FIRST QUANTUM MINERALS LTD.

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 has been prepared by Qualified Persons (QPs) David Gray, Michael Lawlor and Andrew Briggs of First Quantum Minerals Ltd (FQM, the issuer or the Company), for FQM. This Technical Report supersedes the two NI 43-101 Technical Reports dated May 2012 (the 2012 Sentinel Technical Report) and December 2012 (the 2012 Enterprise Technical Report).

In this Technical Report, updated as at 31st May 2015, the respective Mineral Resource and Mineral Reserve aspects were addressed by David Gray (QP) and Michael Lawlor (QP). Metallurgical and mineral processing aspects were addressed by Andrew Briggs (QP).

This Technical Report follows the guidelines and format specified by NI 43-101. Mineral Resource and Reserve estimates have been reported in accordance with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014), which in turn complies with the guidelines of the Australasian JORC Code (JORC, 2012).

1.1 **Property location and ownership**

The Trident Project is located in the southern portion of the Trident Project area, in the north-west province of Zambia, approximately 150 km west of the town of Solwezi. 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 includes two development projects which are the subject of this Technical Report, ie Sentinel and Enterprise. The Sentinel deposit is a copper orebody and the Enterprise deposit is a nickel orebody. The wider Trident Project includes regional exploration targets including the Intrepid deposit. As yet, Mineral Resources have not been defined for these exploration areas and they are not covered in this Technical Report.

1.2.1 **Sentinel**

Initial mine development commenced at Sentinel in 2013 to establish in-pit crushing and conveying infrastructure. To date, material has been excavated to develop boxcuts and crusher pockets, the starter pit, and to provide stockpiled ore for plant commissioning. Open pit mining is carried out using conventional methods, with electric face shovels and hydraulic excavators, and a fleet of 330 t and 240 t capacity haul trucks. Mining capacity will eventually extend to around 65 Mbcm of ore and waste mined per annum. The ultimate 5.4 km long, 1.5 km wide and 375 m deep pit will be mined in stages, with ore crushed in-pit and conveyed overland to the Trident Project process plant.

Development and construction activities for the Trident plant 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 will reach 55 Mt.

The processing plant design is based on a conventional sulphide ore flotation circuit, with a separate 4 Mtpa circuit designed to process nickel ore feed from Enterprise or additional copper ore feed

from Sentinel. Hence, copper ore feed at Sentinel could increase to 59 Mtpa if the Enterprise flotation circuit was to be utilised.

Figure 1-1 is a plan of the Trident Project site showing the development areas at Sentinel, and at Enterprise.

1.2.2 Enterprise

Whilst environmental approval has been granted for the start of mining at Enterprise, pre-strip mining activities are yet to commence and the production start-up time is yet to be determined. The current focus of site development work has been on the construction of water management infrastructure. The Chisola River flows over the top of the Enterprise deposit and has been dammed upstream (the Chisola Dam) to prevent inundation of the future pit and to supply process water to the Trident plant. Other diversion structures are being constructed for the catchment between the dam and the proposed pit.

Once operational, the Enterprise Pit will be an adjunct to the nearby Sentinel Pit, with mining operations at both sites feeding a common Trident Project processing plant. The projected annual average nickel production is around 35,000 tonnes of metal in concentrate.

When operations commence, ore will be road-hauled to the Trident plant (adjacent to the Sentinel Pit) wherein it will be primary crushed and then comminuted in a SAG-ball milling circuit before proceeding to the flotation plant.

1.3 Project approvals

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.

The Sentinel Environmental and Social Impact Assessment (ESIA) was approved by the Zambian Environmental Management Agency (ZEMA) in July 2011. A Sentinel Addendum ESIA, covering the original Project infrastructure as well as amendments to the tailings storage facility, waste dump design and process water facilities, was approved by ZEMA in August 2013. The Enterprise ESIA was approved by ZEMA in September 2014.

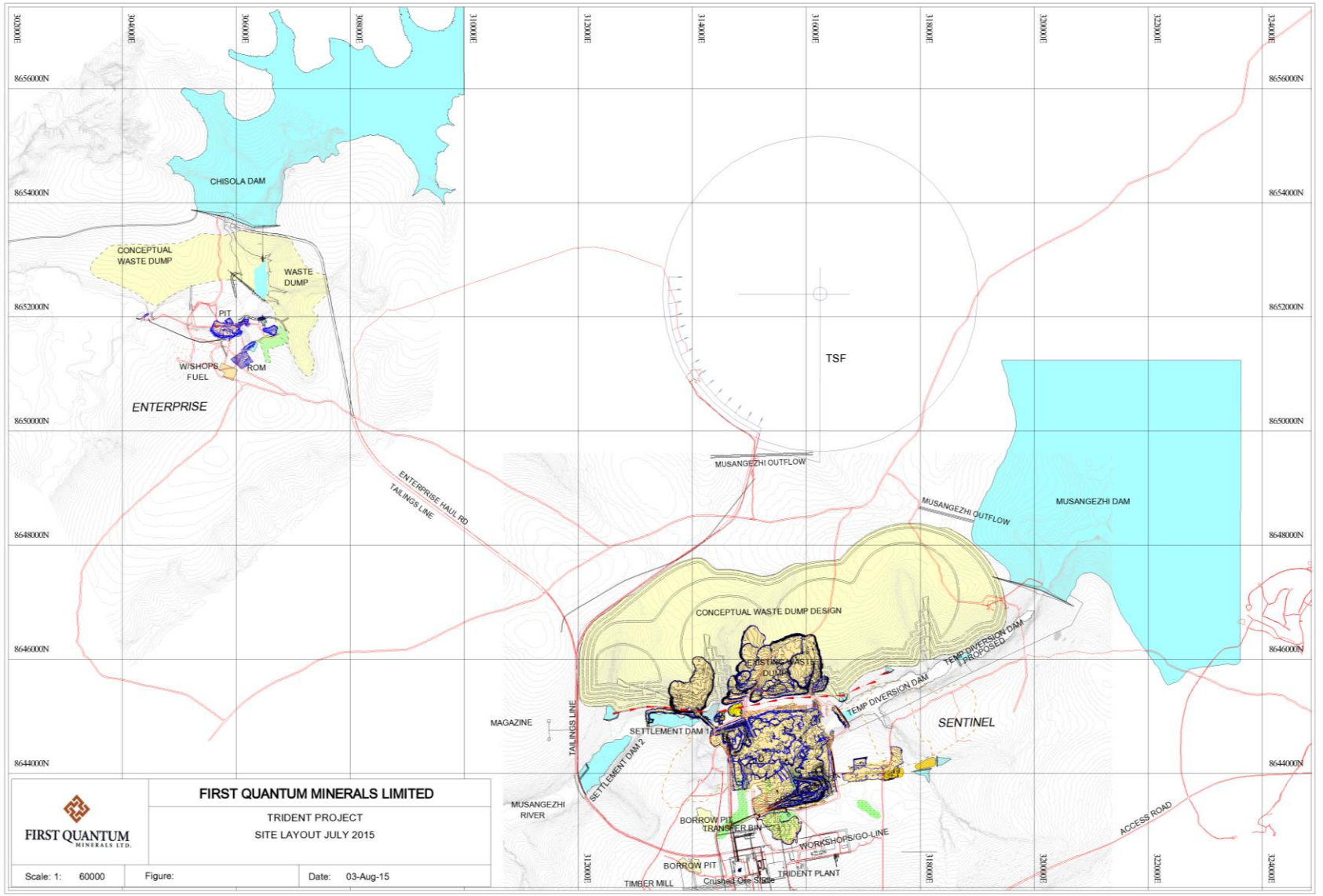
1.4 Geology and mineralisation

The Trident Project is located on the western end of the Lufilian Arc which extends from northern Zambia, across the Katanga Province of 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.880-550 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 mineralisation is hosted within a phyllite package and occurs as a series of layered continuous sheets having a northerly 20 to 30^{o} dip. Copper mineralisation is dominated by chalcopyrite which occurs as fine to coarse disseminations and or veinlets.

Figure 1-1 Trident Project site



1.4.2 Enterprise

The Enterprise nickel deposit is defined as a series of relatively shallow dipping bodies covering an area of 1,000 m by 500 m in the Enterprise main area (Main) and approximately 800 m by 300 m in the Enterprise south-west area (SW). Thicknesses range from a few metres to over 80 m. Enterprise mineralisation has been interpreted to be a hydrothermal nickel deposit with mineralisation hosted in a sequence of siltstone, carbonate and shale units. The shales and shale-carbonates have been preferentially mineralised due to a combination of rheological and geochemical interaction with the mineralising fluids. Nickel mineralisation occurs mainly as sulphide disseminations and or veinlets.

1.5 Exploration status

1.5.1 Sentinel

Drilling at Sentinel dates back to 1959, where only a few holes (31) 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 a nickel resource at Sentinel. However, only holes drilled by Kiwara PLC (Kiwara) (2007 to 2010) and FQM (2010 to 2015) were used in the copper Mineral Resource estimate update for this Technical Report. 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 mineralisation 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 mineralisation. 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.

1.6 Metallurgy summary

1.6.1 Sentinel

The Sentinel ore will be crushed in-pit and conveyed overland, and onto a crushed ore stockpile ahead of two milling trains, each comprising a SAG mill and a single ball mill. Each train will consist of two parallel banks of flotation cells, each comprising seven cells operating in series. Final concentrate at a grade of 24% copper will be thickened and filtered in a dedicated concentrate handling facility.

Based on testwork to date, the recommended metallurgical parameters for mine planning are 92% recovery for primary sulphide, and 70% recovery for the relatively 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 at a grade of 14% to 18% nickel will be thickened and filtered in a dedicated concentrate handling facility.

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

Based on testwork to date, the recommended metallurgical parameters for mine planning are 90% recovery for primary sulphide, and 60% recovery for the relatively smaller proportion of near-surface non-primary sulphide.

1.7 Environmental and social status

The granting of the large scale mining licences (LMLs) for the Trident Project was conditional upon approval by ZEMA of the ESIA which was submitted in early February and approved in July 2011. The Sentinel Addendum EIA, which included modifications to the Project, 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. KML currently tracks the implementation of these commitments on a quarterly basis. In addition to external resettlement and environmental audits, ZEMA will visit site to review all EIA and RAP commitments in 2016.

1.8 Project development status

The following information summarises the status of construction and development at Sentinel and Enterprise.

1.8.1 Plant site

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 tailings storage facility (TSF)), the first of the in-pit crushers and conveyors within the Sentinel Pit from late 2013 through into 2014, and latterly the building of the Enterprise processing circuits (including a dedicated primary crusher).

During this period, construction proceeded on the new Kalumbila townsite for the housing of the operations personnel and their families.

Commissioning of the Sentinel circuits was well advanced during the first half of 2015. The initial power supply connection proceeded in late 2014, with a second power line expected to be

commissioned by Q3 2015. One milling train of the Sentinel circuit has been in operation since the Q2 2015, and the second train is expected to be started by the end of Q3 2015.

1.8.2 Mining sites

Sentinel

In addition to a network of surface haul roads and access roads, the first of three in-pit crushers and overland ore conveyors have been installed from within a starter pit boxcut, with the conveyors extending across to the plant crushed ore stockpile via a surface transfer bin. Site clearing to date has been confined to the stage 1 and 2 starter pit area (and to part of stage 3, to the east). The starter waste dump has been formed and a number of water control and management dams have been constructed around the site. Surface powerlines extend around the initial pit perimeter, connecting to a number of substations powering drills and shovels, and providing power to pit dewatering bores and in-pit sumps.

A heavy vehicle workshop, incorporating refuelling and washdown facilities, is located on the south side of the pit, adjacent and to the east of the processing plant.

Enterprise

Minimal infrastructure is proposed for the Enterprise site. The main components are a small satellite administration office, a recently cleared run-of-mine (ROM) ore pad, a future workshops area and a number of water control and management dams. Some of the proposed surface waste haulroads are in place, and the ore haulroad connecting the ROM Pad with the existing main access road to the Trident plant, is partially complete.

Waste from within the initial starter pit perimeter has been mined to date as a source of material for haulroads and dam construction.

1.8.3 Tailings storage facility

A tailings storage facility (TSF) has been designed for the life of the Trident Project, with a capacity of over a 1,000 million tonnes, and to receive tailings from both of the Sentinel and Enterprise processing circuits. The circular TSF is 5.5 km in diameter and is designed to reach a maximum height of around 40 m.

The TSF stage 1 earthfill embankment provides an initial 15 months of storage capacity. The tailings will be deposited from spigots along the top of the embankment. Over time the TSF will be upstream raised with tailings.

1.8.4 Dams

The Musangezhi River which flows over the top of the Sentinel deposit, has been diverted to allow the Sentinel Pit to be developed. The river has been dammed upstream of the mine, providing a lake alongside the Kalumbila townsite. Excess lake water can overflow 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 into the upper catchment of the Kasombo River.

An additional earthfill dam has been constructed on the Chisola River to the north of the Enterprise deposit, as a source of process water.

1.8.5 Townsite

The Kalumbila townsite has been constructed at a location 19 km from the Trident plant site, on the eastern side of the Musangezhi Dam reservoir. The townsite 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 townsite and includes single person quarters.

1.9 Mineral Resource estimates

1.9.1 Sentinel

Block model grade estimates for the Sentinel deposit were updated in 2014 by David Gray (QP) of FQM with the assistance of FQM exploration and site geologists. The Sentinel Mineral Resource estimates were supported by a grid of diamond drilling and core sampling that cover the extents of mineralisation. An additional 34 infill diamond drilled holes have been included in this estimate update since the May 2012 Technical Report. 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 mineralisation. Block grade estimation is 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 is based upon a smallest mining unit block size of 12 mE by 12 mN by 6 m flitches, which is relevant to the expected scale of mining, minimising mining loss and dilution.

Sentinel Mineral Resource estimates have been classified according to drill grid spacing, geological confidence and confidence in the panel grade estimate as well as consideration of the sampling and preparation methods, analytical techniques and the associated data quality. Additional data and confidences have provided a 4% increase in copper metal for the measured and indicated resource categories due to a 4% increase in grade. Increased confidences have upgraded previous Indicated Mineral Resources into the measured category by 44%.

The resulting Mineral Resources have been reported using a copper cut-off grade of 0.2% as at May 2015, as per Table 1-1. Reported Mineral Resources are inclusive of Mineral Reserves.

Table 1-1 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**

1.9.2 Enterprise

The Enterprise development prospect Mineral Resource estimate update was completed in late 2013 using data from an additional 81 diamond drilled holes. The estimate includes mapping, drilling and sampling data, which adequately defines geological and grade continuity suitable for the estimation of Measured, Indicated and Inferred Mineral Resources. The update does not reflect a material change to the previous Mineral Resource estimate. 474 diamond drilled holes were sampled and logged for geology and structure, allowing for a good understanding of the controls on nickel mineralisation. QAQC information demonstrates acceptable levels of precision and accuracy of sample assay results. Bulk density measurements were obtained from analysis of diamond core and have been assigned to the block model estimate. The estimation method has unfolded the geological model prior to estimation and estimates were derived using ordinary kriging with post processing of parent block estimates into a smallest mining unit block dimension. This minimises mining loss and dilution.

The updated Mineral Resource estimate for Enterprise is presented in Table 1-2. The Mineral Resource statement uses a 0.15% total nickel cut-off grade as at 31st May 2015 and work continues for developing future Enterprise Mineral Resource estimates. The tonnes of combined Measured and Indicated Mineral Resources decreased by 1%, with metal decreasing by 6% due to a decrease in grade. Inferred tonnes increased significantly due to additional drilling at Enterprise SW. Inferred Mineral Resources have good opportunity to be converted into Indicated Mineral Resources with future infill drilling and improved geological understanding. Reported Mineral Resources are inclusive of Mineral Reserves.

Table 1-2 Enterprise Mineral Resource statement as at 31st May 2015 and 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**

1.10 Mineral Reserve estimates

The detailed mine planning for 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 life of mine stage designs for Sentinel, incorporating current and proposed future in-pit crusher positions and conveyor routes, and the scheduling of waste mining to allow optimised surface dumping and pit backfilling, whilst considering ARD management criteria. Both Sentinel and Enterprise now benefit from more accurate mining cost estimates inclusive of depth-increments and modelled ore and waste haul profiles. Sentinel in

particular, benefits from the estimation of mining costs which reflect short hauls to in-pit crusher positions and faster waste hauls under trolley-assist.

1.10.1 Sentinel

The Proved and Probable Mineral Reserve for Sentinel, inclusive of stockpiles, is 985.8 Mt at an average grade of 0.51% TCu (Table 1-3). This is based on a long-term projected copper price of $3.00/lb with an initial cut-off grade of 0.2% TCu to improve the plant feed grade profile within the first half of the LOM production schedule, together with the blending of lower grade feed (0.12% TCu cut-off grade) in the latter part of the mine life. This strategy allows the plant capacity of 55 Mtpa to be sustained over the mine life and is also consistent with the adoption of a design optimal pit shell selected on a maximum undiscounted cashflow basis.

Table 1-3 Sentinel Mineral Reserve statement, in-pit inventory at May 2015

Mineral Reserve statement as at end of May 2015 at $3.00/lb Cu				
Classification	**Material**	**Tonnes (Mt)**	**TCu (%)**	**Cu metal (kt)**
Proved	Non-primary sulphide	80.9	0.44	357.4
Proved	Primary sulphide	738.3	0.51	3,772.6
Proved 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**
Proved and Probable subtotal		**985.3**	**0.51**	**4,985.0**
Probable	Stockpiles	0.5	0.58	3.1
Proved and Probable total		**985.8**	**0.51**	**4,988.1**

The Proved Mineral Reserve is marginally in excess of Measured Mineral Resource for reasons of differing cut-off grade criteria.

1.10.2 Enterprise

The Proved and Probable Mineral Reserve for Enterprise is 35.4 Mt at an average grade of 0.97% Ni (Table 1-4). This is based on a long-term projected nickel price of $7.50/lb. As with the Sentinel strategy, the design optimal pit shell was selected on a maximum undiscounted cashflow basis.

Table 1-4 Enterprise Mineral Reserve statement, at May 2015

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 Proved Mineral Reserve is marginally in excess of Measured Mineral Resource for reasons of differing cut-off grade criteria.

1.11 **Production schedule**

Table 1-5 and Table 1-6 list the life of mine production schedules for Sentinel and Enterprise, respectively.

In the Sentinel schedule (Table 1-5), the tonnage and grade difference between the mined ore and the crusher feed is attributable to the starting ore stockpile balance reclaimed for processing at the outset. Lower grade ore contributes to the crusher feed during the period 2026 to 2033 in order to maintain the plant capacity at 55 Mtpa. In terms of the total crusher feed, the ore above 0.2% cut-off is 895.3 Mt at an average grade of 0.54% TCu, whilst the lower grade ore amounts to 90.5 Mt at an average grade of 0.16% TCu (of which almost 100% is primary sulphide ore). The Sentinel long term ore stockpiles are relatively small and reach a maximum size of 7.2 Mt. The crusher feed listed in Table 1.5 includes current stockpiles of 0.5 Mt.

In the Enterprise schedule (Table 1-6), all mined ore is essentially road-hauled direct to the Trident plant; there are no long term stockpiles.

