell does not include the Enterprise SW deposit.

Table 15-12 Enterprise pit optimisation inputs, Q4 2019

ENTERPRISE OPTIMISATION	Units	2015 INPUTS		2019 INPUTS	
PROCESS		**FLOT**	**FLOT**	**FLOT**	**FLOT**
Ore types		Non-Primary	Primary	Non-Primary	Primary
Optimisation Metal Price					
Nickel Price	$/lb	7.50	7.50	7.50	7.50
Nickel Price	$/tonne	16,535	16,535	16,535	16,535
Mining Parameters					
Mining Recovery	Factor	0.95	0.95	0.95	0.95
Dilution Factor	Factor	1.05	1.05	1.05	1.05
Throughput Rates					
Milling Circuit	Mtpa	4.00	4.00	4.00	4.00
Metal Recovery Factors					
Ni to Concentrate Recovery (Linear Calc)	%	60.0%	90.0%	60.0%	85.0%
Mining Costs					
overall average waste cost	$/t	2.68		2.56	
overall average ore cost	$/t	2.60		2.68	
overall average waste+ore cost	$/t	2.67		2.62	
Treatment and G&A Costs					
Variable Treatment					
processing	$/t process	7.79	7.79	6.21	6.21
reclaim and haulage	$/t process	1.50	1.50	1.50	1.50
Sub-total variable	$/t process	9.29	9.29	7.71	7.71
Fixed Costs					
Sub-total fixed	$/t process	0.00	0.00	2.25	2.25
Total Treatment	**$/t process**	**9.29**	**9.29**	**9.96**	**9.96**
Metal Costs Nickel Concentrate					
Pricing mechanism rate	%			75%	75%
Royalty rate	%	9.0%	9.0%	5.0%	5.0%
Concentrate Grade	% Ni	17%	17%	15%	15%
Moisture Content	Perc	9%	9%	9%	9%
Realisation/freight					
Concentrate Transport (wet)	$/t conc	85.00	85.00	256.00	256.00
Concentrate Transport (dry)	$/t conc	93.41	93.41	281.32	281.32
subtotal	**$/t Ni in conc**	**549.45**	**549.45**	**1,875.46**	**1,875.46**
subtotal	**$/lb Ni in conc**	**0.25**	**0.25**	**0.85**	**0.85**
Concentrate Treatment					
Treatment Cost (wet)	$/t conc	150.15	150.15	564.75	564.75
Treatment Cost (dry)	$/t conc	165.00	165.00	564.24	564.24
subtotal	**$/t Ni in conc**	**970.59**	**970.59**	**3,765.02**	**3,765.02**
subtotal	**$/lb Ni in conc**	**0.44**	**0.44**	**1.71**	**1.71**
Refining Charges					
	c/lb	97.00	97.00		
subtotal	**$/t Ni in conc.**	**2,138.48**	**2,138.48**		
subtotal	**$/lb Ni in conc**	**0.97**	**0.97**		
Price Participation Charges					
	$/t Ni in conc.	**529.11**	**529.11**		
subtotal	**$/lb Ni in conc**	**0.24**	**0.24**		
Smelter Deduction Charges					
	$/t Ni in conc.	**1,322.77**	**1,322.77**		
subtotal	**$/lb Ni in conc**	**0.60**	**0.60**		
Royalties x% Gross					
	$/t Ni in conc.	**1,488.12**	**1,488.12**	**826.73**	**826.73**
subtotal	**$/lb Ni in conc**	**0.68**	**0.68**	**0.38**	**0.38**
Total Concentrate Ni Metal Cost					
	$/t Ni in conc.	**6,998.52**	**6,998.52**	**6,467.21**	**6,467.21**
	$/lb Ni in conc	**3.17**	**3.17**	**2.93**	**2.93**

Table 15-13 Enterprise marginal cut-off grades, Q4 2019

ENTERPRISE OPTIMISATION	Units	2015 INPUTS		2019 INPUTS	
PROCESS		**FLOT**	**FLOT**	**FLOT**	**FLOT**
Ore types		Non-Primary	Primary	Non-Primary	Primary
Marginal Cut-Off Grade Calculation					
Grade Attribute		**Ni**	**Ni**	**Ni**	**Ni**
Process Cost	$/t mill	9.29	9.29	9.96	9.96
Dilution Factor		1.05	1.05	1.05	1.05
Net Return (Metal price - Total metal cost)	$/10kg	95.36	95.36	100.67	100.67
Recovery	%	60.0%	90.0%	60.0%	85.0%
Marginal Cut-Off Grade Direct Feed	% Grade	**0.17**	**0.11**	**0.17**	**0.12**

Figure 15-16 Enterprise pit optimisation results

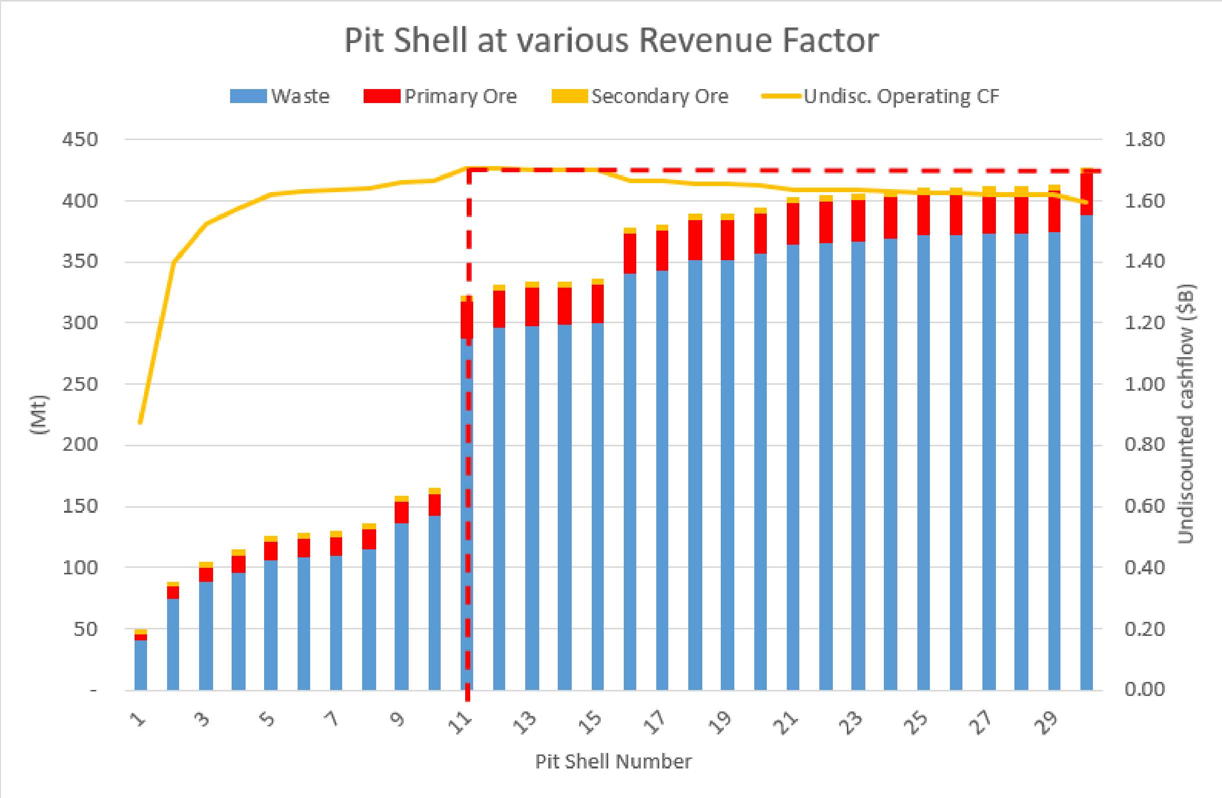


Table 15-14 Enterprise pit optimisation shell sizes and cashflow

Pit Shell Number	Rev factor	Pit Size (Mt)	Waste (Mt)	Plant Feed							Recovered Metal (kt)	Undiscounted Operating CF ($B)
				Total (Mt)	By Resource Classification		by Material Type			Feed Grade (%Ni)		
					Meas. (Mt)	Indic. (Mt)	Primary (Mt)	Non-primary (Mt)	Saprolite (Mt)			
1	0.50	49.0	40.9	8.2	5.0	3.1	4.6	3.6	0.0	1.67	106.2	0.88
2	0.55	88.7	74.1	14.6	7.4	7.2	10.1	4.5	0.0	1.49	173.8	1.40
3	0.60	104.7	88.0	16.7	7.8	8.9	12.1	4.6	0.0	1.43	192.1	1.53
4	0.65	114.3	96.4	17.8	8.3	9.5	13.2	4.6	0.0	1.39	200.2	1.57
5	0.70	125.7	106.3	19.4	8.4	11.0	14.6	4.7	0.0	1.34	209.0	1.62
6	0.75	128.2	108.5	19.7	8.4	11.3	14.9	4.7	0.0	1.32	210.7	1.63
7	0.80	130.2	110.2	20.1	8.4	11.6	15.3	4.8	0.0	1.31	212.0	1.63
8	0.85	136.0	115.3	20.7	8.5	12.1	15.8	4.8	0.0	1.29	214.7	1.64
9	0.90	158.6	136.1	22.5	9.1	13.4	17.6	4.9	0.0	1.23	223.9	1.66
10	0.95	165.3	142.3	23.0	9.3	13.8	18.1	4.9	0.0	1.21	226.3	1.66
11	1.00	322.1	287.4	34.7	9.4	25.3	29.6	5.1	0.0	0.99	278.9	1.70
12	1.05	332.0	296.4	35.6	9.4	26.2	30.5	5.1	0.0	0.97	282.3	1.70
13	1.10	333.4	297.6	35.8	9.4	26.4	30.7	5.1	0.0	0.97	282.8	1.70
14	1.15	334.5	298.5	35.9	9.4	26.5	30.8	5.1	0.0	0.96	283.1	1.70
15	1.20	336.3	300.2	36.1	9.4	26.7	31.0	5.1	0.0	0.96	283.6	1.70
16	1.25	377.9	340.3	37.6	9.4	28.2	32.5	5.1	0.0	0.95	292.2	1.67
17	1.30	380.1	342.3	37.8	9.4	28.4	32.7	5.1	0.0	0.95	292.7	1.67
18	1.35	389.3	351.2	38.2	9.4	28.7	33.1	5.1	0.0	0.94	294.4	1.66
19	1.40	389.6	351.4	38.2	9.4	28.8	33.1	5.1	0.0	0.94	294.5	1.66
20	1.45	394.4	356.0	38.4	9.4	29.0	33.3	5.1	0.0	0.94	295.3	1.65
21	1.50	403.2	364.4	38.8	9.4	29.3	33.4	5.3	0.0	0.94	296.7	1.64
22	1.55	404.6	365.7	38.8	9.4	29.4	33.5	5.3	0.0	0.94	296.9	1.64
23	1.60	405.4	366.5	38.9	9.4	29.4	33.6	5.3	0.0	0.94	297.0	1.63
24	1.65	408.6	369.6	39.1	9.4	29.6	33.7	5.3	0.0	0.93	297.5	1.63
25	1.70	410.3	371.2	39.1	9.4	29.7	33.8	5.3	0.0	0.93	297.7	1.63
26	1.75	411.3	372.2	39.1	9.4	29.7	33.8	5.3	0.0	0.93	297.9	1.62
27	1.80	412.1	372.9	39.2	9.4	29.8	33.9	5.3	0.0	0.93	298.0	1.62
28	1.85	412.4	373.2	39.2	9.4	29.8	33.9	5.3	0.0	0.93	298.0	1.62
29	1.90	413.8	374.6	39.3	9.4	29.8	33.9	5.3	0.0	0.93	298.1	1.62
30	1.95	427.5	387.8	39.7	9.4	30.3	34.1	5.7	0.0	0.92	299.6	1.59

15.4.2.11 Optimisation sensitivity analyses

Several optimisation sensitivity analyses (Table 15-15) were carried out, specifically to assess the risk in selecting shell 11 (revenue factor 1), on the threshold of differing ore and waste mining inventories as indicated by Figure 15-16.

Table 15-15 Enterprise optimisation sensitivity scenarios and analysis

Scenario	Pit Shell Number	Rev factor	Pit Size (Mt)	Waste (Mt)	Plant Feed							Recovered Metal (kt)	Undiscounted Operating CF ($B)
					Total (Mt)	By Resource Classification		by Material Type			Feed Grade (%Ni)		
						Meas. (Mt)	Indic. (Mt)	Primary (Mt)	Non-primary (Mt)	Saprolite (Mt)			
Basecase	10	0.95	165.3	142.3	23.0	9.3	13.8	18.1	4.9	0.0	1.21	226.3	1.66
Basecase	**11**	**1.00**	**322.1**	**287.4**	**34.7**	**9.4**	**25.3**	**29.6**	**5.1**	**0.0**	**0.99**	**278.9**	**1.70**
Basecase	12	1.05	332.0	296.4	35.6	9.4	26.2	30.5	5.1	0.0	0.97	282.3	1.70
$6/lb Ni	8	1.00	130.2	112.4	17.8	7.5	10.3	14.0	3.8	0.0	1.45	209.3	0.94
$9/lb Ni	12	1.00	336.3	296.1	40.3	10.3	29.9	34.3	5.9	0.0	0.87	287.0	2.64
Mining Cost +10%	**11**	**1.00**	**318.8**	**284.4**	**34.4**	**9.4**	**25.0**	**29.3**	**5.1**	**0.0**	**0.99**	**277.8**	**1.63**
Mining Cost -10%	**11**	**1.00**	**331.9**	**296.4**	**35.5**	**9.4**	**26.1**	**30.4**	**5.1**	**0.0**	**0.97**	**282.2**	**1.78**
Processing Cost +10%	**11**	**1.00**	**321.9**	**288.8**	**33.0**	**9.0**	**24.0**	**28.2**	**4.8**	**0.0**	**1.03**	**277.3**	**1.67**
Processing Cost -10%	11	1.00	323.2	286.9	36.2	9.7	26.5	30.8	5.4	0.0	0.95	280.6	1.74
Metal Cost +10%	**10**	**1.00**	**318.7**	**285.6**	**33.1**	**9.1**	**24.0**	**28.2**	**4.9**	**0.0**	**1.02**	**276.5**	**1.52**
Metal Cost -10%	12	1.00	331.9	295.5	36.4	9.6	26.8	31.2	5.2	0.0	0.95	283.0	1.89

This sensitivity analysis shows that, apart from the obvious sensitivity to Ni price:

- A 10% increase in the either of the mining, processing or metal costs, does not significantly change the ultimate pit size, nor the total plant feed (ie, a 1% reduction to the feed due to a mining cost increase, and a 5% reduction due to either of a processing cost or metal cost increase).
- The 10% increases, individually, do not result in a reduction of the plant feed inventory to the magnitude of that of shell 10 in the base case optimisation.
- A 10% increase in mining and processing costs, results in a 4% and a 2% reduction in undiscounted cashflow, respectively.
- Whereas, a 10% increase in the metal costs results in an 11% reduction in the undiscounted cashflow.

15.4.3 Detailed pit designs

Following the pit optimisation, a series of phased pit designs were developed using the selected ultimate (shell 11) and a number of intermediate pit shells.

15.4.3.1 Geotechnical design parameters

Table 15-16 lists the pit slope parameters used for detailed pit design. Further geotechnical information in relation to these design parameters and domains is outlined in Item 16.2.2.

15.4.3.2 Design and planning parameters

General design parameters for haul road layouts were as follows:

- Haul road width with = 25 m (ie, 3 x the width of a 180 t capacity haul truck)
- Maximum haul ramp gradient = 1: 10 = approximately 6^{o}

Table 15-16 Sentinel pit slope design parameters, updated October 2019

Domain	Depth Range (m)	Wall orientation	Bench height (m)	Batter angle (degrees)	Min. berm width (m)	IRA (C to C)
Domain 1 Saprolite south wall	Surface to 1126 mRL	All	6	30	6	20.1
Domain 1 Saprolite north wall	Surface to 1120 mRL	All	6	30	6	20.1
Domain 5 Saprock	1126 mRL to 1084 mRL	All	10	55	6	37.6
Domain 2, 3 & 4 Saprock	1126 mRL to 1110 mRL	All	10	55	6	37.6
Domain 4	1110 mRL to	NE (0° to 90°)	20	75	10	52.5
Fresh rock	1016 mRL	Elsewhere (90° to 360°)	20	70	10	49.2
Domain 2 Fresh rock	1110 mRL to pit floor	All	20	65	10	46.0
Domain 3 Fresh rock	1110 mRL to pit floor	All	20	75	10	52.5
Domain 5 & 6	1084 mRL to	NE (0° to 90°)	20	70	10	49.2
Fresh rock	pit floor	Elsewhere (90° to 360°)	20	65	10	46.0

15.4.3.3 Staged and ultimate pit designs

Figure 15-17 shows the starting point for the mine planning and production scheduling process[6], relative to the design ultimate limits. Pre-stripped waste has been mined to a depth of 25 m to 30 m below surface to provide construction material for the building of surface water diversion dams. This will assist with the reduction of future strip ratios.

Figure 15-18 shows the progression of mining phases throughout the life of mine.

15.4.3.4 Validation between pit shell and design pit

Table 15-17 shows a comparison between the selected shell 11 optimal pit inventory and the inventory for the corresponding detailed ultimate pit design. The overall variances are minimal. Note that the nickel grades are insitu, before the application of unplanned mining dilution.

The reason for the close validation is that the optimisation and design went through a cyclic review and comparison to ensure that the overall slopes and ramp layout were thoroughly considered in the optimisation.

Table 15-17 Comparison between Enterprise ultimate design and optimal pit

		Ult Design Rev8	MI Pit Shell 11	Variance (Design/PitShell)
Pit Size	Mtonnes	323.6	322.1	0.48%
Nickel Ore	Monnes	34.7	34.7	0.15%
	%Ni	1.04	1.04	0.46%
Nickel Contained	ktonnes	362.7	360.4	0.61%
Waste	Mtonnes	288.9	287.4	0.52%
SR	Waste/Ore	8.3	8.3	0.37%

[6]This figure shows the extent of development mining when surveyed at the end of June 2019; there has been no further development work on the site since that time.

Figure 15-17 Enterprise Pit at June 2019

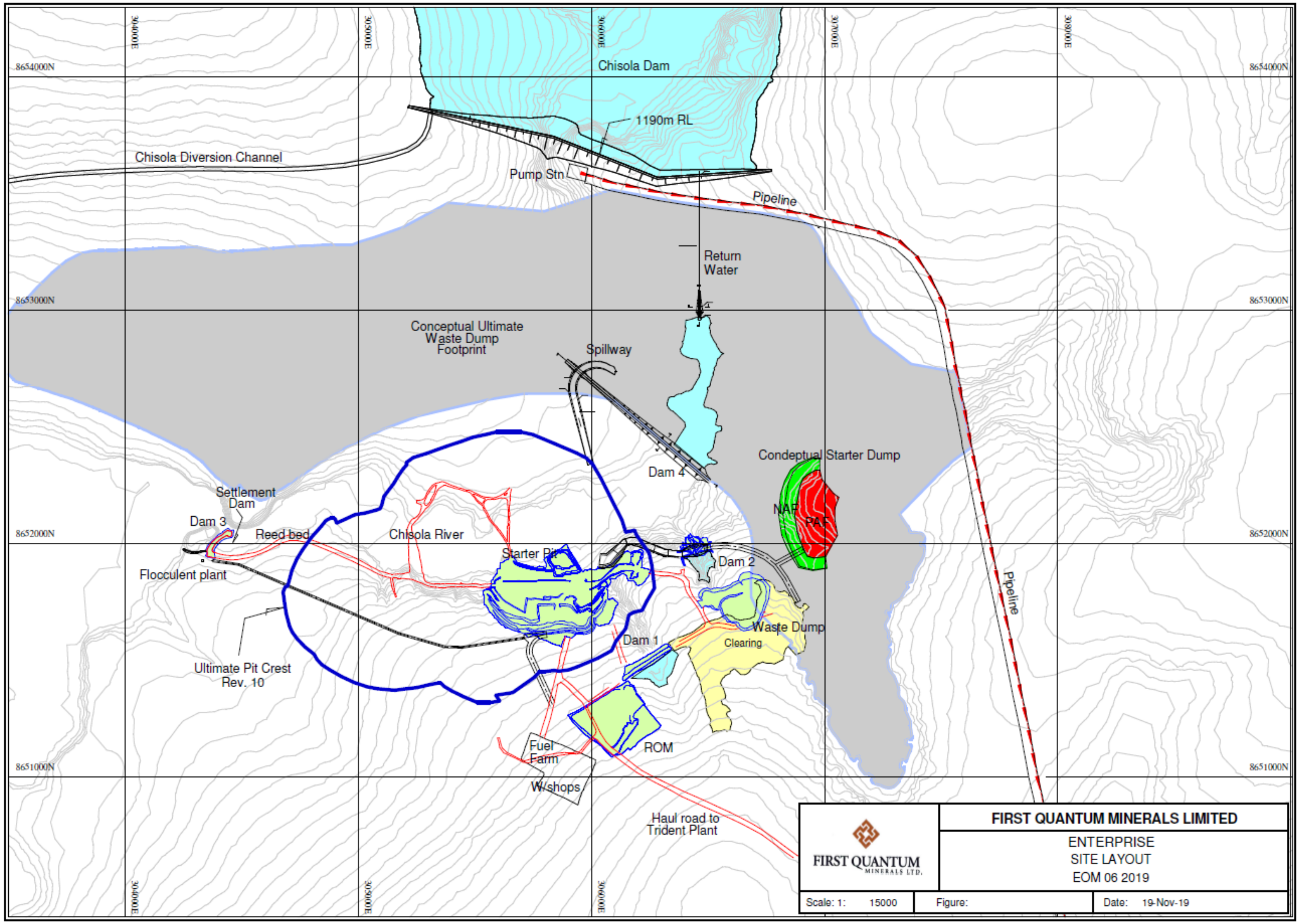
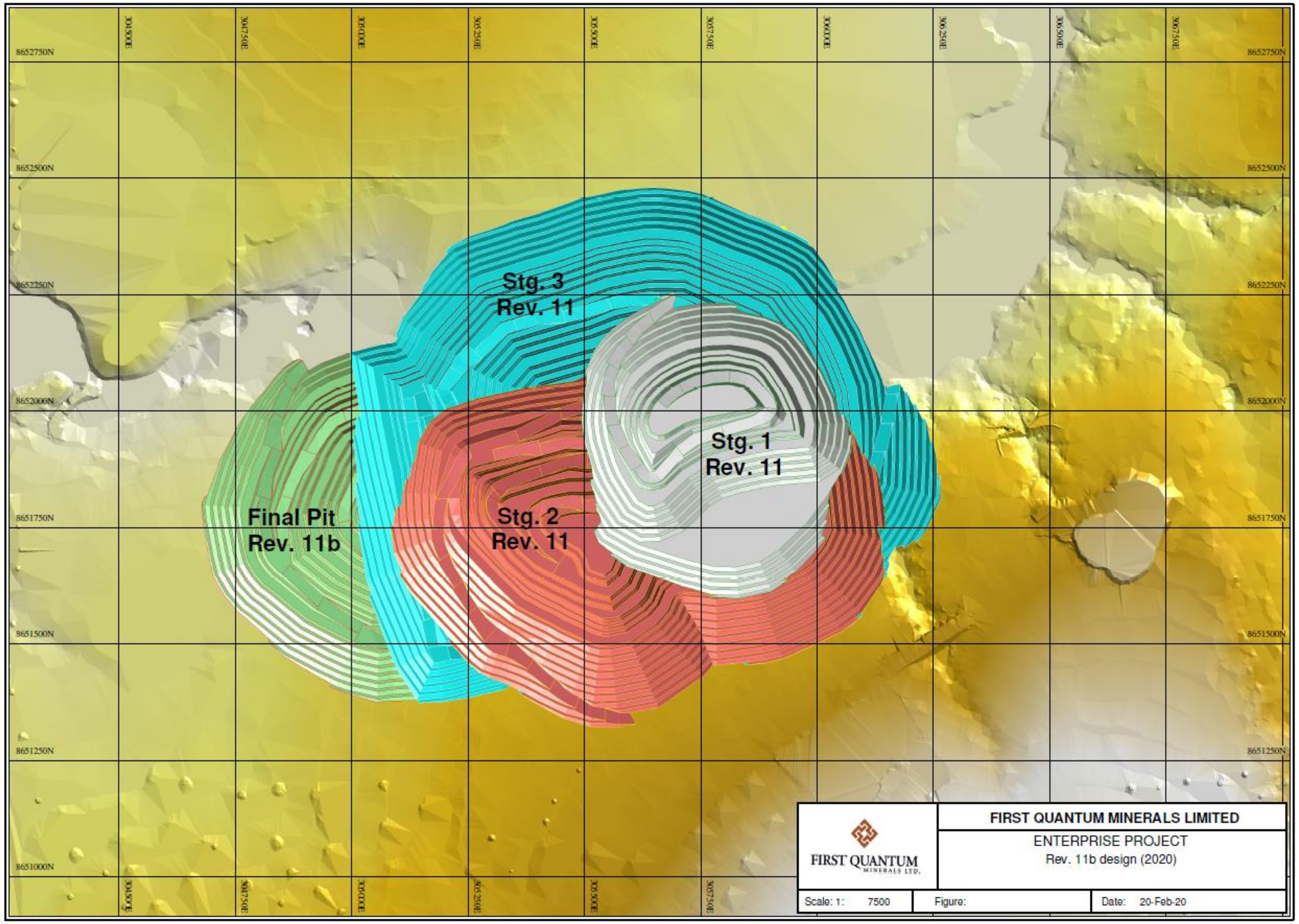


Figure 15-18 Enterprise ultimate and phase pit designs



15.4.4 Mineral Reserve statement

The Mineral Reserve within the designed pit is estimated as 34.7 Mt at an average grade of 0.99% Ni. A breakdown by material type and classification is provided in Table 15-18.

The reported Mineral Reserve is based on an economic cut-off grade which accounts for a longer-term nickel metal price projection of \$7.50/lb (\$16,535/t). The inventory reflects the staged pit designs and the mining production schedule described in Item 16.2.3.

Table 15-18 Enterprise Mineral Reserves statement, in-pit inventory at 31st December 2019

Mineral Reserve statement as at end of December 2019 at \$7.50/lb Ni				
Classification	**Material**	**Tonnes (Mt)**	**Ni (%)**	**Ni metal (kt)**
Proven	Non-primary sulphide	3.8	0.98	37.45
Proven	Primary sulphide	5.7	1.46	84.0
	Proven subtotal	**9.6**	**1.27**	**121.4**
Probable	Non-primary sulphide	2.0	0.44	8.68
Probable	Primary sulphide	23.1	0.93	214.4
	Probable subtotal	**25.1**	**0.89**	**223.1**
	Proven and Probable total	**34.7**	**0.99**	**344.5**

By comparison, the Mineral Reserve statement in the May 2015 Technical Report (Table 6-7) listed a combined Proven and Probable Mineral Reserve of 35.4 Mt at an average grade of 0.97% Ni.

ITEM 16 MINING METHODS

16.1 Sentinel

16.1.1 Mining details

The Sentinel Pit is being mined in a series of terraced phases, using large-scale mining equipment, and with mining costs expected to be minimised through the adoption of bulk mining and ore handling methods featuring electric shovels and drill rigs, trolley-assisted (TA) haulage, and in-pit primary crushing and conveying (IPCC). Waste and ore haul cycle times, and hence fuel consumption, are expected to be reduced through the adoption of TA and IPCC.

Open pit mining at Sentinel commenced in two surface box-cut areas of the Phase 1 Pit; ie, in the north west boxcut in April 2013, and then in the south boxcut from early 2014. Since 2013-2014, mining has proceeded in the Phase 1 pit to a current depth of approximately 200 m (Figure 16-1).

The Phase 2 pit, immediately to the east, was progressively cleared and grade control drilled from 2016. Mining from the southern pit crest limits, 800 m across to the Musangezhi River, now extends along a strike length of 1.2 km and to a depth of approximately 25 m (Figure 16-1).

According to the 2015 Technical Report (FQM, May 2015), the total mining movement capacity was expected to ramp up to about 180 Mtpa by 2016. Consequent to the proposed expanded processing rate, the mining movement capacity will now increase to 190 Mtpa from 2026.

Mine site layout

The current layout of the Sentinel mining site is shown in Figure 16-1. The main features shown are:

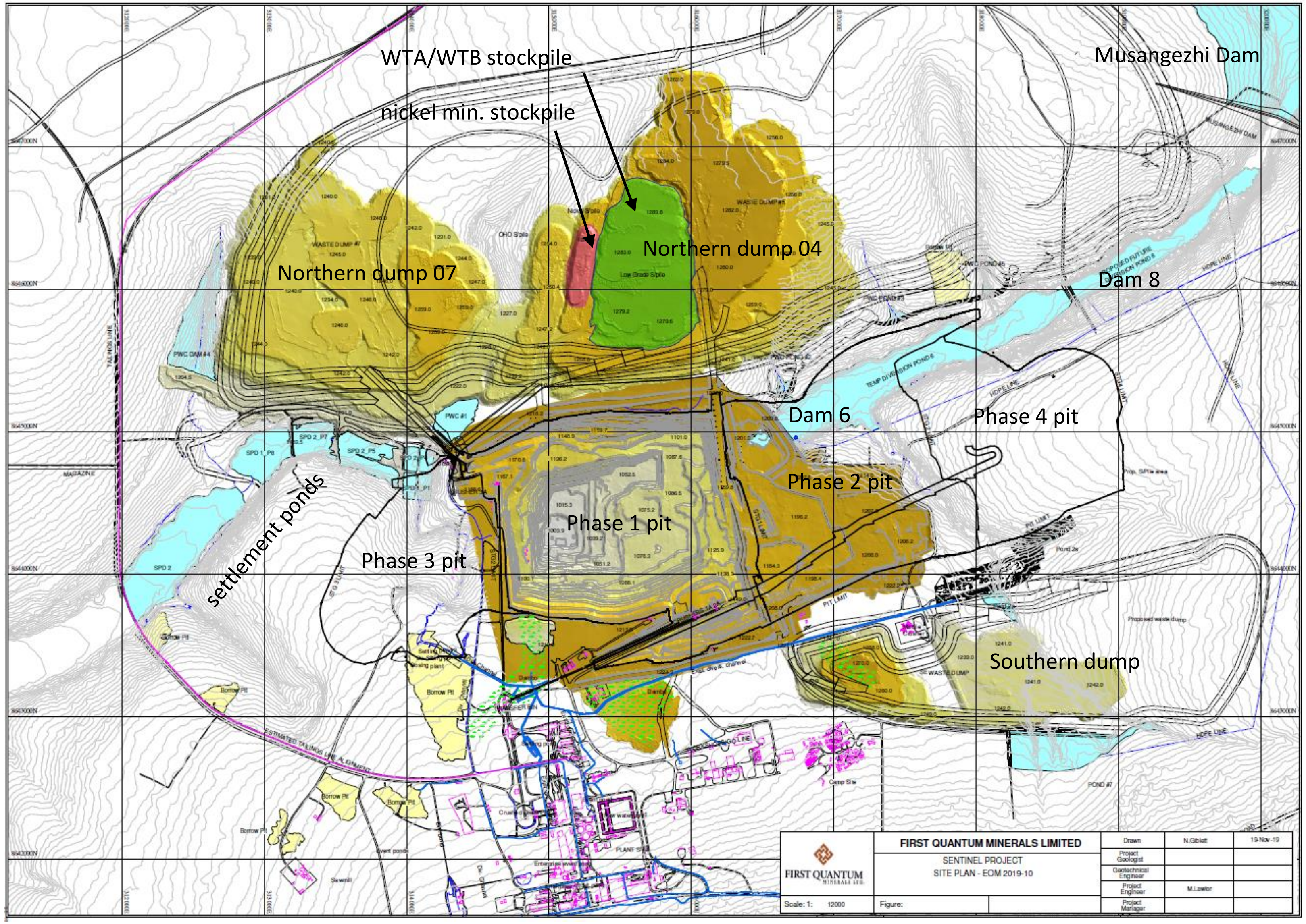
- the extent of mining to date within the Phase 1 (centre) and Phase 2 (east of centre) limits, relative to future phases and the ultimate pit limits
- the extent of waste dumping to the north and south of the pit, relative to the conceptual ultimate dumping limits
- the Musangezhi Dam, on the Musangezhi River approximately 4.5 km upstream from the current mining phases
- an existing temporary diversion/containment dam (Dam 6) located between the pit and the Musangezhi Dam (and another future diversion/containment dam (Dam 8) located further upstream)
- surface run-off settlement ponds located to the west of the current mining phases

Mining method and operations

Mining follows conventional drill and blast, shovel and truck mining practice. The sequence of mining activities is also conventional and is generally as follows:

- RC grade control drilling delineates the ore zones
- a grade control model is developed from which blast limits and digging blocks are designed
- ore and waste blocks are blasted to design, according to layouts based on varying hole patterns and powder factors to suit prevailing ground conditions (Figure 16-2)

Figure 16-1 Sentinel Pit at Q4 2019



- specific blast designs are engineered to suit excavations in close proximity to in-pit crushers and conveyors
- trim blasts and perimeter blasting techniques are used to ensure pit wall profiles are cut to the correct angle and to minimise wall damage
- electric and diesel/hydraulic shovels and excavators load the blasted rock into a fleet of 330 to 360 tonne and 240 tonne capacity haul trucks (Figure 16-3)
- ore is hauled direct to IPCs (Figure 16-4) or to active and long-term stockpiles, whilst waste is hauled to surface dump tip heads
- trolley assisted haulage is currently in use for waste hauls, and is proposed for future ore hauls from increasingly deeper mining elevations (Figure 16-5)

Figure 16-2 Blasthole drilling in Sentinel Pit, Q4 2019

Figure 16-3 Electric face shovel operating in Sentinel Pit, Q4 2019

Figure 16-4 IPCs 1A and 2A, Sentinel Pit, Q4 2019

Figure 16-5 Trolley assisted waste haulage in Sentinel Pit, Q4 2019

16.1.1.1 Drilling and blasting

Production drilling and blasting at Sentinel is carried out by FQM personnel. An explosives supply contractor manufactures emulsion in an on-site emulsion plant and delivers explosives into the pit under a "down-the-hole" supply contract. Two different bulk explosives are supplied, a 100% emulsion explosive and a 70% emulsion/30% AN (ammonium nitrate) explosive. Both are pumped down hole to achieve an average in-hole density of 1.20 g/cm³.

The blast hole drill fleet consists of:

- 7 x Caterpillar MD6640E rotary drills, drilling 270 or 311 mm diameter holes
- 3 x Epiroc PV271E rotary drills, drilling 229 or 251 mm diameter holes
- 2 x Sandvik D25K down-the-hole hammer drills, drilling 165 mm diameter holes

- 2 x Furukawa DCR20 down-the-hole hammer drills, drilling 140 or 165 mm diameter holes

The Caterpillar and Epiroc fleets are used primarily for production drilling, but are also used to drill trim blast patterns. The Sandvik and Furukawa fleets are used to drill wall control, ramp and presplit blast holes. Near surface material classified as 'free-dig' material does not require blasting, and as such is mined insitu using scrapers and excavators.

The northern and southern ore zones are characterised by widely spaced joint sets, and persistent closely spaced joint sets, respectively. One particular joint set runs essentially parallel to the bench surface level, and consequently traditional blasting parameters produce large slabs in the southern zone and large blocks in the northern zone. These circumstances routinely lead to bridging of the IPCs. To overcome this issue, a longer than usual length of subdrill is used to precondition the top 3 to 4 m of the underlying bench to minimise the oversize and bridging propensity.

Production drill and blast patterns are drilled on a 12 m high bench using a combination of 270, 229 and 165 mm diameter production holes which are designed according to the drill fleet size, the rock mass properties and the material classification, ie ore or waste.

Dedicated wall control blasting consists of a single row of 140 mm diameter presplit holes, drilled on the pit wall profile, loaded with packaged explosives, and allowing for the application of a decoupled charge. Trim blasts consisting of four to six rows of 165, 229 or 270 mm diameter holes are drilled parallel to the pit wall. Controlled blasting techniques include the use of air-decks and modified pattern parameters to distribute energy along the pit walls and minimise damage. Current drill and blast parameters are detailed in Table 16-1 and in Table 16-2.

To combat dynamic water inflow and reactive ground conditions (ie, in the presence of acid forming rockypes), every blast hole is sleeved with a plastic hole liner, into which the bulk explosives are pumped. The liner separates the explosive from the reactive ground and prevents water from damaging the charge.

Table 16-1 Sentinel production drilling and blasting parameters

Pattern type	Units	Production			
Material		Waste	Ore	Waste	Ore
Hole diameter	mm	311	270	270	229
Bench height	m	12	12	12	12
Burden	m	8	6	6.5	5
Spacing	m	9	7	7.5	6
Subdrill	m	1.5	4	3	4
Air deck	m	0	0	0	0
Stemming	m	6	5.5	5.5	4.5
Explosive type		Emulsion	Emulsion	Emulsion	Emulsion
Explosive density	g/cm³	1.2	1.2	1.2	1.2
Volume/hole	m³	864	504	585	360
Charge length	m	7.5	10.5	9.5	11.5
Charge mass/hole	kg	684	721	653	568
Powder factor	kg/m³	0.79	1.43	1.12	1.58

Table 16-2 Sentinel wall control drilling and blasting parameters

Pattern type	Units	Wall Control		
Material		Ore/Waste	Ore/Waste	Ore/Waste
Hole diameter	mm	165	229	270
Bench height	m	12	12	12
Burden	m	5	5.5	6.5
Spacing	m	5.8	6.3	7.5
Subdrill	m	0	0	0
Air deck	m	0	0	0
Stemming	m	4	5	5.5
Explosive type		Emulsion	Emulsion	Emulsion
Explosive density	g/cm^3	1.2	1.2	1.2
Volume/hole	m^3	348	415.8	585
Charge length	m	8	7	6.5
Charge mass/hole	kg	205	346	447
Powder factor	kg/m^3	0.59	0.83	0.76

Pattern type	Units	Presplit
Material		Ore/Waste
Hole diameter	mm	140
Bench height	m	12
Burden	m	NA
Spacing	m	1.6
Subdrill	m	0.5
Air deck	m	2
Stemming	m	NA
Explosive type		Splitex
Explosive density	g/cm^3	1.2
Area/hole	m^2	19.2
Charge length	m	10.5
Charge mass/hole	kg	10
Powder factor	kg/m^2	0.53

Design quality and consistency is managed through the use of a design approval document detailing the design parameters and expected blast outcomes. The document is reviewed by the drill and blast, mine planning, geology and geotechnical departments, and is signed-off before being implemented. An in-field quality control process ensures that the design is executed to a high standard and that the metrics associated with each blast, ie hole depth, charge mass and stemming length, are recorded for each hole so that a comparison with the design can be made and blast designs then optimised.

16.1.1.2 Loading and hauling

The current primary loading fleet at Sentinel comprises three x Caterpillar 7495HR electric shovels, two x Komatsu PC5500 electric shovels and a single Komatsu PC5500 excavator. There are also several 250 t and 100 t class excavators, in addition to wheel loaders.

The primary haulage fleet comprises eight x Liebherr T284 trucks (330 t capacity), seventeen x Komatsu 960E trucks (360 t capacity) and fifteen x Komatsu 860E trucks (240 t capacity).

16.1.1.3 In-pit ore crushing and conveying

Engineering and planning around IPCC locations has been a continuously on-going review process since 2012. The initial crushers IPC1A, IPC2A and IPC3A, were installed in the Phase 1 mining boxcuts and these are currently operational. Many design iterations have been produced to consider future relocations, with each one attempting to optimise and account for numerous technical considerations.

A fourth crusher, IPC4A, is now required to ensure crushing continuity when any of the other three are being relocated, and also importantly, to supplement crushing capacity for the proposed 62 Mtpa processing expansion.

The nominal crushing rate of each of IPC1A, IPC2A and IPC3A is 4,000 tonnes per hour. After availability and utilisation factors, plus operational downtime, this rate equates to approximately 18 Mtpa crushed per IPC. The larger capacity IPC4A crusher, when operational, will have a nominal crushing rate of 5,500 tonnes per hour. After similar allowances and factors, this equates to approximately 25 Mtpa crushed.

When the existing three crushers are eventually relocated to new positions in the pit, it is intended that they will each be upgraded to have a similar annual productivity rate as for IPC4A.

16.1.1.4 Waste dumping

The two waste types differentiated are:

- potentially acid forming (PAF), and
- non-acid forming (NAF)

The NAF waste is typically dumped around the planned ultimate dump perimeter, to encapsulate the PAF waste and minimise potential acidic run-off. Further information on the management and separation of the waste types is provided in Item 16.1.2. The current production schedule (described in Item 16.1.3) allows for backfilling of the depleted pit with PAF waste within ten years of pit completion.

16.1.1.5 Mining dilution and recovery

A production recording system has been implemented at Sentinel, which in conjunction with the grade control modelling system and the truck dispatch and monitoring system, makes it possible to track and record ore and waste movements (volume, tonnes and grade) to crushers/stockpiles, and to the waste dump, respectively.

In view of the “planned dilution” incorporated into the Mineral Resource model (Item 14), an additional allowance for “unplanned dilution” has been included in the pit optimisation process and in the mine production schedules. This is an additional 3% of ore tonnes at a nil copper grade. A mining recovery factor of 97% has also been adopted.

With production recording system uncertainties and continuous improvement work acknowledged, there is considered to be a reasonable reconciliation against the Mineral Reserve allowance of 3% dilution tonnage. In regards to the diluent grade, estimated variance figures suggest nil copper content. If the diluent grade was increased to 0.15%Cu to match that assumed in the 2015 Mineral Reserve allowance, the diluent metal content would remain a relatively small inclusion (Table 16-3)[7].

[7] The 3% mining dilution and 97% mining recovery factors are consistent with the optimisation inputs

Table 16-3 Sentinel reconciliation January to December 2018, mining dilution estimate

Reconciliation steps	Adjusted PBI Records[1]		
	Mtonnes	%Cu	Metal (kt)
Ore Control Markout vs	43.37	0.53	230.62
Total Plant Claim	44.99	0.50	225.18
Diluent	1.62	- 0.33	- 5.44
	3.7%		-2.4%
Ore Control Markout vs	43.37	0.53	230.62
Total Plant Claim (recalculated)	44.99	0.51	230.62
Diluent (assuming 0.00%Cu)	1.62	-	-
	3.7%		0.0%
Ore Control Markout vs	43.37	0.53	230.62
Total Plant Claim (recalculated)	44.99	0.52	233.06
Diluent (assuming 0.15%Cu)	1.62	0.15	2.44
	3.7%		1.1%

Note 1: Records adjusted to reflect direct feed ore only

16.1.2 Mine planning and operations

The following information relates to the detail that needed to be considered for designing surface layouts and practical mining phases around the Sentinel optimal pit shell outline. Additional information is provided on technical aspects related to the ongoing mining operations.

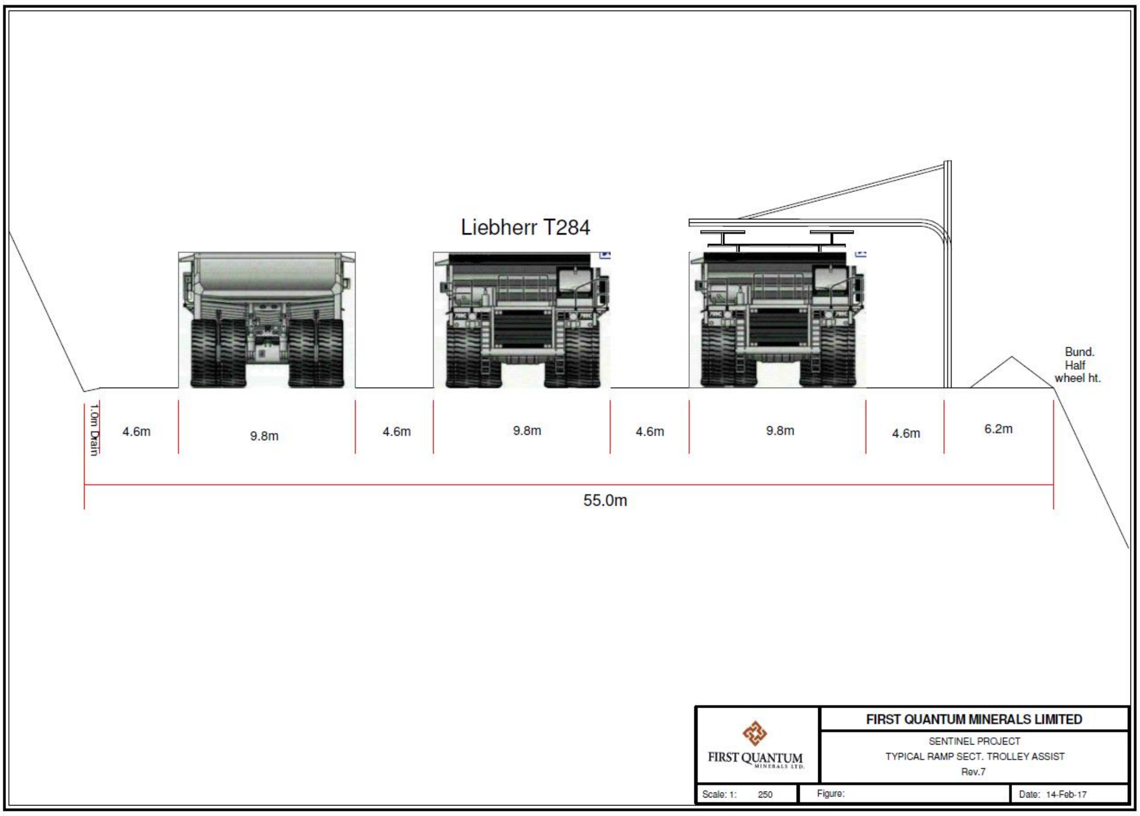
16.1.2.1 Mine design parameters

Basic mine design parameters relating to pit slope design and widths of haul roads are described in Item 15. The following parameters were adopted in designing the pit roads to suit trolley-assist haulage:

- haul road minimum width = 4.9 x truck width (to allow up-haulage off the catenary line) = 47.8 m (equivalent to 3 x 15.9 m wide haulage lanes)
- total haul road width inclusive of catenary pole, bund and side drain = 55 m (refer to schematic in Figure 16-6)
- maximum gradient = 1 : 10
- allowance for suitable catenary run-on and run-off lengths (ie, traction and non-traction transition lengths of the haul road)

Further specific and detailed geotechnical and excavation parameters had to be considered for IPC implementation. ThyssenKrupp engineering drawings provided the required excavation and installation dimensions for each IPC installation. The steep-sided and deep IPC pockets (ie, the bench excavations into which the crushers were installed) required intensive ground support and cable bolt reinforcement.

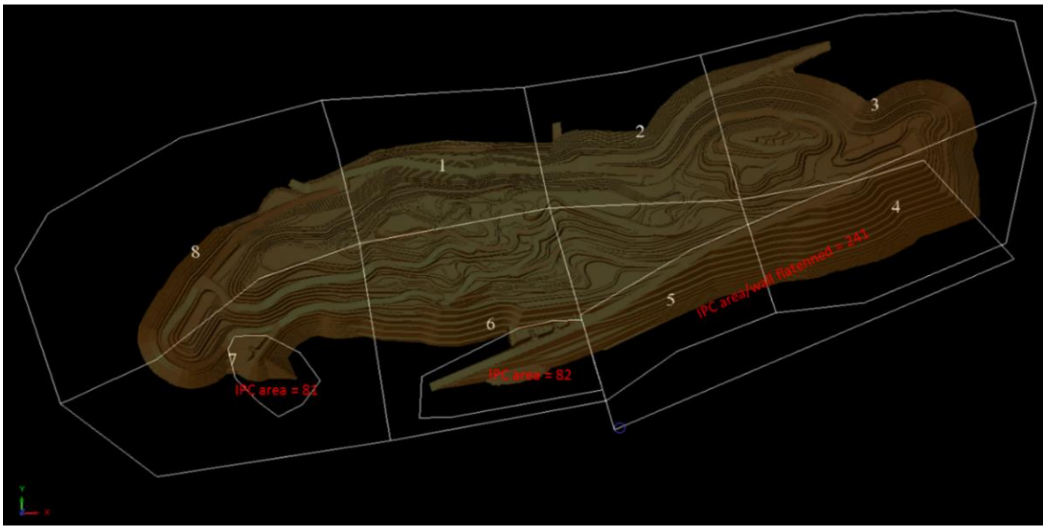
Figure 16-6 Schematic cross section across haul road for trolley-assist haulage



16.1.2.2 Geotechnical engineering

Subsequent to the 2015 Technical Report, the mine geotechnical database was expanded, and updated summary design batter and berm specifications were made available for pit design revisions. The updated geotechnical design sectors were then as shown in Figure 16-7 (XStract, July 2015).

Figure 16-7 Sentinel Pit geotechnical design sectors, updated in July 2015



The revised batter and berm parameters are listed in Table 16-4 and consistent with these details, the additional parameters in Table 16-5 are the respectively updated (July 2015) overall slope angle specifications. The figures shaded in grey at the bottom of this table are the comparative angles as used for the May 2015 pit optimisation. As can be seen, the revised overall angles are somewhat steeper than previously for sectors 1 and 2, are flatter in sector 3, are flatter within subsectors 4 to 7, and steeper in sector 8.

Table 16-4 Recommended pit slope design parameters for Sentinel, XStract July 2015

Domain	Depth Range (m)	Wall orientation	Bench height (m)	Batter angle (degrees)	Min. berm width (m)	IRA (C to C)
Weathered All rock units	0 to 48	All	12	50	8	33.6
MPH	> 48	South wall (330^o to 040^o)	24	60	15	39.8
		All others	24	75	10	55.6
MPHB	> 48	South wall (330^o to 040^o)	24	60	15	39.8
		All others	24	75	10	55.6
MSC	> 48	All	24	75	10	55.6

Table 16-5 Sentinel pit overall slope angle parameters, updated in July 2015

Slope Area		1	2	3	4	5	6	7	8
Weathered bottom RL	mRL	**1160**	**1160**	**1172**	**1172**	**1172**	**1160**	**1172**	**1160**
MPH top	mRL	1076	1016	992	surface	surface	surface	surface	1148
MPHB top	mRL	1052	980	944	1148	1124	1148	1160	1136
MSC top	mRL	from surface	from surface	from surface	n/a	n/a	n/a	n/a	surface
Top	mRL	1220	1208	1208	1232	1220	1220	1208	1196
Bottom	mRL	872	908	836	956	932	882	932	932
Ramp1	mRL	1112	1064	1166	968	1088	1004	1136	1112
Ramp2	mRL		992	896		1070	908	1028	
Ramp width	mRL	60	60	60	55	50	60	60	60
Weathered Rock Base	mRL	1160	1160	1172	1172	1172	1160	1172	1160
Vertical distance to ramp1/Weath. Base	m	60	48	36	60	48	60	36	36
Horizontal distance to ramp1/Weath. Base	m	90	72	54	90	72	90	54	54
Vertical distance to ramp2/Weath. Base	m								
Horizontal distance to ramp2/Weath. Base	m								
MSC Base	mRL	1076	1016	992	ignored	ignored	ignored	ignored	1148
Vertical distance to ramp1/MSC. Base	m	48	96	6					12
Horizontal distance to ramp1/MSC. Base	m	94	128	67					13.69
Vertical distance to ramp2/MSC. Base	m	36	48	174					
Horizontal distance to ramp2/MSC. Base	m	27	34	123					
MPH Base	mRL	1052	980	944	1148	1124	1148	1160	1136
Vertical distance to ramp1/MPH. Base	m	24	24	48	24	48	12	12	12
Horizontal distance to ramp1/MPH. Base	m	16	76	33	29	58	14	14	8
Vertical distance to ramp2/MPH. Base	m		12						
Horizontal distance to ramp2/MPH. Base	m		10						
MPHB Base	mRL	872	908	836	956	932	882	932	932
Vertical distance to ramp1/MPHB Base	m	180	72	48	180	36	144	24	24
Horizontal distance to ramp1/MPHB Base	m	123	55	94	271	98	240	96	81
Vertical distance to ramp2/MPHB Base	m			60	12	18	96	108	180
Horizontal distance to ramp2/MPHB Base	m			41	24	74	180	192	123
Vertical distance to ramp3/MPHB Base	m					138	26	96	
Horizontal distance to ramp3/MPHB Base	m					168	36	120	
Total Vertical Distance	m	**348**	**300**	**372**	**276**	**288**	**338**	**276**	**264**
Total Horizontal Distance	m	**352**	**376**	**412**	**415**	**470**	**561**	**477**	**280**
Overall Slope Angle Input w/o IPC areas	**degrees**	**45**	**39**	**42**	**34**	**31**	**31**	**30**	**43**
Overall Slope Angle Input w/ IPC areas	**degrees**	**45**	**39**	**42**	**24**	**23**	**31**	**30**	**43**
IPC areas	**degrees**				***(footwall flattened)***		**8(Z>1170)**	**8(Z>1166)**	
Overall angle slope 2014/2015 (finsmumodc)	**degrees**	**41**	**35**	**44**	**34**	**29**	**29**	**33**	**29**

Geotechnical design parameters were further reassessed in 2016 (XStract, June 2016) and an inter-ramp and overall slope angle of maximum 27^o specified for the footwall MPBH lithology, where shown in Figure 16-8. The recommended design details are listed in Table 16-6 and incorporated into the overall slope design specifications in Figure 16-9. To note is that design sectors 2 (ie, 21 and 22) and 3 (ie, 31 and 32) were subdivided to reflect a complex slope geometry featuring haul ramps and switchbacks. An additional design sector 9 was included in respect of the Phase 2 and 4 lower slopes extending to the base of the pit.

Figure 16-8 Sentinel Pit footwall design review, after XStract (2016)

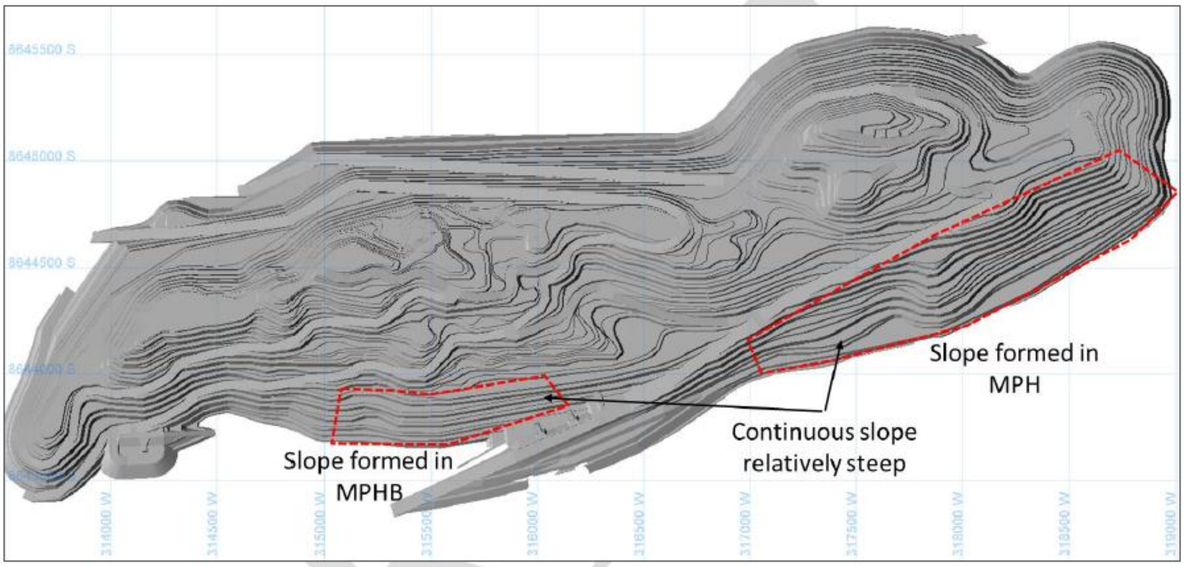
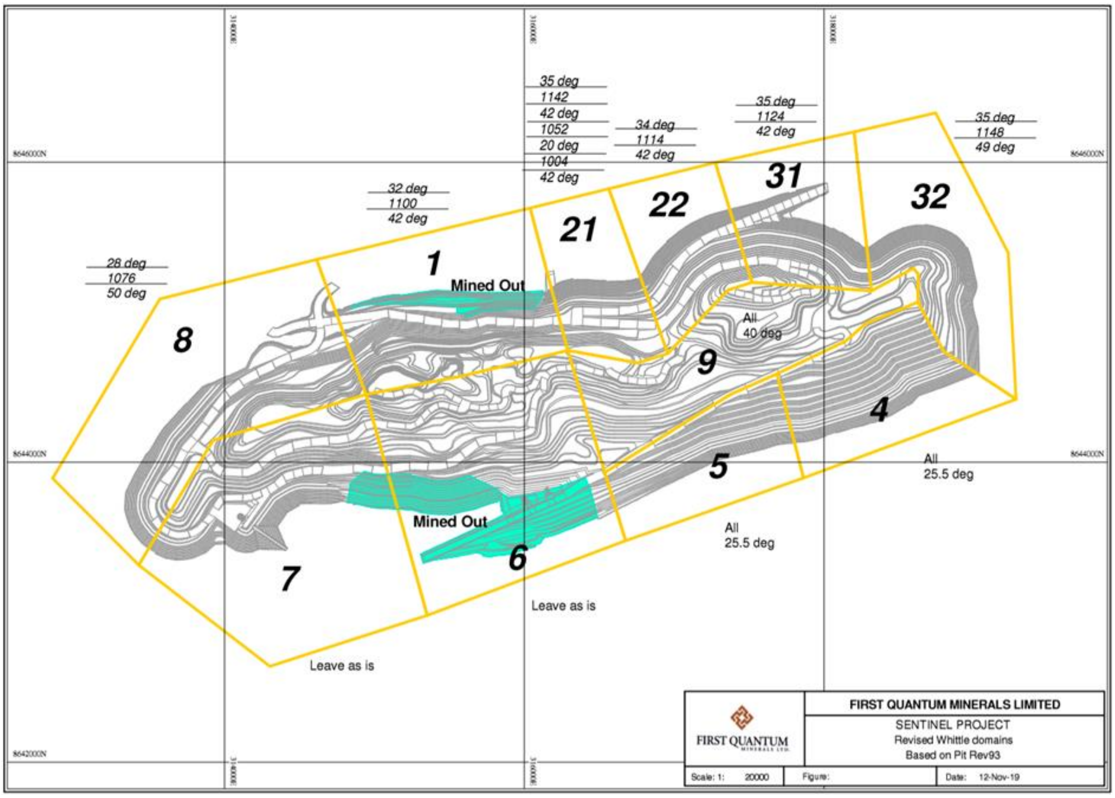


Table 16-6 Sentinel pit slope design parameters, updated June 2016

Domain	Depth Range (m)	Wall orientation	Bench height (m)	Batter angle (degrees)	Min. berm width (m)	IRA (C to C)
Weathered	0 to 60~70	South facing wall Sectors 1-2-3	6	67.0	6	35.0
Weathered	0 to 30~40	South facing wall Sector 8	6	37.2	4	26.0
MPHB	> 60	South facing wall Sector 1-2-3-8	12	34.0	6	27.0
MPH	> 60	South facing wall Sector 1-2-3-8	24	75.0	10	55.6
MSC	> 60	South facing wall Sector 1-2-3-8	24	75.0	10	55.6
Weathered	0 to 35~45	North facing wall Sectors 4-5	6	37.5	4	27.0
Weathered	0 to 35~45	North facing wall Sectors 6-7	6	37.5	6	24.0
MPHB	> 45	North facing wall Sectors 4-5-6-7	12	34.0	6	27.0
MPH	> 45	North facing wall Sectors 4-5-6-7	24	37.6	16	27.0
MSC	> 45	North facing wall Sectors 4-5-6-7	24	75.0	10	55.6

Figure 16-9 Sentinel overall slope parameters for pit design, updated November 2019



To note from these 2016/2019 updated parameters is the flatter inter-ramp angle of 27° for the south wall sectors 4 and 5, relative to 39.8° for the same sectors in the 2015 specifications (Table 16-4). In 2015 planning, the south slope design parameters were more/less conformable with the (steeper) footwall of the mineralization, as shown on the Figure 16-10 cross section.

Subsequently, the south slope design was changed as per the 2016 specification; Figure 16-11 shows the equivalent cross section. The overall slope is flatter than was previously adopted and is now conformable with the detachment zones (and foliation).

Figure 16-10 Cross section through design footwall slope, as per 2015 design parameters

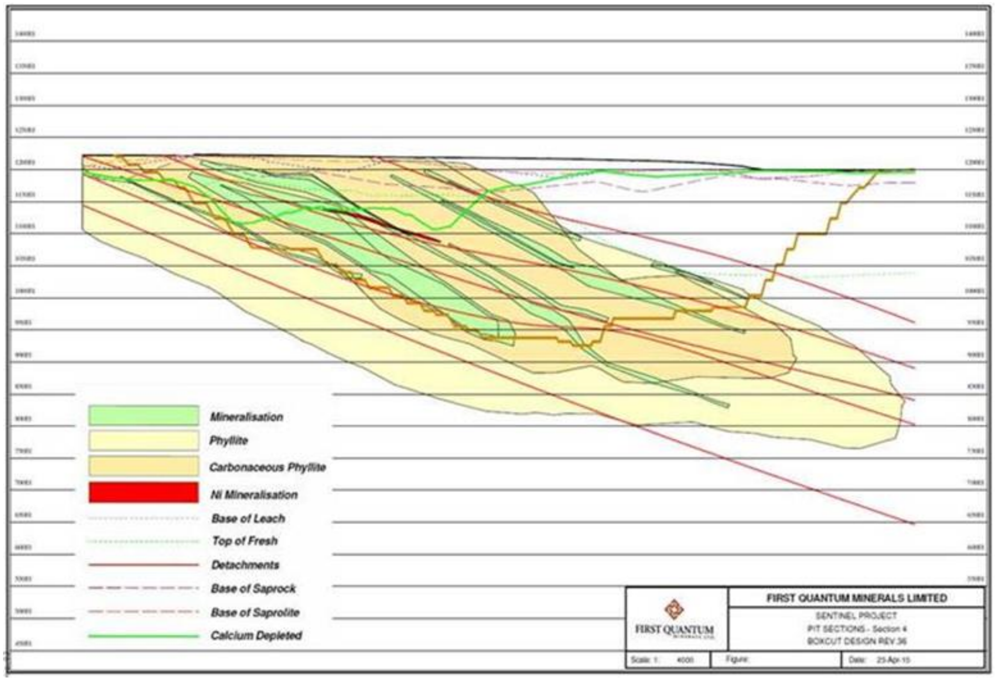
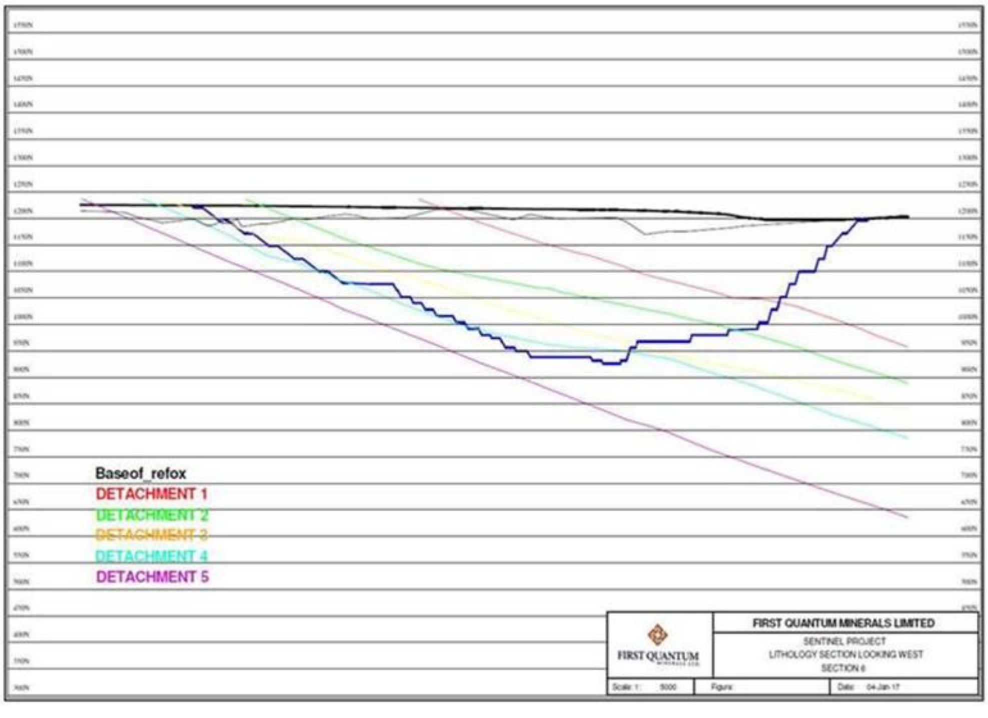


Figure 16-11 Cross section through design footwall slope, as per 2016 design parameters



SLR Consulting (Africa) Proprietary Ltd (SLR, April 2019) conducted a site visit and geotechnical review of the Phase 1 north and south pit walls in April 2019. This visit and review was in response to localised planar sliding failures associated with undulating foliation on the south wall slope, and slumping of the upper north wall in saprolite in the presence of relict structure and groundwater pore pressure.

In respect of the south wall, SLR recommended improved definition in design sector 6 such that zones of differing rock mass classification could be identified to enable locally modified berm widths and slope stack heights. In respect of the north wall, SLR acknowledged the effectiveness of the installed horizontal drains (Figure 16-23) and recommended the installation of piezometers to measure pore pressure reduction.

Table 16-6 lists bench heights of 24 m in certain design sectors. Based on operational performance, and in particular perimeter blasting performance, the bench heights on ultimate phase walls may be reduced to 12 m in future. This modification would not alter the design inter-ramp angle.

16.1.2.3 Loading and hauling

The Liebherr T284 trucks are a more recent addition to the originally procured Komatsu fleet. At the outset of mining, it was envisaged that the Komatsu 960E trucks would be primarily dedicated to waste hauls, whilst the Komatsu 860E trucks were to be used for the ore hauls. In practice, the larger capacity trucks are frequently used for hauling ore and this has led to a realisation that the installed IPCs have an inadequate vault capacity to cope with this situation over a prolonged period.

As mentioned in Item 16.1.1, the outcome of this is that the new IPC4A crusher to be installed in Phase 2 has been manufactured with a larger vault capacity. When it comes time for the existing crushers to be relocated to new positions in the expanding and deepening pit, it is intended that they will each be retrofitted with a larger vault.

A number of the trucks have been fitted with trolley assist haulage pantographs (ie, all of the T284 trucks, fifteen of the 960E trucks and one of the 860E trucks). There is an existing trolley line where shown in Figure 16-12 leading up onto Dump 4, and another where labelled “1” in Figure 16-12. The Dump 7 trolley line shown on this figure is scheduled for construction in Q1 2020, whilst the line labelled “3” is scheduled for construction in Q4 2020.

For longer term planning purposes, and for the sake of updated mining cost estimates, potential trolley line routes have been identified for future ore and waste hauls. These future routes, some of which would be temporary installations (ie, particularly the north-south routes between adjoining phases), are shown on Figure 16-13.

Figure 16-12 Plan showing location of existing and proposed near term trolley assist haul routes

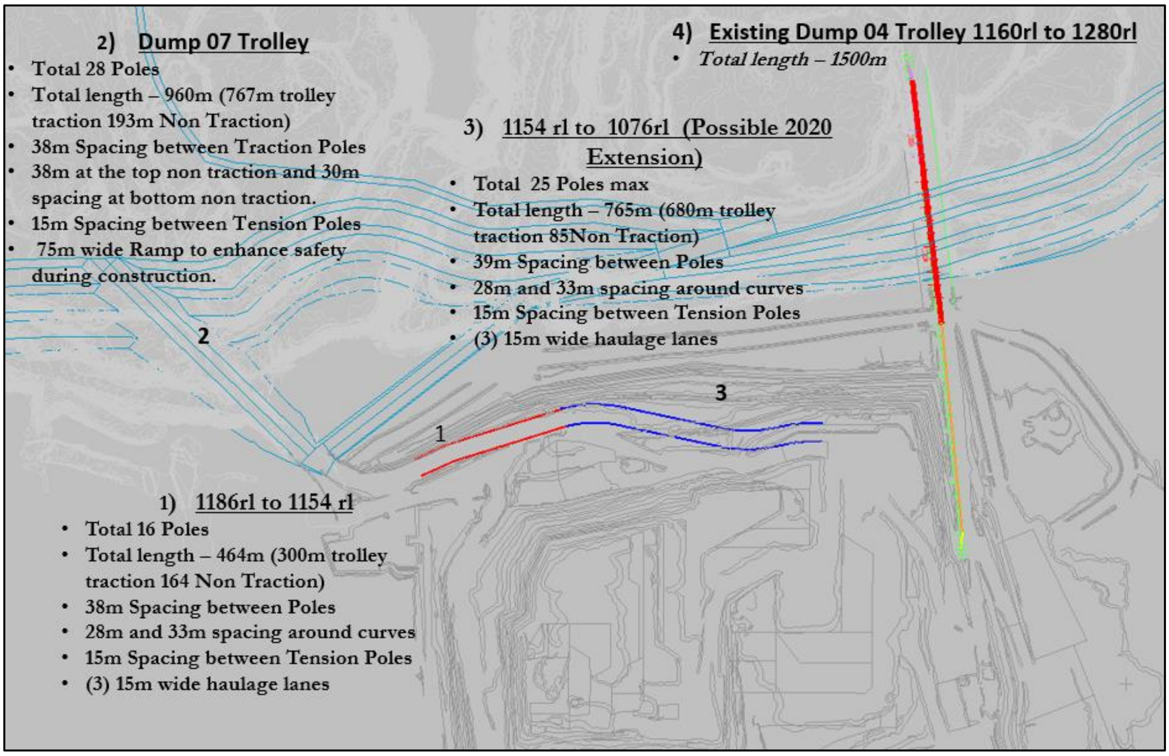
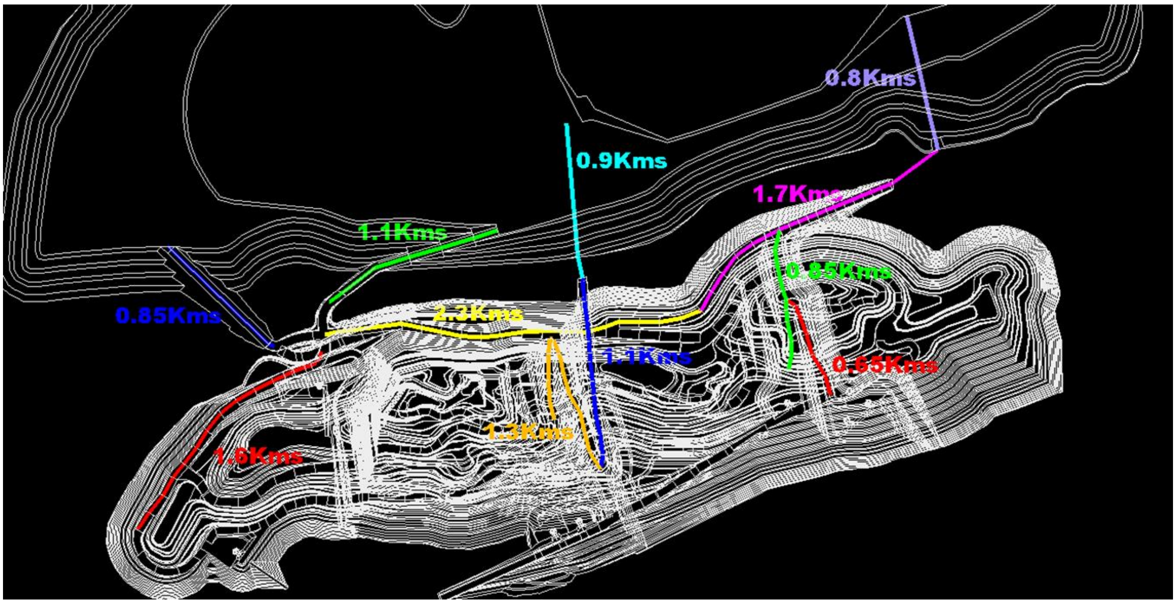


Figure 16-13 Plan showing location of proposed long term trolley assist haul routes



16.1.2.4 In-pit crushing and conveying

The integration of a suitable IPCC layout into the Sentinel phased pit scheme has required an iterative approach to pit design and production scheduling, balancing mining practicality and haulage considerations with plant feed delivery targets and mechanical design considerations.

The existing IPC1A, IPC2A and IPC3A crushers in the Phase 1 Pit are of ThyssenKrupp KB 63 x 89 specification. The proposed new fourth crusher, IPC4A, will be of ThyssenKrupp KB 63 x 130 specification and its planned location is near-surface at the southern western end of Phase 2. The scheduled completion date for the IPC4A crusher pocket is at the end of Q1 2021. The crusher would be part assembled on surface and trammed into position using a purpose built crawler/tractor, in much the same procedure as was done for the IPC1A, IPC2A and IPC3A crushers. Once positioned into the prepared crusher pocket, final assembly of IPC4A and its associated infrastructure (ie, in-pit conveyor extension and transfer stations) would be completed before the end of 2021.

The location of the four crushers is shown in Figure 16-14, whereas Figure 16-15 shows the currently proposed future IPC relocation positions. This relocation concept suits the production schedule described in Item 16.1.3 and takes account of the required mining practicality and associated considerations. The positions and timeframes for IPC relocations are subject to ongoing optimisation.

The respective IPC productivity rates adopted for mine planning are listed in Table 16-7. Whilst the average crusher productivity rate during 2019 to date, for the three existing crushers, has been approximately 3,500 tph this does not take into account the capacity of the discharge conveyor belts nor the rate at which the crusher vaults can be emptied onto the conveyors.

Table 16-7 Design productivity for in pit crushers

ThyssenKrupp IPC	Units	KB 63 x 89	KB 63 x 130	Comments
Nominal productivity	tph	4,000	5,500	for each crusher
Availability	%	92%	92%	routine crusher maintenance
Planned productivity	tph	3,680	5,060	
Utilisation	%	65%	65%	16 hrs/day (discontinuous feed)
Hourly production	tph	2,392	3,289	
Daily capacity	tpd	57,408	78,936	
Operational downtime	days/yr	50	50	blasting, belt maintenance
Annual capacity	**Mtpa**	**18**	**25**	

The design rate for the yet to be installed IPC4A crusher (KB 63 x 130 specification) is based on the same availability, utilisation and downtime factors that apply to the existing crushers (KB 63 x 89 specification). It is intended that the existing three crushers will be retrofitted to the larger capacity during their future relocation and reinstallation.

Figure 16-14 Proposed location of IPC4A in Sentinel Phase 2 Pit

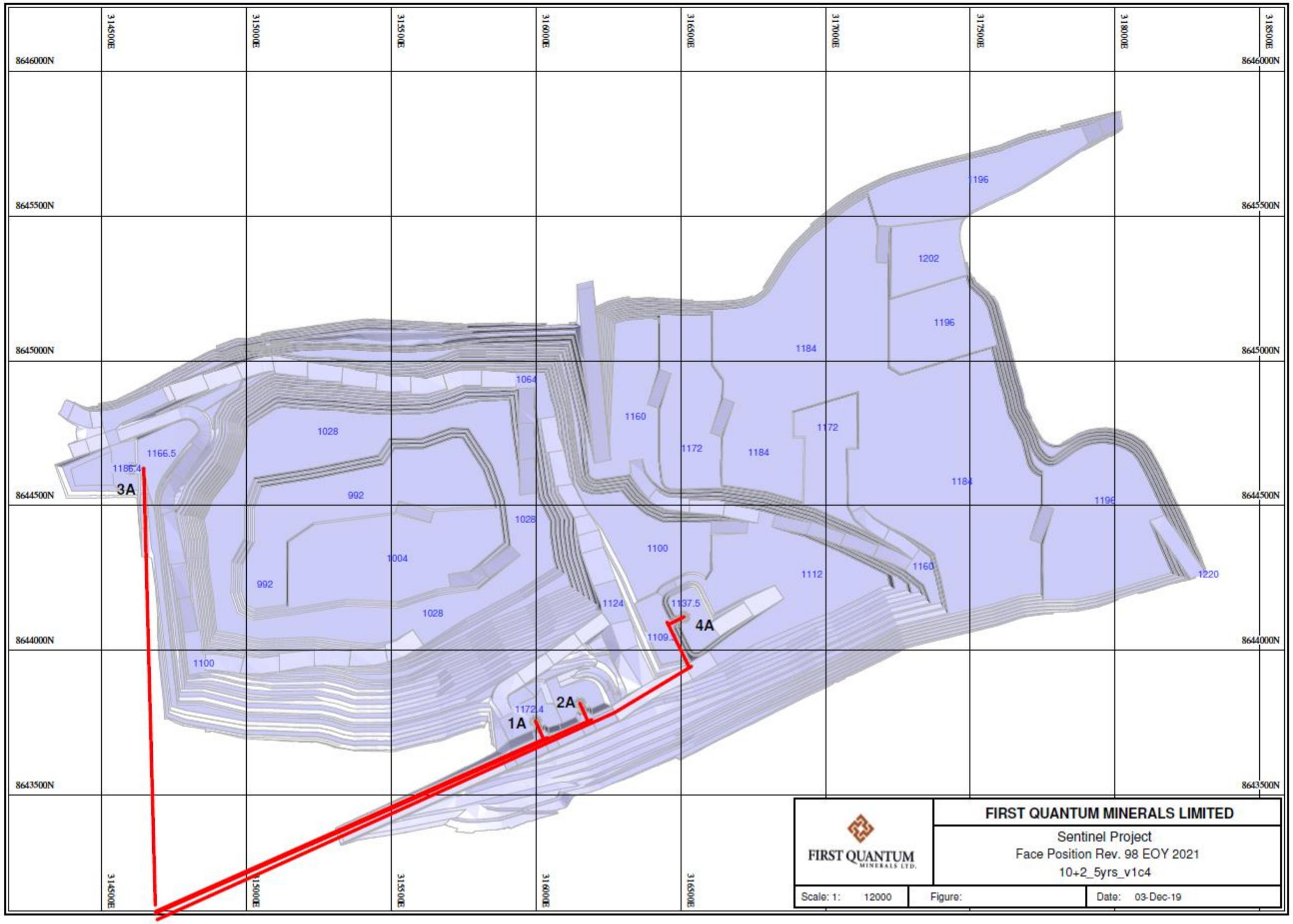
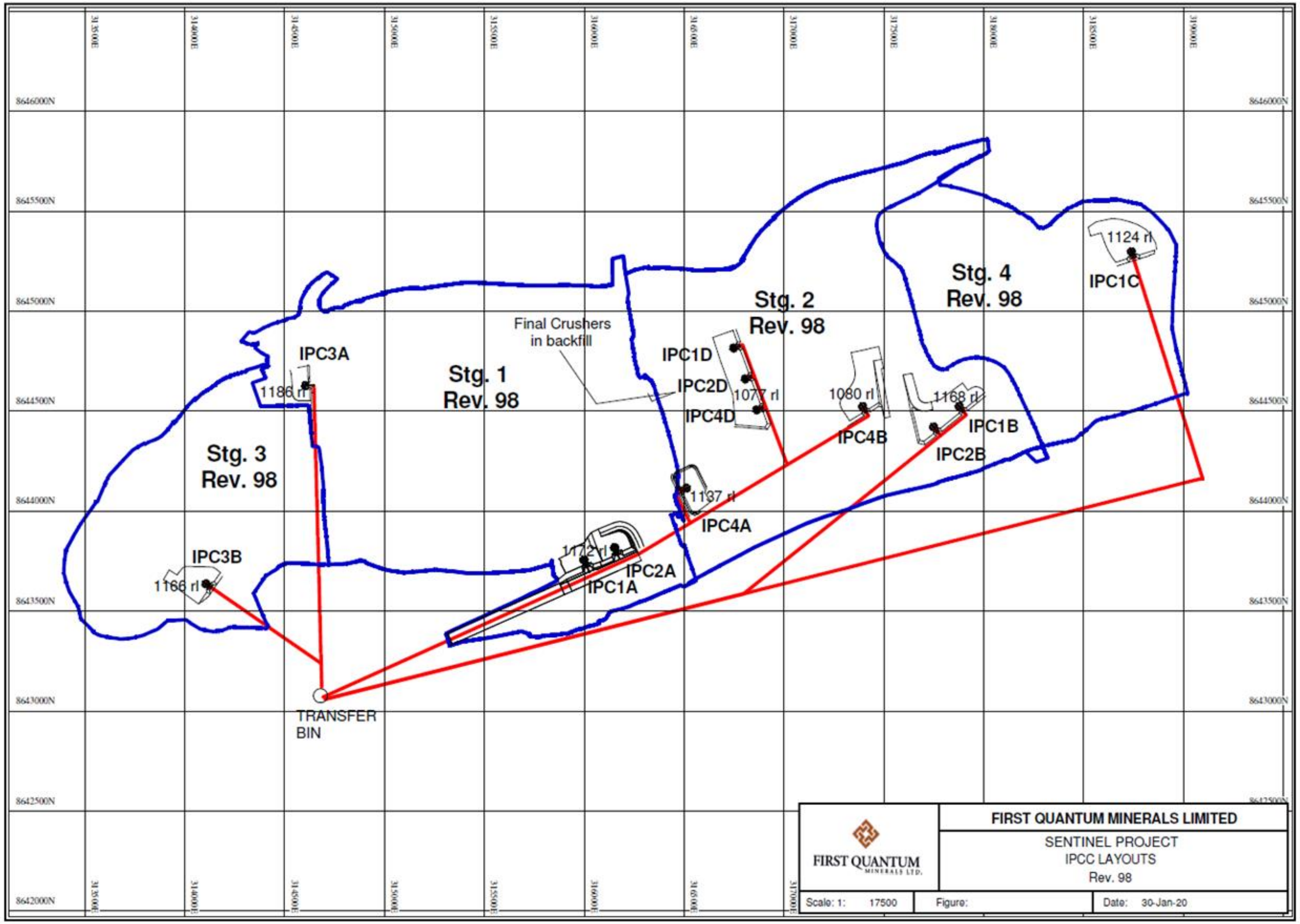


Figure 16-15 Sentinel in pit crusher relocation positions



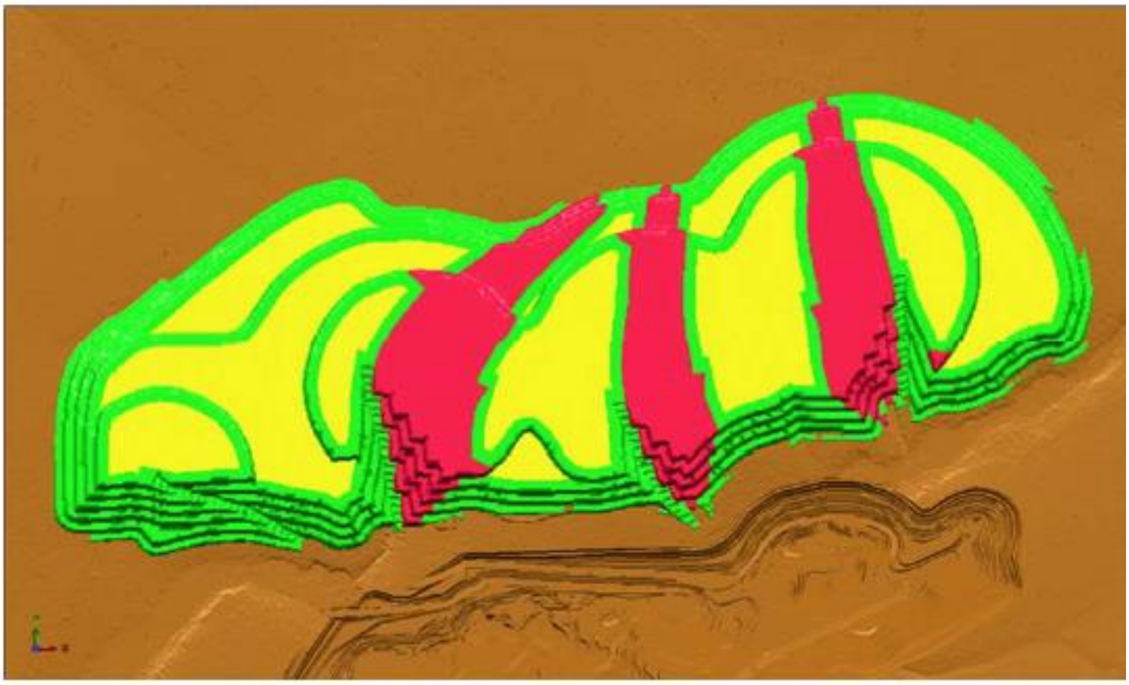
16.1.2.5 Waste dump optimisation and design

An optimised LOM waste dump design was completed in February 2015. The optimisation considered acid rock drainage (ARD) management criteria and established a process for the dumping of potentially acid forming (PAF) waste within a non-acid forming (NAF) waste encapsulation. By virtue of the staged sequence of the mining phases, there was an ability to backfill the depleted pit in the latter half of the mine life.