Table 1-5 Sentinel production schedule

Year	Mined Ore (Mt)	Grade (%Tcu)	Crusher Feed (Mt)	Grade (%TCu)	Waste (Mt)	Total (Mt)	(Mbcm)	Strip Ratio (t:t)
At end May 2015								
2015	27.0	0.54	27.0	0.54	71.8	98.8	38.7	2.7
2016	54.1	0.54	54.5	0.54	131.6	185.7	67.2	2.4
2017	49.8	0.63	49.8	0.63	123.6	173.4	65.4	2.5
2018	54.6	0.58	54.6	0.58	126.2	180.8	65.4	2.3
2019	60.6	0.55	55.0	0.57	121.6	182.2	65.4	2.0
2020	49.9	0.64	55.2	0.62	119.3	169.2	65.6	2.4
2021	56.9	0.52	55.0	0.53	124.6	181.5	65.4	2.2
2022	52.2	0.56	54.4	0.55	128.6	180.8	65.4	2.5
2023	54.0	0.55	54.0	0.55	123.6	177.6	65.4	2.3
2024	59.9	0.57	55.2	0.59	122.8	182.7	65.6	2.1
2025	55.7	0.46	55.0	0.46	117.3	173.0	65.4	2.1
2026	55.1	0.42	55.0	0.42	117.8	172.9	64.9	2.1
2027	50.7	0.46	55.0	0.45	129.2	179.9	65.1	2.5
2028	50.2	0.40	51.5	0.40	131.6	181.8	65.4	2.6
2029	55.4	0.37	55.0	0.37	126.9	182.2	65.4	2.3
2030	57.7	0.41	55.0	0.42	124.6	182.3	65.4	2.2
2031	57.1	0.40	55.0	0.41	85.5	142.6	51.2	1.5
2032	52.7	0.48	50.6	0.49	88.0	140.7	50.5	1.7
2033	31.9	0.58	39.1	0.53	19.2	51.0	18.3	0.6
2034								
LOM (dil'd)	**985.3**	**0.51**	**985.8**	**0.51**	**2,133.8**	**3,119.1**	**1,141.5**	**2.2**

Table 1-6 Enterprise production schedule

Year	Mined Ore (Mt)	Grade (%Ni)	Crusher Feed (Mt)	Grade (%Ni)	Waste (Mt)	Total (Mt)	Total (Mbcm)	Strip Ratio (t:t)
Year 1	0.0	0.33	0.0	0.00	0.4	0.4	0.2	
Year 2	0.0	0.21	0.0	0.23	10.2	10.2	4.9	
Year 3	2.4	1.11	2.4	1.11	21.2	23.6	9.4	8.9
Year 4	4.1	1.27	4.1	1.27	26.7	30.9	11.8	6.5
Year 5	4.0	1.59	4.0	1.59	38.5	42.6	16.5	9.5
Year 6	4.2	1.29	4.2	1.29	40.0	44.1	16.8	9.6
Year 7	4.1	0.97	4.1	0.97	43.2	47.2	16.8	10.6
Year 8	4.0	0.61	4.0	0.61	39.1	43.1	15.9	9.7
Year 9	4.1	0.64	4.1	0.64	38.5	42.6	15.6	9.3
Year 10	4.2	0.62	4.2	0.62	41.8	46.0	16.0	9.9
Year 11	4.2	0.73	4.2	0.73	20.4	24.5	10.0	4.9
Year 12	0.1	0.68	0.1	0.68	0.4	0.5	0.2	4.5
LOM (dil'd)	**35.4**	**0.97**	**35.4**	**0.97**	**320.3**	**355.7**	**134.1**	**9.0**

1.12 **Economic analyses**

Economic analyses in the form of undiscounted cashflow models to support the Mineral Reserve estimates for each of Sentinel and Enterprise are listed separately in Table 1-7 and Table 1-8, respectively. These models show the indicative cashflows and do not replace more comprehensive Project financial models inclusive of taxation, depreciation and amortisation.

Table 1-7 Sentinel Mineral Reserves cashflow model

	UNITS		TOTAL	<2015	2015 to 2019	2020 to 2024	2025 to 2029	2030 to 2034
MINING (after mining dilution and recovery)								
Total ore	Tonnes	Mt	**985.3**	0.0	246.0	272.9	267.1	199.3
Total waste	Tonnes	Mt	**2,133.8**	0.0	574.9	618.9	622.7	317.3
Strip ratio		t : t	**2.2**	0.0	2.3	2.3	2.3	1.6
Total reclaim	Tonnes	Mt	**28.9**	0.0	8.7	7.5	5.5	7.2
TOTAL FEED TO PLANT (after mining dilution and recovery)								
Total ore	Tonnes	Mt	**985.8**	0.0	240.9	273.8	271.5	199.7
Grade	Cu	%	**0.51**	0.00	0.58	0.57	0.42	0.45
Insitu metal	Tonnes	**kt**	**4,988.1**	**0.0**	**1,386.8**	**1,552.8**	**1,141.7**	**906.8**
AVERAGE RECOVERIES								
	Cu	%	**90%**	**0%**	**88%**	**91%**	**90%**	**92%**
METAL RECOVERED								
Rec. metal	Tonnes	kt	**4,491.5**	0.0	1,216.1	1,409.0	1,032.3	834.1
	Pounds	**Mlb**	**9,902.0**	**0.0**	**2,681.1**	**3,106.2**	**2,275.8**	**1,838.9**
GROSS REVENUE								
Cu $/t	6,613.86	**$M**	**29,706.0**	**0.0**	**8,043.2**	**9,318.6**	**6,827.3**	**5,516.8**
CAPITAL COSTS								
Development costs		$M	**1,900.0**	1,900.0	0.0	0.0	0.0	0.0
Sustaining capex allowance		$M	**585.3**	0.0	148.5	164.9	163.9	108.0
subtotal		**$M**	**2,485.3**	**1,900.0**	**148.5**	**164.9**	**163.9**	**108.0**
OPERATING COSTS								
Mining:								
Ore $/t mined	1.77	$M	**1,743.9**	0.0	435.3	483.0	472.8	352.8
Waste $/t mined	1.84	$M	**3,926.2**	0.0	1,057.9	1,138.7	1,145.8	583.8
subtotal mining	**1.82**	**$M**	**5,670.1**	**0.0**	**1,493.2**	**1,621.8**	**1,618.6**	**936.6**
Processing (incl. G&A):								
Ore $/t processed	**6.10**	**$M**	**6,013.3**	**0.0**	**1,469.4**	**1,670.0**	**1,655.9**	**1,218.0**
Stockpile reclaim allow:								
allow	0.80	$M	**23.1**	0.0	6.9	6.0	4.4	5.8
subtotal		**$M**	**11,706.6**	**0.0**	**2,969.5**	**3,297.8**	**3,278.9**	**2,160.3**
METAL COSTS (including 9% royalty)								
TCRCs $/lb	0.39	$M	**3,853.8**	0.0	1,043.5	1,208.9	885.7	715.7
royalty $/lb	0.27	$M	**2,673.5**	0.0	723.9	838.7	614.5	496.5
subtotal	0.66	**$M**	**6,527.4**	**0.0**	**1,767.4**	**2,047.6**	**1,500.2**	**1,212.2**
UNDISCOUNTED CASHFLOW		**$M**	**8,986.7**	**-1,900.0**	**3,157.9**	**3,808.4**	**1,884.2**	**2,036.3**

Table 1-8 Enterprise Mineral Reserves cashflow model

	UNITS		TOTAL	Year 1 to 5	Year 6 to 10	Year 11 & 12
MINING (after mining dilution and recovery)						
Total ore	Tonnes	Mt	**35.4**	10.6	20.6	4.3
Total waste	Tonnes	Mt	**320.3**	97.0	202.5	20.8
Strip ratio		t : t	**9.0**	9.2	9.8	4.9
Total reclaim/haulage	**Tonnes**	**Mt**	**35.4**	**10.6**	**20.6**	**4.3**
TOTAL FEED TO PLANT (after mining dilution and recovery)						
Total ore	Tonnes	Mt	**35.4**	10.6	20.6	4.3
Grade	Ni	%	**0.97**	1.36	0.83	0.73
Insitu metal	**Tonnes**	**kt**	**344.4**	**143.4**	**169.8**	**31.2**
AVERAGE RECOVERIES						
	Ni	%	**88%**	**87%**	**89%**	**86%**
METAL RECOVERED						
Rec. metal	Tonnes	kt	**303.8**	125.1	151.9	26.7
	Pounds	**Mlb**	**669.7**	**275.8**	**335.0**	**58.9**
GROSS REVENUE						
Ni $/t	**16,534.65**	**$M**	**5,023.1**	**2,068.8**	**2,512.2**	**442.1**
CAPITAL COSTS						
Development costs		$M	**90.0**	90.0	0.0	0.0
subtotal		**$M**	**90.0**	**90.0**	**0.0**	**0.0**
OPERATING COSTS						
Mining:						
Ore $/bcm mined	7.55	$M	**92.8**	27.7	53.9	11.2
Waste $/bcm mined	6.77	$M	**797.7**	222.6	521.5	53.6
subtotal mining	**6.64**	**$M**	**890.5**	**250.3**	**575.4**	**64.8**
Processing (incl. G&A):						
Reclaim+haul. $/t proc.	1.50	$M	**53.1**	15.9	30.9	6.4
Ore $/t processed	7.79	$M	**276.0**	82.4	160.3	33.4
subtotal processing	**9.29**	**$M**	**329.1**	98.2	191.1	39.8
subtotal		**$M**	**1,219.6**	**348.5**	**766.5**	**104.6**
METAL COSTS (including 9% royalty)						
TCRCs $/lb	2.50	$M	**1,674.0**	689.5	837.2	147.3
royalty $/lb	0.68	$M	**452.1**	186.2	226.1	39.8
subtotal	**3.17**	$M	**2,126.1**	875.6	1,063.3	187.1
CASHFLOW		$M	**1,587.4**	**754.7**	**682.3**	**150.4**

1.13 Conclusions and recommendations

1.13.1 Geology and Mineral Resource estimates

Sentinel

Sentinel mineralisation occurs as a series of continuous horizons which are often separated and influenced by several sub-parallel structural detachment zones. Close spaced RC grade control drilling will provide additional detail relevant to the scale of mining for these features. Diamond drilled core samples has good recovery and coverage across the extents of Sentinel mineralisation and its respective domains. QAQC results demonstrate sample results are adequate for use in this Mineral Resource estimate and support 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 as relevant to mining and processing. The Sentinel grade estimates validate well against the input data and were believed representative of the prevailing in-situ mineralisation.

Recommendations relevant to the Sentinel Mineral Resource estimate include consideration of sequential copper analysis of RC grade control samples. In-pit mapping and 3D geology modelling

(as practiced) will add rigour to the existing geology models. Multi-element analysis (as practiced) of RC grade control samples will provide clarity to geological domain detail.

Enterprise

The Enterprise estimates of the hydrothermal nickel mineralisation have considered structural deformation with the use of unfolded modelling methods. There is sufficient QAQC to assure confidence that the analysed samples are representative of the in-situ mineralisation. Density data is adequate within fresh material; however, additional data is required for saprolite and saprock in the Enterprise SW area. The additional data from 23,900 m of drilling used for this update has allowed good increases in the Measured and Inferred Resource inventory, with opportunity for Inferred Resources to be moved into the indicated category through future drilling. The estimated model validates well against the used input data.

Recommendations relevant to the Enterprise Mineral Resource estimate were to plan for a suitably close spaced grid of RC grade control drilling with early mining start-up. Additional density data is required for saprolite and saprock in the Enterprise SW area. The recent re-logging data and improved geology understanding needs to be considered in the next resource update. Future diamond drilling at Enterprise needs to consider triple tube methods for PQ diameter coring in order to assure optimal recoveries.

1.13.2 Grade control and reconciliation

The current and early stages of mining start-up at Sentinel include development and implementation of a suitable RC grade control drilling programme, detailed in-pit mapping as well as systems for ore control and reconciliation. These data sources and ore control systems will provide adequate control for supporting optimal mining.

At Enterprise, once initial pre-stripping has commenced and is close enough to mineralisation, grade control RC drilling and supporting ore control and reconciliation systems will be initiated.

1.13.3 Mineral Reserve estimates

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 now takes account of current locations and proposed future in-pit crusher relocations and trolley-assisted waste haulage routes.

In the case of Sentinel, unit mining costs have been estimated from first principles to reflect depth varying increments related to short ore hauls to in-pit crushers and faster waste hauls under trolley-assist. Depth varying unit mining costs have also been estimated from first principles for Enterprise. This is an improvement over the level of estimation detail and accuracy in the 2012 Technical Reports.

In terms of the process operating costs, no significant changes have been made to the Sentinel costs. Enterprise estimates now include an overland ore haulage cost. No changes have been made to the metal costs for either of Sentinel or Enterprise, other than to increase the royalty rate from 6% to 9%. As each operation develops, it is recommended that operating and metal cost estimates are periodically reviewed for future mine planning work, based on actual performance.

Metallurgy

Further metallurgical sample drilling, sampling and testing for Sentinel is currently in the planning stage and is expected to commence later in 2015. Based on plant performance during commissioning, it is expected that grade/recovery relationships will be confirmed under operational conditions.

Further metallurgical sample drilling, sampling and testing for Enterprise is currently in the planning stage and will be carried out prior to commencement of operations. This work will continue to define the optimum flotation conditions for the various mineralogical suites present.

Geotechnical engineering

Since the 2012 Technical Reports, comprehensive geotechnical pit slope stability investigations have now been carried out, arising from which there are new pit slope design specifications provided for Sentinel and Enterprise. As part of the normal cyclic development of operating mine plans and mine design updates, the design specifications should be periodically updated to reflect observations and the additional geotechnical information available from actual pit slope exposures.

Groundwater engineering

Since the 2012 Technical Reports, a considerable amount of work has been done on groundwater (and surface water) investigations, aimed at managing the potential for inflows to the mined pits. In the case of Sentinel, a number of monitoring and abstraction bores are now in place and the drilling/equipping of additional bores is in progress. In the case of Enterprise, an investigation programme and the siting of proposed monitoring and abstraction bores is in progress. Surface water diversion and management dams are in place at Sentinel, and are under construction at Enterprise.

As part of the normal cyclic development of operating mine plans and production schedules, it is recommended that water monitoring and abstraction information be used to reassess the vertical advance rates adopted in all long term mining projections.

Waste dumping

Since the 2012 Technical Reports, waste rock geochemical testwork has been completed and the waste dump design for Sentinel now takes account of ARD management requirements. Testwork has also been completed for Enterprise waste rocks, indicating that the waste will be mostly non-acid forming. Further analysis is recommended for Enterprise and in the meantime, the initial waste dumping materials should be separated and run-off conditions monitored.

In the case of Sentinel, it is recommended that in all subsequent life of mine planning updates, the waste dump design, sequencing and material balance should be continually reviewed.

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 Ltd (FQM, the issuer) for FQM. The purpose of this Technical Report is to document current Mineral Resource and Mineral Reserve estimates for the property, together with supporting information on Project development status and further technical work completed since the last Technical Reports. This Technical Report for the Trident Project covers both Sentinel and Enterprise which have previously been reported separately.

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 May, 2015.

2.3 Qualified Persons and authors

The Mineral Resource estimates were prepared 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.

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

Name	Position	NI 43-101 Contribution
David Gray BSc(Geology), MAusIMM, PrSciNat(SACNASP)	Group Mine and Resource Geologist, FQM (Australia) Ltd	Author and Qualified Person Items 7– 12, 14
Michael Lawlor BEng Hons (Mining), MEngSc, FAusIMM	Consultant Mining Engineer, FQM (Australia) Ltd	Author and Qualified Person Items 1 to 6, 15 and 16, 18 to 26
Andrew Briggs BSc(Eng), ARSM, FSAIMM, PEng (NAPEG)	Group Consulting Project Metallurgist FQM (Australia) Ltd	Author and Qualified Person Items 13 and 17

2.4 Sources of information

Information used in compiling this Technical Report was derived from the previous Technical Reports (FQM, May 2012 and December 2012) and from the reports and documents listed in the references (Item 27).

For Sentinel in particular, repeated reference is made to the two Technical Reports filed under an FQM covering letter in May 2012:

- CSA Global (UK) Ltd (CSA), 2012. NI 43-101 Technical Report Sentinel Prospect, Trident Project, North West Province, Zambia. Report to First Quantum Minerals Ltd. May.
- DumpSolver Pty Ltd (DumpSolver), 2012. NI 43-101 Technical Report Mineral Reserves of Sentinel Prospect, Trident Project, North West Province, Zambia. Open Pit Optimisation, Pit Design, Life-of-Mine Schedule & Mineral Reserves. Report to First Quantum Minerals Ltd. May.

For Enterprise in particular, repeated reference is made to the two Technical Reports filed under an FQM covering letter in December 2012:

- CSA Global (UK) Ltd (CSA), 2012. NI 43-101 Technical Report Enterprise Prospect, Trident Project, North West Province, Zambia. Report to First Quantum Minerals Ltd. December.
- DumpSolver Pty Ltd (DumpSolver), 2012. NI 43-101 Technical Report Mineral Reserves of Enterprise Prospect, Trident Project, North West Province, Zambia. Open Pit Optimisation, Pit Design, Life-of-Mine Schedule & Mineral Reserves. Report to First Quantum Minerals Ltd. December.

2.5 Personal inspections

Authors David Gray and Michael Lawlor, acting as Qualified Persons for this Technical Report, have each personally inspected the Sentinel and Enterprise sites at numerous times since 2013, with the most recent visit by both being in June 2015.

Andrew Briggs, also a Qualified Person for this Technical Report, has visited site numerous times, with his most recent visit being for an essentially continuous period through 2015.

ITEM 3 **RELIANCE ON OTHER EXPERTS**

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

In relation to Item 20 (Environmental Studies, Permitting, Social and Community Impact), and whilst taking QP responsibility for authoring the information reported therein, Michael Lawlor has supplied the commentary and relied on a peer review by Andrew Hester, FQM Environmental Manager, Africa.

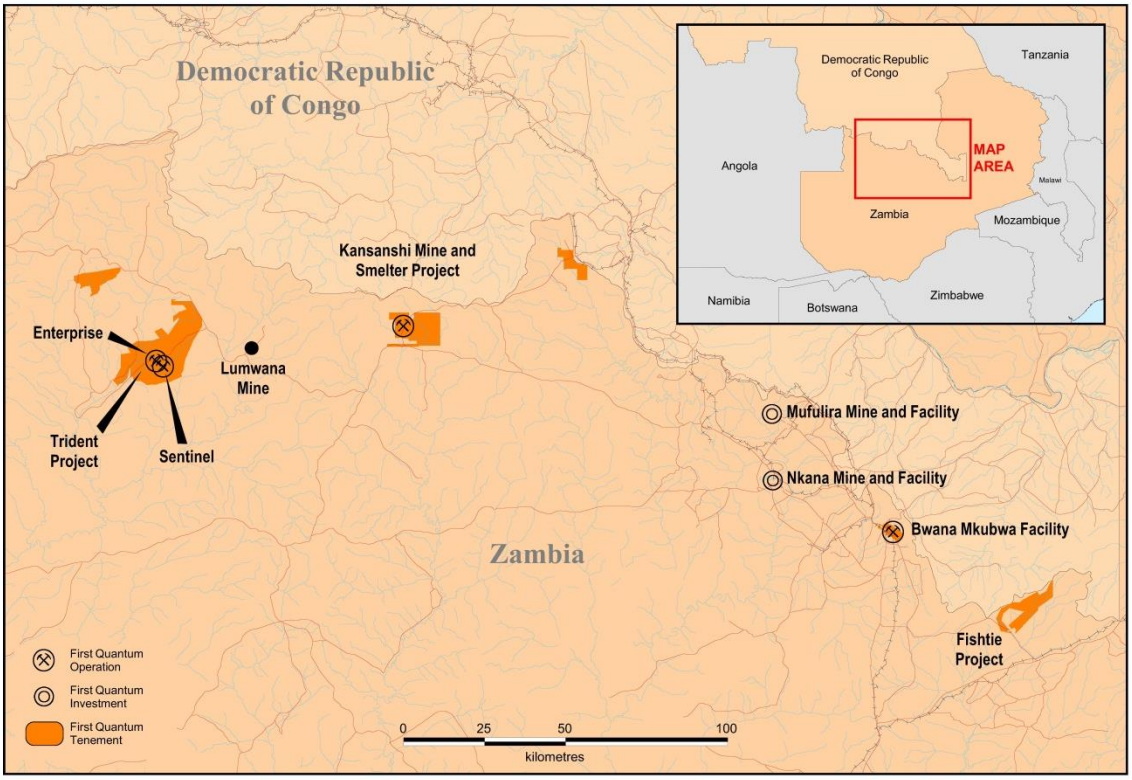
ITEM 4 PROPERTY LOCATION, DESCRIPTION AND TENURE

The following information is reproduced in part from the Sentinel 2012 Technical Report (CSA, May 2012) and from the Enterprise 2012 Technical Report (CSA, December 2012). Other than an update on taxes and royalties by Michael Lawlor (QP), there is no change to this information from that reported in 2012. The information remains current and relevant.