Figure 16-16 shows the ultimate Sentinel waste dump landform as was originally envisaged. Green blocks represent NAF encapsulation, yellow blocks are NAF capping and red blocks are NAF fill within the apache passes. PAF waste is not visible under the encapsulation and capping.

The waste dump slopes were designed with 20 m high benches, and with wall angles at the natural angle of repose (assumed to be 37^{o}) and with 50 m wide berms. The toe of the dump nearest the pit was designed to be at least 250 m from the pit crest to conform with geotechnical recommendations.

Figure 16-16 Sentinel Pit, ultimate waste dump design, 2015 Mineral Reserve plan



An updated waste dump design (and dumping/backfilling schedule, as described in Item 16.1.3) has been produced for the current long term plan. Unlike the optimised dump design that was produced in 2015, this particular design assigns the mined waste into pre-defined dump areas, or into the depleted pit as backfill. It does not optimise lateral haul/dump length *vs* height, nor does it consider the placement of NAF waste sufficient to encapsulate PAF waste. An optimisation which considers these waste haulage aspects remains in progress during Q1 2020.

The design, as it currently stands, has dumping areas defined as shown in Figure 16-17. The north west dump (WRD-NTH-WEST) corresponds to the existing Dump 7, adopting the site nomenclature. The north west centre dump (WRD-NTH-WEST-CENTRAL) corresponds to Dump 5, whilst the north east centre dump (WRD-NTH-EAST-CENTRAL) corresponds to Dump 4. The south east dump is shown in the figure as WRD-STH.

The total design capacity in the four north dumping areas is 509.6 Mlcm. The capacity of the south east dump is 87.5 Mlcm.

The dump design differs from the 2015 version as follows:

- the western extremity has been moved eastwards by 700 m to avoid a watercourse
- the apache passes (intended for last dumping) have been omitted
- the south east dump now exists

Figure 16-17 Sentinel Pit, waste dump design, 2020 Mineral Reserve plan

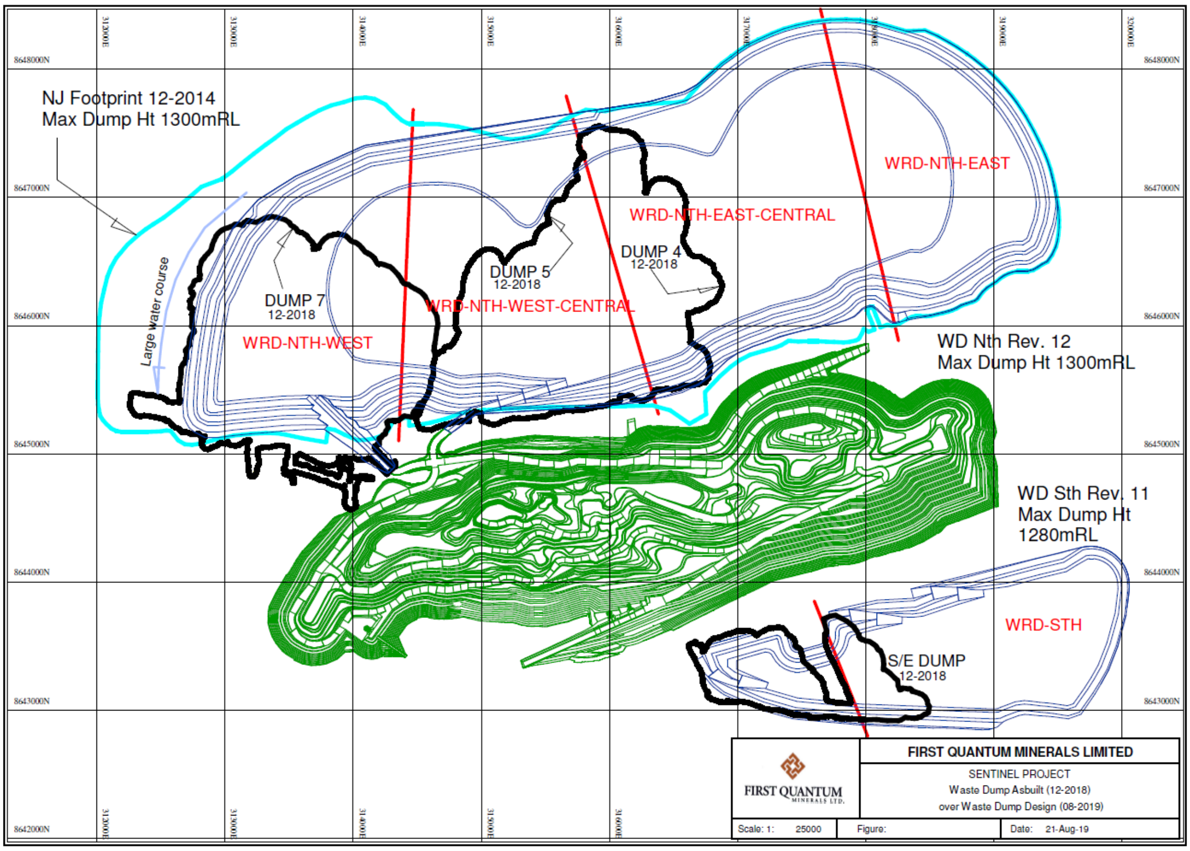
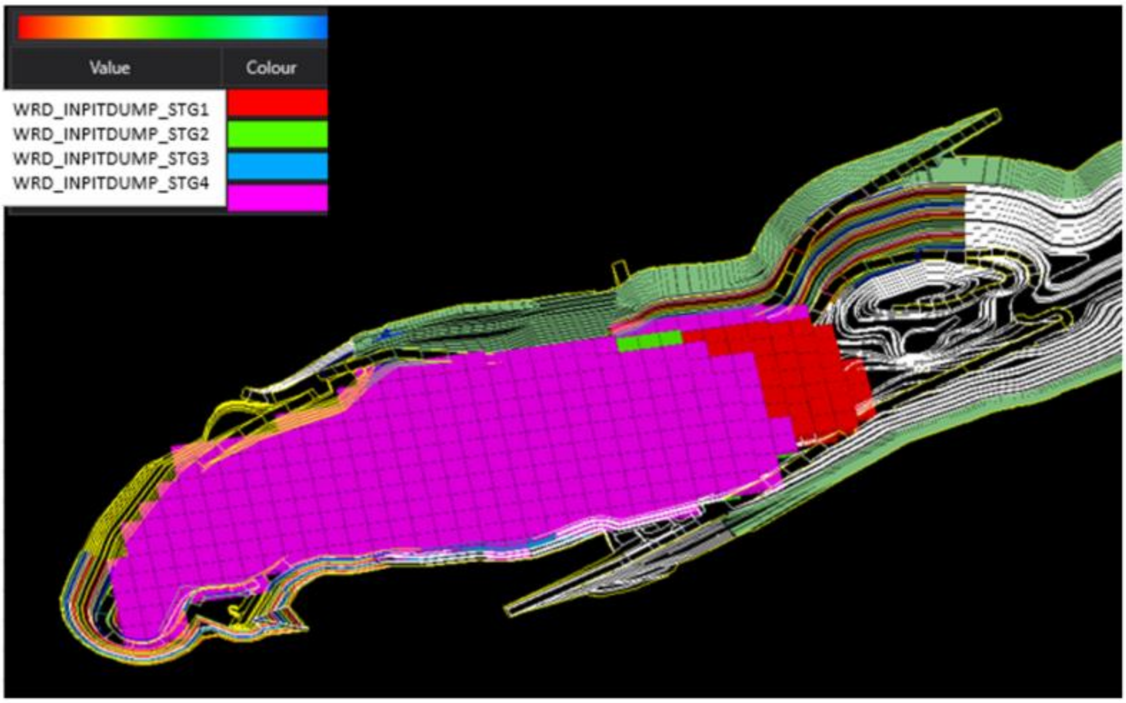


Figure 16-18 shows the inpit backfilling extents, where backfilling is scheduled in four stages. The total pit backfill volume is 226 Mlcm. There is less backfilling in the 2019 long term plan, relative to the 2015 version. The net effect of the differences since 2015 is that the south east dump capacity compensates for the reduction in the north dump footprint and the reduction of inpit backfilling volume. Long term mineralised waste stockpiles displace some Dump 5 and south east dump capacity.

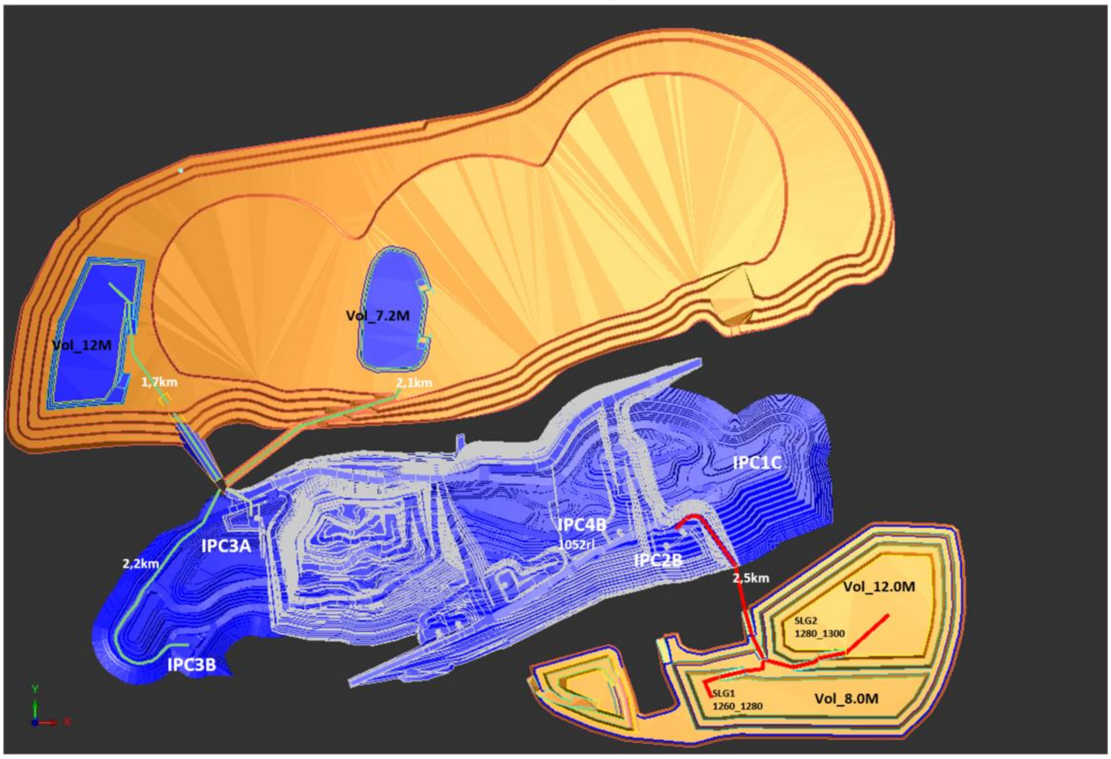
Figure 16-18 Sentinel Pit, in-pit backfill extents, 2020 Mineral Reserve plan



16.1.2.6 Ore Stockpiling

An outcome of the production scheduling described in Item 16.1.3 is the development of ore stockpiles. These are to be located on surface in positions that are proximal to near surface IPCs, as shown in Figure 16-19.

Figure 16-19 Sentinel Pit, surface stockpiles, 2020 Mineral Reserve plan



16.1.2.7 Hydrogeology - pit dewatering bores

There are currently fourteen active dewatering boreholes at Sentinel, with an average cumulative yield of 111 L/sec. Their location and details are shown in Figure 16-20 and Table 16-8, respectively.

Figure 16-20 Existing Sentinel dewatering bores

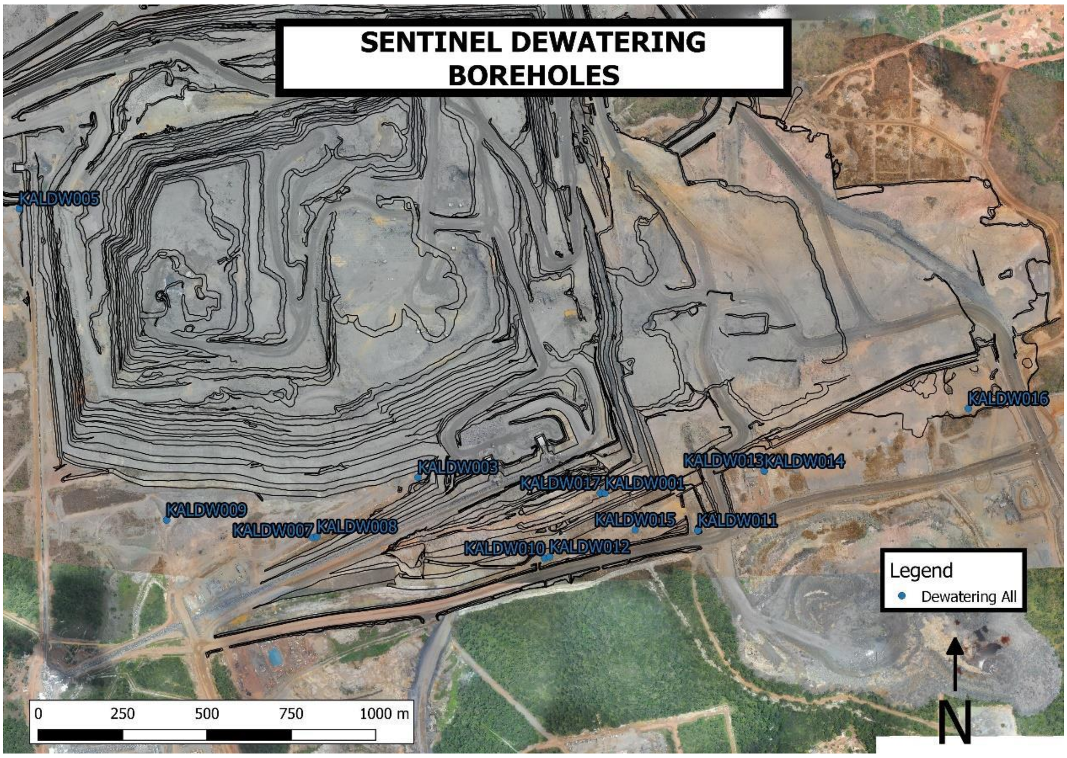
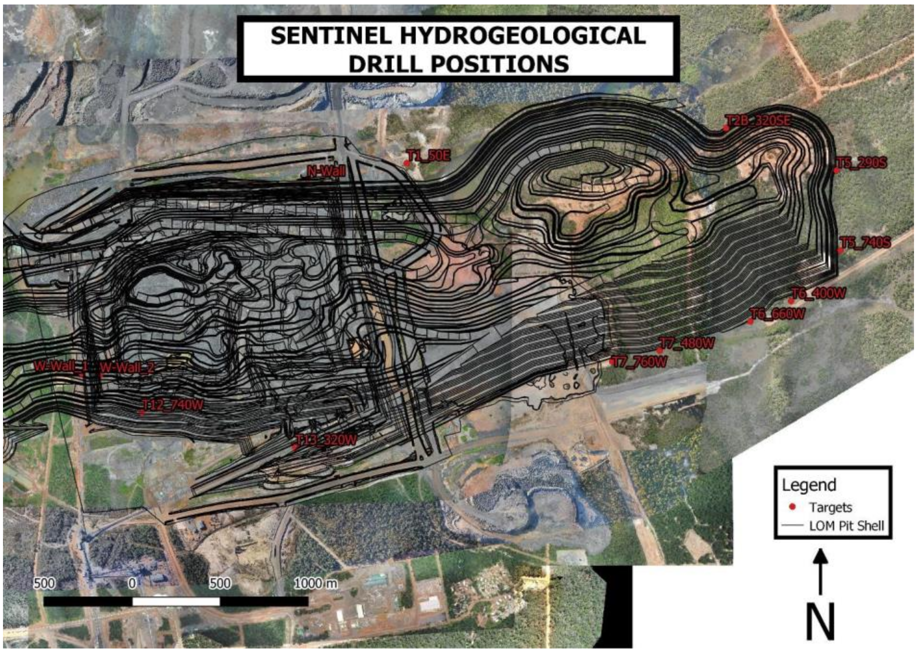


Table 16-8 Sentinel groundwater bores and flow rates, 2019

Borehole ID	m North	m East	m RL	av. flow (L/sec)
KALDW001	8,643,661	316,342	1,187	4.0
KALDW003	8,643,708	315,788	1,213	28.1
KALDW005	8,644,506	314,605	1,201	0.8
KALDW007	8,643,528	315,476	1,213	4.9
KALDW008	8,643,533	315,488	1,213	1.4
KALDW009	8,643,581	315,043	1,210	4.8
KALDW010	8,643,467	316,164	1,222	2.3
KALDW011	8,643,551	316,618	1,223	7.0
KALDW012	8,643,469	316,182	1,223	1.1
KALDW013	8,643,728	316,813	1,225	23.0
KALDW014	8,643,726	316,814	1,225	22.0
KALDW015	8,643,560	316,429	1,224	7.0
KALDW016	8,643,907	317,442	1,224	3.0
KALDW017	8,643,661	316,328	1,187	1.6
			Total	**111.0**

The drilling of fifteen exploratory pilot boreholes was completed during Q3 2019 (Figure 16-21). The purpose of these boreholes was to gather hydrogeological data to be used as further refinement and calibration input to an existing groundwater model. An update of this model will aid in an improved understanding and prediction of future groundwater inflows and movements at the Sentinel operation.

Figure 16-21 Pilot borehole locations, October 2019

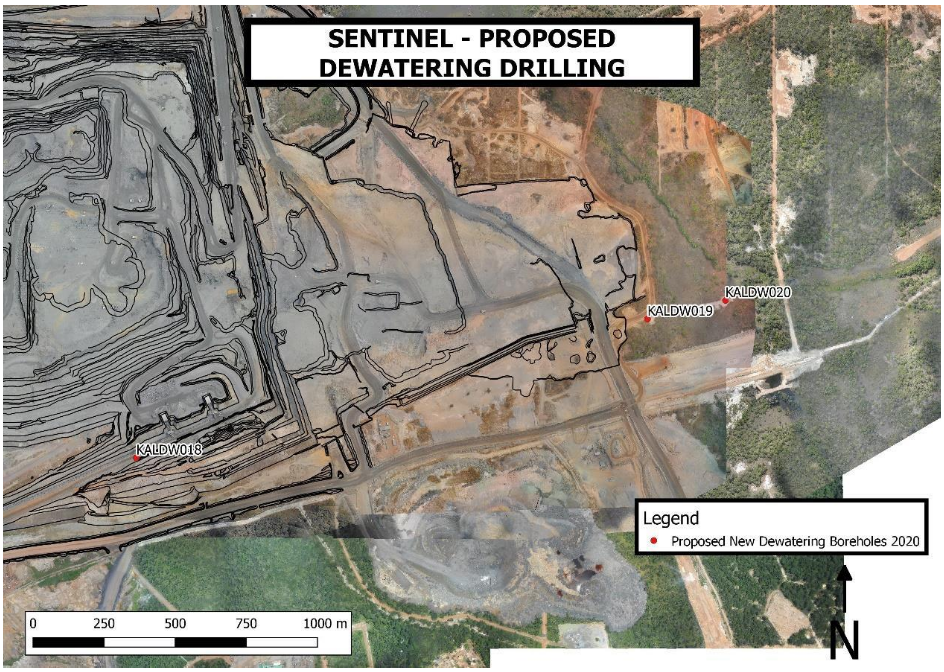


All boreholes were drilled by means of air-percussion method, to enable the measurement of blow-yields during and subsequent to drilling. The average depth of drilling was 200 mbgl. The raw data is currently being processed with the aim of updating the groundwater model in 2020.

Each of the boreholes have been equipped with nested-piezometers. In addition to the enhancement of the monitoring network, hydrochemistry samples from these new boreholes can be used to analyse chemical suite properties.

As a result of the pilot borehole drilling, three sites have been proposed as locations for dewatering bores required in the near future. These locations are shown on Figure 16-22.

Figure 16-22 Proposed future dewatering bore locations

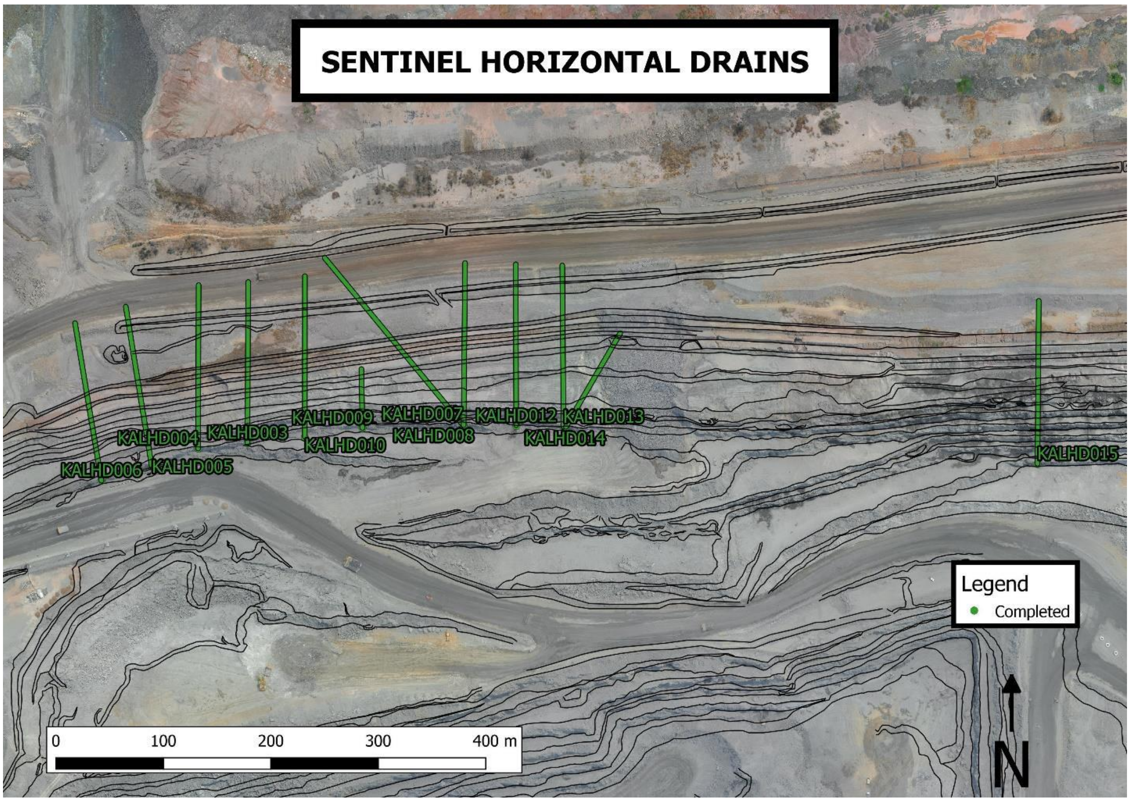


16.1.2.8 Hydrogeology - horizontal drains

A 200 m length of the upper north wall in Phase 1 showed signs of progressive instability over a period up to August 2018, at which time it failed over multiple benches. The failure zone has since been buttressed, the north wall ramp diverted and the site continues to be radar monitored. A contributing cause of this failure in the near-surface weathered horizon was due to pore pressure in the slope.

Subsequent to the failure, horizontal drains were drilled into the face in an effort to reduce the pore pressure. Twelve borehole drains were drilled on the 1160 mRL level to an average depth of 150 m and at a 10° inclination. These drains have subsequently proven effective in reducing the saturation of the saprolitic rock (Figure 16-23).

Figure 16-23 Sentinel Phase 1 Pit north wall horizontal drains



16.1.2.9 Hydrogeology - piezometer monitoring

A number of vibrating wire piezometer (VWP) boreholes have been drilled and fitted with pressure probes and associated telemetry. Piezometer positions were selected to provide pore pressure data in specific areas impacted by pore pressure, ie the upper north wall of Phase 1, and the pit slopes below the Dump 4 trolley line, and opposite crushers IPC1A and IPC2A.

The purpose of these installations is to gather pore pressure data and to track changes over a time-series as mining progresses, with emphasis on measuring the effect that the horizontal drains have on slope depressurisation.

16.1.2.10 Hydrology

Surface water flow management across the mine site has been addressed from three perspectives:

- the management of upstream Musangezhi River flow posing a pit inundation and operational safety risk to the open pit
- the management of potential sediment laden run-off from the site into the downstream Musangezhi River
- the management of waste dump run-off and contact/"dirty water" from the mine

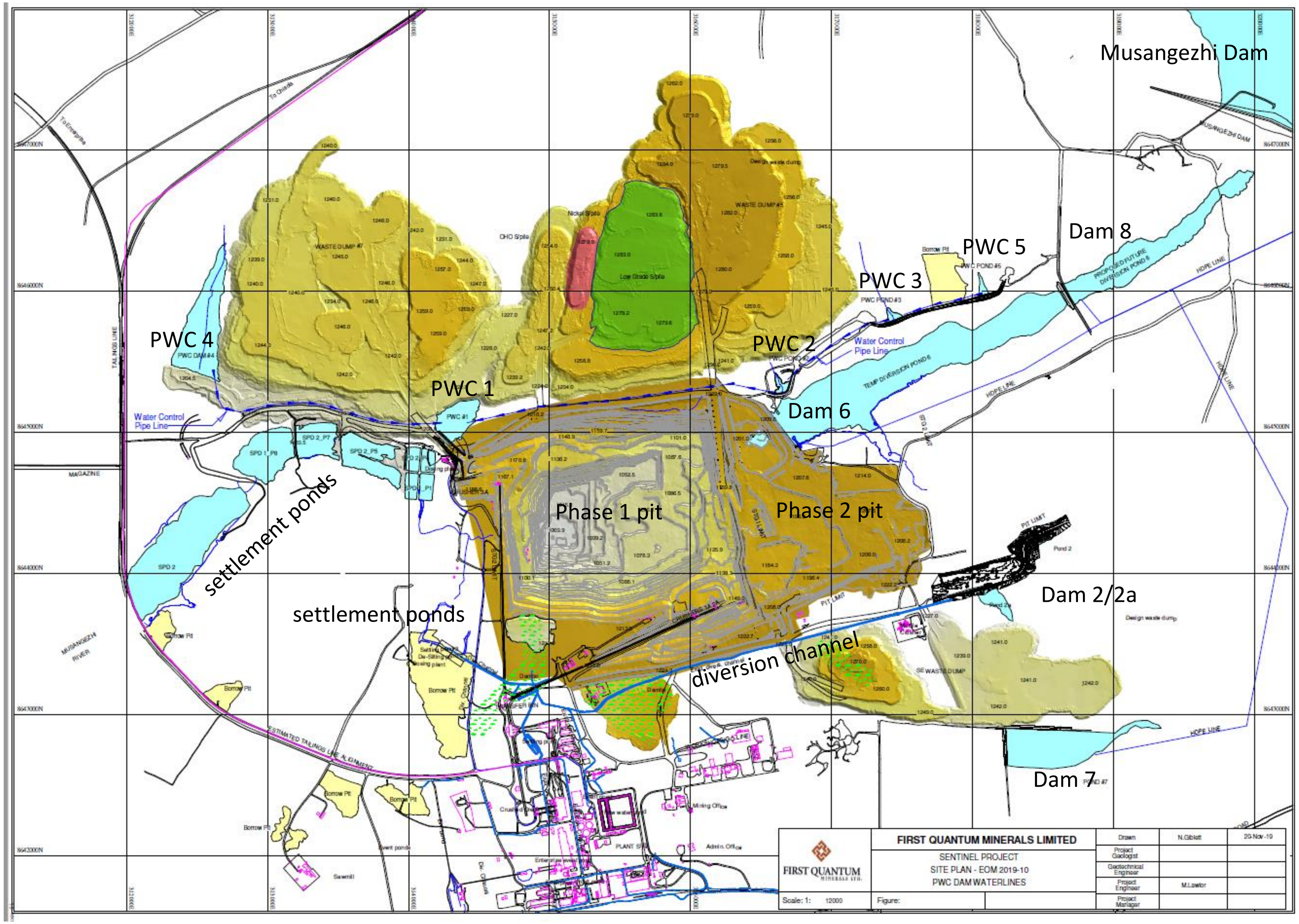
Regarding the first perspective, Figure 16-24 shows the location of several up-stream river diversion dams (Dams 2/2a, 6, 7 and 8). Dams 2/2a, 6 and 7 are in existence whilst Dam 8 is proposed for when mining progresses to the north east and the Phase 2 pit is mined across to its planned northern limits. Clean run off collected in Dams 6 and 7 (and 8, in the future) is pumped back to the Musangezhi Dam.

Regarding the second perspective, Figure 16-24 shows the location of a series of settlement ponds (SPD 1 and 2) which are intended to trap sediments and pollutants and allow them to settle out before discharge to the environment at levels which meet required water quality standards for turbidity and total dissolved solids. A four compartment pond is located at the head of this series of ponds, into which lime is added to manage pH levels. The compartments allow the periodic removal of accumulated sludge.

Regarding the third perspective, Figure 16-24 also shows the location of water control ponds (PWC 1 to 5) to channel/pump surface water flows from the northern dump across and into the series of settlement ponds. PWC 1 is the receptor for these channelled/pumped flows, which then decant into the upper, compartmented settlement ponds.

A diversion channel has been constructed beyond the southern crest line of the pit, where shown on Figure 16-24, to direct run-off from the southern dump and surrounds into a southern series of settlement ponds. These eventually decant across to the northern ponds. The southern diversion channel also intercepts seepage from dambos and receives groundwater pumped from dewatering bores.

Figure 16-24 Sentinel surface water management infrastructure



16.1.2.11 ARD management

A programme of static and kinetic test work to confirm the potential for ARD generation at Sentinel was initiated in 2013 and continues to date. The test work has established that pyrite and pyrrhotite exist throughout the phyllite/hangingwall and the phyllite/footwall, and to a lesser extent, into the footwall schist. Below the weathered/leached horizon there is some acid buffering capacity due to the presence of calcite. In the context of mine planning, the test work has confirmed that:

- Zones of waste material with high sulphide concentrations (pyrite and pyrrhotite) are present; in phyllite/hanging wall and phyllite/footwall contacts, and occasionally in footwall schist.
- Carbonaceous phyllites show substantial sulphide sulphur content with little neutralisation potential; for all phyllite waste, long-term ARD generation is therefore considered likely and this material should be considered potentially acid forming (PAF) material.
- Non-pyritic schist waste, laterite and topsoil are considered non-acid forming (NAF) material.

In order to limit the potential generation of acid rock drainage at Sentinel, an ARD management plan has been developed inclusive of the following control measures:

- Geochemical characterisation of waste, ore and tailings material will continue during the life of the mine.
- All surface water and groundwater around the pit, waste dumps and plant will be monitored regularly during the life of the mine.

In connection with mine planning and in terms of the management plan:

- The design of the waste dump aims to limit the generation of acid through the encapsulation of PAF with NAF material.
- All waste is characterised as either PAF or NAF material before it is hauled to the waste dump.
- All surface runoff and seepage from the waste dumps is collected in water control ponds specifically designed for that purpose.
- All surface runoff from waste dumps and in-pit sumps will be recycled back to the plant where possible, or treated by lime dosage to modify the pH levels.
- Wherever possible, clean water will be prevented from coming into contact with PAF material. This is either through diversion of 'upstream' surface water or as part of the pit dewatering programme.

16.1.3 Mining and processing schedules

With the completion of the detailed ultimate pit designs, detailed life-of-mine (LOM) production scheduling was completed using MineSched software. Scheduling assumptions, inputs and constraints included:

- The original model block size was 6 m x 6 m x 12 m, reblocked to 144 m x 144 m x 12 m.[8]
- The schedule level of detail is monthly for 2020, quarterly for 2021, and annually thereafter.
- The 2020 year schedule was sourced from a site monthly forecast plan/schedule.

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[8] the reblocking does not create additional dilution but rather accelerates the software run time

- The mining sequence in Phase 1 follows an established terrace mining arrangement respecting current short to medium term planning, whereas the subsequent pit phases are divided into multiple squares emulating wide operating terraces (>~300 m width).
- The terrace mining sequence as it progresses through the phases maintains a lowest elevation in Phase 1.
- When mining commences in a new phase, development of an IPC location in that phase is given priority.
- The scheduled sinking rate (ie, vertical mining advance) is four to five benches/year (ie, 4 x 12 m high benches).
- Vertical lag constraint: the minimum horizontal distance to be available on each bench = 144 m.
- Horizontal lag constraint: the minimum mining face width = 24 m on every bench.
- The active stockpile balance is to be minimised (ie, priority is direct feed to IPCs).
- Mineralised waste (ie, cut off grade between marginal and 0.2%Cu) is scheduled as a top-up to direct feed ore when experiencing a plant feed deficiency, otherwise it is scheduled to the long term stockpile.
- Long term stockpile reclaim is scheduled from 2029 onwards.
- The schedule aims to complete phase 2 in early 2029, such that waste rock backfilling can take place from mid-2029 onwards.

The initial production schedule was produced in Q4 2019 reflecting a LOM mining inventory of 858.3 Mt at an average grade of 0.46% TCu (after mining dilution/recovery adjustments), corresponding to the design pit extents as at the end of October 2019 (Table 15-7). The schedule inventory was subsequently updated for mining depletion to the end of December 2019.

16.1.3.1 LOM schedule

Features of the LOM mining and production schedule as listed in Table 16-9 to Table 16-11 are as follows:

- As at the end of December 2019, the Project life is 15 years (ie, to 2034).
- The total material mined amounts to 2,481.3 Mt (908.3 Mbcm), of which 847.9 Mt is ore (primary and non-primary) and 1,633.3 Mt is waste (PAF and NAF). This is on a mining diluted/ recovered basis, assuming a mining dilution factor of 103% (at a nil diluent grade) and a mining recovery factor of 97%.
- Throughout the Project life, the total ore mined to stockpiles amounts to 56.0 Mt at an average grade of 0.19%TCu, whereas that reclaimed is 84.8 Mt at an average grade of 0.21% TCu[9]. This inventory comprises 6.2 Mt of active (OTA and OTB) and 78.7 Mt of longer term (WTA and WTB) stockpiling and reclaim. Of the 78.7 Mt stockpile inventory, 28.9 Mt is the currently existing stockpile balance.

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[9] This differs from the 2015 Technical Report Mineral Reserve strategy which featured an elevated cut-off grade and no reclaim from long term mineralised waste (WTA and WTB) stockpiles.

- The maximum size of the long term stockpiles is 67.8 Mt in 2028. The proposed location of these stockpiles is within the footprint of the waste dumps on the north and south sides of the pit, proximal to the location of near-surface IPCs.
- The 62 Mtpa processing rate is essentially achieved from 2022, and is maintained until 2032.

Figure 16-25 to Figure 16-28 depict the LOM schedule graphical results.

Table 16-9 Sentinel life of mine production schedule, mined tonnages

Year	Mined Ore	Grade	Insitu Cu	Mining Dilution		Mining Diluted		Mining Recovered		Mined Waste	Total Mined		Strip
	(Mt)	(%TCu)	(kt)	(Mt)	(%TCu)	(Mt)	(%TCu)	(Mt)	(%TCu)	(Mt)	(Mt)	(Mbcm)	ratio
2020	63.2	0.50	316.0	2.0	0.00	65.2	0.48	63.2	0.48	97.6	160.8	60.2	1.5
2021	64.4	0.47	305.8	2.0	0.00	66.4	0.46	64.4	0.46	98.3	162.7	60.5	1.5
2022	61.6	0.56	345.6	1.9	0.00	63.5	0.54	61.6	0.54	120.5	182.0	68.3	2.0
2023	63.9	0.52	334.6	2.0	0.00	65.8	0.51	63.9	0.51	116.0	179.8	64.9	1.8
2024	66.1	0.47	311.9	2.0	0.00	68.2	0.46	66.1	0.46	118.6	184.8	68.6	1.8
2025	63.5	0.49	309.5	2.0	0.00	65.5	0.47	63.5	0.47	125.5	189.0	68.5	2.0
2026	69.4	0.47	325.7	2.1	0.00	71.6	0.46	69.4	0.46	120.6	190.0	72.7	1.7
2027	64.6	0.52	335.0	2.0	0.00	66.6	0.50	64.6	0.50	125.2	189.9	70.0	1.9
2028	67.6	0.47	320.6	2.1	0.00	69.6	0.46	67.6	0.46	122.3	189.8	68.2	1.8
2029	47.8	0.38	182.0	1.5	0.00	49.3	0.37	47.8	0.37	121.9	169.7	61.1	2.6
2030	40.6	0.39	160.2	1.3	0.00	41.8	0.38	40.6	0.38	120.9	161.5	58.0	3.0
2031	33.4	0.51	171.1	1.0	0.00	34.4	0.50	33.4	0.50	124.6	158.0	56.8	3.7
2032	60.0	0.38	230.5	1.9	0.00	61.9	0.37	60.0	0.37	119.6	179.6	64.7	2.0
2033	52.3	0.45	235.8	1.6	0.00	54.0	0.44	52.3	0.44	88.8	141.1	50.5	1.7
2034	29.5	0.61	180.9	0.9	0.00	30.4	0.60	29.5	0.60	13.1	42.5	15.1	0.4
LOM	**847.9**	**0.48**	**4,065.2**	**26.2**	**0.00**	**874.2**	**0.47**	**847.9**	**0.47**	**1,633.3**	**2,481.3**	**908.3**	**1.9**

Table 16-10 Sentinel life of mine production schedule, direct feed and stockpile movements

Year	Direct Feed		Stockpile Movements: Active On		Active Off		Active Balance		MW On		MW Off		MW Balance		Crusher Feed	
	(Mt)	(%TCu)	(Mt)	(%TCu)	(Mt)	(%TCu)	(Mt)	(%TCu)	(Mt)	(%TCu)	(Mt)	(%TCu)	(Mt)	(%TCu)	(Mt)	(%TCu)
Opening Balance													28.9	0.24		
2020	52.0	0.53	4.1	0.48	1.8	0.49	2.3	0.48	8.3	0.16	0.6	0.15	36.6	0.22	54.4	0.52
2021	55.9	0.50	0.1	0.24	2.1	0.48	0.3	0.35	8.4	0.16			45.0	0.21	58.0	0.50
2022	61.5	0.54			0.3	0.37	0.0		0.0	0.17			45.1	0.21	61.9	0.54
2023	62.0	0.52	0.4	0.28			0.4	0.28	1.4	0.16			46.5	0.21	62.0	0.52
2024	62.0	0.47	1.2	0.29			1.6	0.28	3.0	0.17			49.4	0.20	62.0	0.47
2025	62.0	0.48			0.0	0.28	1.6	0.28	1.5	0.18			51.0	0.20	62.0	0.48
2026	60.5	0.50			1.5	0.28	0.1	0.28	9.0	0.16			59.9	0.20	62.0	0.49
2027	62.0	0.52	0.1	0.24			0.2	0.26	2.5	0.16			62.4	0.19	62.0	0.52
2028	62.0	0.49	0.2	0.25			0.4	0.26	5.3	0.16			67.8	0.19	62.0	0.49
2029	41.0	0.40			0.4	0.26	0.0		6.8	0.16	20.6	0.22	54.0	0.18	62.0	0.34
2030	37.1	0.40					0.0		3.5	0.16	24.9	0.19	32.6	0.16	62.0	0.32
2031	33.4	0.50					0.0				28.6	0.16	4.0	0.16	62.0	0.34
2032	59.3	0.38					0.0				2.7	0.16	1.3	0.15	62.0	0.37
2033	52.3	0.44													53.7	0.43
2034	28.9	0.61													28.9	0.61
LOM (diluted)	**792.0**	**0.48**	**6.2**	**0.41**	**6.2**	**0.41**			**49.8**	**0.16**	**77.3**	**0.19**			**876.8**	**0.46**

Table 16-11 Sentinel life of mine production schedule, crusher feed and recovered copper

Year	Direct Feed		Reclaim Feed		Crusher feed		Insitu Cu	Rec'd Cu	Recovery	Concentrate
	(Mt)	(%TCu)	(Mt)	(%TCu)	(Mt)	(%TCu)	(kt)	(kt)	(%)	(kt)
2020	52.0	0.54	2.3	0.42	54.4	0.52	283.0	245.1	86.6%	942.6
2021	55.9	0.52	2.1	0.50	58.0	0.50	292.4	259.2	88.6%	996.8
2022	61.5	0.56	0.3	0.38	61.9	0.54	336.0	302.4	90.0%	1,163.2
2023	62.0	0.53	0.0	0.00	62.0	0.52	320.7	292.4	91.2%	1,124.6
2024	62.0	0.49	0.0	0.00	62.0	0.47	293.9	261.2	88.9%	1,004.5
2025	62.0	0.49	0.0	0.29	62.0	0.48	297.3	268.7	90.4%	1,033.3
2026	60.5	0.51	1.5	0.29	62.0	0.49	305.6	277.3	90.7%	1,066.5
2027	62.0	0.53	0.0	0.00	62.0	0.52	320.4	291.5	91.0%	1,121.0
2028	62.0	0.50	0.0	0.00	62.0	0.49	301.5	272.6	90.4%	1,048.3
2029	41.0	0.42	21.0	0.22	62.0	0.34	211.1	182.8	86.6%	703.2
2030	37.1	0.42	24.9	0.20	62.0	0.32	197.8	173.0	87.5%	665.6
2031	33.4	0.51	28.6	0.16	62.0	0.34	211.3	191.1	90.4%	734.9
2032	59.3	0.39	2.7	0.16	62.0	0.37	227.6	205.2	90.2%	789.4
2033	52.3	0.45	1.3	0.16	53.7	0.43	230.6	210.2	91.1%	808.3
2034	28.9	0.62	0.0	0.00	28.9	0.61	175.3	160.0	91.3%	615.4
LOM (diluted)	**792.0**	**0.50**	**84.8**	**0.21**	**876.8**	**0.46**	**4,004.6**	**3,592.6**	**89.7%**	**13,817.6**