4.1 Property 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, North-western Province, Zambia



4.2 Tenure

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 to KML covering an area of 243.69 km^2 over the Enterprise deposit. Additional licences (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, and a plan of their location is shown in Figure 4-2.

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 40,000 m^3/day
  - approved in July 2012
  - valid for five years, and renewable
- Chisola Dam water abstraction rights:
  - rate of 150,685 m^3/day
  - approved in February 2014
  - valid for five years, and renewable

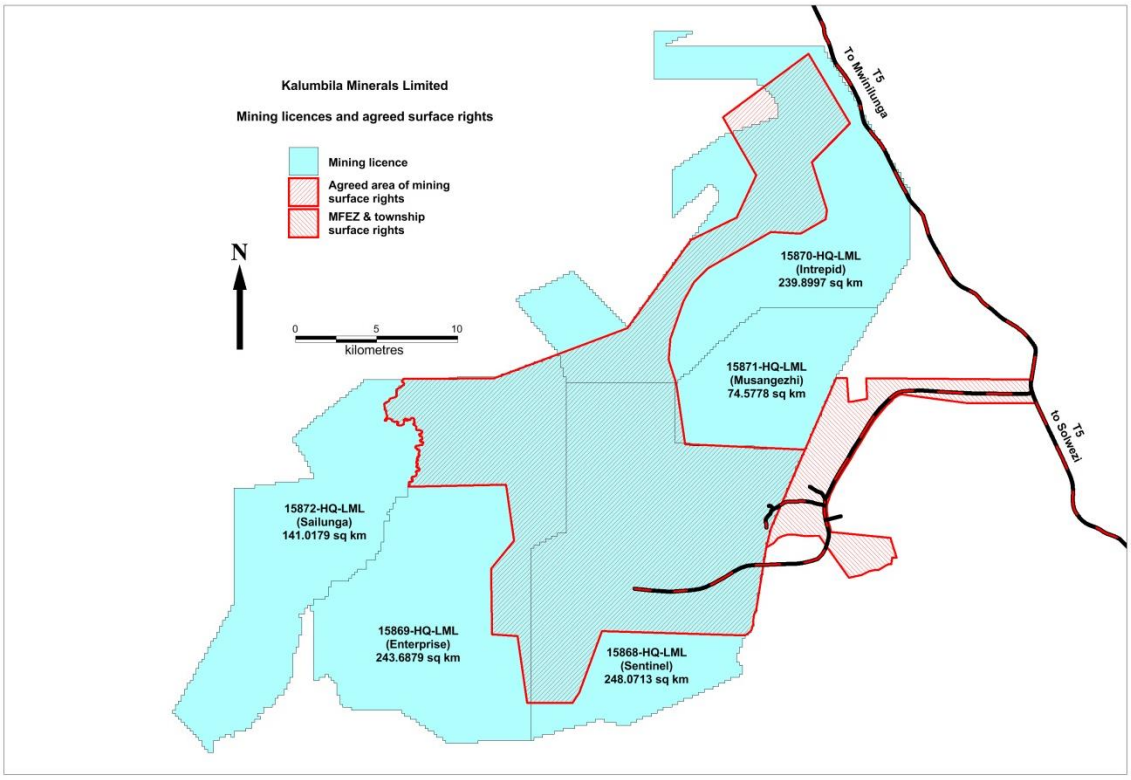
Table 4-1 Coordinates of Sentinel Mining Licence, 15868-HQ-LML (coordinates are in UTM)

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

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

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



4.3 Royalties, rights, payments, agreements

The rate of corporate income tax paid by the Company under Zambian legislation in 2014 has been approximately 30% of earnings plus a variable profits tax of up to 15%, in addition to a mineral royalty of 6% of gross sales on a monthly basis. The mineral royalty rate for copper increased from 3% to 6% of gross sales from April 2012.

In January 2015, the Zambian government proposed an amendment to the corporate tax and mining royalty regime by increasing revenue based royalties from 6% to 20% and reducing corporate taxes to 0% for open pit mining operations. This increase in the mineral royalty rate would result in an increase in the cost of metal sales and a decrease in operating cashflows. In April 2015, the Zambian government revised the royalty increase to 9% and retained the income tax and profits taxes at the 2014 levels. This will become effective from 1st July 2015.

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. Final approval by the President of Zambia and the issue of title deeds is awaited.

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 (Figure 4-2). The Zambia Development Agency has approved the MFEZ subject to granting of title to the land. This is at an advanced stage with the Ministry of Lands currently drawing up title deeds for plots within the township. The MFEZ area lies entirely outside of the mining licences.

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

4.4 Environmental liabilities

There are no known environmental liabilities attached to the property.

The main environmental liabilities at the Trident Project 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 and waste rock dumps.

The Sentinel and Enterprise mine closure plans are reviewed regularly by independent experts. The next review is scheduled for 2017.

4.5 Potential access and exploitation risks

To the extent known, there are no significant factors and risks that may affect access, title to, or the right or ability to perform work on the property.

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

The following information is reproduced in part from the Sentinel 2012 Technical Report (CSA, May 2012). The reproduced information on accessibility remains current and relevant. Information on climate and physiography is provided as updated information from a 2014 hydrogeological investigation report by Schlumberger Water Services (Australia) Pty Ltd (SWS, 2014).

5.1 **Accessibility**

Access to the Trident Project is from a new road of approximately 25 km length, sealed and leading from the highway linking Solwezi to Mwinilunga. An airstrip is located close to the Sentinel site and is suitable for daily commuter plane traffic. A new and substantial airstrip is to be built closer to the Kalumbila townsite.

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 closest meteorological stations to the Trident Project site are located at Solwezi and at Mwinilunga (110 km to the northwest). The annual and average rainfall and temperature data recorded at Solwezi and Mwinilunga between 1960 and 2007 is presented in Figure 5-1.

The mean annual rainfall at Solwezi and Mwinilunga is estimated as 1290 mm and 1380 mm, respectively. Annual average temperature is 19.8^{o}C for Solwezi and Mwinilunga. Overall, the annual potential evaporation exceeds average annual rainfall depths, with potential evaporation estimated to be around 2254 mm and 1729 mm for Solwezi and Mwinilunga, respectively (SWS, 2014).

From experience at the Company's operations at Kansanshi, and considering development activities completed to date, it is expected that wet season rainfall will not significantly hinder mining operations.

5.3 **Physiography**

The Sentinel deposit lies at an elevation of 1200 mRL in a setting of low relief, except for a number of ridges in the north, northeast and at Kalumbila Hill (1397 mRL). The watershed is defined by the Musangezhi River which passes over the top of the Sentinel deposit, and which varies in altitude between 1234 mL and 1204 mRL within the mining footprint (SWS, 2014).

The Enterprise deposit lies at an elevation of 1150 mRL, in a relatively flat to gently undulating setting. The watershed is defined by the Chisola River and varies in altitude from 1180 mRL to 1140 mRL.

The topography and surface drainage of the Trident Project area is shown in Figure 5-2 as being dominated by three parallel river systems which flow in a south-westerly direction (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.

Figure 5-1 Climate conditions, Solwezi and Mwinilunga recording stations (SWS, 2014)

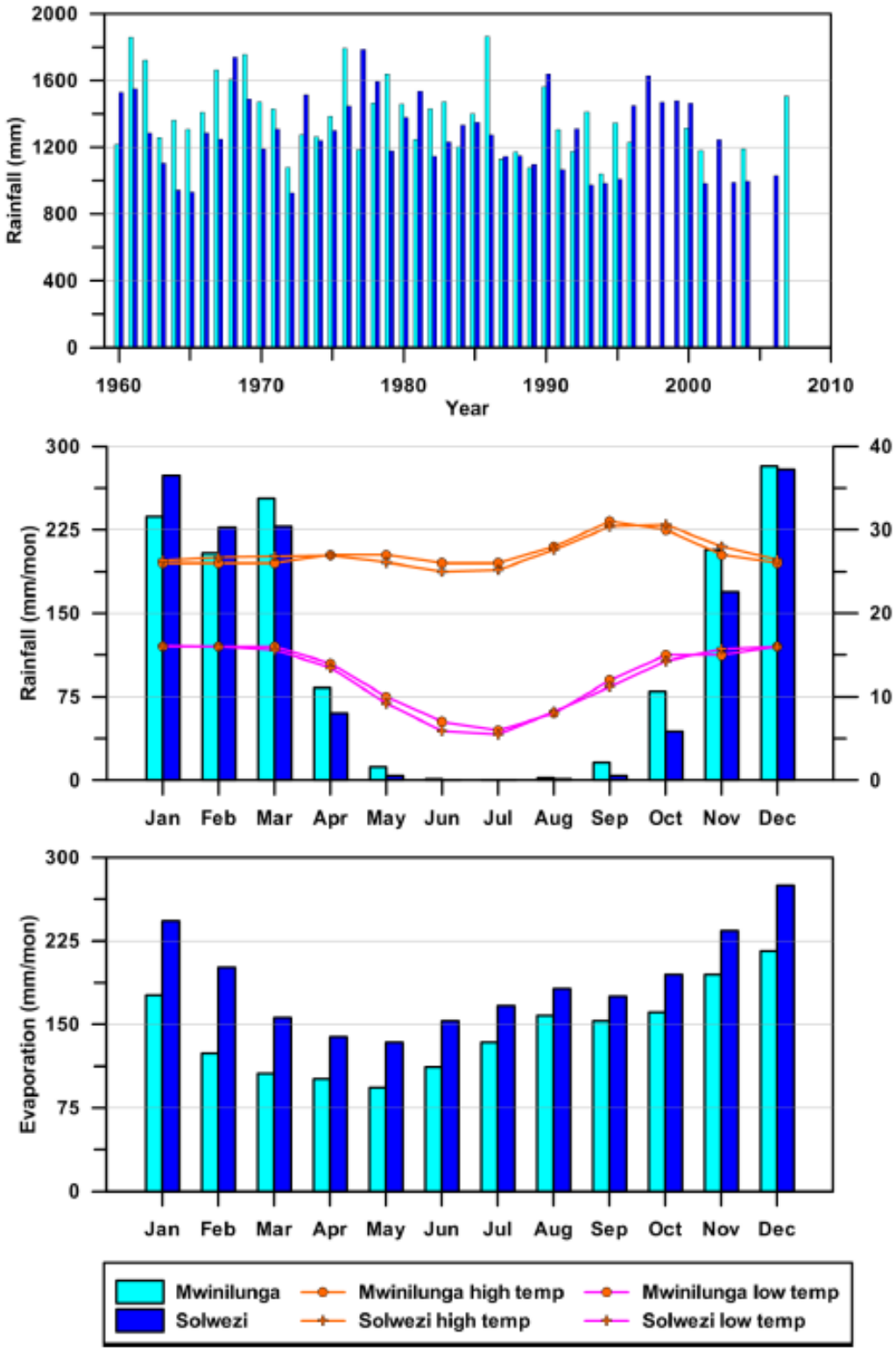
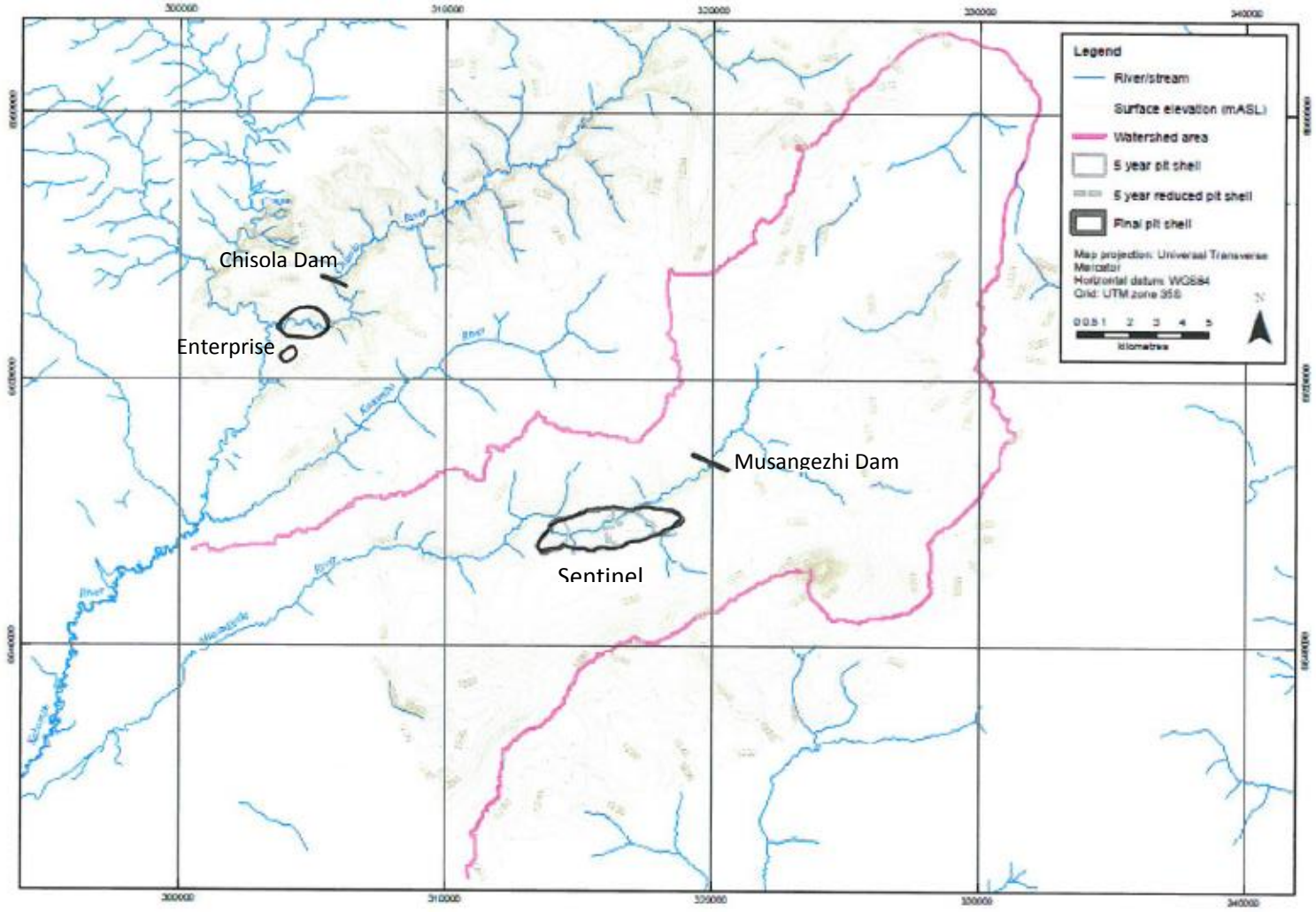


Figure 5-2 Trident Project area, surface drainage (SWS, 2014)



5.4 Vegetation

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

The miombo forest is common throughout Zambia and 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 people and by agricultural activities.

5.5 Local resources

The nearest major population centre is at Solwezi, 150 km to the east. Personnel can be and are recruited from Solwezi and from the rural communities surrounding the Trident Project. 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.

5.6 Infrastructure

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

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.

ITEM 6 HISTORY

The following information is reproduced in part from the 2012 Technical Reports (CSA, May and December 2012; DumpSolver, May and December 2012).

Historical Mineral Resource and Mineral Reserve estimates are provided for information only. The Company is not relying upon nor treating these estimates as current Mineral Resources or Mineral Reserves. David Gray (QP) and Michael Lawlor (QP) have provided updated information and new estimates in Items 14 and 15, respectively.

6.1 Sentinel

6.1.1 Prior ownership, exploration and development work

The Trident Project area was originally investigated by Roan Selection Trust (RST) between 1959 and 1961, then by Anglo American and Equinox in the 1980s and 1990s, to be followed by Kalumbila Minerals Limited (KML, a subsidiary of Kiwara PLC) between 2007 and 2009. Over time, the exploration focus has varied from copper (RST) to nickel (Anglo American) 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 completed 31 wide spaced core holes over the Sentinel area and encountered widespread but relatively low grade copper mineralisation. Anglo American focussed on detailed drilling for localised nickel-copper mineralisation around the Kalumbila Fault, whereas KML subsequently completed the first systematic drilling of the copper mineralisation over an eight kilometre strike length.

Following the acquisition of KML by the Company, exploration was resumed across the Sentinel deposit area in 2010, resulting in the completion of 677 diamond drilled holes and 229,713 m of drilling by December 2013.

6.1.2 Previous Mineral Resource estimate

Prior to ownership by the Company, there were no known Mineral Resource estimates for the Sentinel deposit. In late 2011, following a comprehensive diamond drilling campaign, FQM tasked CSA Global (UK) Ltd (CSA) with a Mineral Resource estimate of the Sentinel deposit. CSA produced an estimate under the supervision of QP, Galen White. This estimate was the subject of the 2012 Technical Report (CSA, May 2012) and used all available geological data as at March 2012. The Mineral Resource statement is presented in **Table 6-1.**

At the stage of this previous estimate the relatively minor oxide mineralisation was considered part of the Mineral Resource. In the Mineral Resource update for this Technical Report, oxide material has been removed from the resource statement due to its high degree of iron oxidation and dominance of refractory copper. Refractory copper will not be processable via the current plant's flotation methods. Removal of this oxide material from the previous May 2012 estimate, results in a total Measured and Indicated Resource of 998.2 Mt at an average grade of 0.51% Cu.

Table 6-1 Trident Project, May 2012 Mineral Resource statement for the 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**

6.1.3 Previous Mineral Reserve estimate

DumpSolver produced the previous Mineral Reserve estimate for Sentinel (DumpSolver, May 2012). The open pit mining inventory statement is reproduced in Table 6-2. The 2012 Mineral Reserve estimate was based on a $3.00/lb copper price, was reported at a 0.2% Cu cut-off grade and included mining tonnage dilution of an additional 5% (at zero grade) and mining recovery losses of 5%.

This estimate followed a similar conventional process as adopted for the updated estimate provided in this Technical Report, in respect of accounting for modifying factors through optimisations, staged pit designs and life of mine production schedules.

The 2012 Mineral Reserve estimate (Table 6-2, DumpSolver, May 2012) is now replaced by the estimate reported and tabulated under Item 15.

Table 6-2 Trident Project, Sentinel Mineral Reserve statement, at March 2012

Mineral Reserve statement as at March 2012 at $3.00/lb Cu			
Classification	**Material**	**Tonnes (Mt)**	**TCu (%)**
Proved	Non-primary sulphide	20.6	0.48
Proved	Primary sulphide	455.2	0.53
Proved subtotal		**475.8**	**0.52**
Probable	Non-primary sulphide	20.0	0.42
Probable	Primary sulphide	278.4	0.47
Probable subtotal		**298.4**	**0.47**
Proved and Probable total		**774.2**	**0.50**

6.2 Enterprise

6.2.1 Prior ownership, exploration and development work

Anglo American Corporation Central Africa (AACCA) first identified the presence of seemingly low level Ni, Cu and Co in the Kawako area (now known as Enterprise), in 1996. In 2008, the project was acquired by Kiwara PLC and in May 2008 reconnaissance was completed. FQM attained the leases in 2011. Little information regarding AACCA's work on the area has survived to the present day, except the drill hole collars for the 73 holes drilled by AACCA.

Reconnaissance was completed in May 2008, which included grab sampling some suspected drill spoil from the surrounding area. Encouraging assays from this work, including Ni assays of greater than 1% Ni, led to a follow up geochemical soil sampling programme.

Follow up drilling was performed based on the results of the soil geochemistry survey. This drilling consisted of two vertical HQ/NQ diameter diamond drill holes, KW1 and KW2, which were sited some 850 m apart from each other. During 2009, Kiwara performed a ground magnetic survey and a fixed loop ground electromagnetic survey of the Enterprise area, as well as a follow up diamond drill core programme.

6.2.2 Previous Mineral Resource estimate

Prior to ownership by the Company, there were no known Mineral Resource estimates for the Enterprise deposit. In late 2012, following a comprehensive diamond drilling campaign, FQM tasked CSA Global (UK) Ltd (CSA) with a Mineral Resource estimate of the Enterprise deposit. CSA produced an estimate under the supervision of QP, Galen White. This estimate was the subject of the 2012 Technical Report (CSA, December 2012), and used all available geological data as at November 2012. The resulting statement is presented in Table 6-3**.**

Note: The Mineral Resource is reported at a 0.15% Ni cut-off and was estimated as a block model within constraining mineralised wireframe. Differences may occur due to rounding errors.