Figure 16-25 Sentinel life of mine schedule – annual material movement tonnes

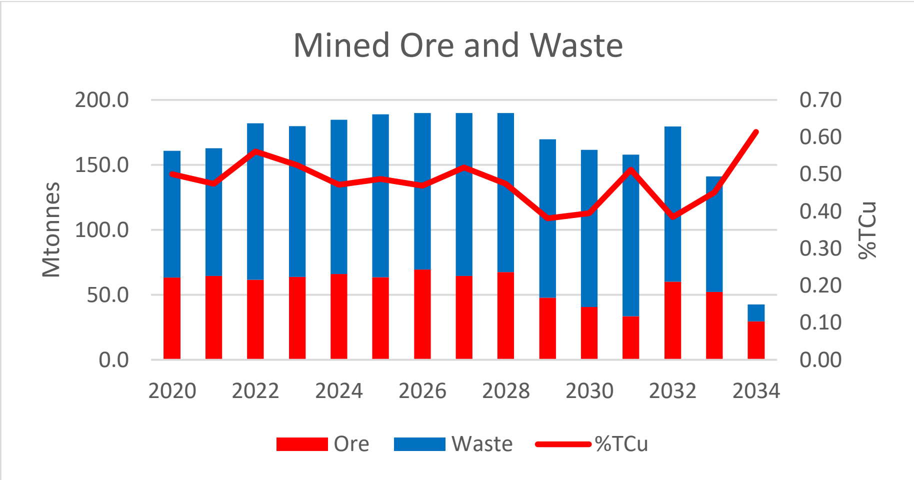


Figure 16-26 Sentinel life of mine schedule – annual plant feed tonnes

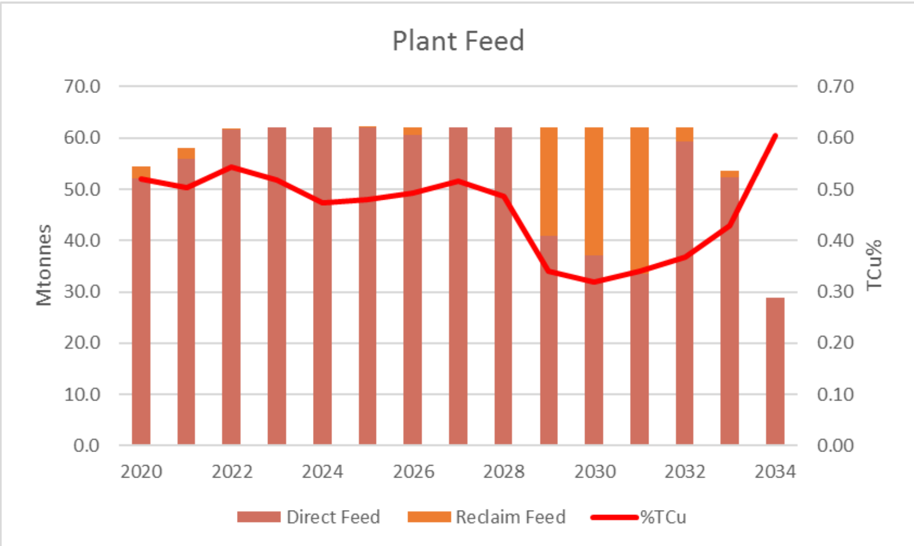


Figure 16-27 Sentinel life of mine schedule – annual copper metal production

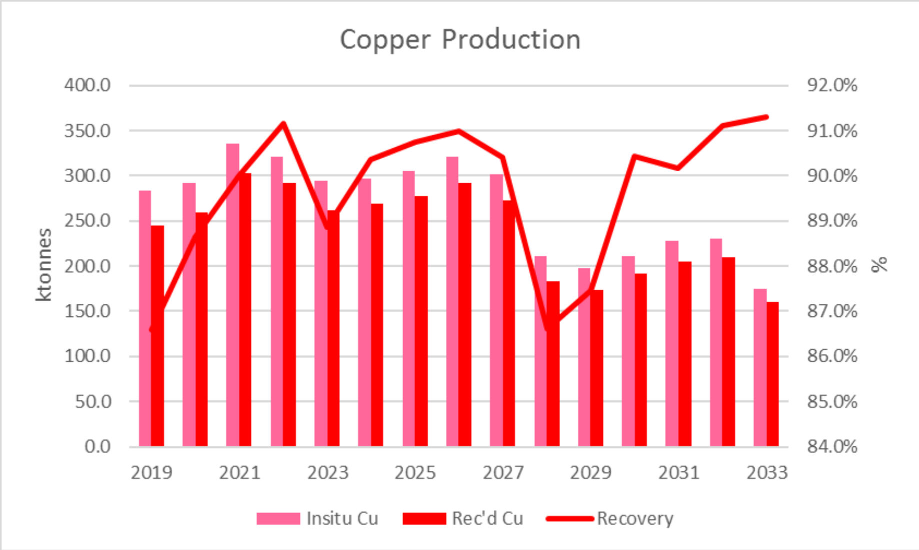
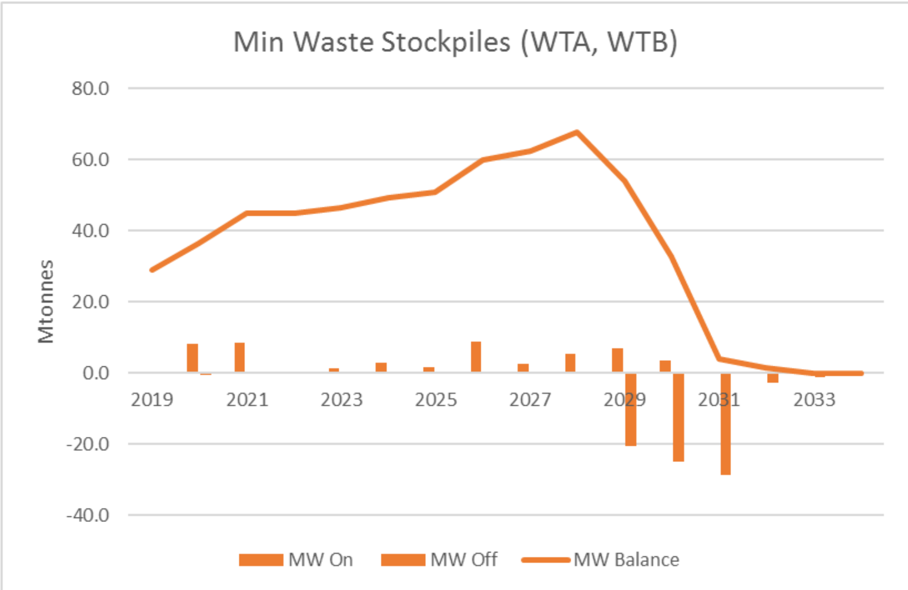


Figure 16-28 Sentinel life of mine schedule – annual stockpile balance tonnes



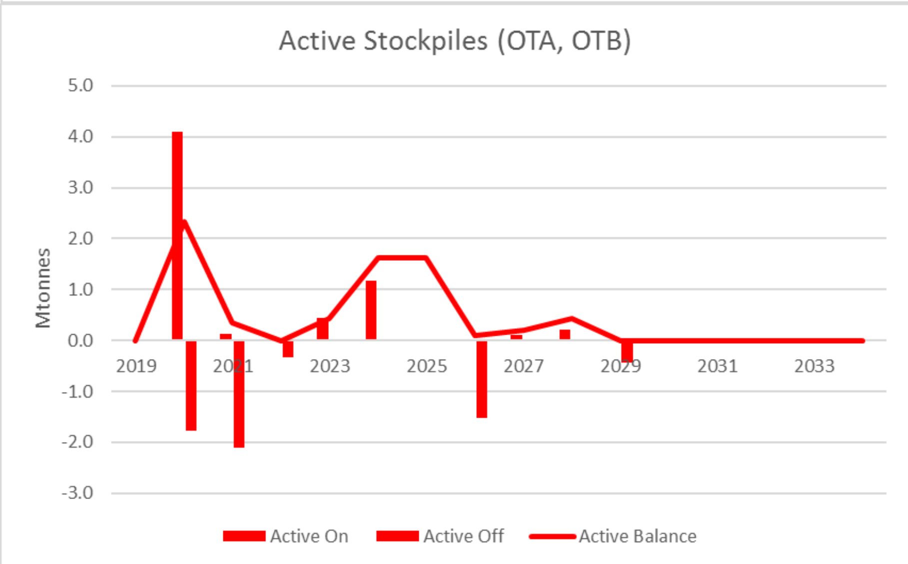


Figure 16-29 shows a comparison between the life of mine production schedule reported in 2015 and that listed in Table 16-11. To note are:

- the pit phases and mining areas differ spatially between the 2015 and 2020 versions of the schedule
- the recovered metal spike in 2022 to 2023 is due to the throughput increase; the head grades are similar
- the significant recovered metal spike in 2025 to 2028 is due to a combination of increased throughput and increased head grades

Figure 16-29 Comparison between Technical Report production schedules

16.1.3.2 Mining sequence

Figure 16-30 shows the mining sequence and combined annual ore/waste production profile.

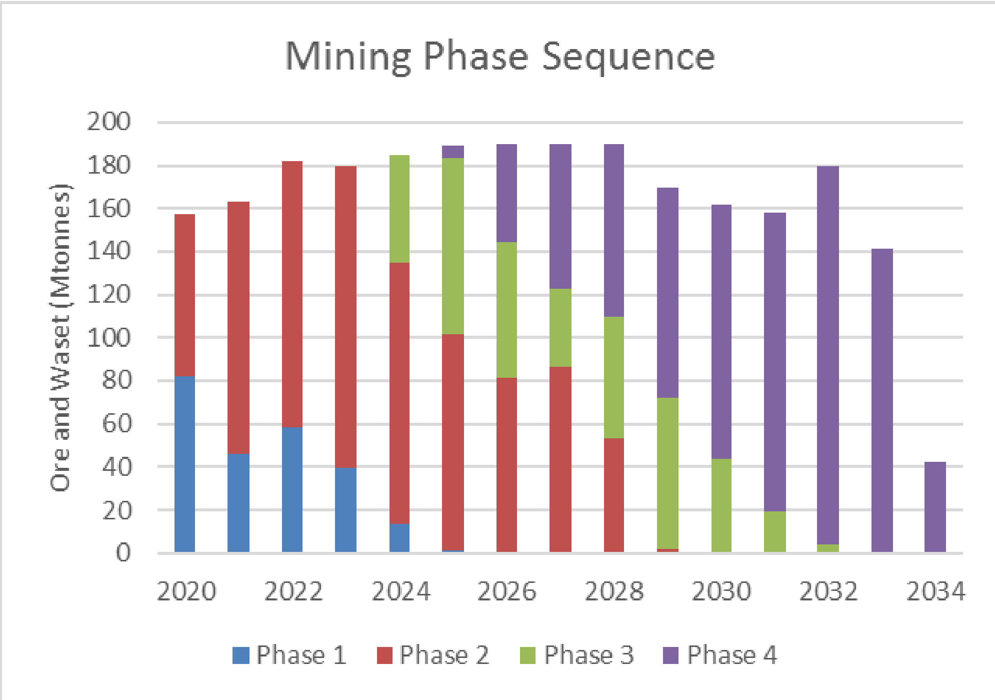
16.1.3.3 IPC relocations

Table 16-12 shows the proposed IPC operating and relocation timeframes, relative to the inpit positions shown in Figure 16-15. The sequence has been devised assuming that there is at least six months of non-operating time when each crusher is moved to a new position. This timeframe assumes that conveyor belts and other infrastructure are erected ahead of, or concurrently with, the relocation/reinstallation time window. The positioning, occurrence and sequence of IPC relocations will continue to be reviewed and optimised following this Technical Report.

Table 16-12 Sentinel life of mine production schedule, IPC relocations

Year	Total Feed (Mt)	In pit crusher relocations 1a	2a	3a	4a	1b	2b	3b	4b	1c	1d	2d	4d	Total crush (Mt)
2020	**54.4**	18.2	18.2	18.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**54.4**
2021	**58.0**	18.0	18.0	17.0	5.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**58.0**
2022	**61.9**	18.0	18.0	6.2	19.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**61.9**
2023	**62.0**	18.0	0.0	10.0	25.0	0.0	9.0	0.0	0.0	0.0	0.0	0.0	0.0	**62.0**
2024	**62.0**	0.0	0.0	10.1	25.0	9.0	17.9	0.0	0.0	0.0	0.0	0.0	0.0	**62.0**
2025	**62.0**	0.0	0.0	0.0	25.6	5.6	5.7	25.1	0.0	0.0	0.0	0.0	0.0	**62.0**
2026	**62.0**	0.0	0.0	0.0	23.7	7.6	9.2	21.5	0.0	0.0	0.0	0.0	0.0	**62.0**
2027	**62.0**	0.0	0.0	0.0	0.0	18.3	18.3	13.7	11.6	0.0	0.0	0.0	0.0	**62.0**
2028	**62.0**	0.0	0.0	0.0	0.0	7.9	7.9	21.7	24.5	0.0	0.0	0.0	0.0	**62.0**
2029	**62.0**	0.0	0.0	0.0	0.0	0.0	21.0	24.3	8.8	7.9	0.0	0.0	0.0	**62.0**
2030	**62.0**	0.0	0.0	0.0	0.0	0.0	24.9	24.2	6.4	6.4	0.0	0.0	0.0	**62.0**
2031	**62.0**	0.0	0.0	0.0	0.0	0.0	17.6	25.0	9.7	9.7	0.0	0.0	0.0	**62.0**
2032	**61.0**	0.0	0.0	0.0	0.0	0.0	0.0	3.2	0.0	13.0	0.0	19.8	25.0	**61.0**
2033	**52.3**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	18.0	18.0	16.3	**52.3**
2034	**29.5**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	18.0	11.5	0.0	**29.5**

Figure 16-30 Sentinel life of mine mining sequence



16.1.3.4 Waste dumping schedule

Table 16-13 lists the scheduled waste dump volumes by mining phase source and by dumping destination. The scheduled dumping volumes account for the footprint that is occupied by the ore stockpiles shown in Figure 16-19.

Table 16-13 Sentinel scheduled waste haulage and dumping (PAF and NAF) volumes

Hauling from		2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	TOTAL
STAGE1																	
PAF	Mbcm	13.5	8.0	7.8	6.7	1.6	0.2	-	-	-	-	-	-	-	-	-	**37.7**
NAF	Mbcm	5.8	0.8	0.3	0.3	0.1	-	-	-	-	-	-	-	-	-	-	**7.2**
subtotal	**Mbcm**	**19.3**	**8.7**	**8.0**	**7.1**	**1.7**	**0.2**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**45.0**
STAGE1_2WALL																	
PAF	Mbcm	-	-	-	-	1.4	3.5	2.9	2.4	1.5	-	-	-	-	-	-	**11.7**
NAF	Mbcm	-	-	-	-	0.3	0.0	0.1	0.0	0.0	-	-	-	-	-	-	**0.5**
subtotal	**Mbcm**	**-**	**-**	**-**	**-**	**1.7**	**3.5**	**3.0**	**2.5**	**1.6**	**-**	**-**	**-**	**-**	**-**	**-**	**12.2**
STAGE2																	
PAF	Mbcm	12.4	10.2	6.7	11.5	7.0	3.7	4.5	7.7	4.0	0.3	-	-	-	-	-	**68.1**
NAF	Mbcm	5.3	18.3	31.4	23.4	18.4	16.1	5.9	3.6	1.8	0.1	-	-	-	-	-	**124.2**
subtotal	**Mbcm**	**17.6**	**28.5**	**38.1**	**34.9**	**25.4**	**19.7**	**10.4**	**11.4**	**5.8**	**0.4**	**-**	**-**	**-**	**-**	**-**	**192.3**
STAGE3																	
PAF	Mbcm	-	-	-	-	4.2	10.3	11.2	6.2	10.2	12.8	3.9	1.5	0.4	-	-	**60.9**
NAF	Mbcm	-	-	-	-	11.7	9.7	2.6	1.4	0.5	1.0	1.7	0.3	0.2	-	-	**29.0**
subtotal	**Mbcm**	**-**	**-**	**-**	**-**	**15.9**	**19.9**	**13.8**	**7.6**	**10.7**	**13.8**	**5.7**	**1.8**	**0.6**	**-**	**-**	**89.9**
STAGE4																	
PAF	Mbcm	-	-	-	-	-	0.0	0.6	2.5	5.0	7.9	4.0	6.4	3.7	0.7	-	**30.9**
NAF	Mbcm	-	-	-	-	-	2.5	20.2	22.8	21.0	22.0	33.9	36.7	12.3	0.4	-	**171.8**
subtotal	**Mbcm**	**-**	**-**	**-**	**-**	**-**	**2.5**	**20.8**	**25.3**	**26.0**	**29.9**	**37.9**	**43.1**	**16.0**	**1.2**	**-**	**202.7**
STAGE2_4WALL																	
PAF	Mbcm	-	-	-	-	-	-	-	-	-	-	-	-	13.0	13.0	3.3	**29.4**
NAF	Mbcm	-	-	-	-	-	-	-	-	-	-	-	-	13.7	17.7	1.5	**32.9**
subtotal	**Mbcm**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**26.7**	**30.7**	**4.9**	**62.3**
Grand Total																	
PAF	**Mbcm**	**25.9**	**18.2**	**14.4**	**18.2**	**14.3**	**17.6**	**19.2**	**18.9**	**20.8**	**21.0**	**7.9**	**7.9**	**17.2**	**13.8**	**3.3**	**238.7**
NAF	**Mbcm**	**11.1**	**19.1**	**31.7**	**23.7**	**30.5**	**28.2**	**28.8**	**27.9**	**23.3**	**23.1**	**35.6**	**36.9**	**26.2**	**18.1**	**1.5**	**365.6**
subtotal	**Mbcm**	**37.0**	**37.2**	**46.1**	**42.0**	**44.7**	**45.9**	**48.0**	**46.8**	**44.1**	**44.1**	**43.5**	**44.9**	**43.4**	**31.9**	**4.9**	**604.3**

Dumping to		2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	TOTAL
North																	
PAF	Mlcm	24.4	11.7	18.1	23.7	18.6	22.9	24.2	21.3	20.5	17.1	5.1	2.0	0.6	-	-	**210.2**
NAF	Mlcm	12.6	22.7	40.9	30.8	39.6	33.4	11.1	21.3	29.8	28.8	44.0	9.5	-	-	-	**324.5**
subtotal	**Mlcm**	**37.0**	**34.4**	**59.0**	**54.6**	**58.1**	**56.3**	**35.3**	**42.6**	**50.3**	**45.8**	**49.2**	**11.5**	**0.6**	**-**	**-**	**534.7**
South																	
PAF	Mlcm	9.3	12.0	0.6	-	-	0.0	0.8	3.3	6.5	3.8	-	-	-	-	-	**36.2**
NAF	Mlcm	1.8	2.1	0.3	-	-	3.3	26.3	15.0	-	-	-	-	-	-	-	**48.7**
subtotal	**Mlcm**	**11.0**	**14.0**	**0.9**	**-**	**-**	**3.3**	**27.1**	**18.3**	**6.5**	**3.8**	**-**	**-**	**-**	**-**	**-**	**84.9**
Backfill																	
PAF	Mlcm	-	-	-	-	-	-	-	-	-	6.4	5.2	8.3	21.8	17.9	4.3	**63.9**
NAF	Mlcm	-	-	-	-	-	-	-	-	0.5	1.3	2.3	38.5	34.1	23.5	2.0	**102.1**
subtotal	**Mlcm**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**0.5**	**7.7**	**7.5**	**46.8**	**55.8**	**41.4**	**6.3**	**166.0**
Grand Total																	
PAF	**Mlcm**	**33.7**	**23.6**	**18.8**	**23.7**	**18.6**	**22.9**	**25.0**	**24.6**	**27.0**	**27.3**	**10.3**	**10.3**	**22.3**	**17.9**	**4.3**	**310.4**
NAF	**Mlcm**	**14.4**	**24.8**	**41.2**	**30.8**	**39.6**	**36.7**	**37.4**	**36.2**	**30.3**	**30.0**	**46.3**	**48.0**	**34.1**	**23.5**	**2.0**	**475.3**
subtotal	**Mlcm**	**48.0**	**48.4**	**59.9**	**54.6**	**58.1**	**59.6**	**62.4**	**60.9**	**57.3**	**57.3**	**56.6**	**58.3**	**56.4**	**41.4**	**6.3**	**785.7**

16.1.4 Primary mining equipment

Table 16-14 lists the original primary equipment purchased for mining start-up, and the inclusion of more recently purchased equipment complementing the current fleet. In addition to these items, there is an ancillary fleet of dozers, graders and water carts. A mining contractor provides a small equipment fleet for site construction activities (eg, roads, dams, diversion channels) and also for pre-stripping and in-pit batter trimming work.

Table 16-14 Sentinel primary mining equipment numbers

Equipment	Make/Model	Original	Current
Drills	Caterpillar MD6640	7	7
	Epiroc PitViper 271E		3
	Sandvik D25K		4
	Furukawa D120	6	5
Shovels	Caterpillar 7495HR	3	3
	Komatsu PC5500	2	2
Excavators	Komatsu PC5500	1	1
	Liebherr 9250	2	2
	Liebherr 9100		2
Loaders	Komatsu WA1200	1	1
	P&H LE2350		2
Trucks	Liebherr T284		8
	Komatsu 960E	15	17
	Komatsu 860E	15	15
	Komatsu HD1500	10	10

Table 16-15, Table 16-16 and Table 16-17 show the typical productivity planned from individual primary fleet items within representative weekly and three monthly periods of the wet and dry seasons. Table 16-18, Table 16-17 and Table 16-20 list the combined fleet productivity, on an annual basis, for the production drills, excavating/loading equipment and haul trucks, respectively. These annual productivity estimates are based on the average individual equipment tonnes/hour productivity rates for each fleet item.

Table 16-15 Productivity of individual production drills

Production drills	Units	Aug18 Weekly Plan	Nov18 Weekly Plan	Feb19 Weekly Plan	May19 Weekly Plan	Average
Drill productivity						
MD 6640	lin. m/hr	32	29	29	37	32
DRC20 & DK25	lin. m/hr	31	30	27	29	29
PV271	lin. m/hr	30	31	26	29	29
Average		**31**	**30**	**27**	**32**	**30**
Drill pattern intensity						
MD 6640	t/ lin m	87	99	76	158	105
DRC20 & DK25	t/ lin m	50	49	62	41	51
PV271	t/ lin m	75	68	65	71	70
Average		**71**	**72**	**68**	**90**	**75**
Production						
MD 6640	Mt/annum	92.0	94.9	72.8	193.3	113.2
DRC20 & DK25	Mt/annum	49.4	46.9	53.4	37.5	46.8
PV271	Mt/annum	35.0	32.8	26.3	32.0	31.5
Total		**176.4**	**174.5**	**152.5**	**262.8**	**191.6**

Table 16-16 Productivity of individual excavating/loading equipment items

Excavating/loading equipment	Jul2018 3MPP (t/hr)	Dec18 3MPP (t/hr)	Mar19 3MPP (t/hr)	Jun19 3MPP (t/hr)	Average 3MPP (t/hr)
Shovels					
SH01 -CAT 7495	5,050	3,844	4,138	4,138	**4,293**
SH02 -CAT 7495	5,050	3,844	4,138	4,138	**4,293**
SH03 -CAT 7495	5,050	3,844	4,138	4,138	**4,293**
SH04 -Komatsu PC5500	2,750	2,100	2,400	2,809	**2,515**
SH05 -Komatsu PC5500	2,750	2,100	2,400	2,452	**2,426**
Excavators					
EX01 -Komatsu PC5500	2,750	2,100	2,400	2,045	**2,324**
EX92 -Liebherr 9250	1,639	1,300	1,550	1,573	**1,516**
EX94 -Liebherr 9250	1,639	1,300	1,550	1,333	**1,456**
Loaders					
FD02 -Komatsu WA1200				0	**0**
FD09 -LeTourneau 2350	930	1,680	1,782	1,728	**1,530**
FD10 -LeTourneau 2350	930	1,680	1,782	1,728	**1,530**

Table 16-17 Productivity of individual haul trucks

Haul trucks	Jul2018 3MPP (t/hr)	Dec18 3MPP (t/hr)	Mar19 3MPP (t/hr)	Jun19 3MPP (t/hr)	Average 3MPP (t/hr)
Liebherr T284	775	620	650	650	**674**
Komatsu 960E	700	580	610	610	**625**
Komatsu 860E	525	450	501	501	**494**
Komatsu HD1500	300	300	312	312	**306**

Table 16-18 Productivity of combined drill fleet

Production drills	No of Units	Availability	Utilisation	Monthly Run Hours	Productivity (m/hr)	Productivity (t/LM drilled)	Production (Mt/mth)	Production (Mt/annum)
MD 6640	6	85%	75%	459	32	105	9.2	110.2
DRC20 & DK25	6	82%	75%	443	29	51	3.9	47.1
PV271	3	80%	75%	432	29	70	2.6	31.5
Total							**15.7**	**188.7**

Table 16-19 Productivity of combined excavating/loading fleet

Excavating/loading equipment	No of Units	Availability	Utilisation	Monthly Run Hours	Productivity (t/hr)	Production (Mt/mth)	Production (Mt/annum)
Shovels							
SH01 -CAT 7495	1	85%	85%	520	4,293	2.2	26.8
SH02 -CAT 7495	1	85%	85%	520	4,293	2.2	26.8
SH03 -CAT 7495	1	85%	85%	520	4,293	2.2	26.8
SH04 -Komatsu PC5500	1	85%	85%	520	2,515	1.3	15.7
SH05 -Komatsu PC5500	1	85%	85%	520	2,426	1.3	15.1
Excavators							
EX01 -Komatsu PC5500	1	85%	85%	520	2,324	1.2	14.5
EX92 -Liebherr 9250	1	85%	85%	520	1,516	0.8	9.5
EX94 -Liebherr 9250	1	85%	85%	520	1,456	0.8	9.1
Loaders							
FD02 -Komatsu WA1200	1	85%	85%	520	0	0.0	0.0
FD09 -LeTourneau 2350	1	85%	50%	306	1,530	0.5	5.6
FD10 -LeTourneau 2350	1	85%	50%	306	1,530	0.5	5.6
Contractor							
Kascco	1					1.4	16.2
Total						**14.3**	**171.7**

Table 16-20 Productivity of combined haul truck fleet

Haul trucks	No of Units	Availability	Utilisation	Monthly Run Hours	Productivity (t/hr)	Production (Mt/mth)	Production (Mt/annum)
Liebherr T284	8	85%	80%	490	674	2.6	31.7
Komatsu 960E	17	80%	80%	461	625	4.9	58.8
Komatsu 860E	15	82%	80%	472	494	3.5	42.0
Komatsu HD1500	10	80%	77%	444	306	1.4	16.3
Total						**12.4**	**148.7**

These tables indicate that in order to achieve the increased mining production rates associated with the increased plant feed rate (ie, an increase in total mining movement capacity to about 180 Mtpa from 2022, and 190 Mtpa from 2026), the following additional primary equipment items would be required:

- excavating/loading – an additional Cat 7495HR shovel, and
- trucks – an additional +6 Leibherr T284 trucks, to adequately serve the additional shovel

The capital cost itemisation in relation to mining equipment is listed in Item 21.1.1. A reduced number of trucks is included amongst the capital purchases for reasons of improved trucking efficiency going forward, when shorter hauls under trolley assist will be apparent in Phase 2.

Whilst current production drilling capacity appears to be adequate, an additional PV271 drill could be justified on the basis of ensuring sufficient blasted stocks for the additional shovel.

16.2 Enterprise

16.2.1 Mining details

The Enterprise Pit is to be mined in a series of phases, using mid-scale open pit mining methods. Initial mining activity will focus on exposing and removing oxide material and transitional ore from the near surface region at the south eastern side of the design pit. Initial pre-stripping in this area of the pit had commenced at the time of the 2015 Technical Report (FQM, May 2015) but was suspended in late 2015. As part of a site development work programme in 2019, remedial work was completed on exposed saprolitic slopes in the initially pre-stripped (Phase 1) starter pit. Existing surface drainage channels and water control dams were also remediated. Some 11.5 Mbcm of waste material remains to be pre-stripped when site development resumes.

16.2.1.1 Mine site layout

The current layout and appearance of the Enterprise site is shown in Figure 16-31 and Figure 16-32, respectively.

The main features shown are:

- the existing starter pit development in waste, providing material for construction of haul roads and surface water diversion and containment structures
- the location of the ultimate pit limits
- the conceptual ultimate waste dump footprint, and the location of the proposed starter waste dump
- existing surface water control dams (Dam 1 and Dam 2)
- a proposed main diversion/containment dam (Dam 4 with spillway) located between the pit and the Chisola Dam
- site workshops and ROM pad
- an existing settlement pond and dam (Dam 3), with a spillway and proposed flocculant dosage plant positioned alongside
- surface drains and bunds
- haulage road to the processing plant

16.2.1.2 Mining method and operations

The Enterprise Pit will be mined using conventional drill and blast, excavator and truck mining methods, with 5 m to 10 m high ore and waste benches.

Up to the time of the 2015 Technical Report (FQM, May 2015) development work had focussed on waste stripping in order to provide suitable materials for surface water control/diversion dams, causeways to the north side of the river (for eventual waste haulage) and surface haulroads. This work was carried out by a contractor.

Under full scale operations, ore and waste will be loaded using 250 tonne diesel-hydraulic excavators, loading into 180 tonne capacity diesel trucks. These trucks will haul waste to the external dump located to the north and east of the pit, and to a surface ROM pad on the south side of the pit (Figure 16-31).

Figure 16-31 Enterprise Pit at June 2019

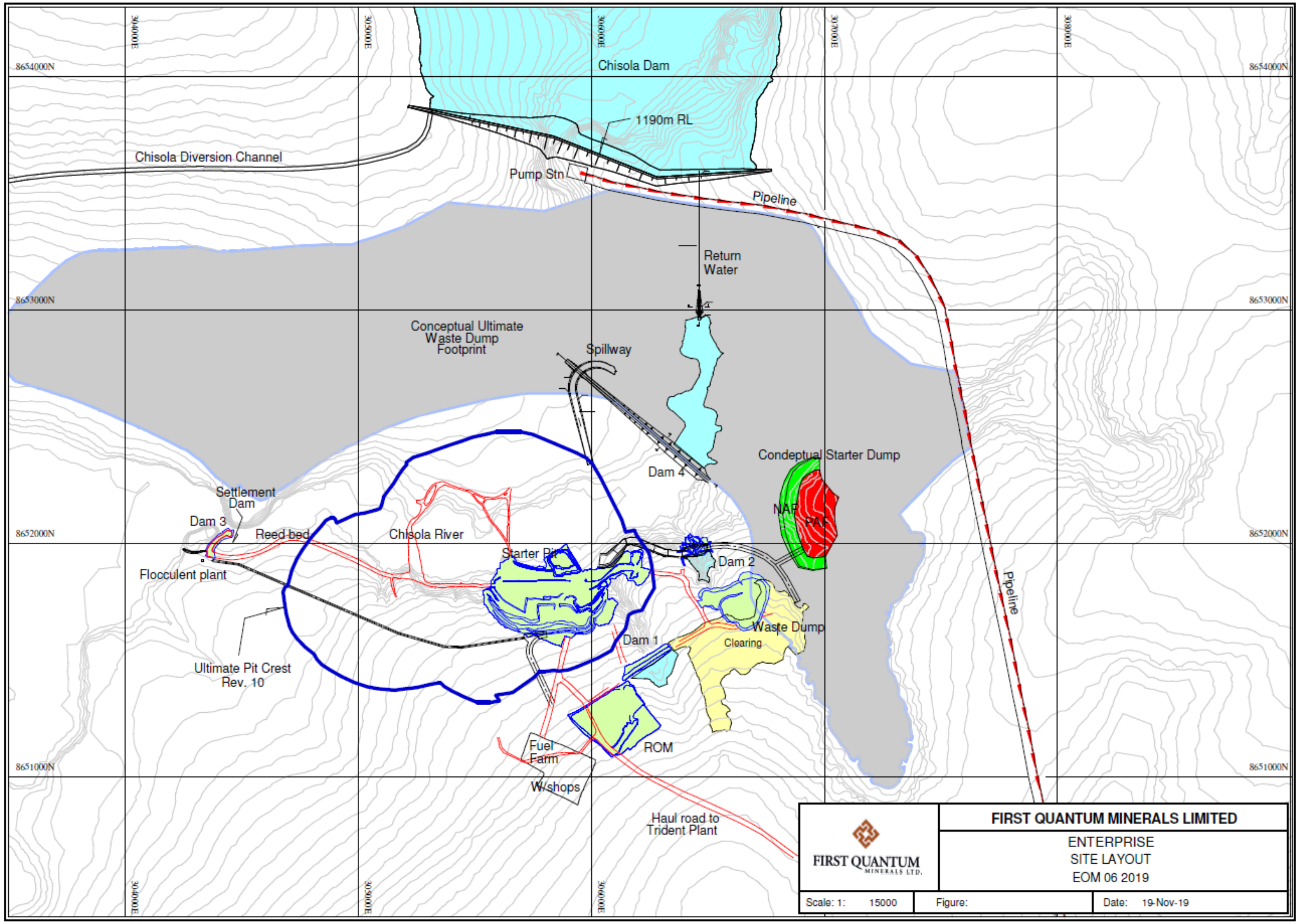


Figure 16-32 View looking north west over the Enterprise starter pit, October 2019

Due to the scale and geometry of the Enterprise Pit, there are no plans for the introduction of in-pit primary crushing and conveying, nor trolley-assisted haulage. In consideration of the perceived reactivity of the nickeliferous ore, mined ore delivered to a surface ROM pad will be reclaimed and road hauled (15 km) directly to the processing plant (ie, there is to be essentially no stockpiling at the mine).

16.2.1.3 Waste dump optimisation and design

The proposed waste dump ultimate footprint is shown in Figure 16-31 and accommodates the total waste volume relating to the pit design phases and production schedule. DumpSolver (2013) produced the optimised life-of-mine waste dump design in May 2013.

Figure 16-31 shows that clearing had commenced (in 2015) on the eastern side of the pit to enable short haulage across from the starter pit. The current pit designs feature haul roads which exit the pit crest on the south and western sides, heading northwards across to the northern dump footprint. A haulage causeway across the river has already been built to service mining to the Phase 3 limits.

16.2.1.4 Mining dilution and recovery

Consistent with the pit optimisation, an allowance for unplanned dilution has been included in the production schedules. This is an additional 5% of ore tonnes at a nil diluent grade. A mining recovery factor of 95% has also been adopted.

16.2.2 Mine planning

The following information relates to the detail that needed to be considered for designing surface layouts and practical mining phases around the optimal pit shell outlines.

16.2.2.1 Mine design parameters

Basic mine design parameters relating to pit slope design and widths of haul roads are described in Item 15, as follows:

- haul road width = 25 m (suitable for 180 t capacity trucks)
- maximum haul ramp gradient = 1: 10

16.2.2.2 Geotechnical engineering

XStract (2015) produced an open pit geotechnical design report for Enterprise as the culmination of a detailed drilling investigation and laboratory testing programme carried out between 2013 and 2014. The geotechnical data collated from this work programme was used by XStract to identify rock mass domains, define rock mass strength properties, undertake pit slope stability assessments and to develop pit slope design criteria, as set out in Table 16-21 (XStract, 2014).

Table 16-21 Pit slope design criteria, XStract 2015

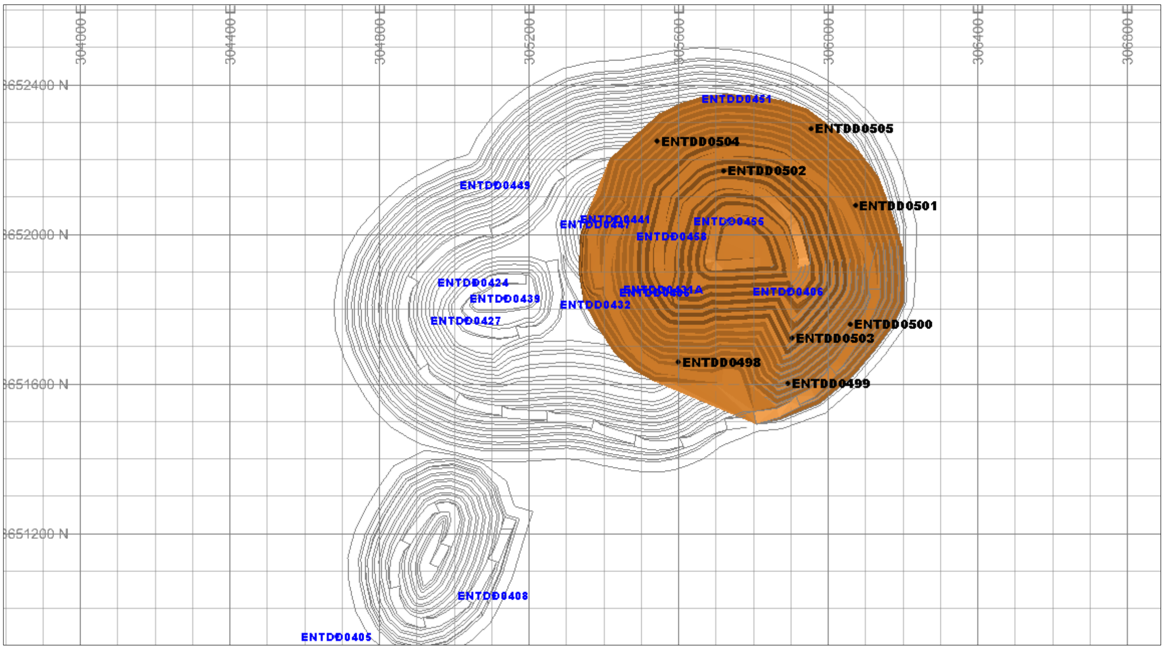
Domain	Depth Range (m)	Wall orientation	Bench height (m)	Batter angle (degrees)	Min. berm width (m)	IRA (C to C)
Saprolite & weathered (All rock units)	0 to 50	All	10	45	6	33.7
Meta Carbonate	> 50	NE wall (0° to 090°) All others	20 20	75 70	10 10	52.5 49.2
Breccia	> 50	All	20	75	10	52.5
Quartzite	> 50	NE wall (0° to 120°) All others	20 20	70 75	10 10	49.2 52.5
Siltstone	> 50	NE wall (0° to 090°) All others	20 20	70 65	10 10	49.2 46.0

Note: the inter-ramp angle (IRA) does not incorporate geotechnical berms

In addition to the criteria listed in Table 16-21, XStract recommended a 20 m wide berm at the base of the weathered slope and then at 80 m vertical intervals thereafter.

A second geotechnical drilling programme was completed at Enterprise between November 2018 and April 2019. Eight PQ diameter holes were drilled for a total of 1,700 m of diamond core. The location of these eight holes is shown in Figure 16-33 (labelled in black font), relative to the location of the holes previously drilled (labelled in blue font). The ultimate pit outline and the pit shaded brown in this figure are the final and the Phase 3 design pits, respectively, as illustrated in the 2015 Technical Report (FQM, May 2015).

Figure 16-33 Location plan showing Enterprise geotechnical drill hole collars



Mechanical test work and geotechnical analyses were completed following the geotechnical drilling programme, and an internal memorandum issued on the findings and the ensuing revised pit slope design criteria. (KML, October 2019). Fundamental to the analyses was the reinterpretation of geotechnical design domains as shown in Figure 16-34, Figure 16-35 and Figure 16-36. The updated design criteria relative to these geotechnical domains is listed in Table 16-22. For the ultimate pit limits to the extents shown in Figure 16-34, the 2015 design parameters remain applicable.

Figure 16-34 Enterprise phase 1 pit geotechnical domains

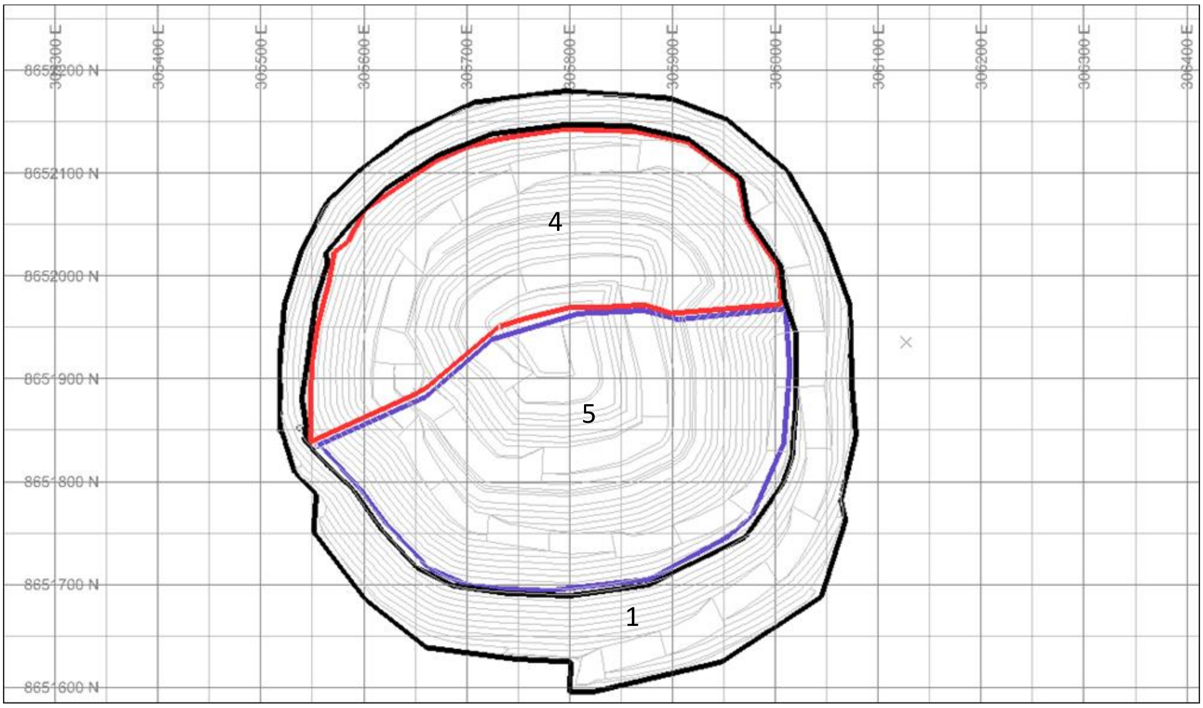


Figure 16-35 Enterprise phase 2 pit geotechnical domains

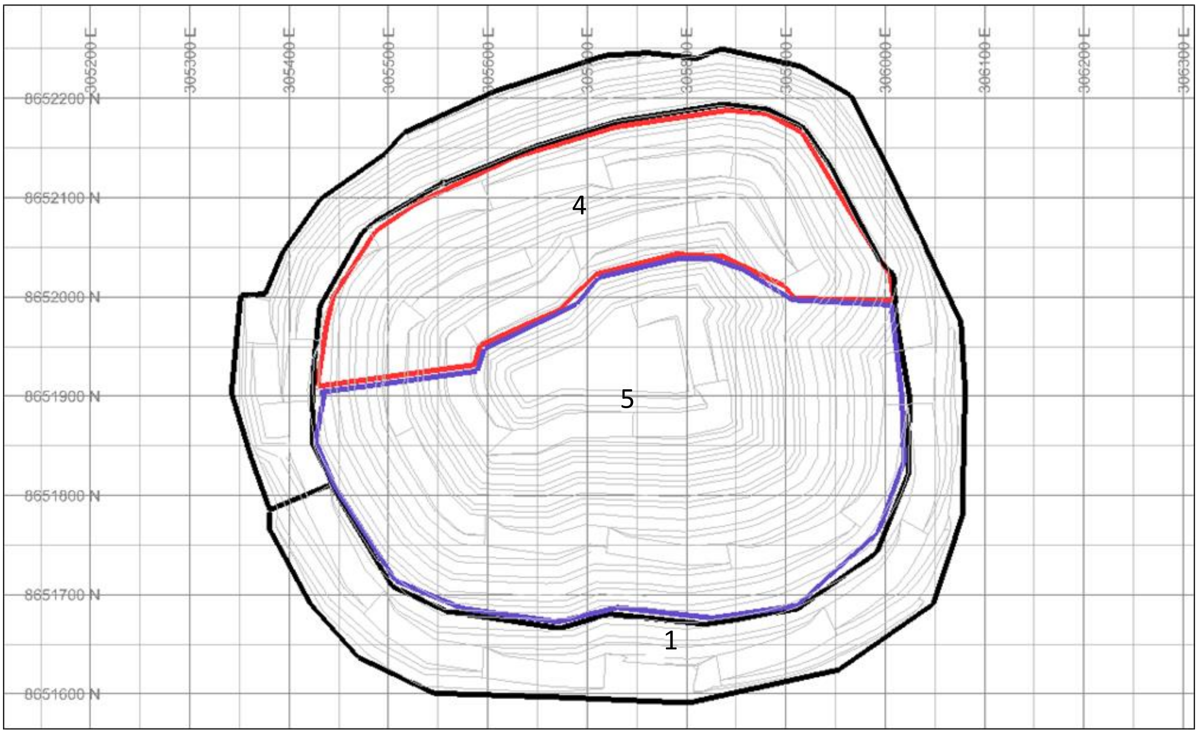


Figure 16-36 Enterprise phase 3 pit geotechnical domains

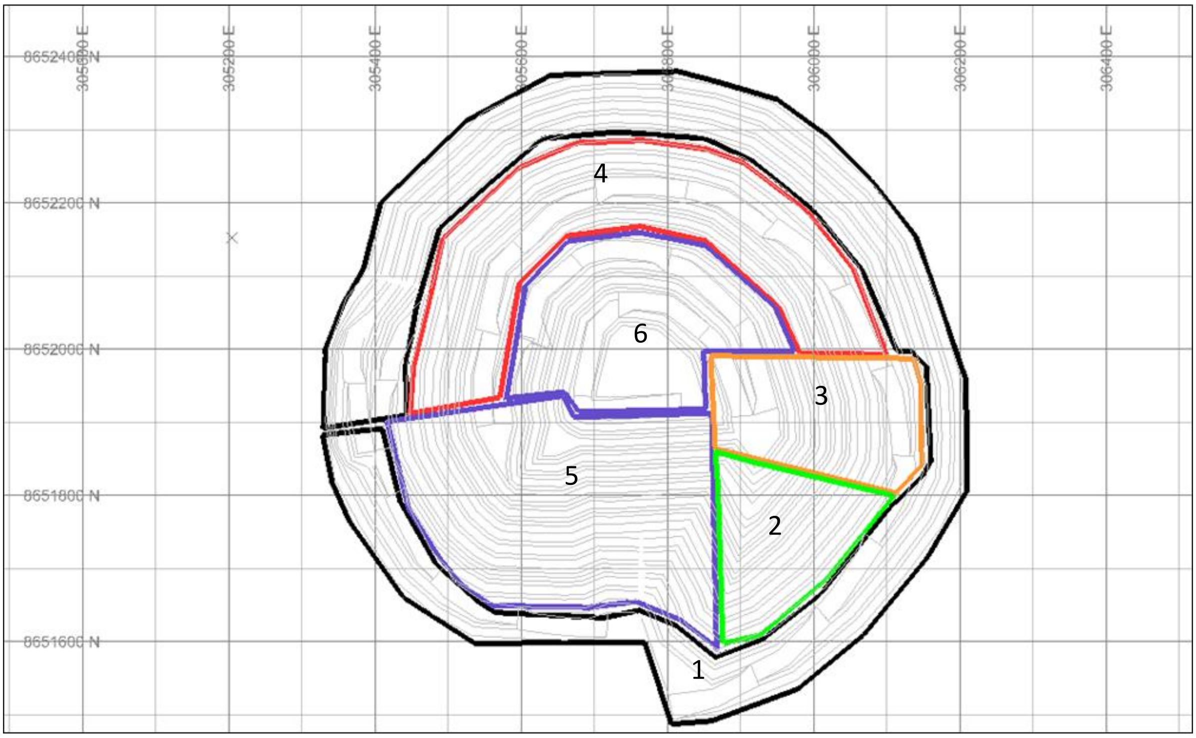


Table 16-22 Updated pit slope design criteria, 2019

Domain	Depth Range (m)	Wall orientation	Bench height (m)	Batter angle (degrees)	Min. berm width (m)	IRA (C to C)
Domain 1 Saprolite south wall	Surface to 1126 mRL	All	6	30	6	20.1
Domain 1 Saprolite north wall	Surface to 1120 mRL	All	6	30	6	20.1
Domain 5 Saprock	1126 mRL to 1084 mRL	All	10	55	6	37.6
Domain 2, 3 & 4 Saprock	1126 mRL to 1110 mRL	All	10	55	6	37.6
Domain 4	1110 mRL to	NE (0^o to 90^o)	20	75	10	52.5
Fresh rock	1016 mRL	Elsewhere (90^o to 360^o)	20	70	10	49.2
Domain 2 Fresh rock	1110 mRL to pit floor	All	20	65	10	46.0
Domain 3 Fresh rock	1110 mRL to pit floor	All	20	75	10	52.5
Domain 5 & 6	1084 mRL to	NE (0^o to 90^o)	20	70	10	49.2
Fresh rock	pit floor	Elsewhere (90^o to 360^o)	20	65	10	46.0

Note: the inter-ramp angle (IRA) does not incorporate geotechnical berms

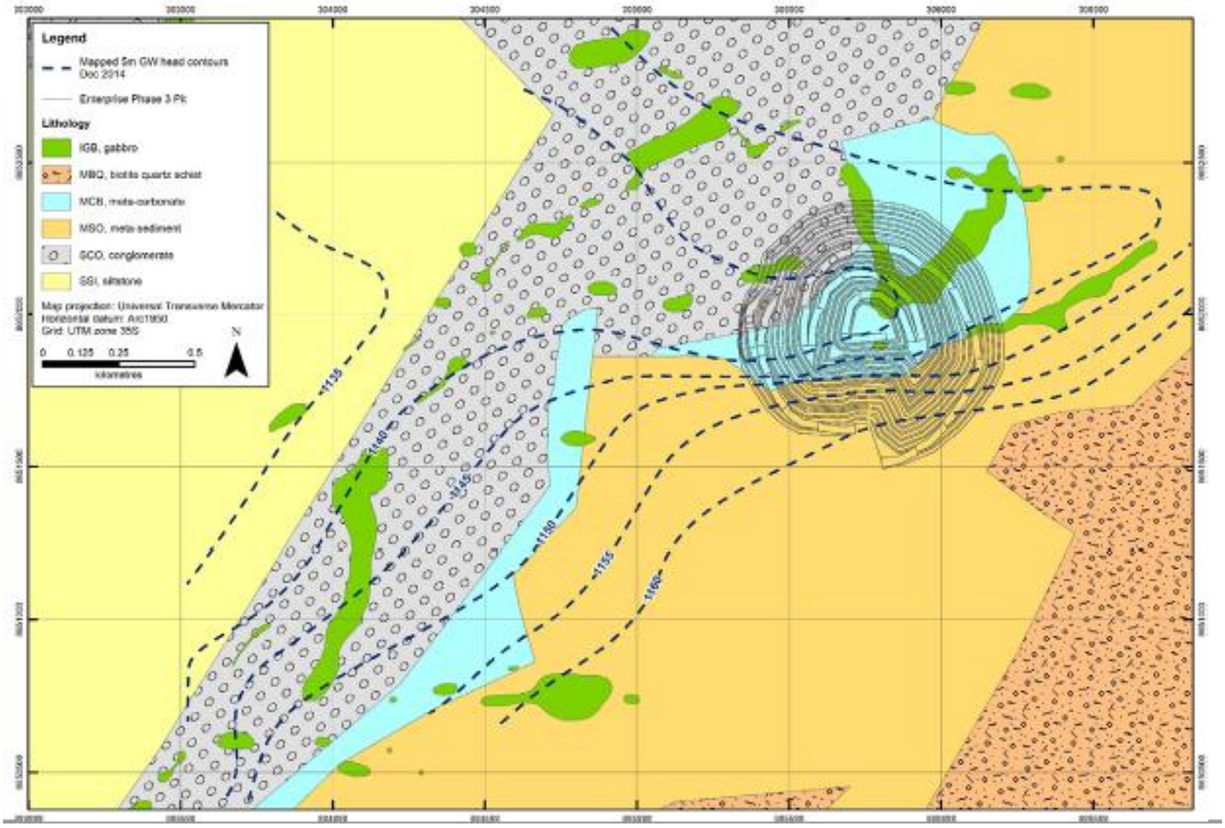
16.2.2.3 Hydrogeology

An initial field investigation and preliminary hydrogeological assessment was carried out by Schlumberger Water Services (Australia) Pty Ltd (SWS, 2015) between November 2014 and April 2015. Prior to that, the groundwater conditions surrounding the Enterprise deposit were undefined and poorly understood.

During the SWS site investigations, an initial survey of groundwater levels was carried out across the Enterprise site. Despite a limited number of exploration boreholes being accessible for depth gauging, a number of general observations could be made (SWS, 2015):

- groundwater flow generally follows the fall of ground towards the Chisola River and westwards into a large and expansive dambo
- steeper hydraulic gradients are apparent within the metasedimentary sequence and at the geological contact between meta sediments and the meta carbonates, on the south side of the deposit (Figure 16-37)
- flatter hydraulic gradients are apparent with the meta carbonate and conglomerate units on the northern side of the deposit, suggesting a higher permeability relative to adjacent lithologies (Figure 16-37)

Figure 16-37 Enterprise groundwater contours and surface geology (SWS, 2015)



At the same time as the SWS site investigation, geophysical surveys consisting of induced polarisation (IP) electrical resistivity measurements were completed by Geophysics GPR Botswana (GPR). This geophysics information was used to correlate low resistivity zones with zones of increased porosity (storage), and this together with rock quality designation (RQD) data from exploration drilling logs, allowed SWS to draw some preliminary conclusions on likely aquifer zones (SWS, 2015).

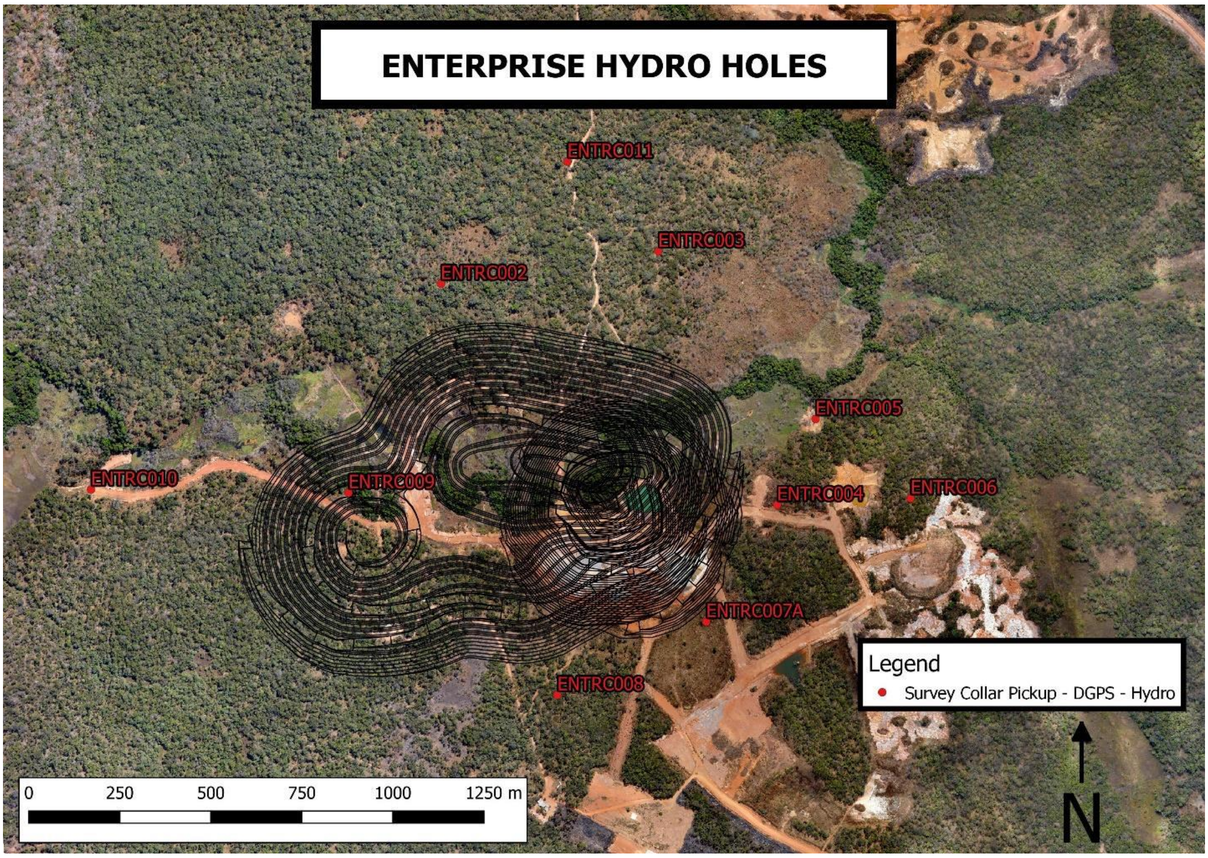
Working with the compiled investigation data, SWS identified twelve targets for pilot (ie, "probe") hole drilling. Planning for drilling of these pilot holes was underway in late 2015 when the Enterprise development work was placed on hold.

Hydrogeological work was resumed in early 2019 and a desktop study was conducted by the Hydrogeological Consultancy (SRK) in January, from which ten drilling sites were subsequently identified as listed in Table 16-23 and shown in Figure 16-38. The boreholes were drilled between January and October 2019 to a target depth of 250 mbgl (metres below ground level) and all of them had six inch diameter steel casing installed. The casing was perforated at pre-determined depths to intersect water-bearing features.

Table 16-23 Enterprise pilot boreholes drilled 2019

Borehole ID ID	Northing (m)	Easting (m)	RL (m)	Target	Water level (mbgl)
ENTRC002	8,652,523	305,313	1,159	Possible fault	18.91
ENTRC003	8,652,613	305,910	1,154	MSO	12.50
ENTRC004	8,651,915	306,237	1,171	Possible fault	16.64
ENTRC005	8,652,151	306,342	1,143	Possible fault	2.29
ENTRC006	8,651,933	306,603	1,177	MSO and fault	5.04
ENTRC007A	8,651,593	306,042	1,179	Possible fault	10.86
ENTRC008	8,651,392	305,633	1,179	Possible fault	11.55
ENTRC009	8,651,948	305,059	1,142	Possible fault	6.13
ENTRC010	8,651,959	304,353	1,144	Low resistivity	3.43
ENTRC011	8,652,859	305,661	1,161	Nested obs point	16.79

Figure 16-38 Enterprise pilot boreholes drilled 2019



A geotechnical diamond drilling programme was completed at Enterprise during the period November 2018 and April 2019. These drillholes were used for packer testing in order to provide data for further groundwater modelling. The testing horizons were mostly within and around the initial starter pit extents. Injection tests were conducted at levels as deep as possible to gather hydrogeological data from below the initial mining floor. VWPs were subsequently installed in several of the packer tested drillholes. The rationale behind these installations was to gather baseline pressure data pre-mining, and then to track the changes over time as mining progresses.

The ten completed pilot boreholes were to be aquifer/pump tested from late Q4 2019. Test data when available will be used as input to the existing groundwater model, in addition to the data from packer testing which is to be used for calibration purposes.

As an update to the observations made by SWS in 2015, the following hydrogeological interpretations have been made following the more recent investigations:

- as evidenced in the exposed starter pit walls, and in the drill core and piezometer bores, groundwater is present in the upper weathered regolith and at depths within the fresh rock domains
- dissolution of carbonates (limestone and dolomitic rocks) in the deposit has resulted in the formation of karstic features, the individual magnitude of which are unknown
- standing water level depths as measured in the pilot boreholes, range between 2.3 m and 18.9 m below surface (Table 16-23)

16.2.2.4 Hydrology

The Chisola River flows over the top of the Enterprise deposit and has been dammed upstream (the Chisola Dam). Between the dam and the north eastern limits of the ultimate pit, there are tributary streams that flow into the Chisola and these need to be contained to prevent inundation of the pit.

Figure 16-31 shows the location of several water management and containment dams (Dams 1, 2 and 4) designed for this purpose. Dams 1 and 2 already exist, whilst Dam 4 is to be constructed in the 2020 dry season. The design of these dams is addressed in an engineering report by consultant Ian Miller of ZMCK Consulting Engineers Ltd (Miller, 2015).

Aside from these containment structures, surface water flow needs to be managed from the perspective of potential sediment laden run-off from the Enterprise site flowing into the downstream Chisola River and dambo system, and ultimately into the Kabombo River. In this regard, Figure 16-30 shows the location of a downstream rock dam (Dam 3) and associated spillway, sedimentation pond and proposed flocculant plant. These facilities are intended to trap sediments and pollutants and allow them to settle out before discharge to the environment at levels which meet required water quality standards for turbidity and total dissolved solids. The design of these facilities is addressed in an engineering report by consultant Peter Townshend of Amanziflow Projects (Townshend, 2014).

The settlement pond with Dam 3, as designed, is large enough to suit the initial pit phases and provide sufficient silty water storage. As the pit phases develop towards the west, however, and this storage volume is lost, Dam 3 can be supplemented downstream with a larger settlement pond and dam. This dam would be constructed in a broad arc across the upper dambo.

16.2.2.5 ARD management

As was done for Sentinel originally, a programme of static test work to confirm the potential for ARD generation was carried out in 2014. A kinetic testing facility was established on the Enterprise site in Q3 2019.

In connection with mine planning and in terms of acid rock drainage management:

- the few ore samples tested (shale and carbonaceous shale) show a mostly long-term potential acid formation (PAF)
- waste rocks (including samples of fluidised breccia, mafic intrusion, meta carbonates and lower siltstones) are mostly non-acid forming (NAF) and are likely to be less of an ARD potential issue, than at Sentinel
- in the presence of significant volumes of basal sandstones and siltstones, and with widespread talc and carbonate alteration, there is higher potential buffering effect from waste rocks
- mostly neutral to alkaline pH values were observed in the sulphuric leach test results, caused by the readily available net neutralisation potential of most static test samples
- a much higher waste strip ratio than at Sentinel provides a relatively easier proposition for encapsulation and containment of any PAF waste

Figure 16-31 shows an initial planned arrangement for separating PAF and NAF waste from initial mine development, allowing conditions to be observed before committing to a formal redesign of the dump should it be necessary.

16.2.3 Mining and processing schedules

With the completion of the detailed ultimate and phased pit designs, detailed life-of-mine (LOM) production scheduling was completed using MineSched software. Scheduling assumptions, constraints and inputs included:

- The original model block size was 10 m x 10 m x 5 m, reblocked to 25 m x 25 m x 5 m, and to 50 m x 50 m x 5 m (in the pre-strip phase)
- The schedule level of detail is monthly for 2020, quarterly for 2021 and 2022, and annually thereafter.
- The first two mining phases are scheduled to commence from the existing prestrip area (Phase 1), without producing any ore, but producing sufficient waste rock for the completion of site infrastructure including Dam 4.
- After the pre-strip, mining progresses to the north and then ultimately to the west.
- The vertical and horizontal lag constraint = 200 m.
- The scheduled sinking rate (ie, vertical mining advance) is six to seven benches/year (ie 6 x 6 m high benches).
- No surface stockpiling is allowed; all ore mined is to be hauled to a surface ROM pad and then reclaimed directly for road haulage to the processing plant.

16.2.3.1 LOM Schedule

Features of the LOM mining and production schedule as listed in Table 16-24 are as follows:

- Beyond 2021, the production schedule timeframe is notional; the commencement time for nickel ore processing is yet to be determined.
- The total material mined is 330.5 Mt (124.3 Mbcm), of which 34.9 Mt is ore (primary and non-primary) and 295.7 Mt is waste (PAF and NAF). This is on a mining diluted/recovered basis,

assuming a mining dilution factor of 105% (at a nil diluent grade) and a mining recovery factor of 95%.

- There are no stockpiles; all ore mined is hauled to surface for ROM reclaim into road transport across to the processing plant.
- Due to the stockpiling limitation, the geometry of the pit, and combined with sensible vertical advance rate restrictions, it is not possible to achieve a consistent 4 Mtpa processing rate.

Figure 16-39 to Figure 16-41 depict the LOM schedule graphical results.

Table 16-24 Enterprise life of mine production schedule

Year	Mined Ore (Mt)	Grade (%Ni)	Crusher Feed (Mt)	Grade (%Ni)	Waste (Mt)	Total Mined (Mt)	Total Mined (Mbcm)	Strip Ratio (t:t)
Prestrip	0.01	0.20	0.0	0.00	11.9	11.9	4.8	
Prestrip/Yr1	1.0	0.50	1.0	0.50	15.8	16.8	7.2	16.5
Year 2	3.3	0.90	3.3	0.90	24.7	28.0	11.7	7.5
Year 3	2.3	1.09	2.3	1.09	38.9	41.2	15.9	17.0
Year 4	3.2	1.44	3.2	1.44	43.2	46.4	17.0	13.6
Year 5	3.9	1.29	3.9	1.29	42.3	46.2	17.0	11.0
Year 6	3.1	1.03	3.1	1.03	39.6	42.7	15.4	12.9
Year 7	4.0	0.71	4.0	0.71	26.8	30.8	10.8	6.7
Year 8	4.0	1.13	4.0	1.13	17.6	21.6	7.5	4.4
Year 9	4.0	1.02	4.0	1.02	14.7	18.6	6.5	3.7
Year 10	4.0	0.70	4.0	0.70	8.7	12.6	4.4	2.2
Year 11	2.1	0.76	2.1	0.76	2.4	4.6	1.6	1.1
LOM (dil'd)	**34.7**	**0.99**	**34.7**	**0.99**	**286.7**	**321.4**	**119.7**	**8.3**

Figure 16-39 Enterprise life of mine schedule – annual material movement tonnes

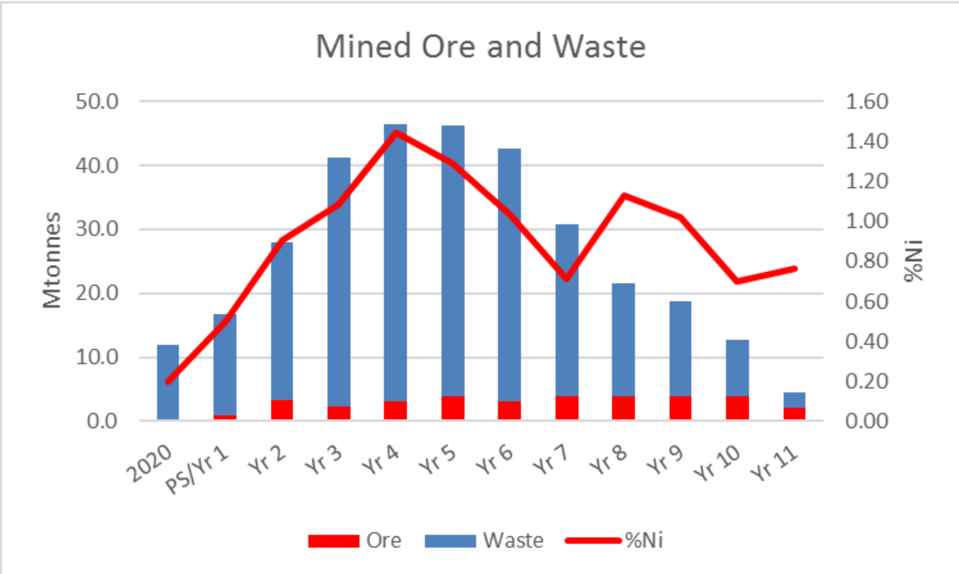


Figure 16-40 Enterprise life of mine schedule – annual plant feed tonnes

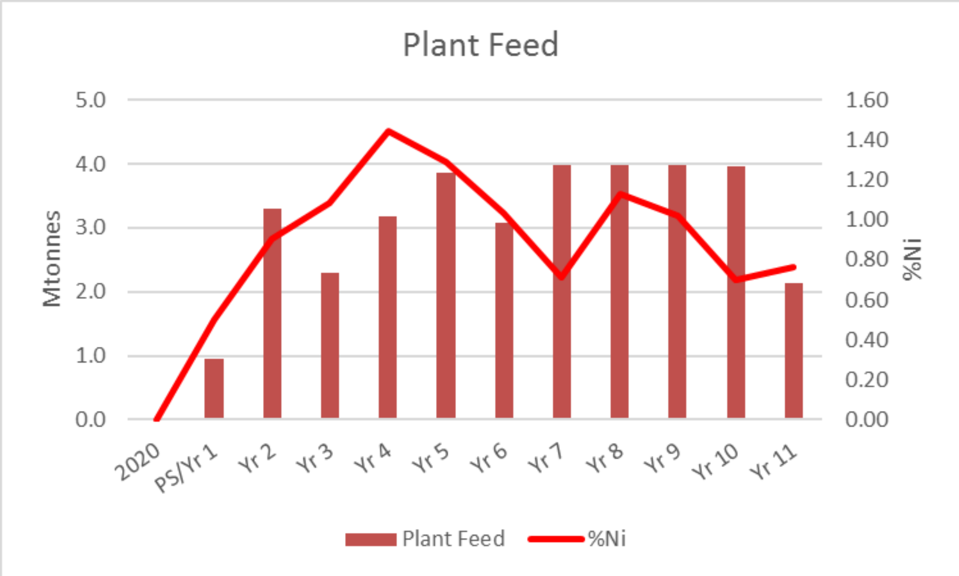
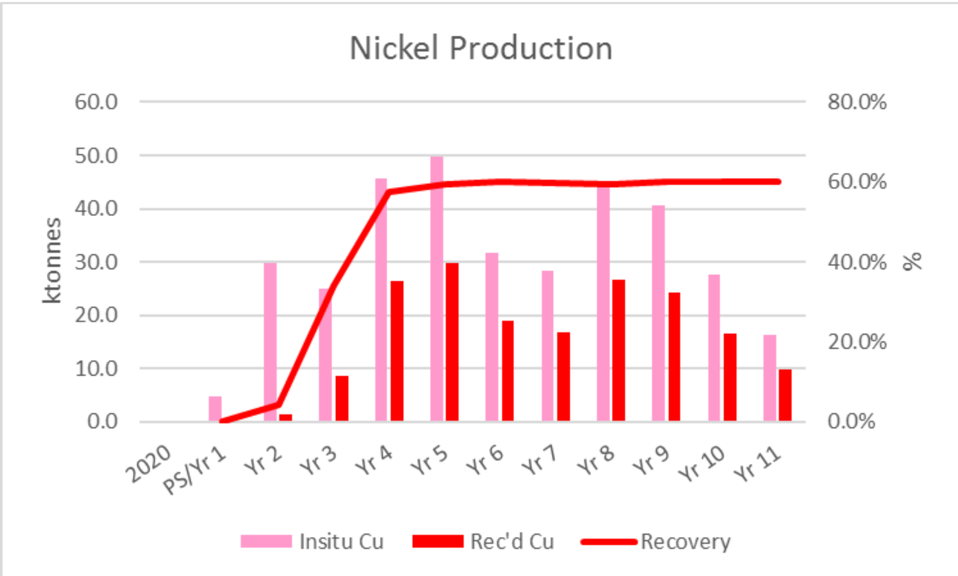


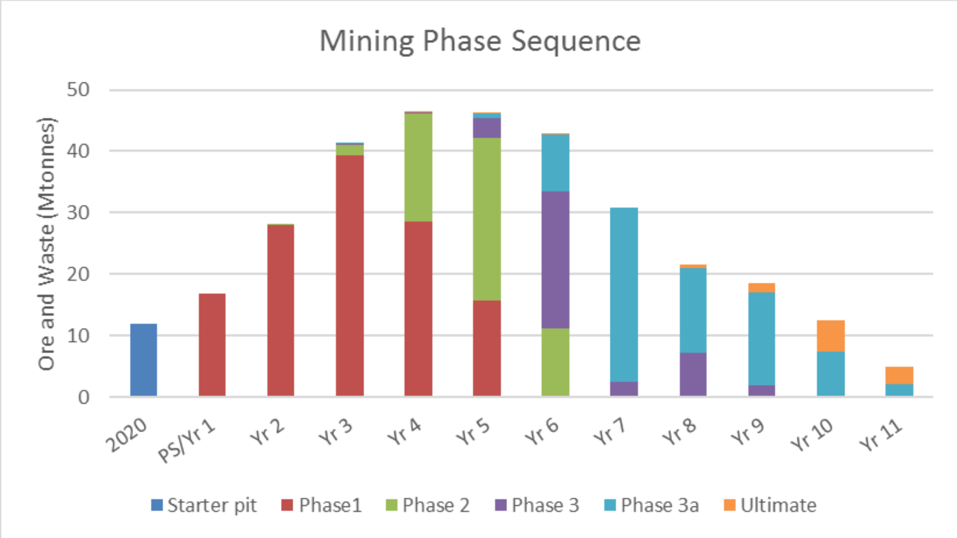
Figure 16-41 Enterprise life of mine schedule – annual copper metal production



16.2.3.2 Mining sequence

Figure 16-42 shows the mining sequence and combined annual ore/waste production profile.

Figure 16-42 Enterprise life of mine mining sequence



16.2.3.3 Waste dumping schedule

Table 16-25 shows the scheduled waste dump volumes (in Mlcm) by mining phase source.

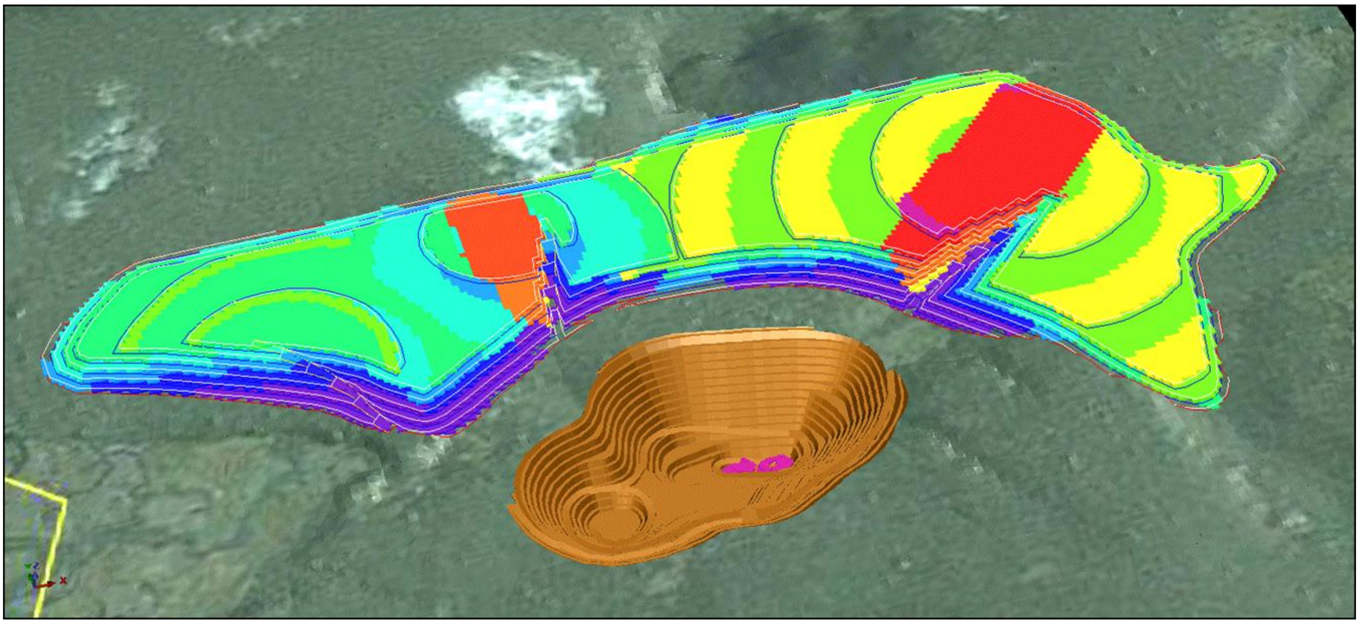
Figure 16-43 shows the proposed ultimate waste dump as optimised and designed by DumpSolver in 2013. With some modifications and design extensions on the south eastern side, as shown in Figure 16-31, there is adequate volume within the design footprint to accommodate the mined waste.

Without the NAF and PAF sequencing that Sentinel requires, the dump development progression is essentially one of progressing and expanding upwards and outwards from the three nodes divided by the apache passes. The initial focus is on developing the southeastern node flanking Dam 4 and the Chisola River channel.

Table 16-25 Enterprise waste dumping (PAF and NAF) volumes

				Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	TOTAL
starter_pit_waste														
PAF	Mlcm	2.56	-	-	-	-	-	-	-	-	-	-	-	**2.56**
NAF	Mlcm	2.96	-	-	-	-	-	-	-	-	-	-	-	**2.96**
phase1_pit_waste														
PAF	Mlcm	-	2.12	3.96	3.88	0.96	0.98	-	-	-	-	-	-	**11.91**
NAF	Mlcm	-	5.64	7.87	12.57	10.26	5.11	-	-	-	-	-	-	**41.45**
phase2_pit_waste														
PAF	Mlcm	-	-	0.01	0.16	0.59	1.66	0.65	-	-	-	-	-	**3.07**
NAF	Mlcm	-	-	0.02	0.53	6.32	8.60	4.09	-	-	-	-	-	**19.56**
phase3_pit_waste														
PAF	Mlcm	-	-	-	0.02	0.01	0.19	1.29	0.01	0.02	0.00	-	-	**1.54**
NAF	Mlcm	-	-	-	0.05	0.07	1.00	8.16	0.87	2.62	0.67	-	-	**13.45**
phase5_pit_waste														
PAF	Mlcm	-	-	-	0.01	-	0.05	0.53	0.12	0.04	0.01	-	-	**0.74**
NAF	Mlcm	-	-	-	0.02	-	0.25	3.32	10.38	5.03	5.48	2.52	0.41	**27.41**
pitshell_final_waste														
PAF	Mlcm	-	-	-	-	0.00	0.01	0.00	-	0.00	0.00	-	-	**0.01**
NAF	Mlcm	-	-	-	-	0.00	0.04	0.02	-	0.25	0.54	1.71	0.56	**3.12**
pitshell_final_SW_waste														
PAF	Mlcm	-	-	-	-	-	-	0.02	0.00	0.00	0.00	-	-	**0.02**
NAF	Mlcm	-	-	-	-	-	-	0.11	0.12	0.10	0.08	-	-	**0.41**
Grand Total														
PAF	**Mlcm**	**2.56**	**2.12**	**3.97**	**4.07**	**1.56**	**2.89**	**2.49**	**0.13**	**0.06**	**0.01**	**-**	**-**	**19.85**
NAF	**Mlcm**	**2.96**	**5.64**	**7.89**	**13.17**	**16.65**	**14.99**	**15.70**	**11.37**	**8.00**	**6.78**	**4.23**	**0.97**	**108.35**

Figure 16-43 Enterprise Pit, ultimate waste dump design (DumpSolver, 2013)



16.2.4 Primary mining equipment

A primary mining equipment fleet comprising 250 tonne diesel-hydraulic excavators, loading into 180 tonne capacity diesel trucks is envisaged. At this time, no plans have been developed for the supply or procurement of dedicated equipment for Enterprise; a suitably scaled mining fleet would be contracted.

ITEM 17 RECOVERY METHODS

The following information for Sentinel is taken from an internal FQM Project Engineering Study dated 2011 (FQM, 2011). The information has been updated and summarised by Andrew Briggs (QP).

The following information for Enterprise is reproduced in part from the 2012 Technical Report (DumpSolver, December 2012). The information has been updated and summarised by Andrew Briggs (QP).

17.1 Sentinel

Figure 17-1 shows a view of the Sentinel processing facilities.

Figure 17-1 View looking west over the processing plant

17.1.1 Key processing design parameters

Sentinel copper ores have an average copper grade of approximately 0.50% Cu. At a processing rate of 55 Mtpa and with 90% recovery, the annual production of copper (in concentrate) is approximately 247,500 tpa Cu.

The overall production data for the last three years of operations are presented in Table 17-1.

Table 17-1 Sentinel production data

Year	Ore (Mt)	Cu grade (%Cu)	Metal insitu (kt)	Metal rec'd (kt)	Recovery (%)	Con. grade (%Cu)
2017	42.1	0.53	221.7	190.9	86.1%	24.3
2018	48.7	0.50	245.0	223.7	91.3%	25.4
2019	49.4	0.50	245.9	220.1	89.5%	26.6
TOTAL	**140.2**	**0.51**	**712.7**	**634.7**	**89.1%**	**25.5**

Mining operations continue to ramp up, as the pit is now being developed into an adjacent Phase 2, towards the east from the initial starter pit and Phase 1. The reduction in recoveries for 2019 relative to 2018 was the result of treating more transitional ores from near surface mineralization in the eastern phase.

17.1.2 Process plant description

The concentrator circuit comprises:

- in-pit crushing of run of mine (ROM) ore
- conveying of primary crushed ore to secondary crushing and to ore stockpiles
- partial secondary crushing of primary crushed ore prior to stockpiles
- SAG and ball milling of crushed ore, with size classification by hydrocyclones. A grind size of 80% passing 212 µm is targeted
- flash Flotation for fast floating coarse chalcopyrite on ball mill cyclone underflow
- pebble crushing on scats generated from the SAG mills
- rougher and scavenger flotation on cyclone overflow slurry
- discharge of scavenger tailings to tailings thickeners, with thickener underflow being pumped to a tailings storage facility (TSF)
- reclaim of decant water from the TSF for usage within the process
- upgrade of the first (high grade) rougher concentrates in a Jameson cell
- cleaner flotation of the rougher concentrates, and Jameson tails, with cleaner scavenger tails being discharged to final tailings
- final cleaner of cleaner cons and Jameson cons in column flotation cells to reduce the entrained carbon levels
- dewatering of copper concentrates by thickening and filtration, followed by bulk transportation to off-site smelters within Zambia
- reagent make-up and dosage systems to support the milling and flotation operations
- water reticulation systems
- compressed air systems to support instrumentation and for automatic valve activation

The Sentinel processing plant is described in more detail in the following commentary.

17.1.2.1 Primary crushing

Figure 17-2 shows the block flowsheet for the comminution circuit at Sentinel, whilst Figure 17-3 is a view of the two SAG and ball mills (discharge end).

The primary crushing circuit consists of three semi-mobile, independent gyratory crushers (IPCs) operating in open circuit. The crushers operate with a nominal open side setting of 165 mm. With all three crushers operating, running times average 16 hours per day, although 24 hour operation is possible. Each crusher is located in-pit, to minimise haulage distances, with crushed ore conveyed to a pit top bin from where the ore is conveyed either directly to the mill feed stockpile, or to secondary crushing. In late 2021, a fourth IPC will be installed, thereby enabling three crushers to continue in operation whilst one is being relocated to a new position deeper in the pit.