Table 6-3 Trident Project, December 2012 Mineral Resource statement for the Enterprise deposit, using a 0.15% Ni cut-off.

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**

6.2.3 Previous Mineral Reserve estimate

DumpSolver produced the previous Mineral Reserve estimate for Enterprise (DumpSolver, December 2012). The open pit mining inventory statement is reproduced in Table 6-4**.** The 2012 Mineral Reserve estimate was based on a $7.50/lb nickel price, was reported at a marginal 0.1%Cu cut-off grade and included mining tonnage dilution of an additional 5% (at zero grade) and mining recovery losses of 5%.

This estimate followed a similar conventional process as adopted for the updated estimate provided in this Technical Report, in respect of accounting for modifying factors through optimisations, staged pit designs and life of mine production schedules.

The 2012 Mineral Reserve estimate (Table 6-4, DumpSolver, December 2012) is now replaced by the estimate reported and tabulated under Item 15.

Table 6-4 Trident Project, Enterprise Mineral Reserve statement, at November 2012

Mineral Reserve statement as at Nov. 2012 at 0.1% Ni cutoff			
Classification	**Material**	**Tonnes (Mt)**	**Ni (%)**
Proved	Non-primary sulphide		
Proved	Primary sulphide		
	Proved subtotal	**2.7**	**1.44**
Probable	Non-primary sulphide		
Probable	Primary sulphide		
	Probable subtotal	**30.0**	**1.08**
	Proved and Probable total	**32.7**	**1.11**

6.3 Production history

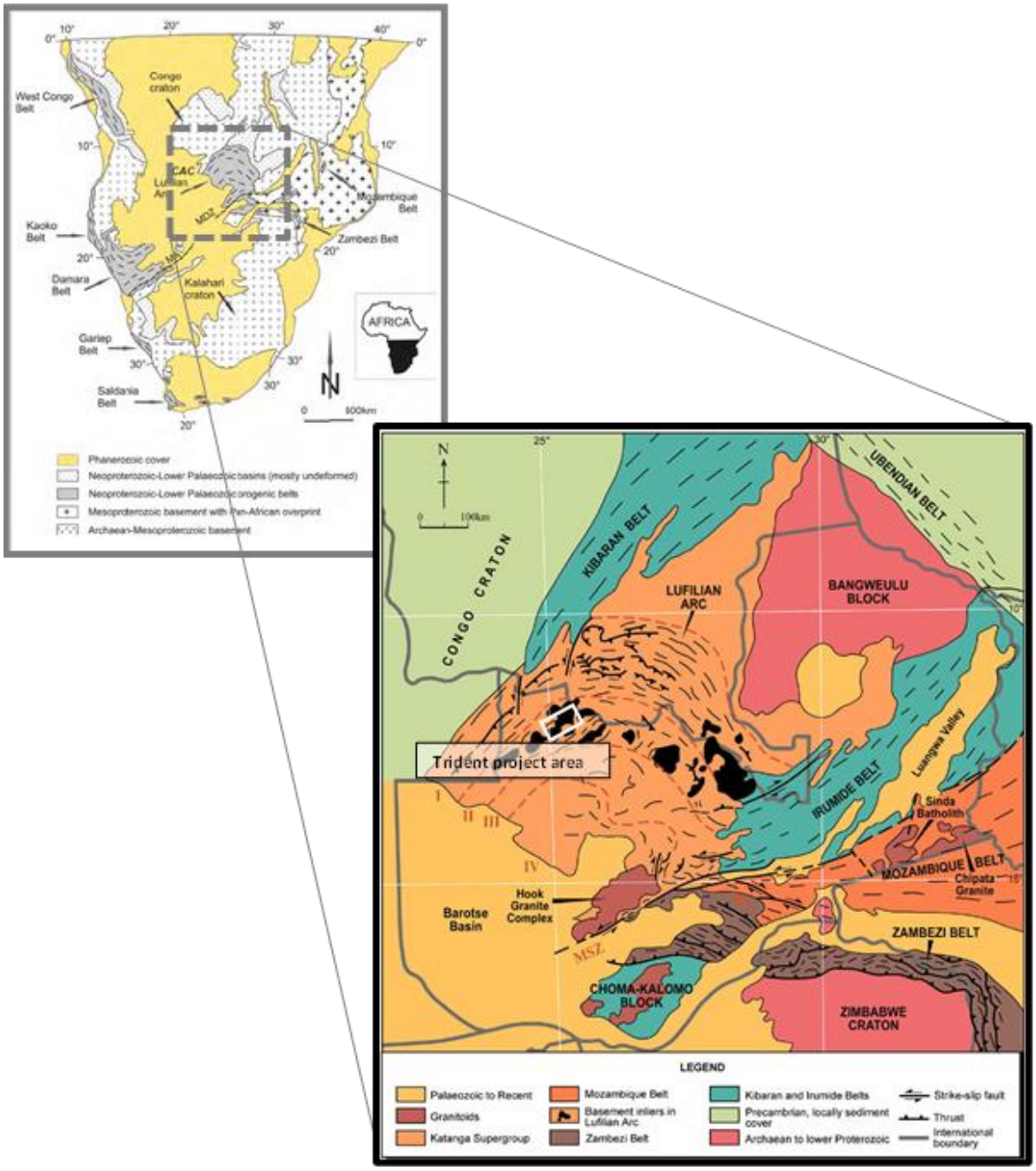
To date the Trident Project has been proceeding through the construction and commissioning phase at Sentinel, together with site development and early mining start-up activities. A start-up time for Enterprise is yet to be determined.

ITEM 7 GEOLOGICAL SETTING AND MINERALISATION

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.

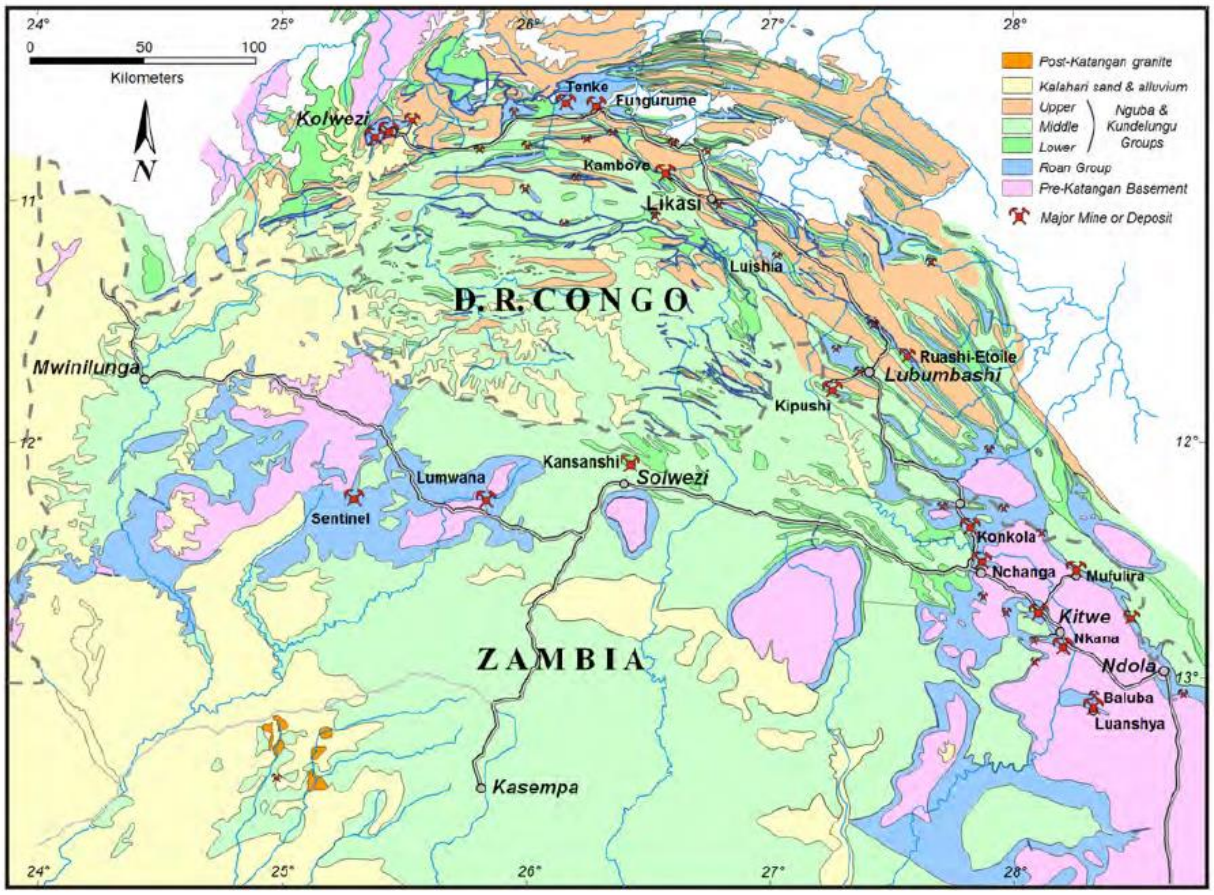
Figure 7-1 Tectonic setting of the Lufilian Arc and Trident Project area



The Project area 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 whiteshist 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)



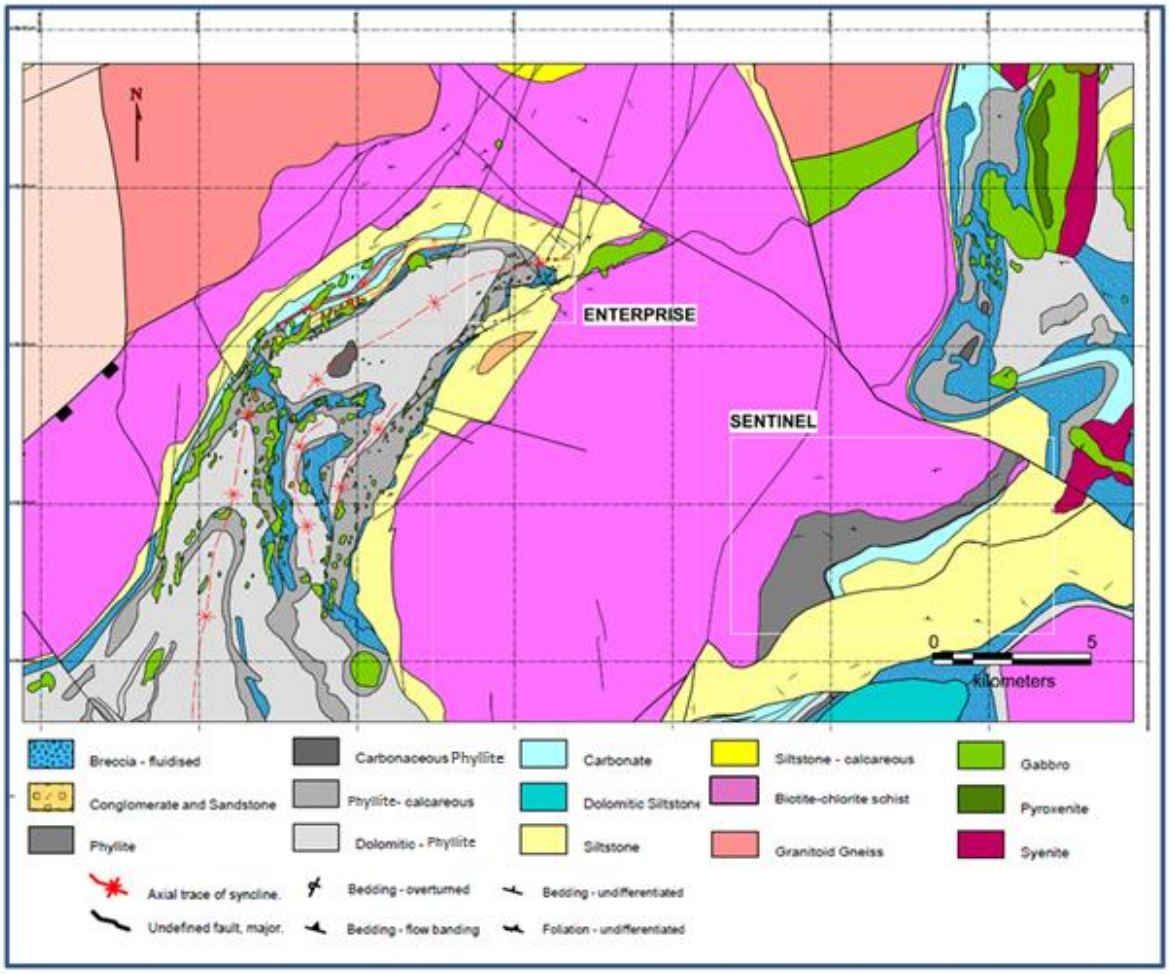
The basement rocks include fine- to coarse-grained biotite gneiss with local amphibolites, ultramafic and granite gneiss bodies, as well as more extensive schist units with varying proportions of phlogopite-muscovite and kyanite. Basement rocks within the Zambian Copperbelt have been interpreted to have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

7.2 Sentinel local and property geology

Sentinel is located to the southeast of the Trident Project area (Figure 7-3), with the deposit hosted within the 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

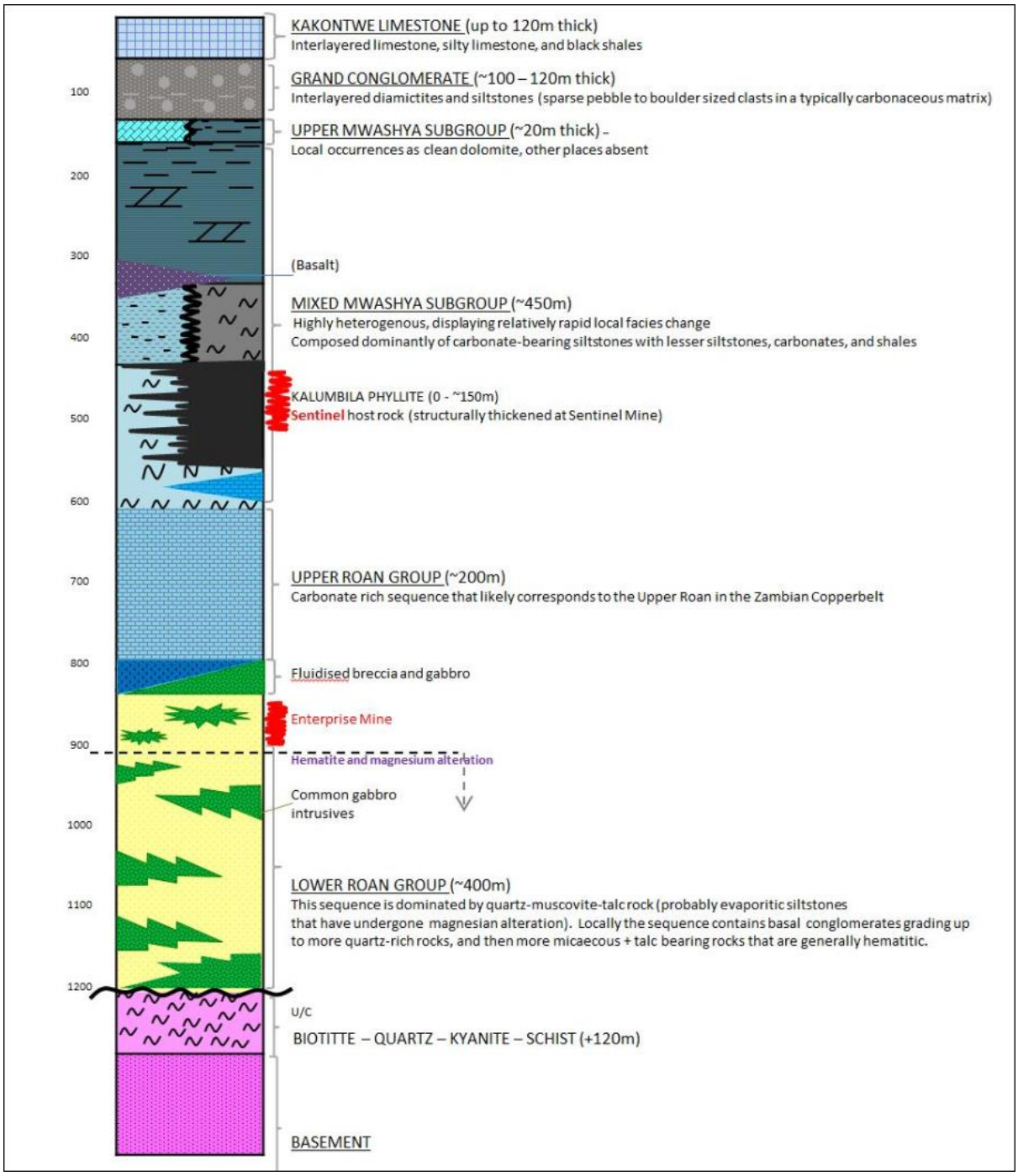


The geometry of the host phyllite is defined by recumbent, typically asymmetric, folds and several detachments that truncate the lower limbs of these folds. The phyllite is terminated to the northeast by the northwest-southeast trending Kalumbila Fault and to the south by a sub east-west cross-cutting 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 hanging-wall and foot-wall rocks are both quartz-feldspar-biotite schists, displaying strong petrographic similarities. Interpretation of diamond drill core drilled within and proximal to the Trident Project area suggests the hanging-wall schist immediately north of Sentinel represents the Mwashya subgroup into Grand Conglomerate, with the thin footwall schist and silt-/sand-stone packages to the south representing the Upper Roan Group.

Overburden and regolith across the deposit is typically 0-5 m, with in situ laterite and saprolitic layers underlying woodland and tall grassland.

7.2.1 Structural geology

The geometry of the Kalumbila phyllite host at Sentinel is characterised by recumbent, typically asymmetric, folds and several detachments that often truncate the lower limbs of these recumbent folds. The S-shape surface expression of the phyllite is indicative of non-cylindrical folding, likely resulting from differences in speeds of transport along the fold axis 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 drag folds. The recumbent folding, detachments, and dominant foliation are likely expressions of top-north directed shearing. Both low-angle thrusting along detachment horizons and shear-related folding resulted in significant local thickening of the original phyllite stratigraphy at Sentinel (Figure 7-5 and Figure 7-6).

The detachment horizons and the dominant foliation likely served as major 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 of the folds. Modelling the distribution of copper was therefore based on following these two structural trends, while respecting their east-west trending intersection lineation as one of the main controls on the generally east-west striking ore-body.

7.2.2 Alteration

Coarse centimetre scale, often euhedral, blebs of kyanite alteration are very common throughout the phyllite matrix. Kyanite occurrence is broadly spatially associated with copper mineralization, though kyanite intensity does not appear to be related 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 hanging-wall schist. These intensely altered lenses are devoid of any mineralization.

Round, mm-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 Mineralisation

Copper mineralisation at Sentinel is limited to the strongly deformed phyllite unit, with rare low-grade mineralisation extending only 1-2 metres into the hanging and foot-wall from the contact. The ore-body strikes approximately east-west for 11 km and mineralised horizons dip 20 to 30 degrees in a northerly direction, generally parallel to the dominant foliation (Figure 14-4).