Figure 17-2 Sentinel crushing and milling circuits, block flowsheet

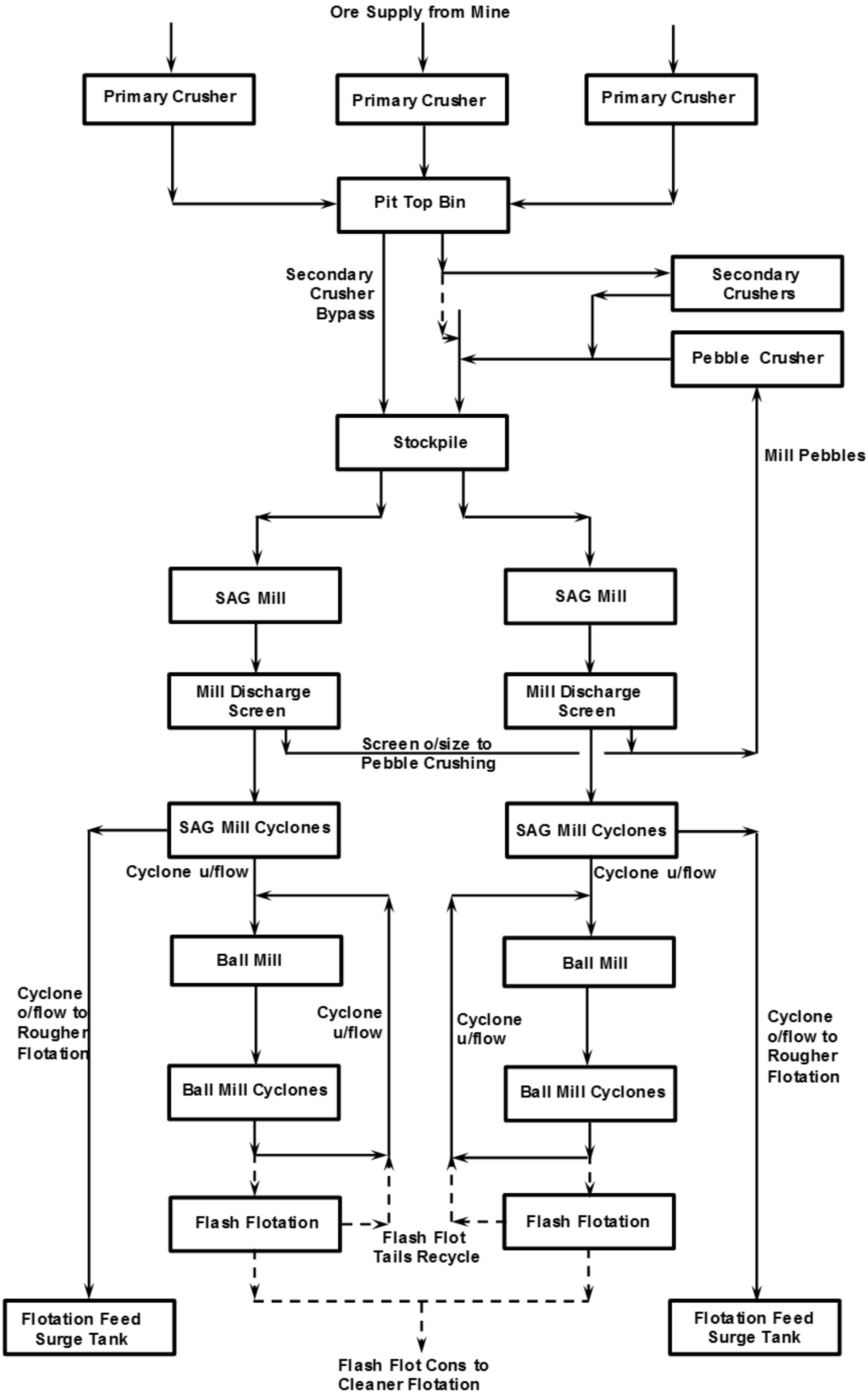


Figure 17-3 View of Sentinel milling circuit

17.1.2.2 Crushed ore stockpiling and reclaim

Two parallel shuttle conveyors are used to deliver crushed material to the stockpile and to distribute the crushed ore along the length of the stockpile. The stockpile has a live capacity of at least 12 hours, approximately 60,000 tonnes. The total capacity of the stockpile can be utilised, if required, by bull-dozing the dead load into the stockpile discharge chutes. Ore is recovered from the crushed ore stockpile by four apron feeders (per mill) located in a tunnel underneath the stockpile.

17.1.2.3 Milling, pebble crushing and flash flotation

The mills were selected on the basis of two milling trains, each comprising the largest SAG and ball mills currently proven at other operations, at the time of design. On that basis, a 28 MW SAG mill (internal diameter of 12.19 m (40 ft)) and a 22 MW ball mill (8.53 m internal diameter (28 ft)) were selected for each milling train. All the mills are equipped with gearless drives.

Each milling train is designed to grind 3,500 tph of material from a feed size of 80% passing 130 mm to a product size of 80% passing 212 µm. A pebble crusher is included in the circuit to crush pebbles ejected from the SAG mills, down to minus 12 mm before recycling them to the stockpile feed conveyor.

Each SAG mill has a dedicated cyclone cluster, with cyclone underflow being directed to the ball mill. Each ball mill has two cyclone clusters, with cyclone overflow mixing with SAG cyclone overflow and gravitating to rougher flotation. The underflow from each ball mill cyclone cluster can be directed to the ball mill feed chute, or alternatively part of the stream can be directed to a dedicated 57 m³ flash flotation cell (two per ball mill).

Flash flotation has been included in the circuit because of the presence of fast floating coarse chalcopyrite. The flash float concentrate is pumped to the third stage of cleaner flotation

17.1.2.4 Rougher/cleaner flotation

Figure 17-4and Figure 17-5 provide a block flowsheet and view of the flotation circuit, respectively.

Figure 17-4 Sentinel flotation crcuit, block flowsheet

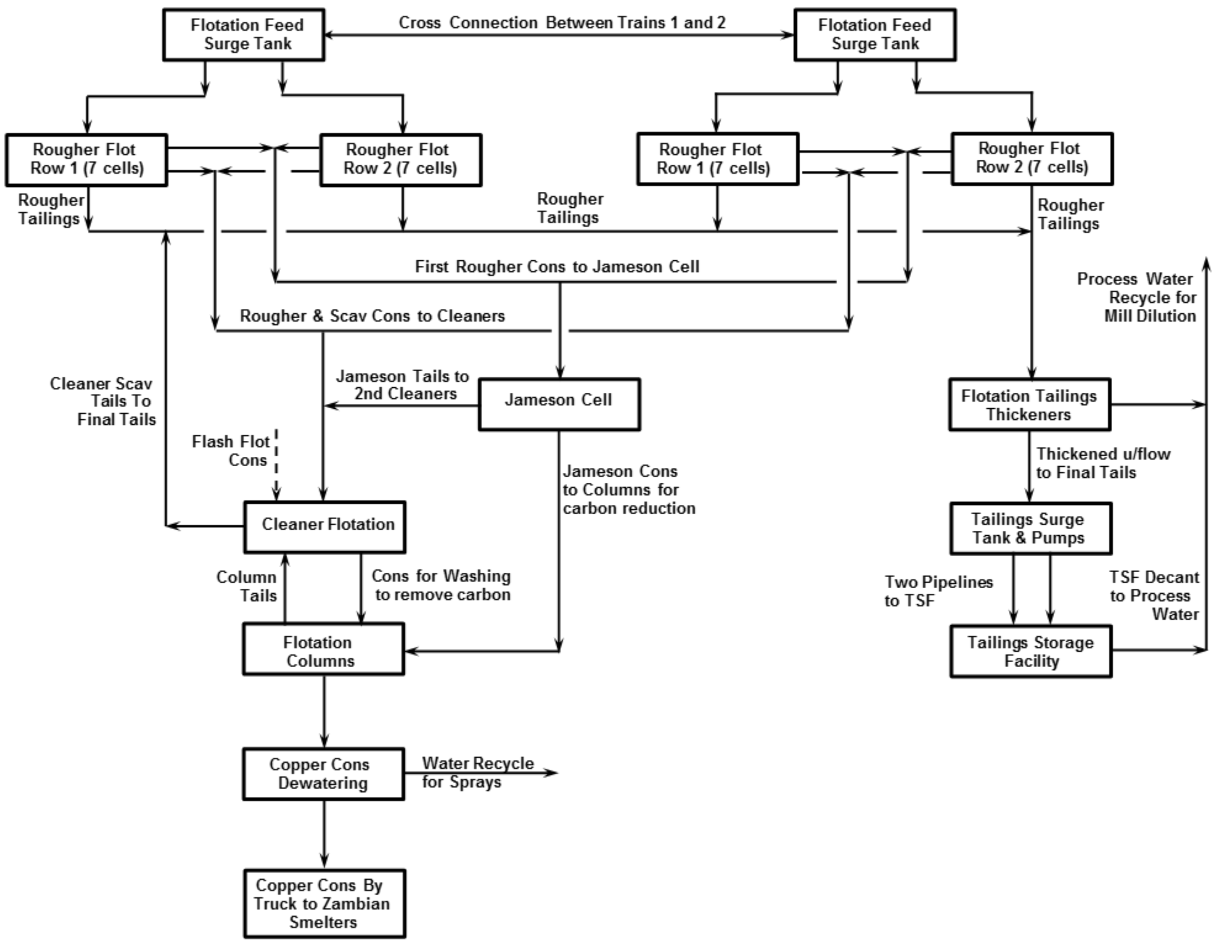


Figure 17-5 View of Sentinel flotation circuit

Four banks of 7 x 300 m^3 rougher/scavenger flotation cells are installed; two banks per milling train.

A single multi-stage cleaner flotation circuit comprising 7 x 300 m^3 and 10 x 200 m^3 cells in total was initially installed. To this a Jameson cell and two flotation columns (a further two are planned for installation in 2020) have been added to improve concentrate grades to 26.5% Cu (design was 24%).

Cyclone overflow from each milling train gravitates to a separate flotation feed surge tank, with a cross over connection installed between the two tanks. Each tank is equipped with dedicated pumps feeding two rougher flotation banks per train. Each bank comprises a conditioning tank and 7 x 300 m^3 flotation cells.

Rougher concentrate from the first cell in each bank is relatively high grade and is pumped to a single Jameson cell that upgrades the concentrate to final grade material. Jameson tails are pumped to the second cleaner flotation cells. Rougher and scavenger concentrates from the remaining rougher cells are pumped to the first cleaners; first cleaner cons is pumped to the second cleaners for upgrading, and cleaner tails gravitate to the cleaner scavengers. Cleaner scavenger cons are recycled to the head of the first cleaners, and cleaner scavenger tails are pumped to final tailings.

Similarly, second cleaner cons are pumped to the third cleaners, and tails are returned to the first cleaners. Third cleaner cons are pumped to the flotation columns, and third cleaner tails are recycled to the second cleaners. Collector, frother and gangue depressant are added to the flotation circuit as required via dedicated systems.

17.1.2.5 Concentrate upgrade

Final concentrates from the cleaning circuit contain high levels of carbon. The carbon is ultra-fine, and is liberated, and can be separated from the concentrate in two stages of desliming cyclones cutting at 10 µm. The deslime circuit was operated during the early phase of operation, but it led to high copper losses (ultra-fine chalcopyrite) in the final cyclone overflow, and this proved very difficult to recover.

Flotation columns can be operated with a deep froth zone, which can be effectively washed. This removes the entrained carbon and other gangue particles, leading to an upgrade in the concentrate to 26.5% Cu, and a drop in carbon levels from about 3% C to less than 1%.

The flotation columns (two installed and two future units planned) are fed with 3rd cleaner cons and Jameson cons. Final concentrate is pumped to a dewatering nthickener, and column tails are recycled to the conventional second or third cleaner cells.

17.1.2.6 On-stream analysis

An on-stream analysis system is installed to provide up-to-date assay information for the critical flow streams in the plant. Additionally, metallurgical samples are collected for accounting purposes.

17.1.2.7 Concentrate handling

Clean concentrate (at less than 1% C, and at 26.5% Cu) is dewatered in a single concentrate thickener and one of two installed pressure filters prior to dispatch by road to smelters within Zambia. The filtered product averages approximately 10% moisture by weight. A concentrate shed that can store a total of 50,000 tonnes of concentrate on site acts as a surge for transport purposes.

17.1.2.8 Tailings disposal

Three 50 m diameter thickeners are installed to thicken flotation tailings. Underflow slurry from each thickener is pumped to a tails surge tanks, from where it is pumped to the tailings storage facility (TSF). Tailings is pumped at 50 to 55% solids, using three stages of centrifugal pumps per line, and two tailings lines. A single overflow tank is used for process water recycle to the milling circuit.

17.1.2.9 Tailings storage facility (TSF)

The TSF is sized for an ultimate capacity of 1,000 Mt of combined tailings from the Sentinel and Enterprise circuits, requiring a storage volume of about 900 Mm^3. Assuming a final height of about 40 m, the design diameter of the circular tailings dam is about 5,500 m (Figure 17-6).

Water is recycled from the tailings dam decant points to the process water tank by submersible pumps feeding surge tanks with forwarding pumps with sufficient capacity for the predicted flow rate during the wet season. Tailings decant water is used as make-up water in the Sentinel milling circuit; the decant pumps deliver to the thickener overflow tank (process water tank), and this tank overflows to the process water pond.

Figure 17-6 Tailings storage facility (photographed March 2019)

17.1.2.10 Reagents and operational spares

Reagent requirements are fairly typical for a copper concentrator, and are limited to grinding media, lime for pH control, flotation reagents, and flocculants for the concentrate and tailings thickeners. Daily and annual requirements are presented in Table 17-2.

Table 17-2 Sentinel reagent consumption

Reagent	g/t treated	kg/day	tpa
SAG mill balls	260	39,178	14,300
Ball mill balls	320	48,219	17,600
Lime	250	37,671	13,750
Frother	75	11,301	4,125
Collector	20	3,014	1,100
Depressant	50	7,534	2,750
NaHS	40	6,027	2,200
Flocculant	20	3,014	1,100

17.2 Enterprise

Figure 17-7 shows a general view over the processing plant. In the left distance can be seen the ROM pad for future nickel ore deliveries from the Enterprise Pit, leading onto conveyors and the Enterprise milling facilities.

Figure 17-7 View looking northwest over the processing plant, Enterprise mills at left

17.2.1 Mineralogy

Nickel occurs as secondary oxides to depths of between 40 m and 80 m and below that as primary sulphides. Nickel sulphides of millerite, vaesite and violarite form relatively coarse crystals intergrown with anhydrite, albite, minor pyrrhotite and rarely chalcopyrite. The secondary supergene nickel oxidation products include garnierite and Ni-rich serpentines. Locally intense iron and carbonate alteration appears to be flanked by serpentine and chlorite haloes.

Millerite, vaesite and nickeliferous pyrite are the main minerals expected to be treated in the Enterprise circuits at the processing plant.

17.2.2 Key processing design parameters

Mined ore at Enterprise will be reclaimed from short term surface stockpiles, located adjacent to the pit crest, and road hauled to a ROM pad located adjacent to the Sentinel copper concentrator. There will be on long term ore stockpiles.

Within the processing plant, a dedicated primary crusher, crushed ore stockpile and conveying system has been provided for the Enterprise ore. The treatment rate will be 4 Mtpa of ore, (500 tph of mill feed), with an anticipated recovery of 60% to 85% Ni, depending on the mineralogy of the feed. Approximately 40,000 tpa of contained nickel metal in concentrates can be produced when high grade ores of 1.15% Ni are treated, early in the mine life.

The crushed ore will be milled in a SAG-ball milling circuit, and the ground product floated in a circuit comprising talc pre-float, nickel rougher and scavenger flotation, and three stages of cleaning. The talc pre-float will be operated without reagent addition and with two stages of cleaning, in order to produce a talc concentrate containing very little nickel, and this will be discarded to tailings. Final concentrate at a grade of between 14% and 16% nickel will be thickened and filtered in a dedicated concentrate handling facility.

The Enterprise circuits share all of the Sentinel processing infrastructure; tailings will be discharged to the Sentinel tailings thickeners and thence to the common TSF.

The processing facilities for Enterprise nickel were constructed at the same time as the Sentinel copper processing facilities. The flotation circuit has been commissioned and operated on Sentinel slurries, but is currently under care and maintenance. The milling circuit is complete, except for conveyor belting installation, and is also on care and maintenance.

17.2.3 Process and plant description

Figure 17-8 shows the block flowsheet for the Enterprise milling and flash flotation circuits, whilst Figure 17-9 provides a photograph of the milling circuit. Figure 17-10 provides a block flowsheet for the nickel flotation circuit, whilst Figure 17-11 is a photograph of the talc prefloat and rougher flotation circuits.

The flowsheet comprises:

- primary crushing of ROM ore accomplished using two jaw crushers, operating in parallel
- a crushed ore stockpile of 10 tonne live capacity
- a single SAG mill (5.8 MW installed power) followed by a single ball mill (3.6 MW), producing a grind size of 80% passing 150 µm
- flash flotation (roughing and cleaning) to make a high grade Ni concentrate
- pre-float for removal of talc in two rougher stages
- scavenging of nickel from the talc concentrate in two stages of cleaner flotation
- rougher flotation of nickel sulphides in seven rougher and scavenger flotation cells arranged in series in a single train
- nickel cleaning in three stages (comprising a total of sixteen cells) for removal of residual gangue minerals
- dewatering of Ni concentrates by thickening and filtration
- pumping of flotation tailings to the Sentinel tailings thickeners
- reagent make-up, storage and dosing systems – integral with the Sentinel processing facilities
- water services – taken from the Sentinel water reticulation systems
- compressed and instrument air – again from the Sentinel services

The talc pre-float concentrate will be pumped directly to the tails disposal tank (bypassing the tailings thickeners) to avoid contaminating the Sentinel process water with fine talc.

The nickel flotation circuit is a standard rougher-cleaner flotation circuit without an intermediary regrind step. However, the flotation circuit design includes a talc removal step. Talc floats very readily, and the majority of the talc present in the ores can be removed in the pre-flotation circuit. Unfortunately, depending on the specific minerals being treated, a significant amount of fine nickel minerals can be lost the talc concentrate. Hence the rougher concentrate will be cleaned in two stages of cleaning to recover the nickel (in the pre-float tails), which will then be recycled to the head of the nickel rougher flotation circuit.

Talc concentrates will be pumped to final tailings, but because talc is hydrophobic, these concentrates will bypass the tailings thickeners, and will be directed into the final tailings pump surge tank.

Final nickel concentrates will be dewatered using a concentrate thickener, followed by filtration, and conveyed to a 30,000 t capacity concentrate storage building.

Figure 17-8 Enterprise crushing, milling and flash flotation circuits, block flowsheet

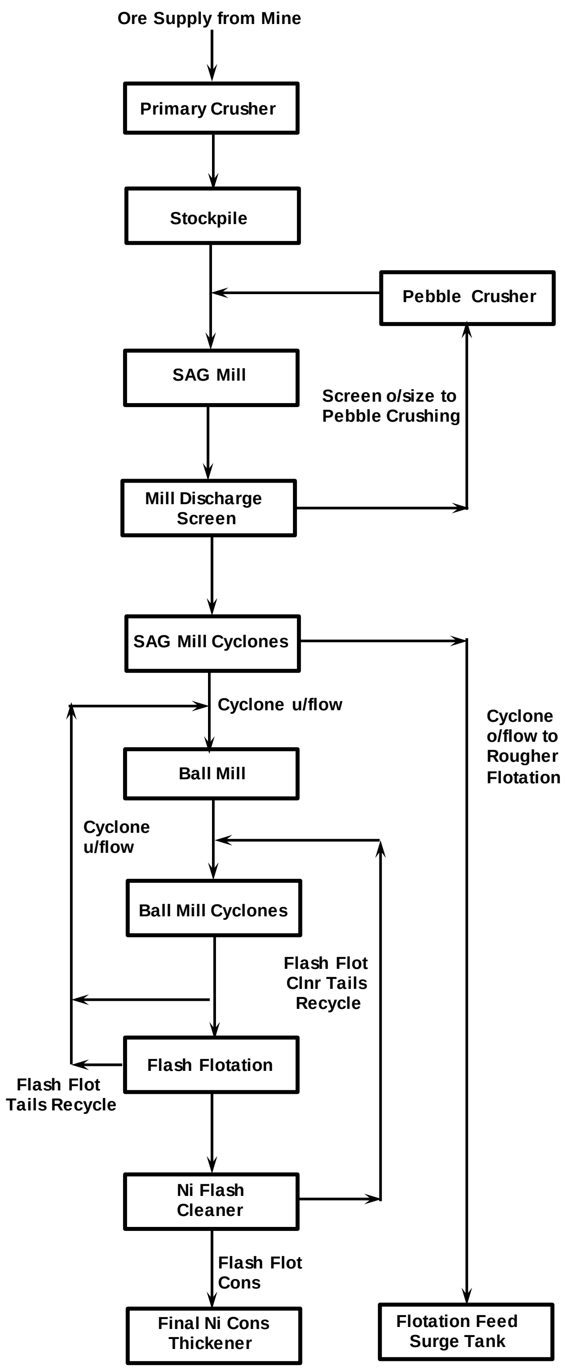
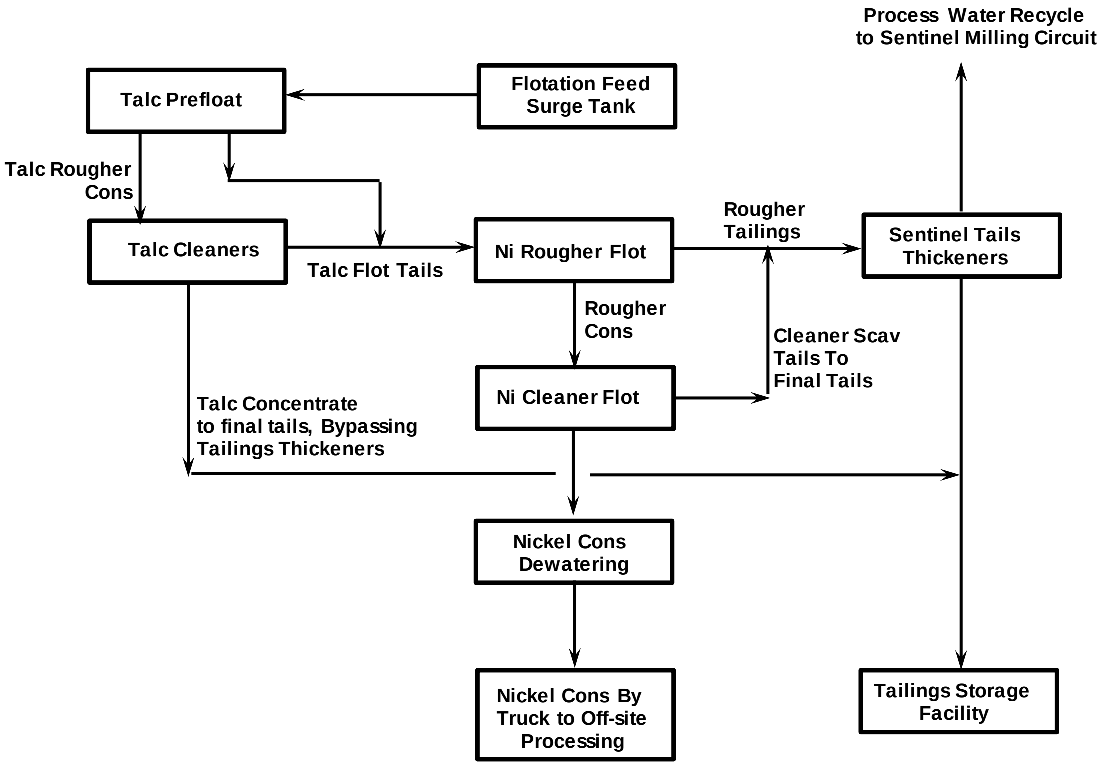


Figure 17-9 Enterprise milling and flash flotation circuit

Figure 17-10 Enterprise talc prefloat and flotation circuit block flowsheet



Concentrate will be transported off site to smelters outside of Zambia. It is assumed that no export levy will apply to nickel concentrates because there is no in-country smelter. Nevertheless transport costs are high, and an internal study is underway, looking at the possibility of treating the concentrates through the pressure leach circuit at Kansanshi, followed by solvent extraction and electrowinning (SXEW). Existing circuits exist at Kansanshi for SX and EW for copper recovery from leaching of oxide ores. The practicality and cost of modifying these circuits for nickel production will be evaluated.

Figure 17-11 Enterprise talc prefloat and rougher flotation circuits

17.2.4 Further testwork

As noted in Item 13.2.7, very little testwork has been completed on some of the mineral types present in the Enterprise ores, and few locked cycle tests have been performed, particularly including the talc pre-float. Therefore, a further round of testwork has been defined on freshly drilled samples from the Enterprise deposit.

The testing is being performed on eight composites intended to represent the mine feed during the initial starting phase (oxidised shale and siltstone) and material later in the mining sequence. The material later in the mining sequence will have less oxidation (fresh shale and siltstone).

This testwork will help define (and optimise) recoveries from the various ore types and nickel mineralogies present at Enterprise and confirm reagent consumptions. Results from this work are expected later in 2020.

ITEM 18 PROJECT INFRASTRUCTURE

18.1 Project site layout

Figure 18-1 shows the existing Trident Project site layout as at Q4 2019. Features of the layout are as follows:

- the Sentinel and Enterprise mining sites, including proposed ultimate waste dump limits
- the processing plant
- existing and new roads, including the haulroad from Enterprise to the plant, which also serves as a corridor for process water supply and tailings delivery pipelines
- the Musangezhi Dam
- the Chisola Dam
- the Kalumbila town site

18.2 Roads and site access

The new roads required for the Project have been as follows:

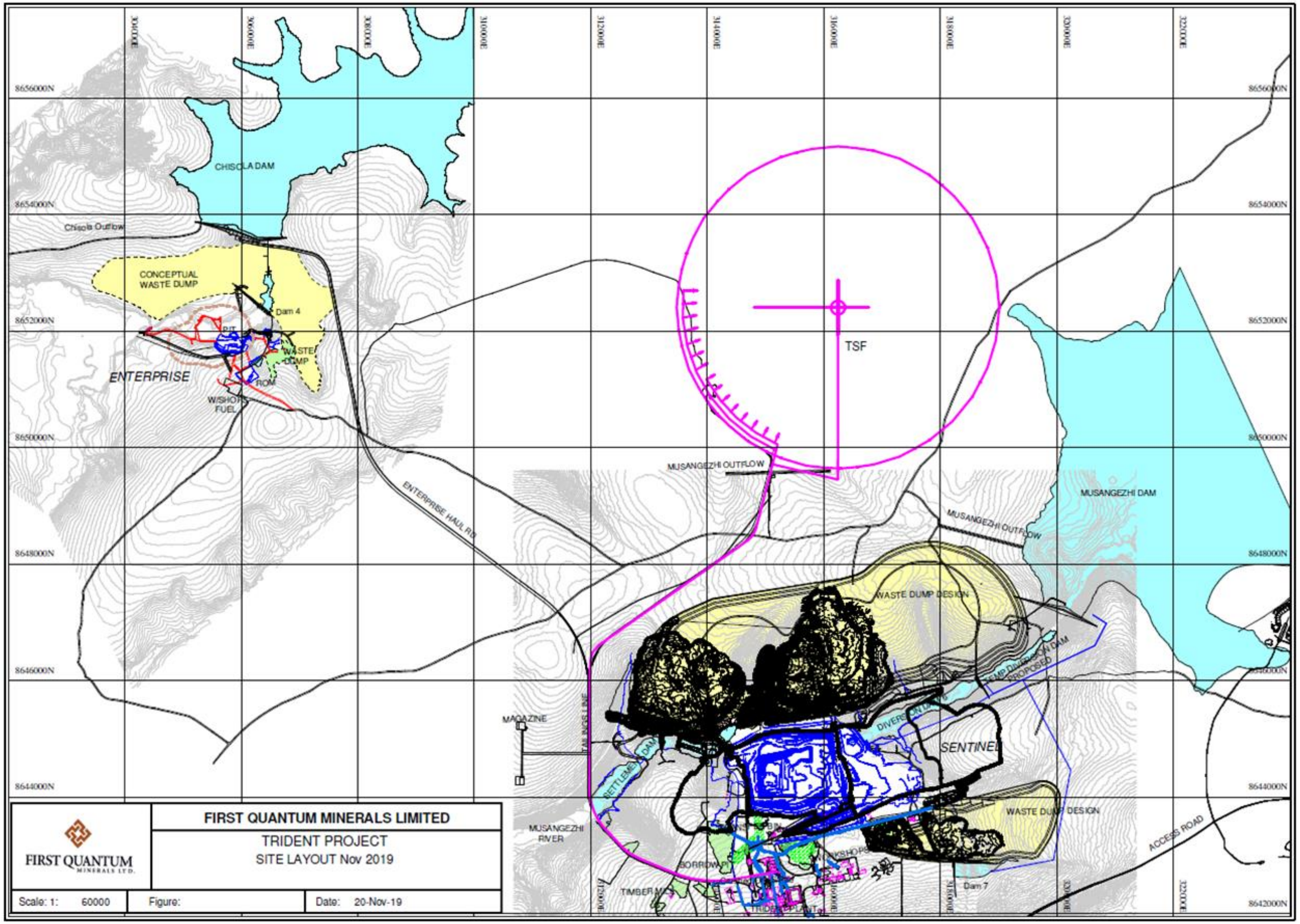
- main access road from the Solwezi road to the site
- access road to the construction camp
- access road to into the Kulumbila town site
- access roads to the tailings storage facility and Chisola Dam
- aggregate quarry haul road
- powerline maintenance track
- borefield maintenance track
- silt pond access tracks
- plant site roads

18.3 Site buildings

The major non-process buildings provided to service the plant, mining and administration functions include:

- an administration office
- plant, mining and light vehicle workshops
- warehouses
- reagent storage sheds
- training offices
- medical clinic
- laboratory

Figure 18-1 Trident Project site, Q4 2019



18.4 Dams

The Musangezhi River which flows over the top of the Sentinel deposit has been diverted to allow the pit to be developed. The river has been impounded upstream of the mine, providing a lake alongside the Kalumbila town site. Excess lake water overflows to the northwest via a spillway system flowing into a channel and directing water in a westerly direction around the southern side of the TSF and eventually into the upper catchment of the Kasombo River.

An additional, large earthfill dam has been constructed on the Chisola River to the north of the Enterprise deposit, as a source of process water. A spillway and channel have also been constructed for this dam to redirect the Chisola River to the north and then westwards into the Kabompo River.

18.5 Town site

The Kalumbila town site has been constructed at a location 19 km from the processing plant site, on the eastern side of the Musangezhi Dam reservoir. The town site comprises housing plots for employees and their families, in addition to commercial businesses such as a supermarket, guesthouse, fuel station and related services. A senior accommodation complex is located as part of the town site and includes single persons quarters.

18.6 Power supply

Power supply to the Trident Project is provided by a 60 km extension of the exiting 330 kV transmission line into the North West Province, via the Luano substation to the Solwezi and Lumwana substations, and connected to a new ZESCO substation at the plant site. Power supply is supplemented from a new 545 km long additional line extending from the Lusaka West substation.

The power supply to the various plant and remote facility load centres is distributed from the main 33 kV distribution board via 33 kV cables and overhead lines.

18.7 Water supply

Three categories of water are used in the processing plant:

- process water
- fresh (raw) water
- potable water

18.7.1 Process water

The water supply required for ore processing is made up from two primary sources:

- recirculation from within the plant
- decant from the TSF

The primary water source for process water make-up is the decant water from the tailings pond with top-up from the raw water pond.

Groundwater abstraction at Sentinel is currently aimed at dewatering the pit and maintaining drained working surfaces and pit slope conditions. Further hydrogeological evaluation is required to identify any significant aquifers and bores that may be installed to assist with the mine dewatering process and provide a further supply of water for the processing plant, should that be required.

18.7.2 Fresh (raw) water

The primary source of fresh water for the processing plant is from the Chisola Dam. A water intake and pump station has been installed at the dam to supply water as required to the raw water pond in the plant.

18.7.3 Potable water

The general quality of groundwater in the area is very good, and it can be treated and used for potable water. Potable water in the plant area is treated Chisola Dam water, whilst township water is treated Musangezhi Dam water.

ITEM 19 MARKET STUDIES AND CONTRACTS

19.1 Markets for Sentinel product

Copper concentrate from the processing plant is transported by truck, primarily to the Company's Kansanshi (KCS) smelter, and to a lesser extent to other copper smelters on the Zambian Copperbelt. Since the advent of on-site smelting at KCS, anode product from the smelter (and cathode product from the Kansanshi SXEW plant) are now the main saleable products.

19.2 Markets for Enterprise product

Under current proposals, nickel concentrate from the processing plant will be transported by truck to a port at Walvis Bay in Namibia, from where it will be exported by ocean freight to potential off-takers. The most likely off-takers are nickel smelters in either Canada, Europe or China.

The concentrate is expected to be attractive to potential off-takers as the concentrate grade of 15% is high relative to other nickel producers. Additionally, the high sulphur grades and low deleterious elements make it an attractive saleable product.

19.3 Contracts

Currently, all anode and concentrate product is being sold through the company's internal marketing division, Metal Corp Trading AG (MCT).

To date, contracts have not been negotiated for the sale of nickel concentrate from the processing plant. These contracts will be negotiated by MCT, who have previously sold nickel concentrates from the Company's former Kevitsa mine to global nickel smelters.

A mining fleet could be contracted for the Enterprise Pit. Preliminary tender documents are being prepared, at least for the continuation of initial pre-strip works. Mining cost estimates for Enterprise optimisation and cashflow modelling reflect an eventual scale of mining fleet that differs from the fleet required for pre-strip works. At this time, a longer term mining contract has not been tendered or awarded.

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

20.1 Environmental setting

The Trident Project includes the existing Sentinel and proposed Enterprise open pits and the existing processing plant. Sentinel comprises a single large operating open pit of multiple cutback phases, two waste rock dumps, a copper ore treatment circuit within the processing plant, various water supply and control dams, plus associated pipelines. Enterprise comprises a proposed smaller design open pit, a single waste dump, a nickel ore treatment circuit within the processing plant, and a number of other infrastructure items including water control dams and a connecting haulroad between the mine and plant site.

A common tailings storage facility serves the copper and nickel processing circuits of the processing plant.

While the Trident Project footprint has been extensively modified in recent years due to mining activity, most of the surface rights area still supports the original indigenous vegetation. The predominant vegetation in the Project area is a mosaic of miombo woodland and termitaria. This vegetation type is common throughout Zambia and especially in the North-West Province. Dambos and floodplains occur scattered throughout the site and often contain riparian forest along the river banks. No endangered flora or fauna species are known from within the surface rights area. A number of large herbivores have been re-introduced into the surface rights area in recent years.

The surface area is drained by three Rivers, namely the Musangezhi (south), Kasombo (centre) and the Chisola (north). All three rivers flow in a westerly direction eventually joining the Kabombo River.

20.2 Status of environmental approvals

Numerous Environmental Impact Assessments (EIAs) for operational infrastructure at Sentinel and Enterprise have been submitted and approved by the Zambia Environmental Management Agency (ZEMA) over the last eight years. These include the original EIAs and subsequent smaller applications for additions and changes to the original project scope.

Each environmental approval is accompanied by a list of commitments. The commitments vary depending on the project, but typically require implementation of a number of control measures, environmental monitoring and adherence to Zambian government legislative requirements. All of the outstanding commitments were recently incorporated into a Consolidated Environmental Management Plan (EMP). The Consolidated EMP was then approved by ZEMA. The commitments are also subject to regular audits by the Zambian Mines Safety Department (MSD) and by ZEMA.

20.3 Environmental management

In addition to the Consolidated EMP, site environmental management is also guided by Company Policy and host country legislative requirements. Legislative requirements include site environmental permits and a number of Zambian government laws and regulations. Site permits are issued every three years and provide requirements for the management of waste, effluent, emissions and chemicals. Effluent permits require regular monitoring of all surface discharges, and compliance with statutory limits. All site permits are up to date and the operations comply with these permits. Relevant legislation includes the Mines and Minerals Development Act No. 11 of 2015 and the Environmental Management Act No. 12 of 2011.

An Environmental Management System (EMS) is in place to improve overall performance and improve adherence to both the policy and legal requirements. The EMS accords with the ISO 14001:2015 standard and is regularly audited by external parties.

20.4 Resettlement

The Trident Project required that a few hundred families needed to be resettled outside of the surface rights area. In order to guide the resettlement process, a Resettlement Action Plan (RAP) was developed. While Zambia does not have any legislated guidelines on involuntary resettlement, the RAP was submitted to ZEMA for their consideration and was subsequently approved. FQM ensured that all resettlement planning complied with the relevant International Finance Corporation (IFC) Performance Standards.

20.5 Community engagement

An open and respectful relationship with all local, regional and national stakeholders is maintained at Trident. Local communication includes group and one-on-one meetings with local government representatives, traditional leaders, village elders, community elected representatives, civil society, non-governmental agencies and community members. Minutes of all meetings are recorded. Local communities have the opportunity to submit grievances to the Company via a number of channels. Grievances are investigated and resolved within a time period acceptable to each party.

20.6 Mine closure

The main environmental liabilities at Trident will arise at closure and are related to the dismantling of the process plants and infrastructure across the site, continuation of water quality management, and rehabilitation of the tailings dams, open pit and waste rock dumps. The closure plan for Sentinel is reviewed externally biennially.

As at the end of 2019, the unscheduled closure cost (or asset retirement obligation (ARO)) for Sentinel was estimated to be $55.7 million. The scheduled closure cost was estimated at $78.2 million. In accordance with National Legislation, Sentinel contributes to an Environmental Protection Fund administered by the MSD.

The Enterprise closure estimate currently stands at $6.5 million. The estimate has been prepared internally and was calculated according to a similar itemisation and with unit rates and allowances as for the Sentinel estimate.

ITEM 21 CAPITAL AND OPERATING COSTS

21.1 Sentinel

21.1.1 Capital costs

The capital cost provisions included in the Sentinel cashflow model as tabulated in Item 22 total $157.2 M (Table 21-1).

Table 21-1 Sentinel capital cost provisions

	2020	2021	2022	2023	2024	TOTAL
Mining capital ($M)						
Inpit crusher IPC4A	$27.0	$32.0				**$59.0**
Trolley assist phases 4 & 5	$6.5					**$6.5**
Additional electric shovel (+10% duty incl.)		$27.5				**$27.5**
2 X additional haul trucks (+10% duty incl.)		$13.2				**$13.2**
1 X additional dozers (+10% duty incl.)		$2.2				**$2.2**
1 X additional drill rigs / diesel PVs (+10% duty incl.)		$3.3				**$3.3**
Geotech - IPC4A ground support						**$0.0**
subtotal ($M)	**$33.5**	**$78.2**	**$0.0**	**$0.0**	**$0.0**	**$111.7**
Processing capital ($M)						
Aggregate crusher upgrade	$0.1					**$0.1**
Float column	$1.7					**$1.7**
Front end rectification (complete)						**$0.0**
Replacement pebble magnet			$10.0			**$10.0**
IPC and conveyor reloc. plus mods and new splitter					$20.0	**$20.0**
SAG discharge screen	$2.0	$2.0	$2.0	$2.0	$2.0	**$10.0**
Tails piping (complete)						**$0.0**
Punch list rectification (complete)						**$0.0**
subtotal ($M)	**$3.8**	**$2.0**	**$12.0**	**$2.0**	**$22.0**	**$41.8**
Other capital ($M)						
Site Projects						**$0.0**
Dam 6B Construction						**$0.0**
Carry over (From 2019)	$0.5	$1.0				**$1.5**
Other Items	$2.2					**$2.2**
subtotal ($M)	**$2.7**	**$1.0**	**$0.0**	**$0.0**	**$0.0**	**$3.7**
Total ($M)	**$40.0**	**$81.2**	**$12.0**	**$2.0**	**$22.0**	**$157.2**

Item 16.1.4 provided an analysis of the primary mining equipment requirements to cater for the increased production requirements. Whilst Table 21-1 shows a provision for an additional electric shovel, the same table shows a provision for two trucks to service that additional shovel. The rationale for only two additional trucks is the future short ore hauling, along proposed trolley assist ramps, to a new IPC position in Phase 2.

21.1.2 Sustaining capital costs

The initial sustaining cost provisions included in the Sentinel cashflow model as tabulated in Item 22 total $488.7 M (Table 21-2).

Beyond the development capital phase from 2024, the cashflow model includes allowances for mining and processing sustaining capital which are equivalent to 5% of the respective mining and processing operating costs.

Table 21-2 Sentinel sustaining cost provisions

	2020	2021	2022	2023	2024	>2024	TOTAL
Mine sustaining ($M)							
Mining eq't - planned maintenance parts	$39.5	$37.8	$37.1	$38.1	$18.0		**$170.5**
MD6640 drill mast	$1.9	$0.0	$0.0	$0.0	$0.0		**$1.9**
Refurbishment of HD1500	$1.0	$0.0	$0.0	$0.0	$0.0		**$1.0**
CAT 7945 dipper handle	$0.0	$1.2	$0.0	$0.0	$0.0		**$1.2**
New bucket and dump trays welding area	$1.0	$0.0	$0.0	$0.0	$0.0		**$1.0**
Mine services - 1 X motivator	$0.0	$1.5	$0.0	$0.0	$0.0		**$1.5**
4 X additional drill rigs	$3.8	$0.0	$0.0	$0.0	$0.0		**$3.8**
Road sweeper for treated haul roads	$0.7	$0.5	$0.0	$0.0	$0.0		**$1.2**
Pit dewatering - 2 X multiflow pumps and MCCs	$0.0	$1.5	$1.0	$1.0	$0.1		**$3.6**
Progressive sustaining costs (at 5% of opex)						$111.0	**$111.0**
subtotal ($M)	**$47.8**	**$42.5**	**$38.1**	**$39.1**	**$18.1**	**$111.0**	**$296.7**
Process sustaining ($M)							
SAG and ball mill segment coils	$0.0	$0.0	$1.1	$0.5	$0.0		**$1.6**
XRF and fusion system	$0.0	$1.0	$0.0	$0.0	$0.0		**$1.0**
Installation of new clear water tank for chillers	$0.0	$0.0	$1.3	$0.0	$0.0		**$1.3**
Progressive sustaining costs (at 5% of opex)						$125.8	**$125.8**
subtotal ($M)	**$0.0**	**$1.0**	**$2.4**	**$0.5**	**$0.0**	**$125.8**	**$129.8**
Other sustaining ($M)							
Capital components (not PCR)	$3.6	$4.4	$4.4	$4.4	$4.4		**$21.2**
ETP plant relocate and new strategy	$0.0	$15.0	$0.0	$0.0	$0.0		**$15.0**
New fabrication shop	$0.0	$1.2	$0.0	$1.2	$0.0		**$2.4**
Light vehicles	$0.6	$0.6	$0.0	$0.0	$0.0		**$1.2**
Carry over (From 2019)	$1.5	$0.0	$0.0	$0.0	$0.0		**$1.5**
Other Items	$5.7	$5.9	$3.1	$4.5	$1.8		**$21.0**
subtotal ($M)	**$11.4**	**$27.1**	**$7.5**	**$10.1**	**$6.2**		**$62.3**
Total ($M)	**$59.2**	**$70.6**	**$48.0**	**$49.7**	**$24.3**	**$236.9**	**$488.7**

21.1.3 Mine closure provisions

Mine closure provisions (also referred to as asset retirement obligations (ARO)) are routinely reviewed and updated by external consultants. This review involves a thorough process of itemising closure and post-closure cost estimates for such as infrastructure decommissioning, site rehabilitation and monitoring. The estimates for these comprehensively listed activities are typically built-up from benchmarked unit costs.