The dominant copper-bearing mineral is chalcopyrite and typically 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 7-5 Schematic model outlining the structural evolution of the Sentinel deposit

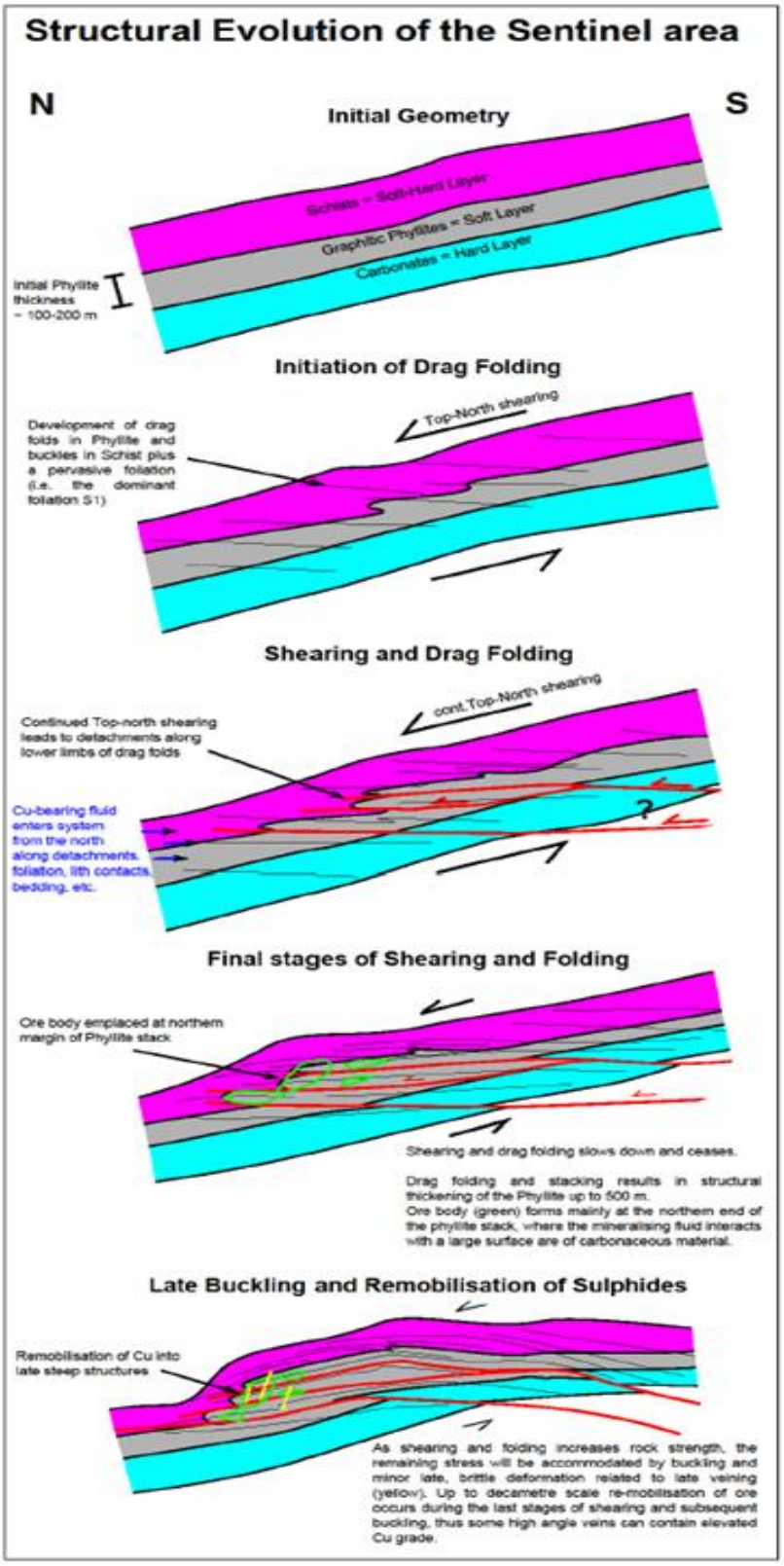
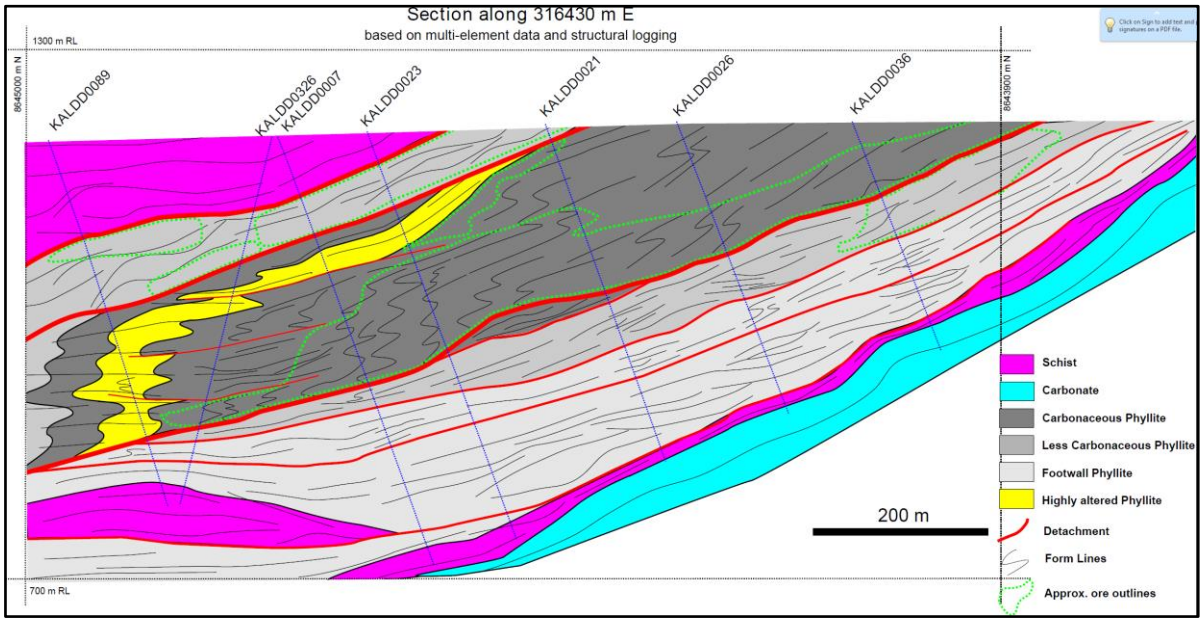


Figure 7-6 Example cross section, looking east, through the centre of the Sentinel deposit – detachment locations and fold form lines derived from diamond drill core logging with support from geochemical multi-element data



The oxidised horizon, up to approximately 70 m in depth, contains non-primary sulphide Cu-minerals, predominantly chalcocite, and tarnished chalcopyrite. The top 5-15 metres from surface is typically leached of copper, or contains mixed refractory copper and trace oxide minerals (Figure 14-4).

Nickel-cobalt mineralisation exists predominantly in the form of cobalt-pentlandite, with trace amounts of vaesite. Apart from rare sporadic metre-scales lenses (likely related to structures) the Ni-Co mineralisation occurs as a discrete horizon within the 'footwall' phyllite. 'Footwall' phyllite refers to the lowermost portion of phyllite that tends to be barren, or very low in copper mineralisation. Ni-Co mineralisation is best developed in the NE 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 mineralisation), 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 to complete modification and recrystallisation during intense metasomatism. Sedimentary textures are largely destroyed. Widespread and intense talc and carbonate alteration suggests that the alteration fluid was a saline brine in composition. This is further supported by the presence of hydrothermal vein and vein selvedge kyanite.

Enterprise lies on the southern limb of a gentle kilometre-scale syncline (Figure 7-3). It is located 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 at Enterprise is strongly altered, and in many places the original composition of the rock is unknown. Black shale is locally recognisable, but has been recrystallised to a granoblastic quartz rock, which grades into a quartz-kyanite-talc rock (± sulphide) where alteration is more intense. What are thought to be mafic rocks are abundant, and spatially associated with sulphide and iron oxides. These mafic remnants have been completely altered to chlorite-actinolite ± magnetite ± hematite. The mafic rocks are sills within the siliciclastic sequence, and form irregular discontinuous bodies. The host package is intensely altered, but a reasonably consistent stratigraphy exists, and most contacts reflect primary contacts. Below the siliciclastic sequence is a quartz-phlogopite/biotite rock that is strongly foliated. 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 following information is summarised from Wood and Outhwaite (2010).

The Kabompo Dome is a gentle doubly plunging, northeast-southwest trending structure that exposes gneissic basement in its core. The basal sequence of sandstones, siltstones and conglomerates grade up into a mixed siliciclastic-carbonate-evaporite sequence now represented by quartz-feldspar-phlogopite schists, dolomites, talc-anhydrite rock and white schists. This sequence appears to grade upwards into marbles and phyllites. 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 phlogophite-muscovite and kyanite.

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.

Upright folds in the basement complex reorient the S1B migmatitic layering. S2B is axial planar to the folds and have open to closed geometry and a gentle plunge. In higher strain S2B zones, the folds are transposed, and have rootless and intrafolial fold geometry. Upright folds in the basement were observed only at the centimetre-scale, the presence of larger upright basement folds has not been confirmed.

Recumbent folds were observed in the phlogopite-quartz horizon, about 50 m above the basement contact. The recumbent folds are intrafolial, and have open to close hinge geometries. S1K is an axial planar fabric to these folds, and is co-planar along the limbs. In the hinges of the recumbent folds, later formed S1K overprints early formed S1K. This fold-fabric relationship is a common feature of folds developed in shear zones. The axes of the recumbent folds plunge shallowly to the south to southeast, parallel to the mica-quartz mineral lineation on S1K. Locally developed asymmetric recumbent folds indicate a north directed tectonic transport direction.

7.3.2 Alteration

The mineralogy associated with large-scale alteration of the host sequence (10's km scale) is not significantly different from alteration proximal to sulphide mineralisation. 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 now difficult to recognise, as it has been locally transformed to a quartz-kyanite rock in mineralised zones. Layered specular hematite–talc rock is characteristic of the host sequence, and is thought to represent an early stage of the alteration paragenesis. Where sulphide mineralisation occurs, hematite has been removed/destroyed. Phlogopite occurs for tens of metres around sulphide mineralisation as clots or in veins.

7.3.3 Mineralisation

Nickel sulphide minerals include vaesite (NiS_2), millerite (NiS), nickeliferous pyrite ($[Fe,Ni]S_2$), bravoite ($[Ni,Fe]S_2$) and carrollite ($Cu[Co,Ni]_2S_4$). Sulphide mineralisation occurs within, or as an alteration halo to, quartz-kyanite ± talc veins and vein breccias. Sulphides are concentrated within and proximal to altered black shales and carbonates, but can occur in lesser amounts in siliciclastic rocks.

Mineralisation is hydrothermal in style, and appears to be related to a series of deep seated structures that have an extensional regime. The dominant control on the location of mineralisation appears to be a combination of structure, rheology and chemistry. The black shales that host the majority of mineralisation are a favourable rheology for brittle deformation, as well as being a reductant for probable oxidised hydrothermal fluids. Carbonate units proximal to black shales sometimes host sporadic bonanza grades of Ni. Other siliciclastic units can be mineralised, but typically have much lower concentrations of Ni than the black shale.

Relatively minor copper mineralisation occurs at 350m below surface and is underlying the known nickel mineralisation. Copper mineralisation of up to 2.80% has also been intersected during drilling at Enterprise, with intercepts of 5 m @ 1.27% Cu as per drill hole ENTDD0120, and 14 m @ 1.72% Cu from ENTDD0019. Copper minerals identified 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. Mineralisation is predominantly primary sulphide copper, with sheet-like horizons of ore dipping north at 20-30^{o}. The mineralisation is parallel to dominant foliation, within the structurally deformed carbonaceous phyllite host.

The geometry of the host phyllite is defined by recumbent, typically asymmetric, folds and several detachments that truncate the lower limbs of these folds.

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 mineralised horizons, to ensure intersection of ore at a high angle.

Detailed drill core logging and measurements, with supporting multi-element data, allowed for the identification of deposit-scale recumbent drag folds truncated by major detachments within the phyllite. The distribution of copper is closely related to this structural framework, with the main volume of chalcopyrite-bearing veinlets occurring parallel to bedding or to the axial planar foliation of the folds. Mineralisation is often truncated along its northern, down-dip extents by the nose of a drag fold, and these fold noses typically contain higher grade mineralisation.

Modelling the distribution of copper, using low grade and high grade wireframes, was therefore based on following these structural trends, while respecting their east-west trending intersection lineation as one of the main controls on the general strike of the deposit.

8.2 Enterprise

8.2.1 Mineral deposit type

Enterprise has been interpreted to be a hydrothermal nickel deposit with mineralisation hosted in a sequence of siltstone, carbonate and shale units. The shales and shale-carbonates have been preferentially mineralised due to a combination of rheological and geochemical interactions with the mineralising fluids. Enterprise is unusual in the lack of a clear spatial control of mafic intrusives on the mineralisation, and the primary source of the nickel mineralisation remains unclear. The presence of post-mineralisation structural deformation (faulting and folding) has been noted from core logging and has influenced geological models with graben-like features and complex folding.

8.2.2 Guiding principles for exploration and modelling

The exploration programme at Enterprise was based upon observations that the mineralisation was sediment-hosted and located within an apparent fault bounded graben block. Mineralisation appears to be related to a series of deep-seated fault structures with an extensional structural framework. The dominant control on location of mineralisation has been interpreted to be a combination of structural, rheological and chemical criterion. The black shales were a favourable

rheology for brittle deformation as well as a reductant for oxidised hydrothermal fluids. The shales tend to host most of the mineralisation. Carbonate units proximal to black shales sometimes host bonanza style grades of Ni sulphides. Other siliciclastic units can be mineralised, but typically have much lower Ni tenor than the black shales. The primary source of the Ni is unclear; however there appears to be a spatial correlation with mafic occurrences and the regional fault intersections.

As such, modelling of mineralisation has been guided by the dominant graben fault controls and the shape of the overlying and underlying lithology controls.

ITEM 9 **EXPLORATION**

The Company has not completed any near deposit exploration at Sentinel since May 2012. Similarly, at Enterprise, the Company has not completed any exploration since December 2012. Infill drilling, however, has been completed within the Sentinel and Enterprise deposit areas since those dates, and is described in Item 10.

The following information, relevant to the development of updated geological models, is reproduced from the previous Technical Reports (CSA, May and December 2012) as a summary of the exploration work completed at Sentinel and Enterprise.

During the exploration phase within the Enterprise and Sentinel deposit areas, a comprehensive soil geochemical sampling programme and multiple geophysical surveys were completed. A continuous programme of outcrop mapping was also carried out. Information from these programmes was used to plan the drilling and sampling phase.

Various geophysical surveys were completed during 2010:

1. A LIDAR survey was flown by Fugro Maps South Africa, providing the mine planning team with a high-resolution topographic elevation surface.
2. 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.
3. 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 these surveys, outcrop mapping and drillhole logging data.

ITEM 10 **DRILLING**

10.1 **Sentinel**

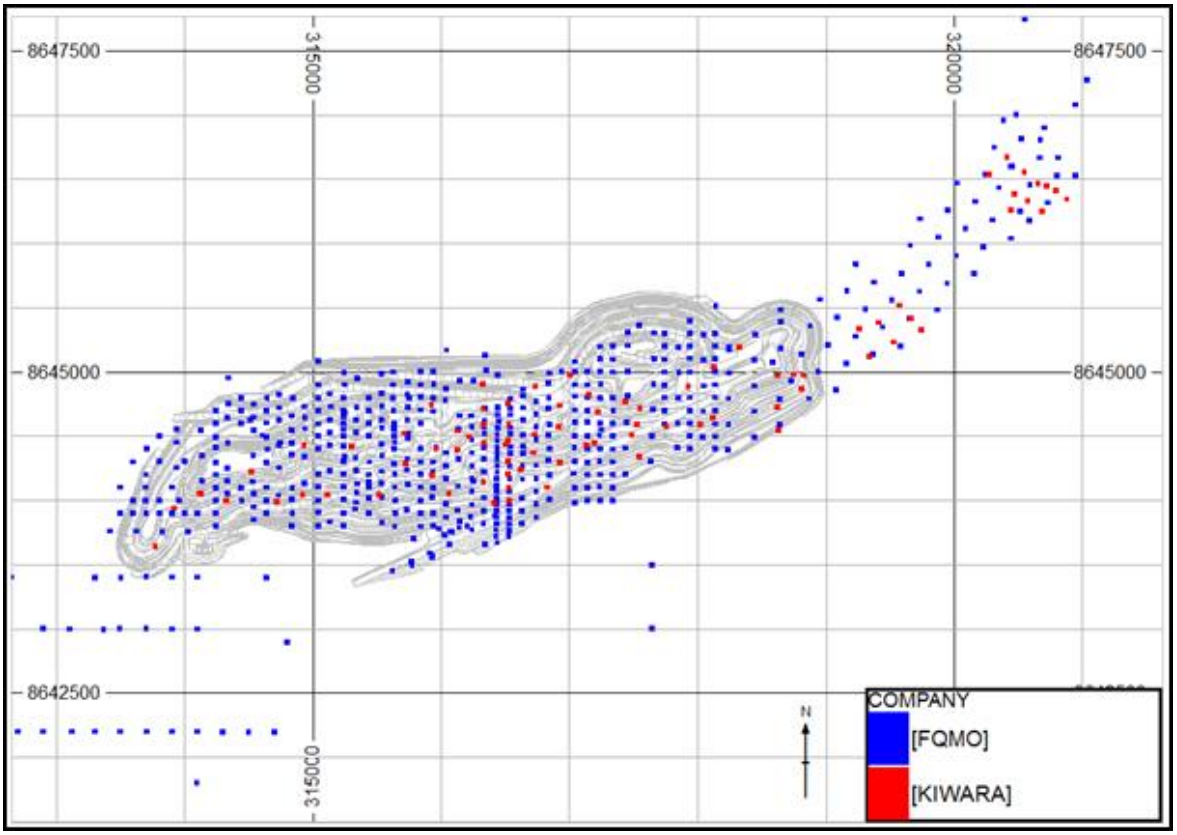
10.1.1 **Historical drilling currently used**

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 a nickel resource at Sentinel. However, only holes drilled by Kiwara PLC (2007 to 2010) and FQM (2010 to current date) were used in the copper Mineral Resource estimate update for this Technical Report.

Historical holes drilled by RST and AA were deemed to be of sub-standard quality to support robust Mineral Resource estimates.

A total of 677 diamond drilled holes for 229,713 m and 146,986 assayed samples were used in the updated estimate (Figure 10-1). Samples 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.

Figure 10-1 Diamond drillhole collar plan across the Sentinel deposit area, showing the Kiwara and FQM holes used in the Mineral Resource estimate



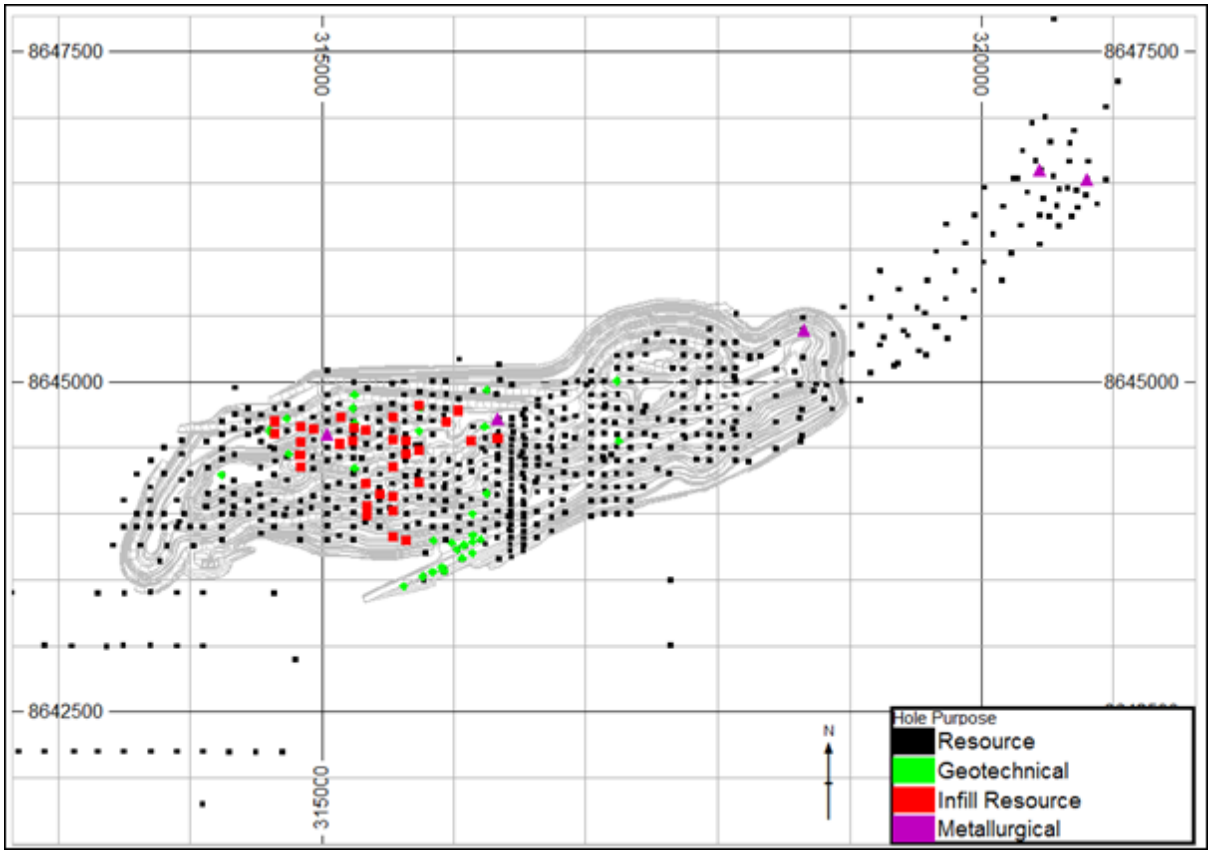
Additionally, as a standard, drillhole core was logged for lithology, regolith, oxidation, alteration, mineralisation and structure. Holes were also logged for geotechnical data, including core recovery and rock quality.

10.1.2 Recent drilling by the Company

Since the 2012 Technical Report, there has been additional resource development drilling of the Sentinel deposit. A total of 72 diamond drillholes were drilled for infill, geotechnical and metallurgical purposes within the life of mine (LOM) area.

Of these 72 holes (Figure 10-2), 34 were drilled as infill, aimed at better defining the extent and continuity of mineralisation in areas of the deposit which required additional data. 33 geotechnical holes were drilled to provide information and data for pit slope design and for inpit crusher pocket design, whilst five metallurgical drillholes provided samples for metallurgical testwork.

Figure 10-2 Plan view of the Sentinel deposit area highlighting the life of mine pit area and the holes drilled since the previous estimate in November 2011



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 around a 150 m depth to the end of the hole. Improved rock competency with depth allowed for the drill run length of both HQ and NQ to be increased to 3m. No drilling utilised a core size less than NQ.

10.1.3 Drilling orientation

The infill and metallurgical holes were drilled at an inclination of 70° towards an azimuth of 180° in order to maximise the angle of intersection to mineralisation. Sentinel mineralisation has a general east-west strike and north-northwest, shallow dipping nature. Geotechnical drillholes were drilled at different inclinations and directions according to their planned targets (eg, crusher pockets or pit wall design).

10.1.4 Collar surveys

Drillhole collars were surveyed by a registered surveyor (Surtech International Limited) using a Differential GPS. All coordinates were surveyed in ARC 1950(mean) 35S coordinate system. All surveyed drillhole collar coordinates were validated against the Lidar topographical survey with no deviations noted.

10.1.5 Downhole surveys

Downhole surveys were conducted on all drillholes by the drilling contractors using the Reflex EZ Shot, an electronic single shot (point), magnetic survey tool. During drilling, surveys were taken at 50 m intervals in order to record downhole deviations. In areas of historical deviations, an initial 25 m survey was taken to monitor deviation at the start of hole. Where deviation was greater than 10 degrees, these surveys were flagged for re-survey or re-drill, depending on the extent and details of the deviation.

10.1.6 Core orientations

The drilling contractors were tasked with ensuring core trays were labelled with the correct drillhole 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' are dependent on the confidence of the orientations, where a minimum of two orientation points must be within 10° to be considered of sufficient confidence. A solid continuous line, drawn using a permanent marker represents high confidence orientations, whilst medium and no confidence were marked as dashed and dotted lines respectively. This was completed each day and monitored by the responsible geologist.

10.1.7 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 on paper and then translated into the Sentinel Datashed database.

In total, 10,024 density measurements were taken allowing for a sufficient number of samples available per key lithology and weathering domain.

Drill core recovery ranged from 76% in the weathered horizons to 98% in fresh rock. Average core recovery was 93%, with 85% of all core having a recovery greater than 90%.

10.1.8 Geological Data

All drillholes were logged by qualified geologists for lithology, regolith, oxidation, alteration and mineralisation. Structural features 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.

All 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.

It is the opinion of the QP, David Gray, that drilling of core samples were adequately monitored and oriented for supporting accurate 3D downhole sample positions. Drilled core was also properly logged for RQD and geology and was sampled for density and element analysis. There are no known drilling factors that could materially affect the accuracy and reliability of this Mineral Resource estimates results

10.2 Enterprise

10.2.1 Summary of recent FQM drilling

Drilling at Enterprise between 2011 and 2013 was exclusively diamond drilling and commenced with a PQ diameter which was reduced to HQ and then to NQ from approximately 200 m depths. All drilling used triple tube to minimise core loss in the upper parts of the hole, which was strongly affected by weathering.

Holes were drilled at a dip of 60°to the south-east with a few steeper drilled holes. The direction (south east) and angle of drilling was used in order to achieve a maximum angle of intersection with mineralisation.

The exploration drilling strategy at Enterprise involved targeting the nickel mineralisation intersected in the historical wide spaced Kiwara holes with infill holes on a 50 to 100 m grid up/down dip and along strike from the past intersections. Drilling covered the initial approximately 2 km zone as defined by anomalous Ni and Cu values from soil sampling. The drilling aimed to connect the intermittent zones of mineralisation.

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 (Figure 7-4), even though this involved drilling through much barren rock beneath the ore zone. This methodology enabled the identification of the Enterprise “graben” like structure, known to have a strong control on the distribution of mineralisation. 50 m infill lines were drilled from late 2011 onwards and the target depth was reduced to the top of the quartzite lithology beneath the mineralised zone.

The number of additional Ni and Cu samples available for the current Mineral Resource estimate is greater than the total metres drilled since the previous estimate. This is due to holes being drilled in 2012, and used to assist the previous interpretation, but with delays in assays being returned from the laboratory preventing their use in the previous estimate. Assays that were unavailable for the previous estimate, but were from holes drilled in 2012, are presented in separate line entries in Table 10-1.

Enterprise comprises two areas – Enterprise Main and Enterprise South West (referred to as Enterprise SW in this Technical Report). The average and maximum drill hole depths used in the

Enterprise Mineral Resource estimate are shown in Table 10-2, whereas a plan view of the drill collar locations are shown in

Figure 10-3.

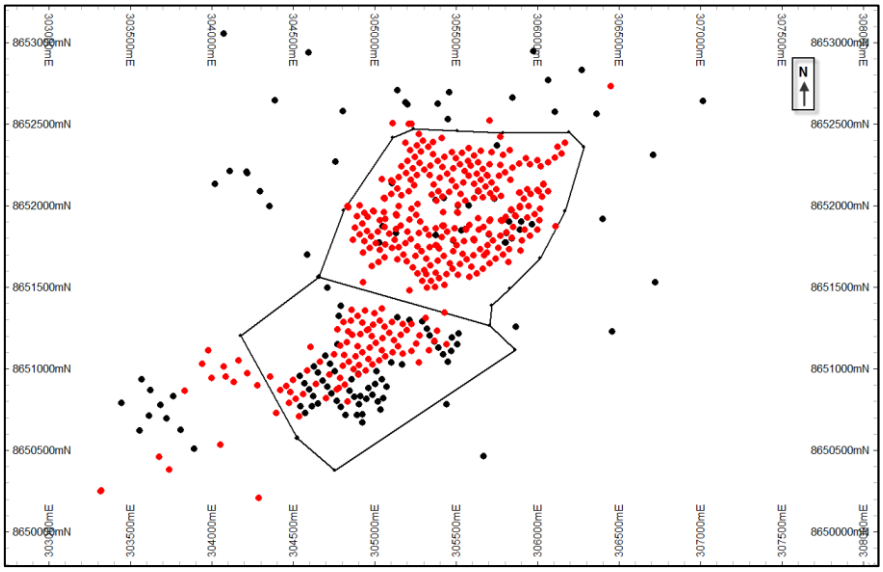
Table 10-1 Table of drilling completed by FQM, and including 11 drill holes completed by Kiwara PLC

Mineral Resource estimate	Area	Number of holes	Number of metres	Number of Ni samples	Number of Cu samples
Previous	Enterprise Main	299	93,035	85,239	75,272
	Enterprise SW	94	31,160	21,991	15,980
	Total	393	124,195	107,230	91,252
Current	Enterprise Main	319	100,294	90,039	90,039
	Enterprise SW	155	47,872	43,905	43,905
	Total	474	148,166	133,944	133,944
Additional data for current update (*including those assays carried forward from 2012)	Enterprise Main	20	7,259	4,800	7,259
	Enterprise SW	61	16,712	16,712	7,508
	Enterprise Main*			-	20,417
	Enterprise SW*			5,202	11,213
	Total	81	23,971	26,714	42,692

Table 10-2 Average and maximum drill hole depths for holes used in the Enterprise Mineral Resource estimate

Area	Average drill hole depth (m)	Maximum drill hole depth (m)
Enterprise Main	314	745
Enterprise SW	308	696

Figure 10-3 Plan view of drill collars for the mineralisation limits of Enterprise Main (top polygon) and Enterprise SW (bottom polygon). The 2013 drill data coloured in black



10.2.2 Core recovery analysis

Core recovery data has been reviewed by the QP. Conclusions were that there is no drilling, sampling, or recovery factors (field errors) that could materially impact the accuracy and reliability of 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. Referring to Table 10-3, Enterprise Main shows higher average grades in PQ diameter core which is attributed to three holes (ENTDD0390, 393 and 406) drilled in 2013, which have intersected high grade near surface mineralisation, and with ENTDD0393 in particular being drilled down the dip of the ore body. At Enterprise SW higher grades are expected in HQ when compared to PQ due to most of the mineralisation being below the highly weathered surface.

Table 10-3 Summary statistics for core diameter versus grade review

Area	Diameter	Count	Average Recovery %	Ni% (Weighted)
Enterprise Main	HQ	5,247	93.8	0.02
	NQ	3,836	96.4	0.01
	PQ	7,045	71.4	0.07
	TOTAL	16,128	88.3	0.03
Enterprise SW	HQ	11,468	96.0	0.13
	NQ	1,1407	97.1	0.07
	PQ	12,072	76.4	0.05
	TOTAL	34,947	92.4	0.09
Waste material	HQ	1,334	93.5	0.01
	NQ	655	96.4	0.01
	PQ	1,784	73.7	0.01
	TOTAL	3,773	88.5	0.01
Grand Total		54,848	91	0.06

10.2.3 Collar surveys

Collar surveys were measured by Surtech International Limited, trading as Ian Robinson Surveys. 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.

10.2.4 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 were to survey drill holes on fifty metre intervals whilst the hole was being drilled in order to monitor down hole deviation as the hole progressed. A re-survey would be demanded if the measured magnetic field at the time of the survey measurement was deemed too high (> 10 degrees), due to magnetic interference as a result of proximity to magnetic rocks or that the core barrel had not been retracted far enough when performing the survey. Surveys were also flagged for investigation if their deviation in either heading or azimuth was greater than five degrees over fifty metres.

10.2.5 Oriented core

All HQ and NQ core was oriented where possible at the end of every drilling run (usually 3 m lengths unless ground conditions dictated otherwise). No attempt to orient PQ core was made as this core was usually in the weathered, upper portions. Initially all orientations were conducted using the 'spear test' technique, however this was abandoned early on in the drilling campaign due to unsatisfactory results. The decision was made to revert to digital orientation instruments, namely the Reflex ACT orientation instrument and the Reflex ACT II RD orientation instrument.

10.2.6 Core handling, density measurements and RQD determination

All core was returned to the safe and secure (locked) core shed at the Enterprise exploration camp.

Suitably trained field technicians would measure and then calculate the Recovery and Rock Quality Designation (RQD), and measure and mark every 1 m point down the hole. Magnetic susceptibility measurements were obtained on the metre marks for the entirety of the each hole, and specific gravity (SG) of selected intervals of no less than 20 cm in length at a frequency of one interval per ten metres were also routinely collected. Full core photography, both wet and dry, was captured by a suitably trained technician using a SLR digital camera to obtain high resolution digital photographs. The photos are renamed according to the drill hole ID and the depth interval displayed on the core tray. Core Photographs are available on the site server and were linked to the Datashed database.

10.2.7 Geological and structural logging of core

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

ITEM 11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sentinel

11.1.1 Diamond core logging and sample preparation

Drilled diamond core was returned to the core shed at the Enterprise exploration camp and logged for lithology, mineralisation, alteration and structure by suitably qualified geologists using a standardised logging code system to ensure consistency. This data was managed and validated in the site DataShed database.

Recovery and Rock Quality Designation (RQD) was calculated and diamond drilled core was marked out by the geologist using 1 m sample lengths and the prevailing geology contacts. The full length of each hole was sampled. Full (un-cut) core was photographed wet and dry prior to any sampling. The marked diamond drilled core was cut in half using a diamond saw and then bagged according to and with their 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 been sampled. Bagged samples were also assigned a batch number per hole. Each holes' samples were packed into several larger polyweave bags, detailing the batch number and the individual samples contained.

The polyweave bagged samples were transported to the ALS preparation laboratory facility at Kansanshi mine for preparation.

Samples were prepared by crushing the drill core to 90% passing 2 mm using a jaw crusher. The crushed sample was split through a single tier splitter to a 500 g sample mass. LM2 pulverisers were used to produce a pulp sample with 90% passing 106 microns. Approximately 250 g of pulp were submitted for ALS Chemex for analysis.

The pulp samples were despatched via Mercury couriers and DHL to the ALS Chemex laboratory in Kansanshi or Johannesburg. All pulp rejects were returned to and stored on site. It is David Gray's (QP) opinion 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.

11.1.2 Diamond core sample analysis

The additional samples from November 2011 were all assayed at the ALS Chemex Kansanshi laboratory in Zambia and used a 50 g charge subjected to a 4 acid digest and followed by an ICPES finish. While the ALS Kansanshi laboratory is not accredited, it is independently managed by ALS personnel. In addition, and in order to minimise risk of inadequate sample QAQC, FQM completed a comprehensive programme of umpire check analysis of pulp duplicates using the accredited ALS laboratory in Johannesburg. The umpire duplicate results demonstrated adequate precision with more than 90% of sample pairs having values within 10%. The results from inserted quality samples also demonstrate quality analyses that are repeatable and representative of the material sampled.

11.1.3 QAQC procedures

Diamond drilled core samples submitted from November 2011 have followed good QAQC practices with a blank and range of Certified Reference Material (CRM) samples each inserted every 40th sample. Field duplicates were completed for a sub set of the data with 685 field duplicate samples taken. Coarse crush duplicates were inserted every 28th sample and pulp duplicates were inserted every 46th sample.

Returned blank insert sample values demonstrate that contamination has been adequately controlled during sample preparation. The inserted CRM's have highlighted acceptable 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. The inserted coarse crush duplicates 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.

11.1.4 QP's opinion

Safe and secure sampling, sample data management and preparation have provided a solid foundation for consistent and repeatable sample analysis. Additionally all samples since November 2012 were analysed using the same analytical technique at ALS laboratories at Kansanshi or Johannesburg. FQM's blank, CRM, duplicate and umpire laboratory sample inserts have assured representative and repeatable results with contamination maintained to a minimum, duplicates having adequate precision and CRM samples demonstrating accurate results. It is David Gray's (QP) opinion that the Sentinel sample preparation, analysis and security is adequate for use in this Mineral Resource estimate.

11.2 Enterprise

11.2.1 Diamond core logging and sample preparation

Diamond drilled core was returned to the secure core shed at the Enterprise exploration camp and logged for lithology, mineralisation, alteration and structure by suitably qualified geologists using a standardised logging code system to ensure consistency. This data was managed and validated in the site DataShed database.

Suitably trained field technicians calculated the Recovery and Rock Quality Designation (RQD).. Specific gravity (SG) samples of at least 20 cm in length at a frequency of one per ten metres were routinely collected. Full core, both wet and dry, was photographed using a SLR digital camera to obtain high resolution digital photographs.

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. The results and the original α, β and γ measurements were stored in the Datashed database.

From the beginning of 2012, sampling of the quartz-biotite rock at the base of many holes was stopped as it was proven to be completely barren. Holes were sampled on the basis of lithology and observed mineralisation. Standard sampling lengths were 1 m in mineralised lithologies and 1 m in visually un-mineralised lithologies. No sub-metre samples were taken over higher mineralised zones by FQM, a practice which was used by Kiwara.

The soil and saprolite horizons were sampled from a halved split of the core tray material using a sharp edged 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 and to ensure that the consistent left hand half of the core was sampled.

Cut half core samples were placed into plastic bags with a sample ticket and marked using a permanent marker with a batch number, the hole and sample number. Bags were stapled shut and

samples from an entire hole were packed into a number of cable tie-sealed polyweave sacks labelled with the batch number for transportation to the preparation laboratory.

As core is dominantly aligned perpendicular to the mineralisation and the samples are half core, they are expected to be representative of the deposit with no suspected sample bias. Sample preparation was carried out at the Intertek Genalysis laboratory in Ndola, Zambia. Sample preparation consisted of drying samples at 100°C +/- 20°C, and crushing the drill core to sub-10 mm through a jaw crusher. All samples above 3 kg were riffle split to obtain sample weight between 0.6 kg to 1.2 kg. The split sample was pulverised using the Essa LM2 pulveriser so that more than85% of the material passed 75 micron. . About 150 g of pulp was packed into a pulp envelope and sent for analyses. All coarse and pulp rejects from the sample preparation process were returned to site.

The transportation and handling of samples was carried out by a representative of FQM or its designated contractors. It is Mr Gray's (QP) opinion that there is minimal chance of interference or degradation of sample quality and that samples have been adequately secured from their source to destination.

11.2.2 Diamond core sample analysis

Enterprise samples were assayed by Intertek Genalysis in Ndola, Zambia using a 25 gram charge subjected to a 3 acid digest and followed by AAS finish. Standard elements analysed were nickel and copper. The QP, David Gray notes that the Genalysis laboratory in Ndola was not accredited. As a result 10 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 4 acid digest followed by an ICP-OES finish was used. Results of the umpire checks have confirmed that the primary laboratory accuracy is adequate for using these samples in this Mineral Resource estimate update. In addition, FQM has employed comprehensive QAQC with blanks, CRM's and duplicate samples. QAQC practices and results are described in the next section, where results are noted to demonstrate quality analyses that are repeatable and representative of the material sampled.

11.2.3 QAQC procedures

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. In addition, FQM randomly selected and submitted 10% of samples to ALS Chemex in Perth, Australia for multi-element assaying and further check/umpire assessments. 64% of these check assays were on pulps from original assay samples, and 36% were re-split bulk reject samples. Multi-element analysis was carried out on all samples sent for check assays. FQM has also inserted blind standards (AMIS and OREAS certified reference material samples) and blanks at a rate of 1 in 50 as well as field duplicates during the core sampling programme. Returned standard analysis results demonstrate that the primary laboratory has acceptable precision with most values been within three standard deviations of the certified value. There was some evidence for mislabelling of CRM sample inserts. Results of the field duplicates demonstrate repeatability and precision. Inserted blank samples demonstrate that contamination was contained during sample preparation. All QAQC data was stored in the Datashed system with QAQC'R software been used for analysis of QAQC sample results.

Mislabelling of CRM sample inserts was rectified by using a coloured sticker system which has reduced the number of misallocations.

11.2.4 **QP's opinion**

It is David Gray's (QP) opinion that the sample preparation checks; check assays and analytical results, both in terms of the laboratory QAQC protocols and FQM blind inserts have been satisfactory and thus are believed to have adequate controls on sampling preparation and analysis. Enterprise sample and assay results were supported by good QAQC, the results of which support that contamination was adequately controlled and that assay values were both accurate and precise. Sample values were believed to be representative of the prevailing mineralisation 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 seven occasions over the last two and a half years, with the most recent visit in May 2015. During these visits and subsequent studies for this resource estimate, Mr Gray has gained good familiarity and confidence in the available diamond data, the geology models and prevailing mineralisation as exposed in the start-up pit exposures.