Golder Associates Africa Pty Ltd (Golder) carried out the most recent estimate of scheduled closure provisions as at September 2018 (Golder, 2018). A summary of the Golder estimate is provided in Table 21-3.

21.1.4 Mining costs

Forward looking Sentinel mining costs have been estimated based on an operating performance equation which accounts for:

- the expected use of primarily larger capacity trucks in the future, hauling ore to retrofitted IPCs with increased vault capacity, and hauling waste to the dumps
- haulage along defined ore and waste routes applicable to the current life of mine plan, with an explicit identification of routes of where trolley assisted haulage could be adopted
- current excavating/loading costs reflecting the use of higher utilisation face shovels plus smaller hydraulic excavators and loaders operating at lower utilisation

Table 21-3 Sentinel closure cost provisions

	Progressive 2020 to 2031	Final 2032 to 2034	($M)
Closure components			
Infrastructural aspects:			
dismantling of structures	$0.0	$12.3	$12.3
rehabilitation of roads	$0.0	$0.2	$0.2
removal of linear structures	$0.0	$1.5	$1.5
disposal of demolition waste	$0.0	$2.2	$2.2
subtotal	**$0.0**	**$16.3**	**$16.3**
Mining aspects:			
pit rehabilitation	$0.4	$4.6	$5.0
waste dump rehabilitation	$27.0	$0.0	$27.0
surface water pond rehabilitation	$0.0	$0.8	$0.8
TSF rehabilitation	$12.6	$0.0	$12.6
subtotal	**$40.0**	**$5.4**	**$45.5**
General surface rehabilitation	$0.0	$3.0	$3.0
Water management	$0.0	$0.3	$0.3
Subtotal closure costs	**$40.0**	**$25.0**	**$65.0**
Post-closure components			
Surface water monitoring	$0.0	$0.1	$0.1
Groundwater monitoring	$0.0	$0.0	$0.0
Rehabilitation monitoring	$0.0	$0.1	$0.1
Care and maintenance	$0.0	$1.3	$1.3
Contingencies for post-closure	$0.0	$0.2	$0.2
Subtotal post-closure costs	**$0.0**	**$1.7**	**$1.7**
Additional allowances			
Preliminary and general	$0.0	$9.8	$9.8
Contingencies	$0.0	$0.0	$0.0
Additional studies	$0.0	$1.6	$1.6
Subtotal additional costs	**$0.0**	**$11.4**	**$11.4**
Total closure costs	**$40.0**	**$38.2**	**$78.2**

Assumptions implicit in the mining cost equation, and based on operating performance, are:

- truck fuel consumption costs reflect an assumed fuel cost of $1.00/L and a consumption rate of 750 L/hour
- truck fuel consumption costs reflect engine load factors of 10% when idling (eg, when trucks are being loaded), 20% when unladen and travelling flat, 30% when laden and travelling flat, 20% when unladen and hauling up a ramp, 100% when laden and hauling up a ramp
- a 9 minute allowance for non-travel time, plus estimated times for flat and ramp travel under prevailing speed limitations, resulting in the indicative times listed in Table 21-4
- maintenance costs of $220/hour inclusive of major rebuilds and key maintenance overhaul requirements, resulting in the unit costs listed in Table 21-5
- an allowance of $180/hour for labour costs, depreciation and overheads, and resulting in the other unit costs listed in Table 21-5
- a unit excavating/loading cost of $0.60/bcm based on current information reflective of the mix of equipment items and their respective utilisation
- a unit drill and blast cost of $1.65/bcm inclusive of consumables
- a 20% loading on load and haul costs to cover the use of ancillary mining equipment

Table 21-4 Estimated truck non-travel and travel times for Sentinel

Item	Units	Value
Non travel time	**minutes**	**9.00**
Travel time, no TA		
unladen, flat	minutes/km	1.50
laden, flat	minutes/km	2.00
subtotal	**minutes/km**	**3.50**
unladen, on ramp	minutes/km	2.40
laden, on ramp	minutes/km	5.00
subtotal	**minutes/km**	**7.40**
Travel time, under TA		
unladen, on ramp	minutes/km	2.40
laden, on ramp	minutes/km	3.16
subtotal	**minutes/km**	**5.56**

Table 21-5 Estimated truck haulage operating costs for Sentinel

Item	Fuel ($/bcm)	Maintenance ($/bcm)	Other ($/bcm)	Total ($/bcm)
During non travel time	**$0.09**	**$0.20**	**$0.21**	**$0.49**
During travel time, no TA				
unladen, flat	$0.03	$0.04	$0.03	$0.11
laden, flat	$0.06	$0.06	$0.05	$0.16
subtotal	**$0.09**	**$0.10**	**$0.08**	**$0.27**
unladen, on ramp	$0.05	$0.07	$0.06	$0.17
laden, on ramp	$0.48	$0.14	$0.12	$0.74
subtotal	**$0.53**	**$0.21**	**$0.17**	**$0.91**
During travel time, under TA				
unladen, on ramp	$0.06	$0.08	$0.07	$0.20
laden, on ramp	$0.04	$0.11	$0.09	$0.23
subtotal	**$0.09**	**$0.19**	**$0.15**	**$0.43**

Specific trolley assist costs, such as electrical power and trolley capital/maintenance costs have been excluded. At the same time, the benefit of reduced truck maintenance due to trolley assist (engine and component life) has been ignored. It is considered that the relative costs and benefits would more or less balance each other.

The derived mining cost equation in terms of $/bcm is shown below, composed of the unit costs and haulage coefficients listed in Table 21-6:

The total cost per bcm mined =

- drilling/blasting cost of $1.65/bcm, plus
- loading cost of $1.31/bcm, plus
- flat travel cost of $0.32/bcm/km multiplied by the flat travel distance (in km), plus
- ramp travel cost of $1.09/bcm/km multiplied by the non-TA ramp travel distance (in km), plus
- ramp travel cost of $0.52/bcm/km multiplied by the TA ramp travel distance (in km)

Table 21-6 Estimated unit mining costs and haulage coefficients for Sentinel

Item	Units	Value
Drilling/blasting costs	**$/bcm**	**$1.65**
Loading costs		
Excavating/loading costs	$/bcm	$0.60
Total hourly cost	$/bcm	$0.49
Ancillary eq't allowance	%	20
subtotal	**$/bcm**	**$1.31**
Hauling costs		
Flat travel		
Total hourly cost	$/bcm	$0.27
Ancillary eq't allowance	%	20
coefficient	**$/bcm/km**	**$0.32**
Ramp travel, no TA		
Total hourly cost	$/bcm	$0.91
Ancillary eq't allowance	%	20
coefficient	**$/bcm/km**	**$1.09**
Ramp travel, under TA		
Total hourly cost	$/bcm	$0.43
Ancillary eq't allowance	%	20
coefficient	**$/bcm/km**	**$0.52**

A haulage optimisation study was completed in order to derive forward looking, applicable ore and waste haulage distances, on a phase-by-phase and bench-by-bench basis. These haulage distances account for ramps of suitable length for trolley assist implementation. The equation and unit cost/coefficient information above was used to translate the distances into an average ore and waste mining unit cost for every bench in the mine planning model.

The resulting database was able to be plotted such that a regression relationship could be determined, relating load and haul unit cost to mining bench RL, as shown for ore and waste, in Figure 21-1 and Figure 21-2, respectively. From the relationships in Figure 21-1 and Figure 21-2, the overall average load and haul costs are as follows:

- average waste mining cost = $3.41/bcm ($1.26/t)
- average ore mining cost = $3.83/bcm ($1.37/t)

For pit optimisation purposes, a fixed cost of $1.65/bcm was added to these figures for drill and blast, irrespective of mining bench level. This equated to a total waste mining cost of $1.86/t mined and a total ore mining cost of $1.96/t mined.

For the purposes of the cashflow model described in Item 22, the unit cost relationships were applied to the respective total ore and waste mining volumes for each year of the production profile, reflecting the varying mining bench RLs in each year of the profile. The ore relationships were modified to account for, and average, the haulage costs direct to the IPCs, to stockpiles and from stockpiles (without a drill and blast component).

Figure 21-1 Ore load and haul unit cost by bench RL

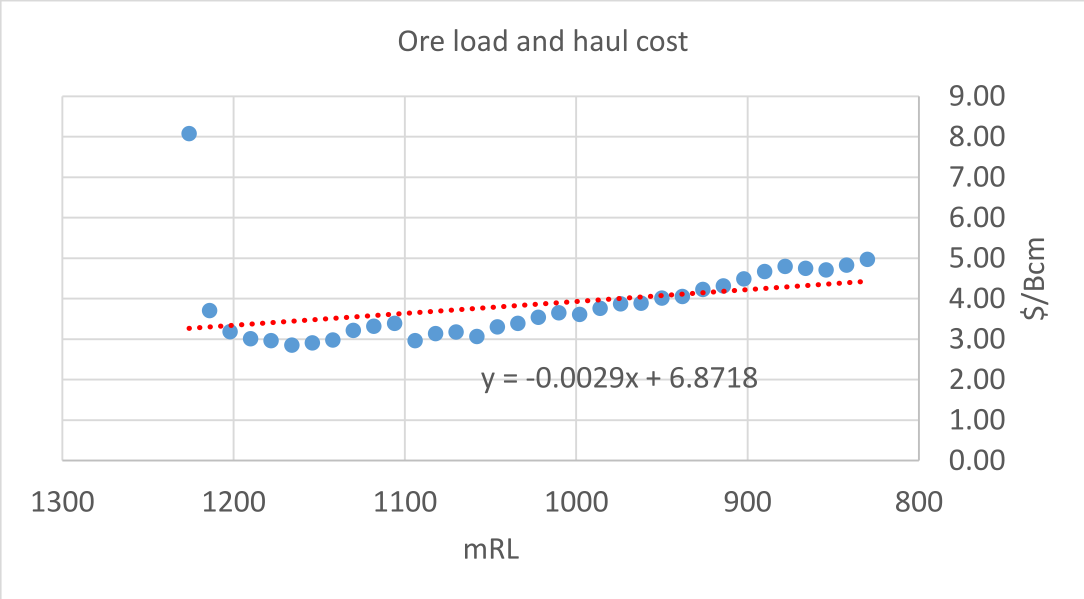
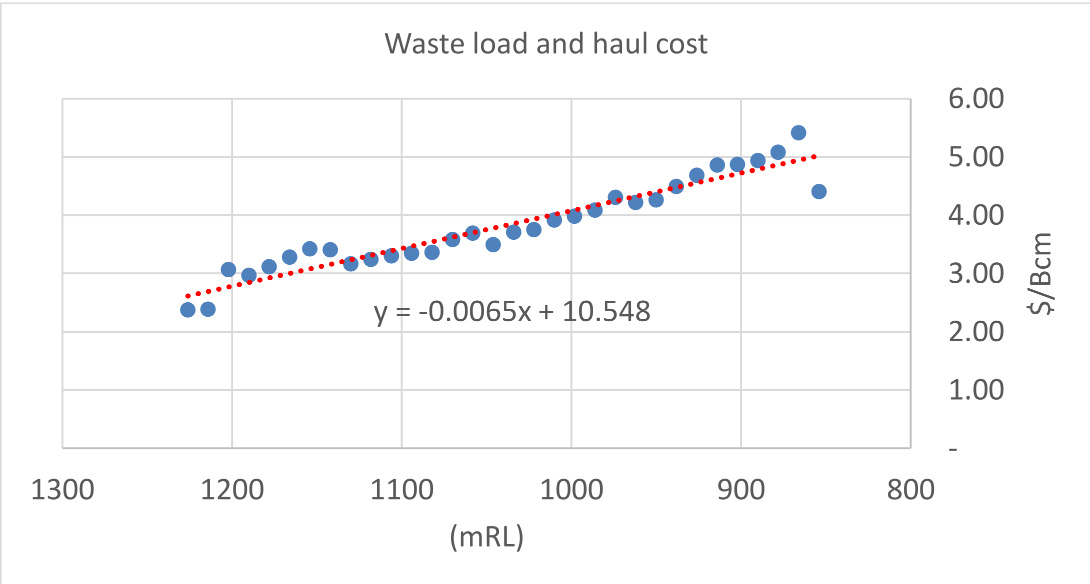


Figure 21-2 Waste load and haul unit cost by bench RL



This more rigourous approach to modelling of the mining costs led to a reduction in the overall life of mine ore mining cost from $1.96/t mined (for optimisation input) to $1.66/t mined (for cashflow modelling), as listed in Table 21-7. Considering the magnitude of the difference, a check re-optimisation was carried out to test the sensitivity of the reduced ore mining unit cost to the fundamental selection of the optimal shell forming the basis of the design ultimate pit, the production schedule and the Mineral Reserve estimate. The sensitivity analysis outcome, indicating a minimal impact to the pit shell size, is discussed in Item 15.3.3.

Table 21-7 Mining costs for Sentinel cashflow modelling

Year	Ore $/t	Waste $/t	Total $/t
2020	$1.60	$1.81	$1.74
2021	$1.58	$1.81	$1.72
2022	$1.68	$1.86	$1.80
2023	$1.66	$1.83	$1.77
2024	$1.59	$1.84	$1.75
2025	$1.53	$1.81	$1.71
2026	$1.63	$1.90	$1.80
2027	$1.68	$1.82	$1.77
2028	$1.70	$1.79	$1.76
2029	$1.61	$1.81	$1.75
2030	$1.73	$1.81	$1.79
2031	$1.77	$1.96	$1.92
2032	$1.63	$1.95	$1.83
2033	$1.82	$2.10	$2.00
2034	$2.04	$2.29	$2.09
Average	**$1.66**	**$1.86**	**$1.80**

21.1.5 Operating costs

Forward looking Sentinel operating costs have been derived following a review of actual costs incurred at site for 2019 year-to-date, as recorded in a September 2019 Business Review report (KML, September 2019). This derivation and projection was produced by QP Andrew Briggs.

21.1.5.1 Processing and engineering costs

The 2019 year-to-date records indicate an overall variable processing cost of $3.07/t processed, accounting for power supply, reagent usage, milling balls and operating spares. The same records indicate an overall fixed processing cost of $0.36/t processed, and hence a total fixed+variable processing cost of $3.43/t.

To be added to these figures are the additional costs for plant engineering, accounting for electrical spares, mechanical spares, lubricants and additional reagents. These recorded figures indicate an overall variable cost of $0.91/t processed, and an overall fixed cost of $0.72/t processed, yielding a combined total unit cost of $1.63/t processed.

For 2019 then, the applicable unit cost would be as follows:

- variable = $3.07 + $0.91 = $3.98/t processed
- fixed = $0.36 + $0.72 = $1.08/t processed
- total = $5.06/t processed

It is considered that the actual 2019 variable unit processing cost is sufficiently representative to be carried forward for each continuing year of processing. The 2019 fixed unit cost, however, will reduce over time as the plant expands to the proposed 62 Mtpa capacity. The projected processing costs beyond 2019 are as listed in Table 21-8.

Table 21-8 Sentinel 2019 actual and projected unit operating costs

Year	G&A	Processing			Opt'n	Cashflow
	Fixed $/t	Variable $/t	Fixed $/t	Total ($/t)	Total $/t	Total $/t
2019	$1.24	$3.98	$1.08	**$5.06**	**$6.30**	
2020	$1.10	$3.98	$0.95	**$4.93**	**$6.03**	**$6.25**
2021	$1.10	$3.98	$0.95	**$4.93**	**$6.03**	**$6.02**
2022	$0.97	$3.98	$0.85	**$4.83**	**$5.80**	**$5.97**
2023	$0.97	$3.98	$0.85	**$4.83**	**$5.80**	**$5.97**
2024	$0.97	$3.98	$0.85	**$4.83**	**$5.80**	**$5.97**
2025	$0.97	$3.98	$0.85	**$4.83**	**$5.80**	**$5.97**
2026	$0.97	$3.98	$0.85	**$4.83**	**$5.80**	**$5.97**
2027	$0.97	$3.98	$0.85	**$4.83**	**$5.80**	**$5.97**
2028	$0.97	$3.98	$0.85	**$4.83**	**$5.80**	**$5.97**
2029	$0.97	$3.98	$0.85	**$4.83**	**$5.80**	**$5.97**
2030	$0.97	$3.98	$0.85	**$4.83**	**$5.80**	**$5.97**
2031	$0.97	$3.98	$0.85	**$4.83**	**$5.80**	**$5.97**
2032	$0.97	$3.98	$0.85	**$4.83**	**$5.80**	**$6.00**
2033	$0.97	$3.98	$0.85	**$4.83**	**$5.80**	**$6.28**
2034	$0.97	$3.98	$0.85	**$4.83**	**$5.80**	**$6.28**
2035	$0.97	$3.98	$0.85	**$4.83**	**$5.80**	
2036	$0.97	$3.98	$0.85	**$4.83**	**$5.80**	
Average	**$1.00**	**$3.98**	**$0.87**	$4.85	$5.85	$6.02

The overall average $5.85/t figure was adopted for the pit optimisation process as described in Item 15.3.2. Between the time of the optimisation and the cashflow modelling process, these costs were further reviewed and updated to an overall average of $6.02/t. The (minimal) impact of the 3% higher unit cost is addressed in the pit optimisation sensitivity analyses.

21.1.5.2 General and administration costs

The 2019 year-to-date records indicate a total expenditure on general and administration (G&A) costs which equate to an overall unit cost of $1.24/t processed. The expenditure relates to general management, security services, environmental management, commercial/finance functions, safety and support services, and reflects a combination of variable costs and (mostly) fixed costs.

The 2019 fixed unit cost will reduce over time as the plant expands to the proposed 62 Mtpa capacity. On this basis the projected G&A costs beyond 2019 are as listed in Table 21-7. The overall average $1.00/t figure was included with the variable and fixed costs in the original pit optimisation.

21.1.6 Metal costs

In addition to royalties, Project metal costs include concentrate transport charges to the Kansanshi smelter (KCS), smelting costs, anode transport costs and subsequent refining charges. The individual unit costs are listed in Table 21-9. This table also shows the unit costs that apply to the concentrate that is not smelted at KCS. This information can be translated into combined overall average costs as listed in Table 21-10.

Table 21-9 Projected Sentinel unit metal costs

Transport, Smelting and Refining Charges	Units	$/unit
Costs associated with KCS smelting		
Concentrate freight cost	$/t con.	27.29
KCS smelter cost	$/t con.	82.50
Anode transport cost	$/t metal	183.64
Refining charge	c/lb Cu	6.08
Costs associated with smelting elsewhere		
Concentrate freight cost	$/t con.	130.99
Smelter cost	$/t con.	80.72
Refining charge	c/lb Cu	8.09

The combined average TCRC cost of $0.37/lb is estimated from a 63% to 37% split on the concentrate that is delivered to the KCS and that which is delivered to smelter(s) elsewhere on the Zambian Copperbelt. The additional royalty charge is equivalent to 7.5% of the $3.00/lb copper price.

Table 21-10 Projected Sentinel metal costs, including royalties

Transport, Smelting and Refining Charges		Units	Q4 2019 PARAMETERS			
PROCESS			Non-Primary	Primary	Non-Primary	Primary
Ore types			OTB, WTB	OTA, WTA	OTB, WTB	OTA, WTA
Smelter			**Kansanshi**	**Kansanshi**	**Copperbelt**	**Copperbelt**
Metal Costs Copper Concentrate						
Royalty rate		%	7.5%	7.5%	7.5%	7.5%
Concentrate Grade		% Cu	26.5%	26.5%	26.5%	26.5%
Moisture Content		%	10%	10%	10%	10%
Realisation/freight						
Concentrate Transport (wet)		$/t conc	24.56	24.56	117.89	117.89
Concentrate Transport (dry)		$/t conc	27.29	27.29	130.99	130.99
	subtotal	**$/t Cu**	**102.97**	**102.97**	**494.32**	**494.32**
	subtotal	**$/lb Cu**	**0.05**	**0.05**	**0.22**	**0.22**
Kansanshi Smelter Treatment						
Treatment Cost (wet)		$/t conc	74.25	74.25		
Treatment Cost (dry)		$/t conc	82.50	82.50		
	subtotal	**$/t Cu**	**311.32**	**311.32**	**0.00**	**0.00**
	subtotal	**$/lb Cu**	**0.14**	**0.14**	**0.00**	**0.00**
Concentrate Shipped to Copper Belt						
Treatment Cost (wet)		$/t conc			72.65	72.65
Treatment Cost (dry)		$/t conc			80.72	80.72
	subtotal	**$/t Cu**	**0.00**	**0.00**	**304.59**	**304.59**
	subtotal	**$/lb Cu**	**0.00**	**0.00**	**0.14**	**0.14**
Anode Transport Costs						
Transport (dry)	subtotal	**$/t Cu**	**183.64**	**183.64**		
	subtotal	**$/lb Cu**	**0.08**	**0.08**	**0.00**	**0.00**
Refining Charges						
		c/lb	6.08	6.08	8.09	8.09
	subtotal	**$/t Cu**	**134.00**	**134.00**	**178.33**	**178.33**
	subtotal	**$/lb Cu**	**0.06**	**0.06**	**0.08**	**0.08**
Total Concentrate Cu Metal Cost						
	subtotal	**$/t Cu**	**731.93**	**731.93**	**977.24**	**977.24**
	subtotal	**$/lb Cu**	**0.33**	**0.33**	**0.44**	**0.44**
Average Concentrate Cu Metal Cost						
Concentrate proprtion to smelters		%	63%		37%	
	subtotal	**$/t Cu**	**823.29**			
	subtotal	**$/lb Cu**	**0.37**			
Royalties x% Gross						
	subtotal	**$/t Cu**	**496.04**			
	subtotal	**$/lb Cu**	**0.23**			
Average Concentrate Cu Metal Cost						
	subtotal	**$/t Cu**	**1,319.32**			
	subtotal	**$/10kg Cu**	**0.60**			

21.2 Enterprise

21.2.1 Capital costs

The capital cost provisions included in the Enterprise cashflow model as tabulated in Item 22 total $65.4 M (Table 21-11).

Table 21-11 Enterprise capital cost provisions

		2019	2020	2021	2022	2023	TOTAL
Site capital ($M)							
	Powerline and transformers		$0.1	$0.6	$0.0	$0.0	**$0.8**
	Civil works on existing dams		$0.1	$0.0	$0.0	$0.0	**$0.1**
	Dam 4 earthworks		$1.2	$0.0	$0.0	$0.0	**$1.2**
	Dam 4: 6 x 90 kW pumps + 1.8 km x 630 mm pipe		$0.8	$0.0	$0.0	$0.0	**$0.8**
	Dam 2: 2 x 90kW + pipes		$0.0	$0.1	$0.0	$0.0	**$0.1**
	Fuel storage facility		$1.0	$1.0	$0.0	$0.0	**$2.0**
	subtotal ($M)	**$0.0**	**$3.3**	**$1.8**	**$0.0**	**$0.0**	**$5.0**
Mining capital ($M)							
	Mining pre-strip		$20.7	$31.5	$1.8	$0.0	**$54.0**
	Haul road upgrade		$1.8	$0.0	$0.0	$0.0	**$1.8**
	Pit dewatering bores		$0.4	$1.1	$0.0	$0.0	**$1.5**
	Geotechnical: slope monitoring radar		$0.0	$0.5	$0.0	$0.0	**$0.5**
	Two-way radio network upgrade		$0.2	$0.0	$0.0	$0.0	**$0.2**
	subtotal ($M)	**$0.0**	**$23.1**	**$33.1**	**$1.8**	**$0.0**	**$58.0**
Processing capital ($M)							
	Trident plant works		$0.4	$0.0	$0.0	$0.0	**$0.4**
	Other						**$0.0**
	subtotal ($M)	**$0.0**	**$0.4**	**$0.0**	**$0.0**	**$0.0**	**$0.4**
Other capital ($M)							
	Re-approved technical work		$1.0	$0.0	$0.0	$0.0	**$1.0**
	Resource drilling and assaying		$0.9	$0.0	$0.0	$0.0	**$0.9**
	Metallurgical testwork		$0.1	$0.0	$0.0	$0.0	**$0.1**
	subtotal ($M)	**$0.0**	**$1.9**	**$0.0**	**$0.0**	**$0.0**	**$1.9**
	Total ($M)	**$0.0**	**$28.7**	**$34.9**	**$1.8**	**$0.0**	**$65.4**

21.2.2 Sustaining capital costs

Sustaining capital cost provisions amount to $10.4 M over the life of the Project. This is based on an allowance of 5% of the process operating costs in each year from Year 1 to Year 8. Mine operating costs are excluded on the basis that the fleet will be contracted.

21.2.3 Mine closure provisions

A mine closure provision has been estimated internally, although reflecting the itemisation of costs and listed activities identified by external consultants for the Sentinel closure estimate.

A summary of the $6.5 M internal estimate is provided in Table 21-12.

21.2.4 Mining costs

Forward looking Enterprise mining costs have been derived based on an operating performance equation which accounts for:

- contracted mining operations
- the expected use of primarily mid-capacity trucks (180 t), more suited to smaller operating pit phases than those at Sentinel
- haulage along defined ore and waste routes applicable to the current life of mine plan

- excavating/loading costs reflecting the use of compatibly sized hydraulic excavators

Table 21-12 Enterprise closure cost provisions

		($M)
Closure components		
	Enterprise process area	$2.2
	Open pit	$1.8
	Waste dump	$1.0
	ROM pad	$0.3
	Mine services area	$0.0
	Access road	$0.0
	Pit roads	$0.0
	Berms and road- south	$0.0
	Sediment ponds	$0.0
	Dams	$0.2
	Camp	$0.0
	Preliminaries and general (15%)	$0.5
	Subtotal closure costs	**$6.2**
Post-closure components		
	Environmental monitoring	$0.3
	Care and maintenance	$0.0
	Contingencies	$0.0
	Subtotal post-closure costs	$0.4
	Total closure costs	**$6.5**

Assumptions implicit in the equation, and based on FQM operating performance data adapted for contracted mining, are:

- truck fuel consumption costs reflect an assumed fuel cost of $1.00/L and a consumption rate of 440 L/hour
- truck fuel consumption costs reflect engine load factors of 10% when idling (eg, when trucks are being loaded), 20% when unladen and travelling flat, 30% when laden and travelling flat, 20% when unladen and hauling up a ramp, 100% when laden and hauling up a ramp
- a 7 minute allowance for non-travel time, plus estimated times for flat and ramp travel under prevailing speed limitations, resulting in the indicative times listed in Table 21-13
- contracted maintenance costs of $121.50/hour inclusive of major rebuilds and key maintenance overhaul requirements, resulting in the unit costs listed in Table 21-14
- an allowance of $108/hour for contracted labour costs, depreciation and overheads, and resulting in the other unit costs listed in Table 21-14
- a unit excavating/loading cost of $0.86/bcm based on information from the Kansanshi operations reflective of the mix of equipment items and their respective utilisation that would be applicable at Enterprise, and adjusted for contract mining
- a contracted unit drill and blast cost of $2.45/bcm inclusive of consumables
- a 20% loading on load and haul costs to cover the use of ancillary mining equipment

Table 21-13 Estimated truck non-travel and travel times for Enterprise

Item	Units	Value
Non travel time	**minutes**	**7.00**
Travel time, no TA		
unladen, flat	minutes/km	1.50
laden, flat	minutes/km	2.00
subtotal	**minutes/km**	**3.50**
unladen, on ramp	minutes/km	2.40
laden, on ramp	minutes/km	5.00
subtotal	**minutes/km**	**7.40**

Table 21-14 Estimated truck haulage operating costs for Enterprise

Item	Fuel ($/bcm)	Maintenance ($/bcm)	Other ($/bcm)	Total ($/bcm)
During non travel time	**$0.08**	**$0.22**	**$0.20**	**$0.50**
During travel time, no TA				
unladen, flat	$0.03	$0.05	$0.04	$0.12
laden, flat	$0.07	$0.06	$0.06	$0.19
subtotal	**$0.10**	**$0.11**	**$0.10**	**$0.31**
unladen, on ramp	$0.05	$0.08	$0.07	$0.20
laden, on ramp	$0.57	$0.16	$0.14	$0.87
subtotal	**$0.62**	**$0.23**	**$0.21**	**$1.06**

The derived mining cost equation in terms of $/bcm is shown below, composed of the unit costs and haulage coefficients listed in Table 21-15:

The total cost per bcm mined =

- drilling/blasting cost of $2.45/bcm, plus
- loading cost of $1.63/bcm, plus
- flat travel cost of $0.37/bcm/km multiplied by the flat travel distance (in km), plus
- ramp travel cost of $1.28/bcm/km multiplied by the non-TA ramp travel distance (in km)

Table 21-15 Estimated unit mining costs and haulage coefficients for Enterprise

Item	Units	Value
Drilling/blasting costs	**$/bcm**	**$2.45**
Loading costs		
Excavating/loading costs	$/bcm	$0.86
Total hourly cost	$/bcm	$0.50
Ancillary eq't allowance	%	20
subtotal	**$/bcm**	**$1.63**
Hauling costs		
Flat travel		
Total hourly cost	$/bcm	$0.31
Ancillary eq't allowance	%	20
coefficient	**$/bcm/km**	**$0.37**
Ramp travel, no TA		
Total hourly cost	$/bcm	$1.06
Ancillary eq't allowance	%	20
coefficient	**$/bcm/km**	**$1.28**

A preliminary schedule was produced in order to derive forward looking, applicable ore and waste haulage distances, on a bench-by-bench basis. The equation and unit cost/coefficient information above was used to translate the distances into an average ore and waste mining unit cost for every bench in the mine planning model.

The resulting database was able to be plotted such that a regression relationship could be determined, relating load and haul unit costs to mining bench RL, as shown for ore and waste, in Figure 21-3 and Figure 21-4, respectively. A fixed cost of $2.45/bcm must be added to these figures for drill and blast, irrespective of mining bench level.

From these relationships, the overall average drill, blast, load and haul costs are as follows:

- average waste mining cost = $6.41/bcm ($2.39/t)
- average ore mining cost = $6.26/bcm ($2.26/t)

Figure 21-3 Ore load and haul unit cost by bench RL

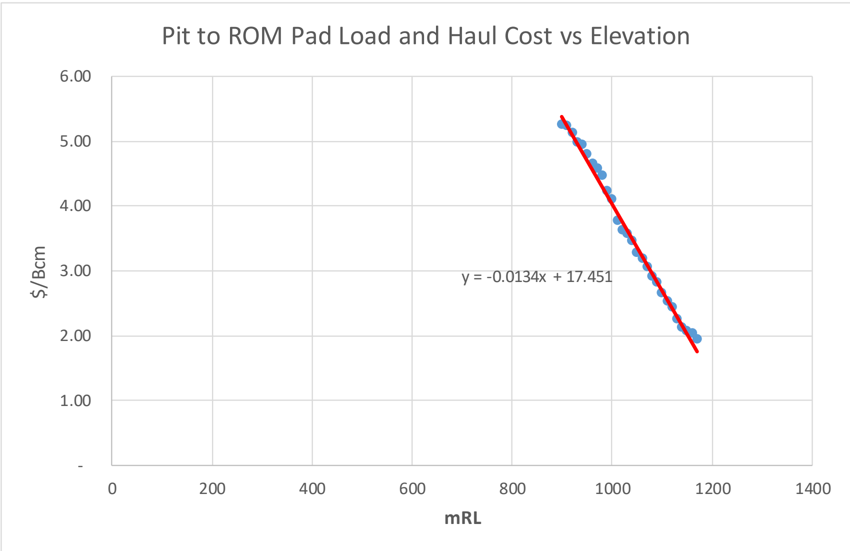
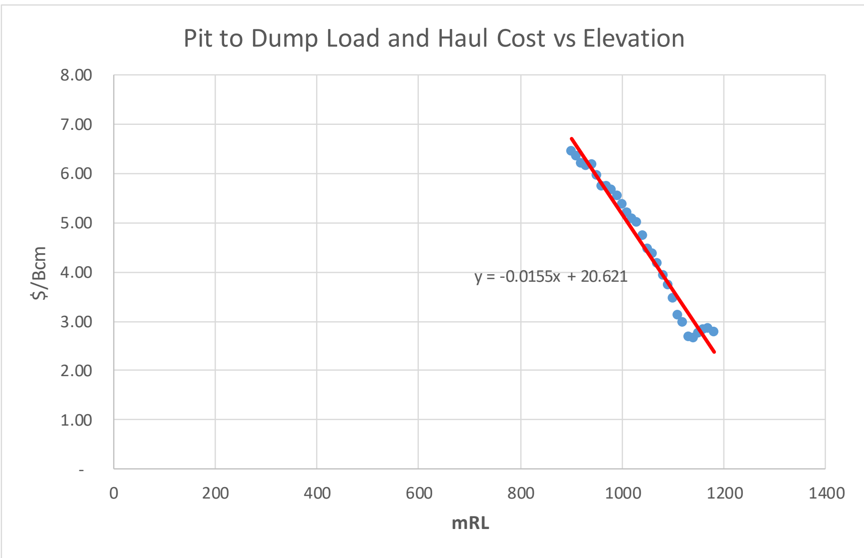


Figure 21-4 Waste load and haul unit cost by bench RL



For the purposes of the cashflow model described in Item 22, the unit cost relationship was applied to the respective total ore and waste mining volumes for each year of the production profile, reflecting the varying bench RLs in each year of the profile. The modelled unit costs as they would apply during the mining pre-strip period were ignored, in place of the capital cost allowance of $54.0 M equating to approximately $4.65/bcm of waste stripped.

21.2.5 Operating costs

The operating costs as listed for the 2015 Technical Report (FQM, May 2015) were reviewed and updated for the 2019 optimisation and cashflow model. The starting point for this update was an estimate of the power consumption and milling steel consumption (Table 21-16), followed by an estimate of the process and maintenance labour headcount (Table 21-17), and finally by an estimate of the processing consumables costs (Table 21-18).

Table 21-19 summarises the overall estimated operating costs. No additional G&A costs are included, on the basis that such overheads would already be covered by the Sentinel operations.

Table 21-16 Estimated power and milling steel consumption for Enterprise

Power consumption	
Power draw (MW)	14.85
Power cost (c/kWh)	8.8
Power cost ($/MWh)	88.0
Annual operating hours	8,000
Annual power draw (MWh)	118,800
Total cost ($pa)	10,454,400
Cost per tonne ore ($/t)	**2.614**
Steel consumption	
Plant throughput (Mtpa)	4.0
Annual operating hours	8,000
Plant throughput (tph)	500
Abrasion index (Ai)	0.16
SAG mill power draw (kW)	4,520
Ball mill power draw (kW)	2,740
Total power draw (kWh/t)	14.52
Steel consumption (kg/t)	0.084
Steel consumption (kg/kWh)	1.221
Annual consumption (tpa)	**4,882.1**

Table 21-17 Estimated process labour numbers and costs for Enterprise

Labour	Operators	Maintenance		
		Mech.	Elect.	Instr.
Primary Crushers, Conveyors, Pebble Crusher	4	4	2	
Milling Circuit	4	4	2	1
Flotation	4	3	2	1
Control Room (additional)	1			
Reagents (additional)	1			
Total (four operating shifts)	**56**	**11**	**6**	**2**
Grand Total	**75**			
Annual cost @$40k each ($pa)	**3,000,000**			

Table 21-18 Estimated consumables cost for Enterprise

Consumables	Addition (g/t)	Consumption (tpa)	Unit cost ($/t)	Total cost ($)	Cost ($/t ore)
Power				**10,454,400**	**2.614**
Labour				**3,000,000**	**0.750**
Reagents					
Frother	15	60.0	3,000	180,000	0.045
Collector	135	540.0	2,000	1,080,000	0.270
Depressant (Guar)	500	2,000.0	1,000	2,000,000	0.500
Lime	300	1,200.0	150	180,000	0.045
Flocculant	50	200.0	4,100	820,000	0.205
			Subtotal	**4,260,000**	**1.065**
Steel					
Steel Balls (SAG & Ball)	1,221	4,882.1	1,400	6,834,876	1.709
Mill Liners				3,281,379	0.820
			Subtotal	**10,116,254**	**2.529**
Other					
Maintenance			**Subtotal**	**6,000,000**	**1.500**

Table 21-19 Summary operating cost estimate for Enterprise

Variable Costs	$/t
Power	2.61
Reagents	1.07
Steel	2.53
subtotal	**6.21**
Fixed Costs	**$/t**
Labour	0.75
Maintenance	1.50
subtotal	**2.25**
Total	**8.46**

Finally, added to the total operating cost in Table 21-19 was an allowance of $1.50/t for surface ore reclaim and road haulage to the processing plant. This equates to an equivalent unit cost estimate of $0.10/tonne/km.

21.2.6 Metal costs

In addition to royalties, the expected Project metal costs will include concentrate road transport charges to Walvis Bay, Namibia and separate ocean freight charges for transport thereon.

As advised by the Company's metals marketing group (MCT), concentrate treatment charges are now typically simplified through the adoption of a single payability percentage which accounts for the separate cost items of treatment and refining charges, plus smelter deductions.

On this basis, Table 21-20 lists projected metal cost estimates for Enterprise.

Table 21-20 Projected Enterprise metal costs

Nickel price		**$/lb Ni** **$/t Ni**	**7.50** **16,535**
Nickel concentrate			
	grade	%Ni	15%
	moisture content	%	9%
Concentrate transport			
	to Walvis Bay	$/wmt	168.00
		$/dmt	184.62
	ocean freight	$/wmt	88.00
		$/dmt	96.70
		$/t Ni	1,875.46
	subtotal	**$/lb Ni**	**0.85**
Concentrate pricing mechanism			
	payability	%	75%
Total TCRCs			
		$/wmt	564.8
		$/dmt	564.2
		$/t Ni	3,765.0
	Subtotal	**$/lb Ni**	**1.71**
Royalty			
	rate	%	5%
	gross	$/t Ni	826.73
	Subtotal	$/lb Ni	**0.38**
	Total metal cost	**$/lb Ni**	**2.93**
	Net metal price	**$/lb Ni**	**4.57**

ITEM 22 ECONOMIC ANALYSES

22.1 Principal assumptions

In accordance with Part 2.3 (1) (c) of the Rules and Policies of the Canadian National Instrument (NI 43-101), the economic analyses set out below do not include Inferred Mineral Resource.

The economic analyses in the form of a pre-tax cashflow model for each of Sentinel and Enterprise are intended to support the Mineral Reserve estimates, and in order to demonstrate a positive cashflow for each production year of mining and processing. The development capital costs, sustaining capital costs and longer term rehabilitation costs are included in the models for completeness. The cashflow models form part of a more comprehensive Project financial model which extends to depreciation and tax etc.

22.1.1 Metal pricing

The annual revenues in the Sentinel cashflow model are calculated using late 2019 consensus average pricing information from a number of banks and financial service companies, as listed in Table 22-1. A long term price of $3.00/lb copper was adopted for the pit optimisations (Item 15.3.2).

Table 22-1 Consensus copper pricing information for cashflow modelling

Pricing Date	2019E ($/lb)	2020E ($/lb)	2021E ($/lb)	2022E ($/lb)	2023E ($/lb)	LT ($/lb)
18/11/2019	2.74	3.00	3.00	3.00	.	3.00
11/11/2019	2.73	2.93	3.13	2.90	2.83	3.25
11/11/2019	2.72	2.88	3.00	3.00	3.00	3.00
8/11/2019	2.68	2.54	2.71	3.11	3.36	2.84
8/11/2019	2.74	2.82	2.72	2.81	.	2.90
2/12/2019	2.80	2.70	2.50	2.15	.	3.00
1/11/2019	2.73	2.75	3.00	3.00	3.15	3.55
15/10/2019	2.70	2.75	3.00	3.25	3.25	3.00
14/10/2019	2.71	2.75	2.80	2.85	2.90	3.00
11/10/2019	2.71	2.88	3.28	.	.	3.10
4/10/2019	2.72	2.88	3.10	3.33	.	3.09
24/07/2019	2.91	2.94	2.94	.	.	2.82
4/06/2019	2.93	3.18	3.40	3.63	3.63	.
2/12/2019	2.72	2.87	2.83	.	.	.
Consensus Average	**2.75**	**2.85**	**2.96**	**3.00**	**3.16**	**3.05**
Minimum (nominal)	2.68	2.54	2.50	2.15	2.83	2.82
Median (nominal)	**2.80**	**2.86**	**2.95**	**2.89**	**3.23**	**3.19**
Maximum (nominal)	2.93	3.18	3.40	3.63	3.63	3.55
Minimum (real)	2.68	2.49	2.40	2.03	2.61	2.82
Median (real)	**2.80**	**2.80**	**2.84**	**2.72**	**2.98**	**3.19**
Maximum (real)	2.93	3.11	3.27	3.42	3.35	3.55

The annual revenues in the Enterprise cashflow model are also calculated using late 2019 consensus median pricing information, as listed in Table 22-2. A long term price of $7.50/lb nickel was adopted for the pit optimisations (Item 15.4.2).

Table 22-2 Consensus nickel pricing information for cashflow modelling

Pricing Date	2019E ($/lb)	2020E ($/lb)	2021E ($/lb)	2022E ($/lb)	2023E ($/lb)	LT ($/lb)
18/11/2019	6.00	6.00	6.50	7.00	.	8.00
11/11/2019	6.94	8.62	7.26	7.14	7.94	7.50
8/11/2019	6.36	7.14	7.43	8.39	8.62	7.53
8/11/2019	6.54	8.22	8.85	7.94	.	6.48
7/11/2019	5.95	5.85	6.00	6.50	6.63	6.70
6/11/2019	6.58	7.94	7.96	8.41	9.11	9.00
1/11/2019	6.68	8.25	8.00	8.00	8.00	8.00
15/10/2019	6.50	7.50	8.00	8.00	8.00	7.50
14/10/2019	6.46	6.88	7.00	7.50	8.00	8.00
11/10/2019	6.57	7.50	7.00	.	.	7.00
4/10/2019	6.57	7.83	8.20	8.30	.	6.92
26/09/2019	6.10	7.50	7.50	8.00	7.50	6.00
26/09/2019	6.42	7.00	7.00	.	.	7.00
24/07/2019	5.56	5.92	7.45	.	.	7.46
4/06/2019	5.90	6.35	6.80	7.71	8.16	.
2/12/2019	6.43	7.52	7.63	.	.	.
Consensus Average	**6.35**	**7.25**	**7.41**	**7.74**	**8.00**	**7.36**
Minimum (nominal)	5.56	5.85	6.00	6.50	6.63	6.00
Median (nominal)	**6.25**	**7.24**	**7.42**	**7.45**	**7.87**	**7.50**
Maximum (nominal)	6.94	8.62	8.85	8.41	9.11	9.00
Minimum (real)	5.56	5.74	5.77	6.13	6.13	6.00
Median (real)	**6.25**	**7.09**	**7.13**	**7.03**	**7.27**	**7.50**
Maximum (real)	6.94	8.45	8.50	7.92	8.42	9.00

22.1.2 Royalties and other levies

Changes to the mining tax regime, announced in September 2018, were implemented by the Zambian government from January 1st 2019, as follows:

- the sliding scale mineral royalty rate on copper was increased by 1.5% to between 5.5% and 7.5% depending on the LME monthly average price
- an 8% royalty rate is applicable if the LME monthly average price is greater than $7,500 per tonne (ie, $3.40/lb) and less than $9,000 per tonne (ie, $4.08/lb)
- a 10% royalty rate is applicable if the LME monthly average price is greater than $9,000 per tonne
- a nickel royalty rate of 5%

Effective 1st June 2016, a 10% export duty was suspended in relation to ores and concentrates for which there are no processing facilities in Zambia (eg, nickel concentrate). In January 2019, a 15% export levy was reimposed on precious metals, including gold, and an import duty of 5% imposed on copper and cobalt concentrates.