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 visual 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
  - assay data in the database reflect data in the original assay certificates
  - a check on the relative magnitudes of downhole survey data in order to identify erroneous data.
- QAQC data was investigated together with the processes used for analysis and were verified as robust for assuring assay accuracy, precision with evidence for control of contamination.
- In-pit observations have served to verify the prevailing geology and its association with the different styles of mineralisation 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 core sampling markings. Mineralisation and geology as modelled for this resource estimate update, was verified from mapped data and visual observations of in-pit exposures. David Gray (QP) believes that the available data used for the Sentinel Mineral Resource estimate is of adequate quality to represent the mineralisation of the Sentinel deposit.

12.2 **Enterprise**

David Gray (QP) has visited the site of the Enterprise deposit on several occasions over the last two and half years, with his most recent visit being in May 2015. During this visit and subsequent studies, Mr Gray has gained confidence in the available data, geology and prevailing mineralisation. The following verifications were completed:

12.2.1 **Database verification**

All geological logging data and assays are imported into the site Datashed database. This process is an automatic import of the data file as sent by the laboratory or the logging and sampling sheets as completed by the geologist responsible for logging each hole. Access to the database is restricted to read-only access to all staff with the exception of the database administrator. Assay data is completely locked and cannot be manually modified once imported into the database. Superseded results are also stored in the database to preserve the audit chain. All actions performed on the

data within the database are automatically recorded with details as to the user who modified the data as well as a time and date that the data was modified so as to enable results stored within the database to be auditable.

Data verification carried out includes checking of the assay results against the laboratory certificates for approximately 1% of all samples stored in the database. This is achieved by selecting around 10% of the holes at random and then verifying the assay result on the certificate against that stored in the site database. No data loading errors were discovered in this process.

12.2.2 2012 vs 2013 data comparison

Grade distributions comparing 2012 and 2013 drill data show 2012 data reporting higher grades than 2013. This is due to the focus of 2013 drilling being on lower grade extensional drilling, particularly in Enterprise SW, rather than any data bias.

12.2.3 QAQC data

The QP supervised the CSA review of the Enterprise QAQC reports. FQM's QAQC reports combined Enterprise and Sentinel diamond drill core sampling. No fatal flaws were noted, however, it is recommended that separate QAQC reports be completed and reviewed for each project to ensure reliable conclusions can be drawn from the appropriate dataset. Results of the QAQC review for the Trident Project are summarised below.

The majority of the analyses for the Trident Project were completed at Genalysis Ndola with ALS Perth used as an umpire laboratory. While the Genalysis laboratory was uncertified, a comprehensive QAQC programme was used in order to provide sufficient confidence to the assay results. QAQC reports were generated and reviewed monthly by FQM with any issues highlighted, usually dealt with in a timely manner.

Table 12-1 (compiled from monthly QAQC reports provided by FQM) summarises the QAQC material inserted and analysed.

Table 12-1 QAQC Summary for Trident Project samples (%'s are of Drill Samples).

Lab Code	Total Batches	Total Samples	Drill Samples	Field Duplicates[1]		Standards		Lab Pulp Splits		Lab QC	
GEN_TRI	433	67,441	54,086	2,304	4.26%	3,203	5.92%	531	0.98%	4,759	8.80%

[1] Mixture of quarter core and half core

12.2.4 QP's Opinion

Review of the QAQC results, data management procedure and database security supports quality data representative of in-situ mineralisation. In addition, the QP, David Gray has completed several phases of visual inspections of drill core, drill collars and outcropping geology. David Gray is confident that the data and information available is 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 2012 Technical Reports (DumpSolver, May and December 2012) and updated by Andrew Briggs (QP).

13.1 Sentinel

From an 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 copper belt, 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 deslime
- concentrate handling
- tailings disposal
- reagent mixing, storage and distribution
- water and power supply
- other services

Metallurgical testwork was carried out in three phases:

1. The first phase of the testwork was conducted at the Company's Kansanshi operation and were scoping in nature. The sample locations for this programme are shown on Figure 13-1. The samples were selected from holes L028, L043, L057, L063 and L065. Holes L043 and LO65 are located within the initial stage 1 and 2 mining limits.
2. The second phase of the testwork 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 are located within the initial stage 1 and 2 mining limits. This work was performed by SGS Lakefield in Perth, beginning in May 2011, and continuing through to 2014.
3. A third phase of testwork 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

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

plant during ramp up. Figure 13-2 is a cross-section showing metallurgical sample holes within the initial stage 1 and 2 mining area.

13.1.1 Mineralogy

Mineralisation is almost all sulphidic in nature with oxide derivatives generally insignificant and limited to 10 m to 20 m below surface in the weathering profile. Copper mineralisation is exclusively chalcopyrite, with extensive zones of pyrite, particularly in the hanging wall, and with pyrrhotite occurring throughout.

Primary sulphide mineralisation 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.

Non-primary sulphide mineralisation is predominantly chalcocite and tarnished chalcopyrite.

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. Nickel mineralisation appears to be limited to the north-eastern end of the deposit.

13.1.2 Metallurgical testwork summary

Testwork conclusions

The first programme showed that a concentrate grade of 28% copper with a copper recovery of 85.7% could be produced under the following conditions:

- primary grind of 80% passing 150 µm
- starch addition as a gangue depressant of 300 to 500 g/t
- SEX (sodium ethyl xanthate) as a collector at 40 to 50 g/t
- pH in roughers of 10
- pH in cleaners of 11
- rougher flotation time (ie, in the laboratory) of about twelve minutes

These results were used as the basis for the second and third programme and allowed confirmation of the above preliminary design conditions, with additional detail regarding grinding characteristics.

Figure 13-1 Sentinel metallurgical testwork, drillhole sample locations

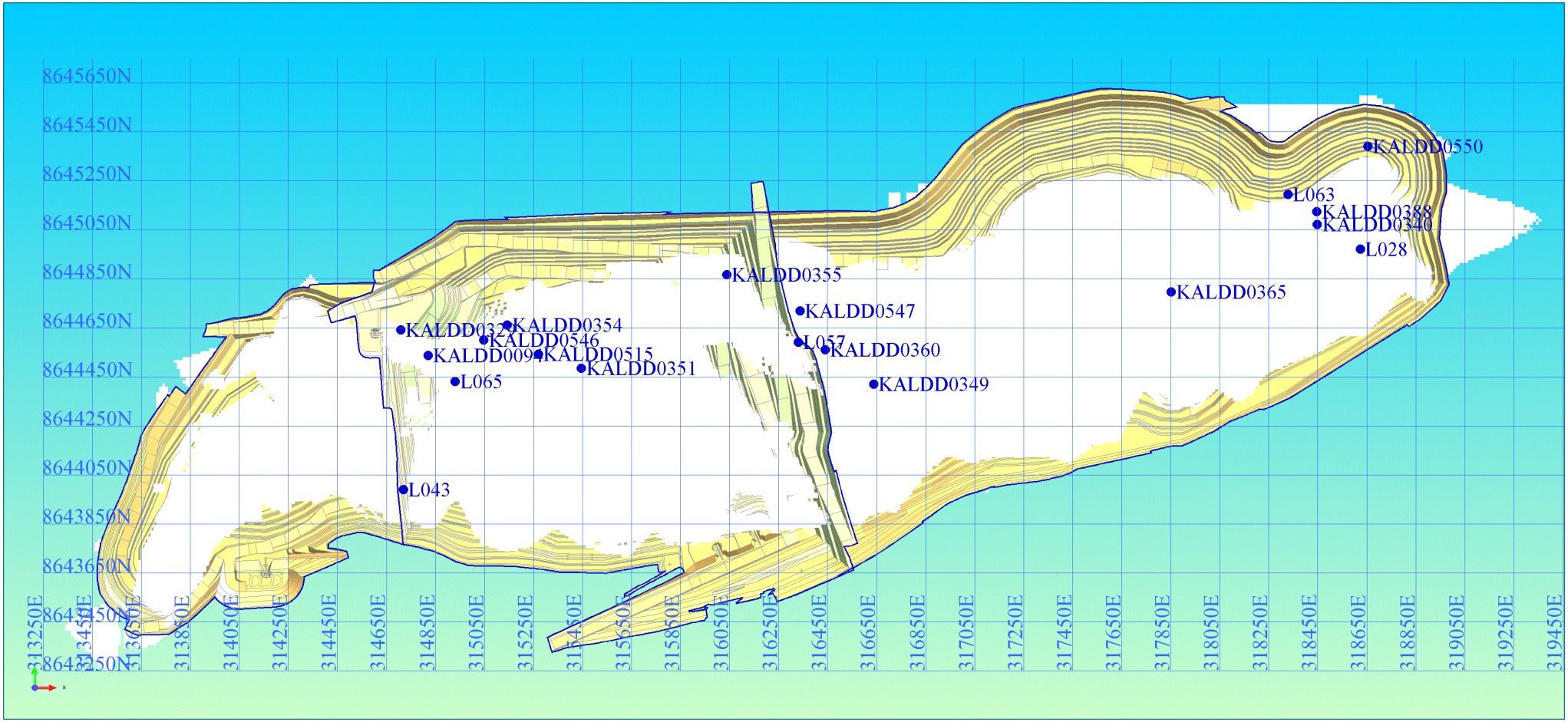
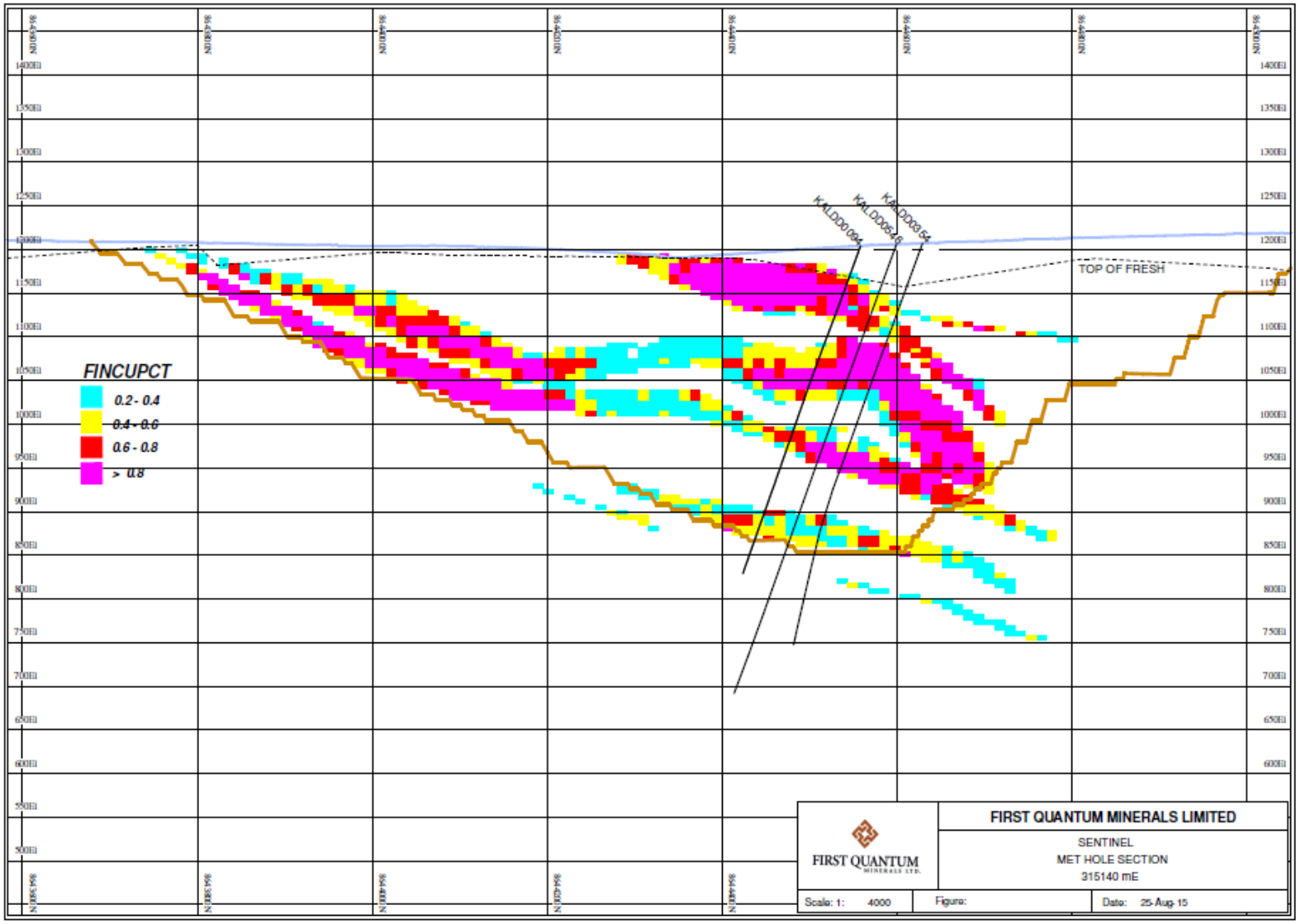


Figure 13-2 Sentinel metallurgical testwork, cross-section at 315,140 mE



The flotation testwork provided the following revised 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%

During this programme, evaluation of concentrate regrind showed no benefit and as such was not included in the final flowsheet. Testwork 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 or improve concentrate grades, and hence this step was not included in the final flowsheet for the processing of primary sulphides. However, as part of the commissioning phase, pre-flotation will be evaluated for the processing of unblended non-primary sulphide feed containing elevated levels of carbon and mica.

The presence of elevated carbon levels in the final concentrate resulted in a carbon removal circuit being included in the flowsheet (refer to the detail below).

The grinding testwork 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.

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; and 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.

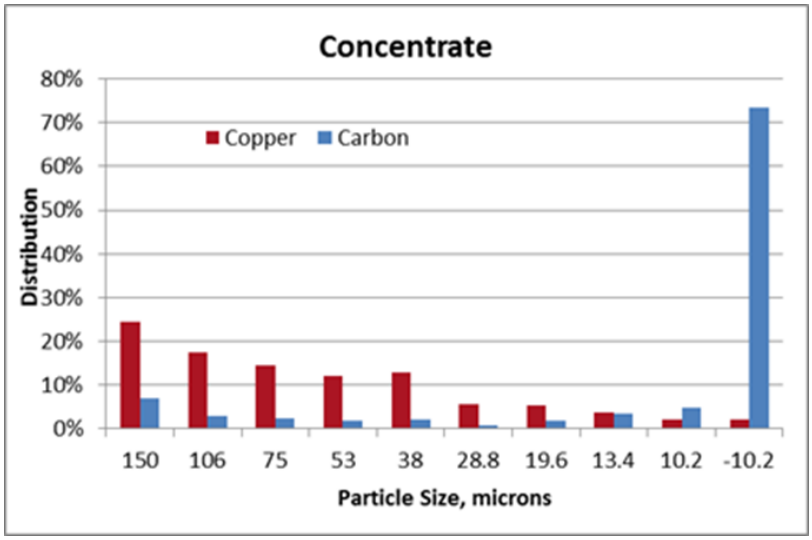
The conclusion from these tests is that whilst the Sentinel primary ore does show some variability in terms of flotation response and final concentrate grade, the range of results for primary ores is restricted. However, the flotation response for the non-primary ores is not as consistent, recovery is 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 it is expected that with blending, their impact will be limited.

The comminution testwork 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.

Carbon in concentrates and deslime

Analyses of concentrates produced from the testwork indicated elevated carbon content in the samples, which could be problematic to smelter operations, limiting capacity due to high heat generation. Sizing work on final concentrates (Figure 13-3) has shown that the majority of this carbon is less than 10 µm in size and liberated. The carbon in concentrates can be reduced to less than 1% using deslime cyclones, and these have been included in the design.

Figure 13-3 Carbon content in concentrates



13.1.3 Potential for deleterious elements

A full elemental analysis was provided by SGS for concentrate samples produced in locked cycle testwork. These analyses indicated low levels of deleterious elements, listed as follows, and therefore are 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

13.1.4 Design metallurgical parameters

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

- mill throughput of 55 Mtpa
- primary grind of 80% passing 212µm
- primary sulphide recovery = 92%
- non-primary sulphide recovery = 70%
- concentrate grade = 24% Cu

13.2 Enterprise

13.2.1 Mineralogy

Nickel (Ni) occurs as secondary sulphides to a depth of 40 m to 80 m, and below that as primary sulphides. Ni sulphides include vaesite, millerite, nickeliferous pyrite, bravoite and carrollite, and generally occur 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 when stockpiling. Secondary supergene Ni minerals include garnierite. Minor Ni mineralisation is contained within silicates including chlorite and talc. Mineralisation style is variable and found in veins, parallel to foliation and as disseminations. The majority of high grade Ni mineralisation 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 testwork programme

Preliminary testwork on initial samples from Enterprise was conducted at the Company's metallurgical laboratories at Kansanshi, between June and August 2010. The testwork was conducted on a quarter core 10 kg sample of high grade (5% Ni), taken from 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 13-4. Figure 13-5 is a cross section showing metallurgical sample holes within the stage 1 and 2 initial mining area.

Scoping testwork results and conclusions

Flotation testwork 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.

Figure 13-4 Enterprise metallurgical testwork, drillhole sample locations

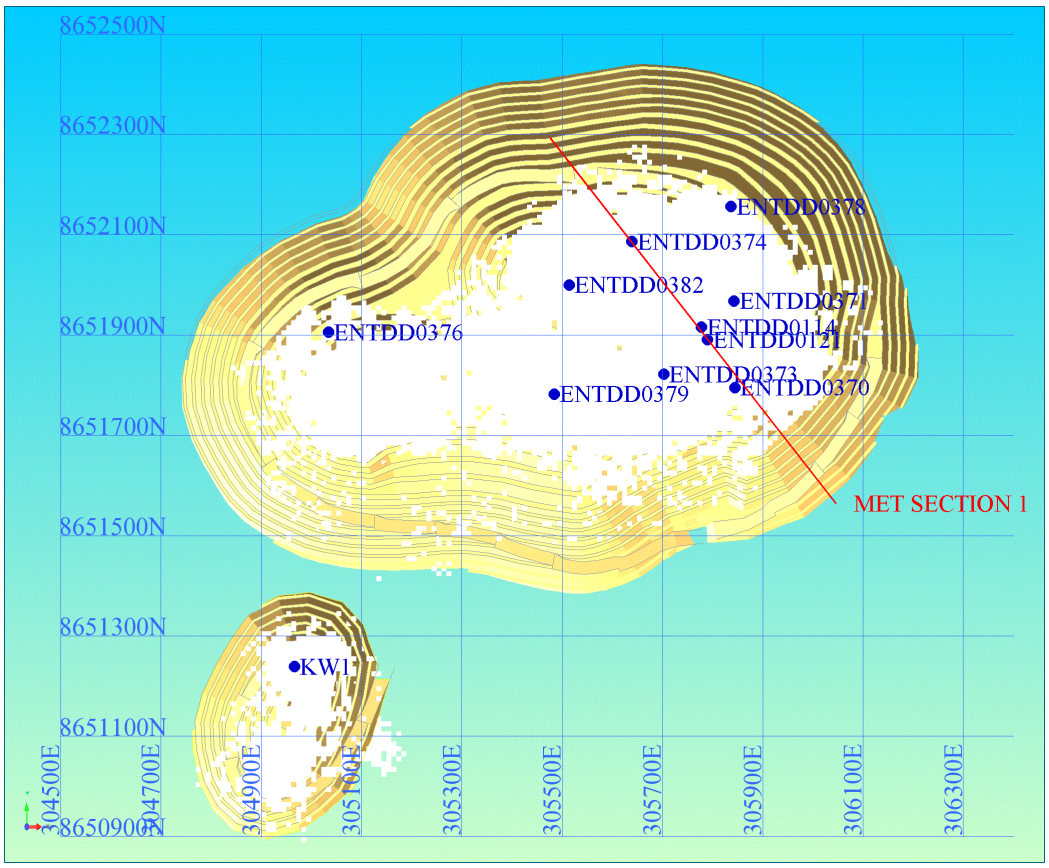
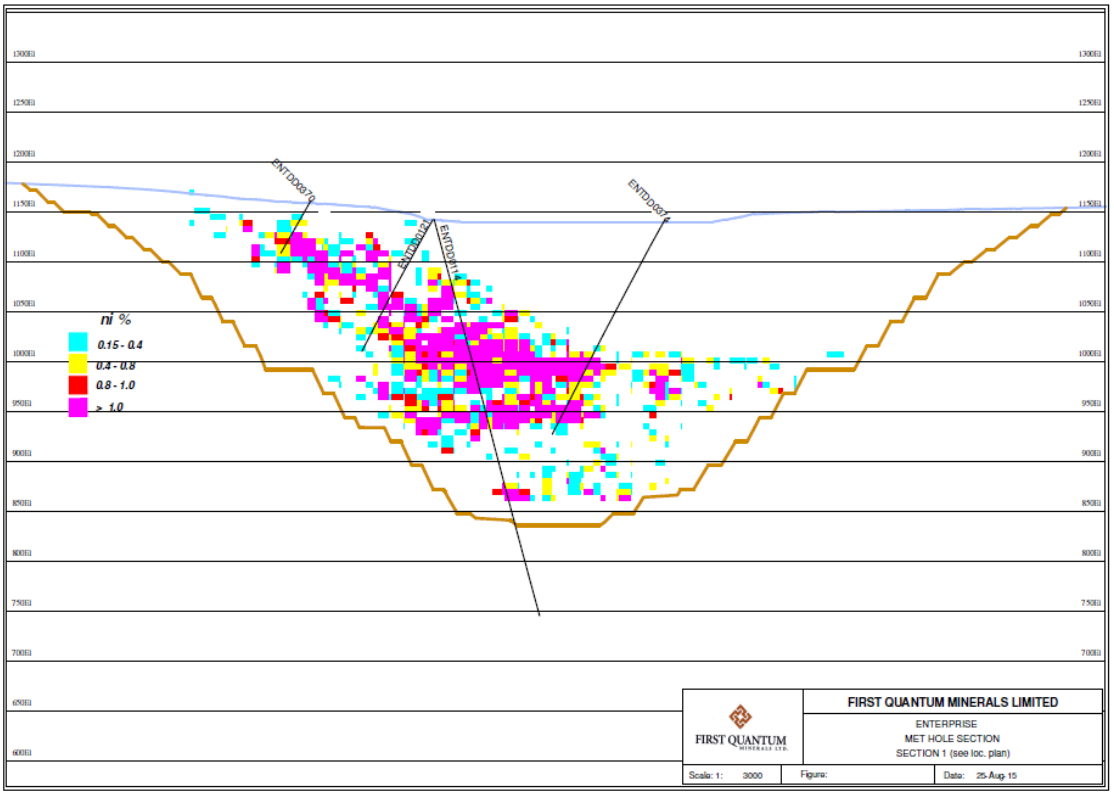


Figure 13-5 Enterprise metallurgical testwork, cross-section through stage 1 and 2 pit



This scoping testwork 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 talc prior to nickel recovery. Specific conclusions from the scoping testwork were:

- nickel losses to the talc pre-float are grind dependant – 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 – 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

Initial comminution work

Two samples were sent to JKTech Pty Ltd in April 2011 for drop weight testwork, 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.