22.2 Sentinel

22.2.1 Production schedule

The production schedule forming the basis of the cashflow model is the same as that listed in Table 16-9, Table 16-10 and Table 16-11 in Item 16.

22.2.2 Revenue inputs

In addition to the metal pricing information in Tables 22-1, other revenue related inputs to the Sentinel cashflow model are:

- The overall average copper metal recovery is 90% (after mining dilution, and where metal recoveries for each individual block are calculated according to recovery algorithms that apply to differing ore types).
- For metal in concentrate, which is largely destined for the Company's smelter at Kansanshi (KCS), the copper payability factor is 96.6%.

22.2.3 Capital costs inputs

According to the itemisation provided in Table 21-1, a total Project capital cost of $157.2M was included in the cashflow model, along with an additional $236.9M of immediate sustaining cost provisions for the period 2020 to 2024 (Table 21-2). Thereafter, a 5% of operating costs provision amounting to $251.8M is modelled for continuing Project sustaining costs.

An amount of $78.2M is included for Project closure costs, inclusive of a progressive $40.0 M spent equally over twelve years, and then $38.2 M spent equally over the final three years of Project life.

22.2.4 Operating and metal cost inputs

Figure 21-1 and Figure 21-2 in Item 21 show the incremental ore and waste unit mining cost algorithms applicable by mining RL. These algorithms were applied to the Item 16.1.3 mining production schedule physicals (by mining bench RLs) to yield the average annual unit mining costs listed in Table 22-3. These annual unit mining cost rates were then adopted in the cashflow model.

From the information in Table 21-7, total processing costs (ie, fixed, variable and G&A) were included in the cashflow model, varying from $6.03/t in 2020 to $5.80/t in 2034.

From the information in Table 21-9, annual unit metal costs of $0.60/lb were applied in the cashflow model, inclusive of royalties.

22.2.5 Cashflow model outcomes

The pre-tax cashflow model to support the Mineral Reserve estimate is listed in Table 22-3.

Under the circumstances of a proposed expansion of production at Sentinel, the undiscounted cashflow value of the Project is $7,989.8 M.

The indicative net present value (NPV) from January 2020, at a nominal discount rate of 10%, is $4,356.8 M. At an 8.5% discount rate, the NPV from January 2020 would be $4,713.9 M.

Table 22-3 Sentinel Mineral Reserve undiscounted cashflow model

	UNITS	TOTAL	Q4 2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
MINING (after mining dilution and recovery)																		
Total ore	Mt	**847.9**		63.2	64.4	61.6	63.9	66.1	63.5	69.4	64.6	67.6	47.8	40.6	33.4	60.0	52.3	29.5
Total waste	Mt	**1,633.3**		97.6	98.3	120.5	116.0	118.6	125.5	120.6	125.2	122.3	121.9	120.9	124.6	119.6	88.8	13.1
Total mined	Mt	**2,481.3**		160.8	162.7	182.0	179.8	184.8	189.0	190.0	189.9	189.8	169.7	161.5	158.0	179.6	141.1	42.5
Strip ratio	t : t	**1.9**		1.5	1.5	2.0	1.8	1.8	2.0	1.7	1.9	1.8	2.6	3.0	3.7	2.0	1.7	0.4
Reclaim																		
Active (OTA/OTB) reclaim	Mt	**6.2**		1.8	2.1	0.3	0.0	0.0	0.0	1.5	0.0	0.0	0.4	0.0	0.0	0.0	0.0	0.0
Long Term (WTA/WTB) reclaim	Mt	**78.7**		0.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	20.6	24.9	28.6	2.7	1.3	0.0
Total reclaim	Mt	**84.8**		2.3	2.1	0.3	0.0	0.0	0.0	1.5	0.0	0.0	21.0	24.9	28.6	2.7	1.3	0.0
PIT TO IPCs DIRECT																		
Total ore	Mt	**792.0**		52.0	55.9	61.5	62.0	62.0	62.0	60.5	62.0	62.0	41.0	37.1	33.4	59.3	52.3	28.9
Grade	%	**0.50**		0.54	0.52	0.56	0.53	0.49	0.49	0.51	0.53	0.50	0.42	0.42	0.51	0.39	0.45	0.62
PIT TO STOCKPILE																		
Total ore	Mt	**56.0**		12.4	8.5	0.0	1.9	4.1	1.5	9.0	2.6	5.6	6.8	3.5	0.0	0.0	0.0	0.0
Grade	%	**0.19**		0.27	0.17	0.18	0.20	0.21	0.18	0.17	0.17	0.17	0.16	0.16	0.00	0.00	0.00	0.00
STOCKPILE TO IPCs																		
Total ore	Mt	**84.8**		2.3	2.1	0.3	0.0	0.0	0.0	1.5	0.0	0.0	21.0	24.9	28.6	2.7	1.3	0.0
Grade	%	**0.21**		0.42	0.50	0.38	0.00	0.00	0.29	0.29	0.00	0.00	0.22	0.20	0.16	0.16	0.16	0.00
STOCKPILE BALANCE																		
Total ore	Mt		28.9	38.9	45.4	45.1	46.9	51.1	52.6	60.0	62.6	68.2	54.0	32.6	4.0	1.3	0.0	0.0
Grade	%		0.24	0.24	0.21	0.21	0.21	0.21	0.21	0.20	0.20	0.20	0.18	0.17	0.21	0.31	0.00	0.00
TOTAL FEED TO PLANT (before mining dilution and recovery)																		
Total ore	Mt	**876.8**		54.4	58.0	61.9	62.0	62.0	62.0	62.0	62.0	62.0	62.0	62.0	62.0	62.0	53.7	28.9
Grade	%	**0.47**		0.54	0.52	0.56	0.53	0.49	0.49	0.51	0.53	0.50	0.35	0.33	0.35	0.38	0.44	0.62
Insitu metal	**kt**	**4,127.9**		**291.8**	**301.4**	**346.4**	**330.6**	**303.0**	**306.5**	**315.0**	**330.3**	**310.8**	**217.6**	**204.0**	**217.8**	**234.5**	**237.8**	**180.4**
TOTAL FEED TO PLANT (after mining dilution and recovery)																		
Total ore	Mt	**876.8**		54.4	58.0	61.9	62.0	62.0	62.0	62.0	62.0	62.0	62.0	62.0	62.0	62.0	53.7	28.9
Grade	%	**0.46**		0.52	0.50	0.54	0.52	0.47	0.48	0.49	0.52	0.49	0.34	0.32	0.34	0.37	0.43	0.61
Insitu metal	**kt**	**4,001.7**		**280.1**	**292.4**	**336.0**	**320.7**	**293.9**	**297.3**	**305.6**	**320.4**	**301.5**	**211.1**	**197.8**	**211.3**	**227.6**	**230.6**	**175.3**
AVERAGE RECOVERIES																		
	%	**90%**		**87%**	**89%**	**90%**	**91%**	**89%**	**90%**	**91%**	**91%**	**90%**	**87%**	**87%**	**90%**	**90%**	**91%**	**91%**
METAL RECOVERED																		
Rec. metal	kt	**3,592.6**		245.1	259.2	302.4	292.4	261.2	268.7	277.3	291.5	272.6	182.8	173.0	191.1	205.2	210.2	160.0
	Mlb	**7,920.3**		**540.3**	**571.4**	**666.7**	**644.6**	**575.8**	**592.3**	**611.3**	**642.6**	**600.9**	**403.1**	**381.5**	**421.2**	**452.5**	**463.3**	**352.8**
CONCENTRATE GRADE																		
26.5%	%	**26.5%**		26.5%	26.5%	26.5%	26.5%	26.5%	26.5%	26.5%	26.5%	26.5%	26.5%	26.5%	26.5%	26.5%	26.5%	26.5%
METAL PAYABLE																		
96.55%	**kt**	**3,468.6**		**236.6**	**250.2**	**292.0**	**282.3**	**252.1**	**259.4**	**267.7**	**281.4**	**263.2**	**176.5**	**167.1**	**184.5**	**198.2**	**202.9**	**154.5**
	***10^{6} lbs**	**7,646.9**		**521.7**	**551.6**	**643.7**	**622.4**	**555.9**	**571.9**	**590.2**	**620.4**	**580.1**	**389.2**	**368.3**	**406.7**	**436.9**	**447.3**	**340.6**
GROSS REVENUE																		
Cu \$/t 3.00	**\$M**	**\$22,940.7**		**\$1,565.0**	**\$1,654.9**	**\$1,931.2**	**\$1,867.1**	**\$1,667.7**	**\$1,715.6**	**\$1,770.7**	**\$1,861.1**	**\$1,740.4**	**\$1,167.6**	**\$1,105.0**	**\$1,220.1**	**\$1,310.6**	**\$1,342.0**	**\$1,021.8**
CAPITAL COSTS																		
Mining capital	\$M	**\$111.7**		\$33.5	\$78.2	\$0.0	\$0.0	\$0.0										
Processing capital	\$M	**\$41.8**		\$3.8	\$2.0	\$12.0	\$2.0	\$22.0										
Other capital	\$M	**\$3.7**		\$2.7	\$1.0	\$0.0	\$0.0	\$0.0										
Closure cost provisions	\$M	**\$78.2**		\$3.3	\$3.3	\$3.3	\$3.3	\$3.3	\$3.3	\$3.3	\$3.3	\$3.3	\$3.3	\$3.3	\$3.3	\$12.7	\$12.7	\$12.7
subtotal	**\$M**	**\$235.3**		**\$43.3**	**\$84.5**	**\$15.3**	**\$5.3**	**\$25.3**	**\$3.3**	**\$3.3**	**\$3.3**	**\$3.3**	**\$3.3**	**\$3.3**	**\$3.3**	**\$12.7**	**\$12.7**	**\$12.7**
SUSTAINING CAPITAL																		
Mining sustaining, initial	\$M	**\$185.6**		\$47.8	\$42.5	\$38.1	\$39.1	\$18.1										
Process sustaining, initial	\$M	**\$3.9**		\$0.0	\$1.0	\$2.4	\$0.5	\$0.0										
Other sustaining, initial	\$M	**\$62.3**		\$11.4	\$27.1	\$7.5	\$10.1	\$6.2										
Sustaining, ongoing	\$M	**\$237.1**							\$34.2	\$35.1	\$34.8	\$34.6	\$32.8	\$32.4	\$33.1			
subtotal	**\$M**	**\$488.9**		**\$59.2**	**\$70.6**	**\$48.0**	**\$49.7**	**\$24.3**	**\$34.2**	**\$35.1**	**\$34.8**	**\$34.6**	**\$32.8**	**\$32.4**	**\$33.1**	**\$0.0**	**\$0.0**	**\$0.0**
OPERATING COSTS																		
Mining:																		
Ore \$/t mined	\$/t	**\$1.66**		\$1.60	\$1.58	\$1.68	\$1.66	\$1.59	\$1.53	\$1.63	\$1.68	\$1.70	\$1.61	\$1.73	\$1.77	\$1.63	\$1.82	\$2.04
	\$M	**\$1,409.4**		**\$103.2**	**\$101.8**	**\$103.2**	**\$106.2**	**\$105.1**	**\$97.0**	**\$113.0**	**\$108.3**	**\$114.6**	**\$77.1**	**\$70.0**	**\$59.0**	**\$96.6**	**\$95.3**	**\$59.1**
Waste \$/t mined	\$/t	**\$1.86**		\$1.81	\$1.81	\$1.86	\$1.83	\$1.84	\$1.81	\$1.90	\$1.82	\$1.79	\$1.81	\$1.81	\$1.96	\$1.95	\$2.10	\$2.29
	\$M	**\$3,044.6**		**\$176.5**	**\$177.8**	**\$223.8**	**\$212.3**	**\$218.5**	**\$226.8**	**\$229.6**	**\$228.4**	**\$218.7**	**\$220.2**	**\$218.7**	**\$243.7**	**\$233.0**	**\$186.7**	**\$29.9**
	\$/t	**\$1.80**		\$1.74	\$1.72	\$1.80	\$1.77	\$1.75	\$1.71	\$1.80	\$1.77	\$1.76	\$1.75	\$1.79	\$1.92	\$1.83	\$2.00	\$2.09
subtotal mining	**\$M**	**\$4,454.0**		**\$279.7**	**\$279.6**	**\$327.0**	**\$318.5**	**\$323.6**	**\$323.7**	**\$342.6**	**\$336.7**	**\$333.3**	**\$297.3**	**\$288.7**	**\$302.7**	**\$329.5**	**\$282.1**	**\$89.0**
Processing (incl. G&A):																		
	\$/t	**\$5.83**		\$6.03	\$6.03	\$5.80	\$5.80	\$5.80	\$5.80	\$5.80	\$5.80	\$5.80	\$5.80	\$5.80	\$5.80	\$5.80	\$5.80	\$5.80
subtotal processing	**\$M**	**\$5,111.3**		**\$327.9**	**\$349.7**	**\$358.8**	**\$359.6**	**\$359.6**	**\$359.6**	**\$359.6**	**\$359.6**	**\$359.6**	**\$359.6**	**\$359.4**	**\$359.6**	**\$359.6**	**\$311.2**	**\$167.9**
Stockpile reclaim allow:																		
	\$/t	**\$1.00**		\$0.99	\$0.99	\$0.99	\$0.00	\$0.00	\$0.99	\$0.99	\$0.99	\$0.00	\$1.02	\$1.02	\$0.99	\$0.98	\$0.98	\$0.00
subtotal processing	**\$M**	**\$85.1**		**\$2.3**	**\$2.1**	**\$0.3**	**\$0.0**	**\$0.0**	**\$0.0**	**\$1.5**	**\$0.0**	**\$0.0**	**\$21.4**	**\$25.3**	**\$28.2**	**\$2.6**	**\$1.3**	**\$0.0**
subtotal operating costs	**\$M**	**\$9,650.5**		**\$609.9**	**\$631.4**	**\$686.1**	**\$678.1**	**\$683.2**	**\$683.3**	**\$703.7**	**\$696.3**	**\$692.9**	**\$678.3**	**\$673.4**	**\$690.5**	**\$691.8**	**\$594.6**	**\$256.9**
METAL COSTS																		
	\$/lb	**\$0.37**		\$0.37	\$0.37	\$0.37	\$0.37	\$0.37	\$0.37	\$0.37	\$0.37	\$0.37	\$0.37	\$0.37	\$0.37	\$0.37	\$0.37	\$0.37
TCRCs	**\$M**	**\$2,855.6**		**\$194.8**	**\$206.0**	**\$240.4**	**\$232.4**	**\$207.6**	**\$213.6**	**\$220.4**	**\$231.7**	**\$216.6**	**\$145.3**	**\$137.5**	**\$151.9**	**\$163.1**	**\$167.0**	**\$127.2**
	\$/lb	**\$0.22**		\$0.23	\$0.23	\$0.23	\$0.23	\$0.23	\$0.23	\$0.23	\$0.23	\$0.23	\$0.23	\$0.23	\$0.23	\$0.23	\$0.23	\$0.23
royalty (7.5%)	**\$M**	**\$1,720.6**		**\$117.4**	**\$124.1**	**\$144.8**	**\$140.0**	**\$125.1**	**\$128.7**	**\$132.8**	**\$139.6**	**\$130.5**	**\$87.6**	**\$82.9**	**\$91.5**	**\$98.3**	**\$100.6**	**\$76.6**
	\$/lb	**\$0.60**		\$0.60	\$0.60	\$0.60	\$0.60	\$0.60	\$0.60	\$0.60	\$0.60	\$0.60	\$0.60	\$0.60	\$0.60	\$0.60	\$0.60	\$0.60
subtotal metal costs	**\$M**	**\$4,576.2**		**\$312.2**	**\$330.1**	**\$385.2**	**\$372.4**	**\$332.7**	**\$342.2**	**\$353.2**	**\$371.3**	**\$347.2**	**\$232.9**	**\$220.4**	**\$243.4**	**\$261.4**	**\$267.7**	**\$203.8**
UNDISCOUNTED CASHFLOW	**\$M**	**\$7,989.8**	**\$0.0**	**\$540.4**	**\$538.3**	**\$796.4**	**\$761.5**	**\$602.2**	**\$652.5**	**\$675.3**	**\$755.4**	**\$662.3**	**\$220.2**	**\$175.4**	**\$249.7**	**\$344.7**	**\$466.9**	**\$548.4**

22.2.6 Cashflow sensitivity analysis

A basic sensitivity analysis was completed for several cashflow model variables (ie, varied independently) with the results listed in Table 22-4. Whilst copper price is an obviously sensitive variable, and comparable in impact with the variance in overall processing recovery, the analysis highlights the particular sensitivity of the mining and processing costs (including G&A) relative to the capital and sustaining costs.

Table 22-4 Sentinel Mineral Reserve cashflow model sensitivity analysis

Variable		Cashflow ($M)	Delta (%)
Base Case		**$7,989.8**	**100%**
Metal Recovery	+5%	$8,908.1	111%
	-5%	$7,071.6	89%
Gross Revenue	+5%	$9,136.9	114%
	-5%	$6,842.8	86%
Capital Costs	+10%	$7,966.3	100%
	-10%	$8,013.4	100%
Sustaining Costs	+10%	$7,941.0	99%
	-10%	$8,038.7	101%
Mining Costs	+10%	$7,533.3	94%
	-10%	$8,446.4	106%
Process & GA Costs	+10%	$7,466.1	93%
	-10%	$8,513.6	107%
Metal Costs	+10%	$7,704.3	96%
	-10%	$8,275.4	104%

22.3 Enterprise

22.3.1 Production schedule

The production schedule forming the basis of the cashflow model is the same as that listed in Table 16-24 in Item 16.

22.3.2 Revenue inputs

In addition to the metal pricing information in Table 22-2, other revenue related inputs to the Enterprise cashflow model are:

- The overall average nickel metal recovery is 82% (after mining dilution, and where metal recoveries for each individual block are calculated according to fixed values that apply to the two differing ore types).
- For metal in concentrate, which is envisaged for delivery to smelting/refining facilities outside of Zambia, the copper payability factor is 75%.

22.3.3 Capital cost inputs

According to the itemisation provided in Table 21-10 a total Project capital cost of \$65.4 M was included in the cashflow model, along with an additional \$10.4 M for sustaining cost provisions equivalent to 5% of process operating costs.

An amount of \$6.5 M is included for Project closure costs in the final year of Project life.

22.3.4 Operating and metal cost inputs

Figure 21-3 and Figure 21-4 in Item 21 show the incremental ore and waste unit mining cost algorithms applicable by mining RL. These algorithms were applied to the Item 16.2.3 mining production schedule physicals (by mining bench RLs) to yield average annual unit mining costs which were then applied to the cashflow model physicals. The overall average ore mining cost is \$2.26/t mined, whilst the overall average waste mining cost is \$2.19/t mined. These figures exclude the cost of pre-strip waste mining.

From the information in Table 21-18 , total processing costs of \$8.26/t (ie, fixed, variable and <u>no</u> G&A allowance) were included in the cashflow model for every year of processing. On top of this, \$1.50/t was included to cover surface reclaim of mined ore and road haulage to the processing plant.

From the information in Table 21-19, annual unit metal costs of \$0.85/lb and \$0.38/t were applied in the cashflow model for concentrate transport and royalty charges, respectively. The cost of the TCRCs is \$1.71/lb reflective of the 75% payability on metal recovered.

22.3.5 Cashflow model outcomes

The pre-tax cashflow model to support the Mineral Reserve estimate is listed in Table 22-5. Assuming a continuous mining and processing period from the pre-strip through to the final year of processing, the undiscounted cashflow value of the Project is \$1,698.9M, including capital expenditure allocated for the pre-strip works.

Table 22-5 Enterprise Mineral Reserve undiscounted cashflow model

	UNITS	TOTAL	PRESTRIP / Year 1		Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
MINING (after mining dilution and recovery)														
Total ore	Mbcm	**12.3**	0.01	0.4	1.3	0.8	1.1	1.3	1.1	1.4	1.4	1.4	1.4	0.7
	Mt	**34.7**	0.01	1.0	3.3	2.3	3.2	3.9	3.1	4.0	4.0	4.0	4.0	2.1
Total waste (incl. min. saprolite)	Mbcm	**107.4**	4.8	6.7	10.4	15.1	15.9	15.7	14.3	9.4	6.1	5.1	3.0	0.8
	Mt	**286.7**	11.9	15.8	24.7	38.9	43.2	42.3	39.6	26.8	17.6	14.7	8.7	2.4
Total mined	**Mbcm**	**119.7**	**4.8**	**7.2**	**11.7**	**15.9**	**17.0**	**17.0**	**15.4**	**10.8**	**7.5**	**6.5**	**4.4**	**1.6**
	Mt	**321.4**	**11.9**	**16.8**	**28.0**	**41.2**	**46.4**	**46.2**	**42.7**	**30.8**	**21.6**	**18.6**	**12.6**	**4.6**
Strip ratio	**bcm:bcm**	**8.7**	872.7	16.2	8.3	18.7	14.4	11.6	13.4	6.8	4.4	3.7	2.2	1.1
TOTAL FEED TO PLANT (after mining dilution and recovery)														
Non-primary ore	Mt	**5.8**	0.0	1.0	3.1	1.0	0.6	0.1	0.0	0.0	0.1	0.0	0.0	0.0
	% Ni	**0.79**	0.00	0.50	0.89	1.11	0.33	0.39	0.43	0.71	0.81	0.15	0.00	0.00
Primary ore	Mt	**28.9**	0.0	0.0	0.2	1.3	2.6	3.8	3.1	4.0	3.9	4.0	4.0	2.1
	% Ni	**1.03**	0.00	0.00	1.23	1.07	1.69	1.31	1.03	0.71	1.13	1.02	0.70	0.76
Total ore	**Mt**	**34.7**	**0.0**	**1.0**	**3.3**	**2.3**	**3.2**	**3.9**	**3.1**	**4.0**	**4.0**	**4.0**	**4.0**	**2.1**
Grade	% Ni	**0.99**	**0.00**	**0.50**	**0.90**	**1.09**	**1.44**	**1.29**	**1.03**	**0.71**	**1.13**	**1.02**	**0.70**	**0.76**
Insitu metal	**Ni kt**	**344.5**	**0.0**	**4.8**	**29.8**	**25.0**	**45.8**	**49.8**	**31.8**	**28.3**	**45.0**	**40.5**	**27.6**	**16.3**
METAL RECOVERD														
Non-primary ore (60%)	Ni kt	**27.7**	0.0	2.9	16.6	6.5	1.1	0.2	0.0	0.1	0.3	0.0	0.0	0.0
Primary ore (85%)	Ni kt	**253.6**	0.0	0.0	1.8	12.1	37.3	42.1	27.0	23.9	37.8	34.4	23.4	13.8
Total recovered metal	**Ni kt**	**281.3**	**0.0**	**2.9**	**18.4**	**18.5**	**38.4**	**42.3**	**27.0**	**24.0**	**38.1**	**34.4**	**23.4**	**13.8**
GROSS REVENUE														
At $7.50/lb Ni	**$M**	**$4,651.5**	**$0.0**	**$47.4**	**$304.1**	**$306.4**	**$635.5**	**$698.9**	**$446.8**	**$397.2**	**$629.8**	**$569.5**	**$387.3**	**$228.5**
CAPITAL COSTS														
Mining prestrip	$M	**$54.0**	$21.7	$32.3										
Site capital works	$M	**$5.0**	$3.3	$1.8										
Mining capital	$M	**$4.0**	$2.4	$1.6										
Processing capital	$M	**$0.4**	$0.4	$0.0										
Other capital	$M	**$1.9**	$1.9	$0.0										
Closure cost provisions	$M	**$6.5**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$6.5
Total development capital	**$M**	**$71.9**	**$29.7**	**$35.7**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$6.5**
SUSTAINING CAPITAL COSTS														
Mine sustaining	$M	**$0.0**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Process sustaining	$M	**$10.4**	$0.0	$0.4	$1.4	$1.0	$1.3	$1.6	$1.3	$1.7	$1.7	$0.0	$0.0	$0.0
Other sustaining	$M	**$0.0**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Total sustaining capital	**$M**	**$10.4**	**$0.0**	**$0.4**	**$1.4**	**$1.0**	**$1.3**	**$1.6**	**$1.3**	**$1.7**	**$1.7**	**$0.0**	**$0.0**	**$0.0**
OPERATING COSTS														
Mining ore	$M	**$78.5**	$0.0	$2.0	$6.5	$4.5	$6.5	$8.2	$6.8	$9.1	$9.5	$9.9	$9.9	$5.7
Mining waste	$M	**$627.9**	$0.0	$0.0	$58.1	$88.3	$97.9	$99.4	$96.3	$67.5	$46.4	$40.8	$25.5	$7.8
Processing variable costs	$M	**$215.4**	$0.0	$5.9	$20.4	$14.3	$19.7	$24.0	$19.1	$24.7	$24.8	$24.7	$24.6	$13.2
Processing fixed costs	$M	**$78.1**	$0.0	$2.2	$7.4	$5.2	$7.1	$8.7	$6.9	$9.0	$9.0	$9.0	$8.9	$4.8
Reclaim and overland ore haul	$M	**$52.0**	$0.0	$1.4	$4.9	$3.4	$4.8	$5.8	$4.6	$6.0	$6.0	$6.0	$5.9	$3.2
Total operating costs	**$M**	**$1,051.9**	**$0.0**	**$11.5**	**$97.4**	**$115.7**	**$136.0**	**$146.0**	**$133.7**	**$116.1**	**$95.6**	**$90.3**	**$74.8**	**$34.8**
METAL COSTS (TCRCs AND ROYALTIES)														
Nickel concentrate	kt	**1,875.4**	0.0	19.1	122.6	123.5	256.2	281.8	180.1	160.2	253.9	229.6	156.1	92.1
	% Ni	**15%**	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%
Concentrate transport	$M	**$527.6**	$0.0	$5.4	$34.5	$34.8	$72.1	$79.3	$50.7	$45.1	$71.4	$64.6	$43.9	$25.9
TCRCs (75% payability)	$M	**$1,058.2**	$0.0	$10.8	$69.2	$69.7	$144.6	$159.0	$101.6	$90.4	$143.3	$129.6	$88.1	$52.0
Royalties	$M	**$232.6**	$0.0	$2.4	$15.2	$15.3	$31.8	$34.9	$22.3	$19.9	$31.5	$28.5	$19.4	$11.4
Total metal costs	**$M**	**$1,818.4**	**$0.0**	**$18.5**	**$118.9**	**$119.8**	**$248.4**	**$273.2**	**$174.7**	**$155.3**	**$246.2**	**$222.6**	**$151.4**	**$89.3**
CASHFLOW	**$M**	**$1,698.9**	**-$29.7**	**-$18.7**	**$86.4**	**$70.0**	**$249.7**	**$278.1**	**$137.2**	**$124.1**	**$286.3**	**$256.6**	**$161.1**	**$97.8**

Under these assumed continuous operating circumstances, the notional net present value (NPV) from 2020, at a nominal discount rate of 10%, is $818.5 M. At an 8.5% discount rate, the notional NPV would be $906.8 M.

22.3.6 Cashflow sensitivity analysis

A basic sensitivity analysis was completed for several cashflow model variables (ie, varied independently) with the results listed in Table 22-6. Whilst nickel price is an obviously sensitive variable, the analysis highlights the particular sensitivity of the metal costs (as a function of the 75% payability) relative to the capital and operating costs.

Table 22-6 Enterprise Mineral Reserve cashflow model sensitivity analysis

Variable		Cashflow	
		($M)	Delta (%)
Base Case		**$1,698.9**	100%
Metal Recovery	+5%	$1,840.5	108%
	-5%	$1,557.2	92%
Gross Revenue	+5%	$1,931.4	114%
	-5%	$1,466.3	86%
Capital Costs	+10%	$1,691.7	100%
	-10%	$1,706.0	100%
Operating Costs	+10%	$1,593.7	94%
	-10%	$1,804.0	106%
Metal Costs	+10%	$1,517.0	89%
	-10%	$1,880.7	111%

ITEM 23 ADJACENT PROPERTIES

There are no adjacent properties or relevant information pertaining to adjacent properties that are material to this Technical Report.

ITEM 24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant information or explanation required to make this Technical Report understandable and not misleading.

ITEM 25 INTERPRETATIONS AND CONCLUSIONS

25.1 Mineral Resource modelling and estimation

25.1.1 Sentinel

The conclusions relevant to this Technical Report are as follows:

- Sentinel mineralization is associated with strongly developed phyllite host rock foliation and zones of folding. The mineralization occurs as a series of continuous horizons which are often separated and influenced by several sub-parallel structural detachment zones.
- Diamond drilled core samples have good coverage across the extents of Sentinel mineralization and its respective domains. Average core recovery is above 93%.
- There is sufficient data to adequately define the limits and extents of the Sentinel mineralization.
- Reverse circulation drilling now covers a significant volume of the Sentinel deposit providing valuable data for estimates and reconciliation. Comparisons between wide spaced diamond drill estimates and close spaced RC estimates suggest that mineralization volumes may be overstated by around 5%.
- QAQC practices adequately demonstrate that diamond core and RC chip sampling and analysis are of acceptable quality and are suitable for use in Mineral Resource estimation, supporting the applied Mineral Resource classification.
- There is sufficient density data for applying robust mean values per geological domain.
- Detail geology, from close spaced drilling and in-pit mapping has supported good geology models relevant to the scale of mining.
- The applied geological domains relevant to high, medium and low grade domains, as well as the separation of non-primary sulphides from primary sulphides, have provided grade distributions suitable for ordinary kriging and post processing by localised uniform conditioning.
- Infill grade control sampling will continue to serve to improve domain definition and short range grade continuity. In addition, close spaced RC data has increased accuracy in the position, shape, volume and grades of mineralization for mining.
- Grade estimates were validated visually in cross section, with comparative statistics and by using swath plots. Estimates validate well with the input data used and support the applied Mineral Resource classification.
- Sentinel Mineral Resource estimate classification has reduced Measured and Indicated Mineral Resource volumes by around 6% with marginal grade increases.
- Inferred Mineral Resources have been removed from areas no longer economically viable. The inferred resources removed are distal to the ultimate pit shells and are located in areas of narrower, less continuous and lower grade mineralization.

25.1.2 Enterprise

The conclusions relevant to this Technical Report are as follows:

- Nickel mineralization is hydrothermal, and appears to be related to a series of deep seated structures that have an extensional architecture.

- Improved structural and lithological framework from interpretation of multi-element geochemistry has provided an improved framework to control the Main deposit estimates.
- The Mineral Resource estimate has focussed on the Main deposit with no new data or updates to the South West deposit.
- The Main deposit estimate has used dynamic anisotropy to improve definition of ore volumes as well as to support improved sample selection during grade estimation.
- The QAQC programme implemented by FQM supports confidence in drilled sample assay results being representative of in-situ mineralization.
- There is sufficient density data within fresh material for both Main and SW deposits. However, there is insufficient density data for saprolite and saprock in SW due to the density sampling difficulty of highly friable and porous material.
- Diamond core recovery for Enterprise was good, with the shallower PQ drilled core having lowest average recoveries (75%). Core recovery data shows no recognisable relationship between grade and sample recovery.
- The estimated model was validated statistically by comparing mean grades, visually, by viewing cross sections and using swaths plots to validate local variability. Validation methods support the Enterprise estimates as representative of the input sample data and prevailing mineralization.
- Mineral Resource estimates were largely unchanged apart from the removal of Inferred Mineral resources no longer economically viable.

25.2 Mine planning and Mineral Reserve estimation

The Mineral Reserve estimates for Sentinel and Enterprise are the product of a thorough and conventional process reflecting detailed ultimate pit designs constrained by appropriate optimal pit shells. Volume comparisons between the design ultimate pits and the corresponding pit shells indicate acceptable minor differences.

The optimisation process incorporates the best available information, including latest geotechnical information and updated mining, processing and metal costs.

The Sentinel pit designs take account of a longer-term IPCC concept and incorporate crusher pocket layouts, haulage/tipping access and suitable in-pit conveyor routes. Waste dumps and ore stockpiles have been included into the site layout plan and haulage simulations have been undertaken to provide updated incremental mining costs reflecting haulage to crusher positions and waste haulage under trolley-assist.

A detailed production schedule has been prepared for the Sentinel life of mine, along with a detailed waste dumping schedule which satisfies ARD management requirements and allows partial in-pit backfilling in the latter half of the mine life.

The Enterprise pit designs have also been updated, but with minimal difference relative to the 2015 Mineral Reserve estimate (FQM, May 2015). Detailed phase designs been completed and haulage simulations have been undertaken to provide updated incremental mining costs reflecting haulage to a short-term surface ROM pad and to a surface waste dump.

A detailed production schedule has also been prepared for the Enterprise life of mine. A high-level waste dumping schedule has been prepared, which based on current geochemical information, does not require the level of material separation and management as required at Sentinel.

In the opinion of Michael Lawlor (QP), therefore, the Mineral Reserve estimates reflect achievable mining plans and production sequencing, taking into account staged mining progression and reasonable total material movement profiles (and hence equipment usage). In the case of the Sentinel Pit, the long term mine plan takes account of proposed future in-pit crusher relocations and trolley-assisted waste haulage routes.

In the Sentinel production schedule, efforts have focussed on minimising the size of long-term stockpiles. Whilst sizeable stockpiles are generally inconsistent with an IPCC concept, the Sentinel layout allows some ore to be stockpiled and reclaimed later into adjacent near-surface crushers. At Enterprise there will be no long-term stockpiles and mined ore will be hauled direct to the processing plant.

The following information relates to risks and uncertainties around the Mineral Reserve estimates.

25.2.1 Mining

There is considered to be minimal risk attributable to the mining method and to the primary equipment in use at Sentinel and proposed for Enterprise. The methods and equipment are conventional and suitable for large-scale and mid-scale bulk mining operations, respectively.

25.2.2 Water management

Large water storage dams are located upstream of Sentinel and Enterprise. Each of these dams is an engineered structure with a core trench, select fill embankments and an impervious core.

In addition to these large dams, are a number of smaller downstream dams preventing inflow to the pits from lesser tributaries in the catchment area adjacent to the mine. These are also engineered structures with core trenches, select fill embankments and impervious cores. For the dam that exists at Sentinel, water is being pumped back to the Musangezhi Dam, with the intention of minimising water retention and obviating the risk of breaching and inundating the pit. This is also the intent for the dams at Enterprise.

As mining has progressed to the east into a new mining phase at Sentinel, surface dams have already been constructed to prevent inflow from the south into the phase workings. A new surface dam to replace the one described above, is to be constructed in the 2020 dry season to enable the phase to be developed across its entire footprint.

25.2.3 Environmental and social

To the Company’s knowledge, the Trident Project is not considered by any applicable environmental regulatory authority to be a risk to the environment and the continued development of the Project.

ITEM 26 RECOMMENDATIONS

26.1 Mineral Resource recommendations

26.1.1 Sentinel

Recommendations relevant to the Sentinel Mineral Resource estimate are as follows:

- Mining of Sentinel mineralization should continue to be supported with close spaced RC grade control drilling.
- QAQC protocols should be reviewed for optimal insert rates and use of representative CRMs. The frequency of umpire sample dispatches should be increased.
- Adequate lead time for RC drilling coverage of the eastern areas of the Sentinel deposit outcrop should be considered in order to best define the horizons of weathering and oxidation.
- Mine to Mill reconciliation should continue in order to provide continuous feedback of estimate performance.
- In-pit geology mapping with 3D modelling should continue with a focus on providing improved controls on mineralization.

26.1.2 Enterprise

Recommendations relevant to the Enterprise Mineral Resource estimate are as follows:

- Multi-element analysis of the SW deposit core samples should continue in order to support geochemical interpretation of lithology and structure.
- 3D geology modelling, as per the Main deposit should be completed for the SW deposit.
- Drill grid spacing studies should be completed in order to guide reverse circulation grade control drilling.
- Rate of oxidation sample analysis should be completed in order to understand how the respective nickel minerals are likely to behave during mining and stockpiling.

26.2 Mineral Reserve recommendations

26.2.1 Sentinel

Recommendations in respect of the Sentinel Mineral Reserve estimate are as follows:

- The 2015 Technical Report (FQM, May 2015) made mention of the need to dispense with the fixed processing recovery values then in use and move towards the determination and use of variable relationships reflecting operational performance. This important step has since been implemented and such relationships have now been used to model recovery in the current life of mine planning and Mineral Reserve process. An obvious recommendation is that the relationships continue to be developed and refined for use in ongoing mine planning.
- The 2015 Technical Report (FQM, May 2015) also made mention of the need to review and update geotechnical design criteria. The current pit optimisation and mine design have certainly made use of updated geotechnical information and as such, it is recommended that routine review and update remains ongoing.

- Phase designs for the Sentinel Pit have changed since 2015, as has the concepts for proposed future IPCC layouts. Planning around this important aspect must remain an iterative process to assess and continually re-evaluate optimum crusher locations and the timeframe and duration for their relocation.
- Item 26.1.1 mentions Mine to Mill reconciliations, and continuous improvement in this regard is endorsed. In the context of “unplanned” mining dilution and recovery, continuing efforts are required to validate and improve the accuracy of the substantial dispatch and tracking data records, analyse the trends and rectify mining practices where appropriate.

26.2.2 Enterprise

Recommendations in respect of the Enterprise Mineral Reserve estimate are as follows:

- The Enterprise SW deposit does not currently optimise and convert to Mineral Reserve, on the basis of the defined saprolitic nickel mineralisation (deemed unrecoverable in the processing plant) and the limited tonnage of non-primary nickel mineralisation (which is recoverable). If the recommended geochemical work mentioned in Item 26.1.2 should change the interpreted mineralization volumes, then a re-optimisation is recommended.
- The assigned fixed processing recovery value for Enterprise primary ore has been reduced from 90% as documented in the 2015 Technical Report (FQM, May 2015) to 85% for use in the current life of mine planning and Mineral Reserve process. Testwork remains in progress on metallurgical samples recently collected, and as such it is recommended that updated information, when available, should be used to review the life of mine plan and production schedule.
- Beyond the initial phases of mining there is a waste stripping burden to access deeper ore in the later cutbacks. In conjunction with an improved metal price outlook, improved primary ore recovery projections arising from the testwork in progress would enhance the value of the later pit cutbacks. It is recommended that this aspect should be reviewed in the context of the processing recovery that is recommended when the testwork programme is completed.
- Since 2015, geotechnical data gathering has continued and pit slope design criteria has been updated accordingly. New information has been used for the optimisation and detailed mine design work that is a basis of the current Mineral Reserve update. As the starter pit is developed and batter exposures become available for mapping, it is recommended that the geotechnical design criteria continues to be reviewed and updated.
- Since 2015, groundwater drilling and investigations have continued, and modelling and analysis of potential pit inflows is currently in progress. It is recommended that the pit designs and dewatering costs (as applicable) be reviewed and updated when the information becomes available.

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 "Trident Project, North West Province, Zambia, NI 43-101 Technical Report" dated effective 30th March 2020 (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 Fellow Member of the Australian Institute of Geoscientists (FAIG).

5. I have worked as a geologist for a total of thirty years since my graduation from university. I have gained over fifteen years experience in production geology, and over five years of exploration management of precious, base metal and copper deposits. Over 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 Trident property described in the Technical Report in August 2019.

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 3 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 mineralization 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 2020 at West Perth, Western Australia, Australia.

David Gray

Michael Lawlor
First Quantum Minerals Ltd
24 Outram St, West Perth, Western Australia, 6005
Tel +61 8 9346 0100; mike.lawlor@fqml.com

I, Michael Lawlor, do hereby certify that:

1. I am a Consultant Mining Engineer employed by First Quantum Minerals Ltd.

2. This certificate applies to the technical report entitled "Trident Project, North West Province, Zambia, NI 43-101 Technical Report" dated effective 30th March 2020 (the "Technical Report").

3. I am a professional mining engineer having graduated with an undergraduate degree of Bachelor of Engineering (Honours) from the Western Australian School of Mines in 1986. In addition, I have obtained a Master of Engineering Science degree from the James Cook University of North Queensland (1993), and subsequent Graduate Certificates in Mineral Economics and Project Management from Curtin University (Western Australia).

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

5. I have worked as mining and geotechnical engineer for a period in excess of thirty years since my graduation from university. Within the last ten years I have held senior technical management positions in copper mining companies operating in Central Africa and Central America, and before that, as a consulting mining engineer working 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 Trident property described in the Technical Report in November 2019.

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, and for Items 1, 2, and 18 to 26.

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 mine planning and the preparation of long term mining plans and production schedules, commencing in 2013.

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 2020 at West Perth, Western Australia, Australia.

M. Lawlor

Michael Lawlor

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

I, Andrew Briggs, do hereby certify that:

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

14. This certificate applies to the technical report entitled "Trident Project, North West Province, Zambia, NI 43-101 Technical Report" dated effective 30th March 2020 (the "Technical Report").

15. 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.

16. 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.

17. 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.

18. 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.

19. I most recently personally inspected the Trident property described in the Technical Report in November 2019.

20. 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. I am also responsible for the estimates in Item 21 pertaining to processing, plus general and administration costs.

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

22. 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. I was involved in the commissioning and ramp up of the Trident plant, and was on site for the majority of 2015.

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

24. 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 2020 at West Perth, Western Australia, Australia.

Andrew Briggs