13.2.3 Second testwork programme

The second testwork 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 13-4. The testwork was carried out by SGS Lakefield (Perth) in 2012.

Comminution testwork

The samples selected for comminution testwork 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 testwork are proposed to assess the comminution of the harder domains.

The testwork 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)

Flotation testwork

Flotation tests were performed under conditions similar to those used during the initial testwork, using MIBC (methyl isobutyl carbinol) as the frother with 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.

Variability testwork

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 the table of test results, to try and link the minerals to the flotation response (Table 13-1).

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.
- 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 testwork programme

Five master composites and one other sample were prepared from drilling samples taken in late 2012 (Table 13-2). 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 core excluding sample MC-5, which was PQ. The drillhole sample locations are shown on Figure 13-4.

Table 13-1 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	0.040	4.58	3.03	1.17	17.3	92.5	9.42	14.0	95.0	12.0
159	129-144	Millerite, Ni-Pyrite	8.23	0.086	48.6	47.5	8.05	18.6	50.2	22.2	17.1	51.7	24.9
160	151-158	Millerite, Vaesite	3.29	0.152	0.73	1.35	6.09	18.8	91.9	16.1	15.3	95.3	20.6
161	163-166	Vaesite, Ni-Pyrite	3.98	0.230	0.52	1.69	12.9	16.8	82.6	19.6	15.1	94.7	25.1
162	168-174	Vaesite, Millerite, Ni-Pyrite	4.43	0.209	2.71	5.30	8.67	18.1	65.8	16.1	15.2	91.7	26.7
163	198-216	Ni-Pyrite, bit Millerite	3.02	0.185	0.41	0.87	6.41	19.4	82.9	12.9	14.2	94.6	20.1
164	282-291	Vaesite	1.74	0.126	0.53	1.91	6.27	15.8	88.4	9.75	14.0	92.1	11.5
	ENT 0121												
165	72-76	All 3 samples Ni-Pyrite with minor Vaesite	0.556	0.050	0.26	9.35	20.0	2.99	81.7	15.2	2.57	85.1	18.4
166	78-85		6.58	0.30	1.53	1.70	7.30	18.2	80.4	29.0	17.0	95.7	36.9
167	92-98		3.51	0.27	0.65	3.00	16.1	6.08	38.5	22.3	9.39	93.2	34.9

Table 13-2 Master Composite criteria and sampling for third testwork programme

Master Composite	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.

Flotation testwork

The flotation testwork performed on the five Master Composites can be summarized 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 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, the rougher flotation (@ up to 500 g/t) had a significant impact in suppressing the flotation of magnesium minerals.

Comminution testwork

Each of the composite samples in Table 13-2 were sent for full comminution testwork. Testwork was conducted at the SGS Lakefield lab in February 2013, excluding the JK drop weight work which was sent to JKTech Pty Ltd.

The testwork 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 siltstone lithology with kyanite alteration. These results were used in the sizing of the comminution circuit.

13.2.5 Representivity of metallurgical samples

Due to the volume of high grade samples that have been sent for flotation and grindability testwork it is recommended that further testwork includes a higher quantity of lower grade samples. Thus far a good range of the mineralised lithologies and alteration styles have been represented in the testwork 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 testwork is proposed in due course.

For the comminution work it would be useful to analyse samples of only quartz-kyanite material with variable grades to ascertain the minimum A*b value expected from the Enterprise rocks. This could have an effect on the mill throughput which could be modelled in advance with this additional information.

13.2.6 Potential for deleterious elements

The flotation testwork 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, 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 0.1 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, and desliming 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

13.2.7 Design metallurgical parameters

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

- mill throughput of 4 Mtpa, 500 tph
- primary grind of 80% passing 150 µm
- primary sulphide recovery = 90%
- non-primary sulphide recovery = 60%
- concentrate grade = 14% to 18% 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, ie Datamine Studio version 3.0 and Snowden Supervisor version 8.3. Estimates were completed in April 2014 by David Gray (QP) of FQM with the assistance of FQM exploration and site geologists. 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 mineralisation. Interpolation parameters were based upon the geology, styles of mineralisation, drill hole spacing and geostatistical analysis of the data. Mineral Resource estimates were classified according to geological continuity, QAQC, density data, drillhole grid spacing, grade continuity and confidence in the panel grade estimate and have been reported in accordance with the guidelines of the Australasian JORC Code (JORC, 2012), which in turn complies with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014).

The April 2014 Sentinel Mineral Resource estimate was depleted for early start-up material mined as at 31st May 2015 and was reported using a 0.2% copper cut-off grade. The Mineral Resource statement (**Table** 14-1) is inclusive of Mineral Reserves and is reported within a life of mine pit shell. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

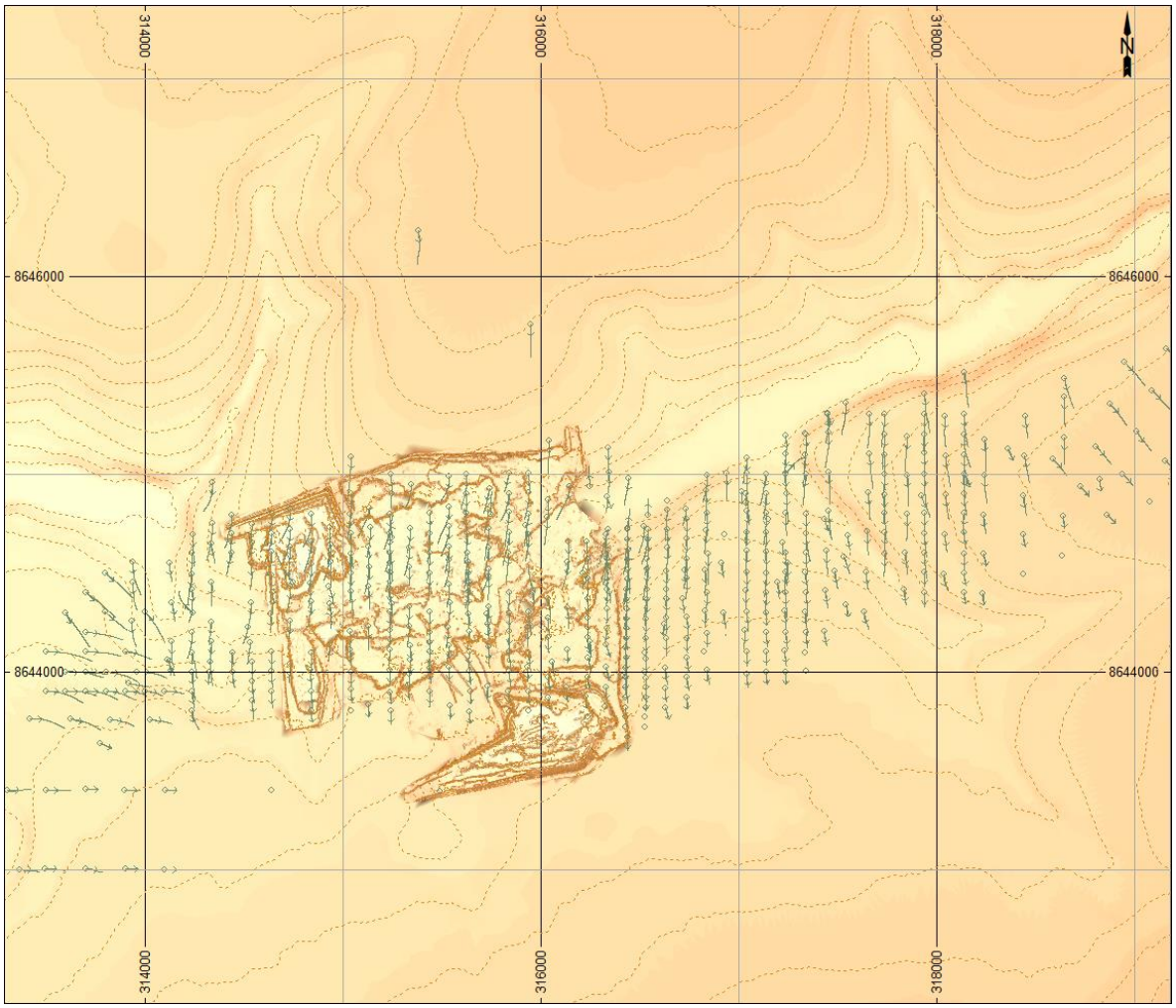
Table 14-1 Mineral Resource statement for Sentinel Copper mine using a 0.2% copper cut-off and depleted for mining as at 31st May 2015

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**

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 May 2015) was used to define the upper limit of unmined Mineral Resources (Figure 14-1).

Figure 14-1 A plan view of the Sentinel operations topography and the early start-up pit floor survey as at 31st May 2015 together with the distribution of diamond drilled holes.



Only holes drilled by Kiwara PLC (2007 to 2010) and FQM (2010 to current date) were used in this Mineral Resource estimate update. 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, 677 diamond drilled holes for 229,713 m and 146,986 assayed samples were used in this estimate (Figure 14-1). Samples 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.

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

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:

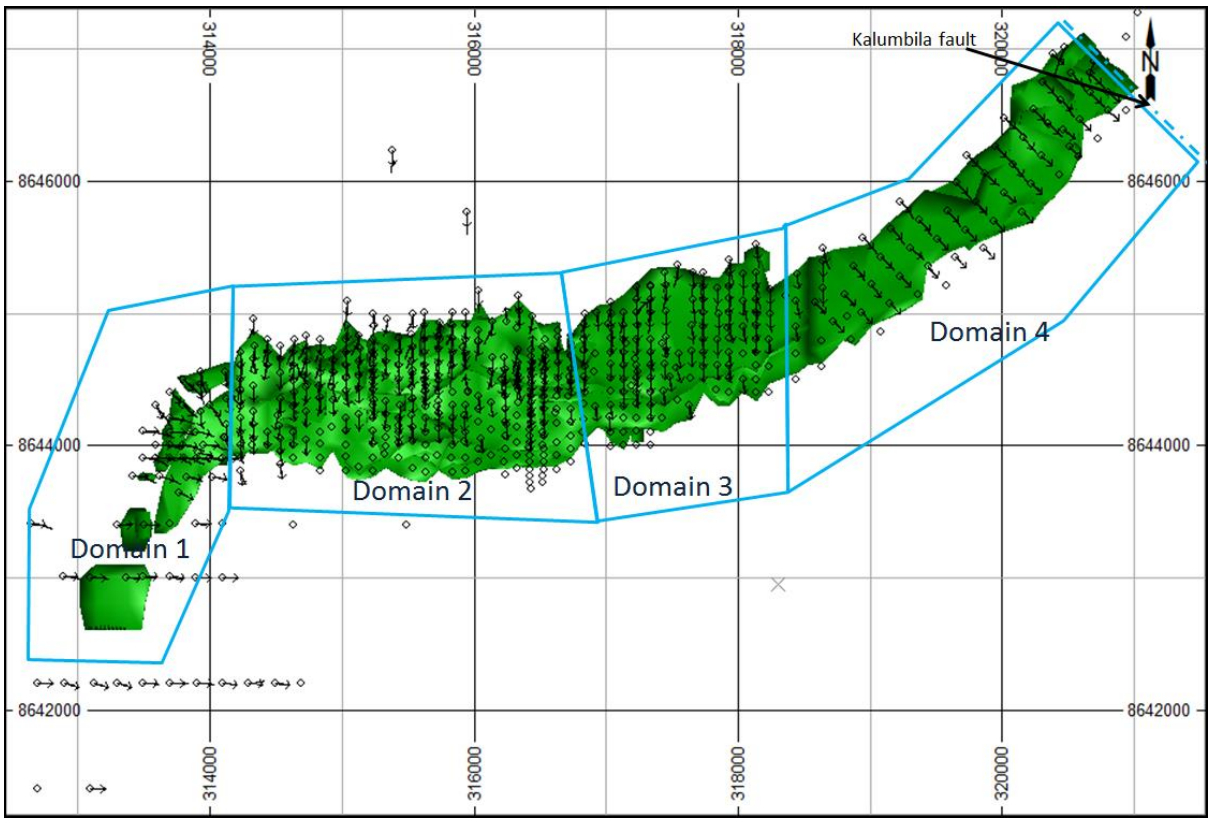
- visual checks of collar coordinates against topography
- investigation for overlaps, duplication and gaps in the logging and sampling data.
- assay values were checked for excessively high values.
- downhole survey data was assessed for excessive deviations.

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

14.1.3 Geological and mineralisation models

The Sentinel deposit strikes approximately east west for about 11 km and, on plan, has a lazy "S" sinusoidal shape. Mineralised horizons dip 20 to 30^{o} in a northerly direction (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 sub-divided into four geologically distinct domains (Figure 14-2) with different mineralisation characteristics.

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

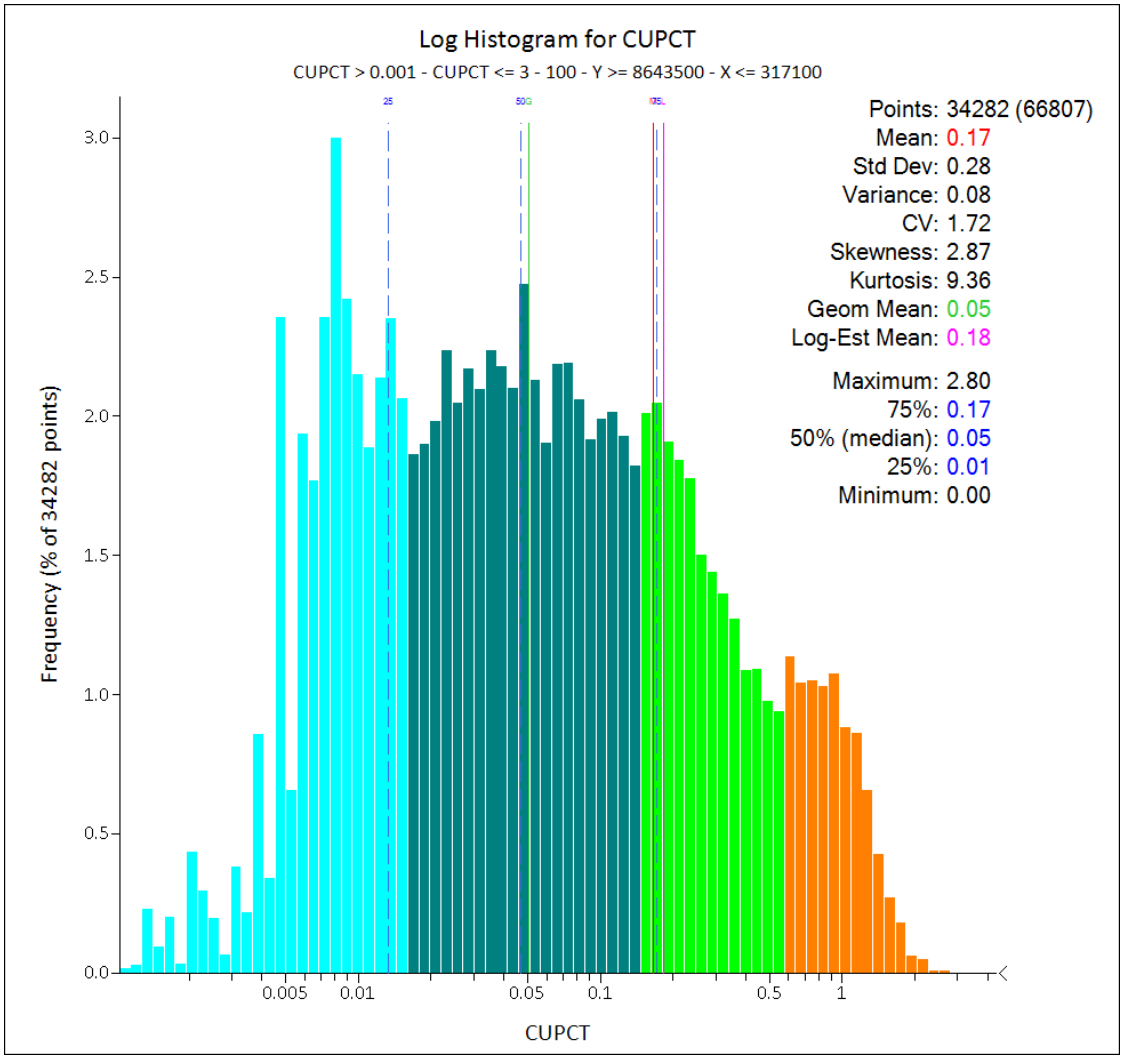


The dominant primary sulphide (chalcopyrite) copper mineralisation is hosted by a structurally deformed and metamorphosed phyllite which has a strong foliation overprinting older remnant

folding and sedimentary layering. Mineralisation 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 mineralisation, 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 mineralisation (Figure 14-4). A notable characteristic of Sentinel mineralisation is that it is often truncated along its northern, down dip extents by an apparent fold nose.

3D geology modelling (Figure 14-4) has defined these shallow dipping sub parallel zones of high and lower grade mineralisation. Vertical sections (Figure 14-4) along each drill line were used to digitise copper mineralisation as string envelopes. A 0.15% copper cut-off grade was used to position string envelopes to the exact sample top or bottom along the de-surveyed 3D drillholes. The 0.15% cut-off was chosen from visual inspection of the drill hole sample values in section as well as from review of the distribution of sample copper values (Figure 14-3).

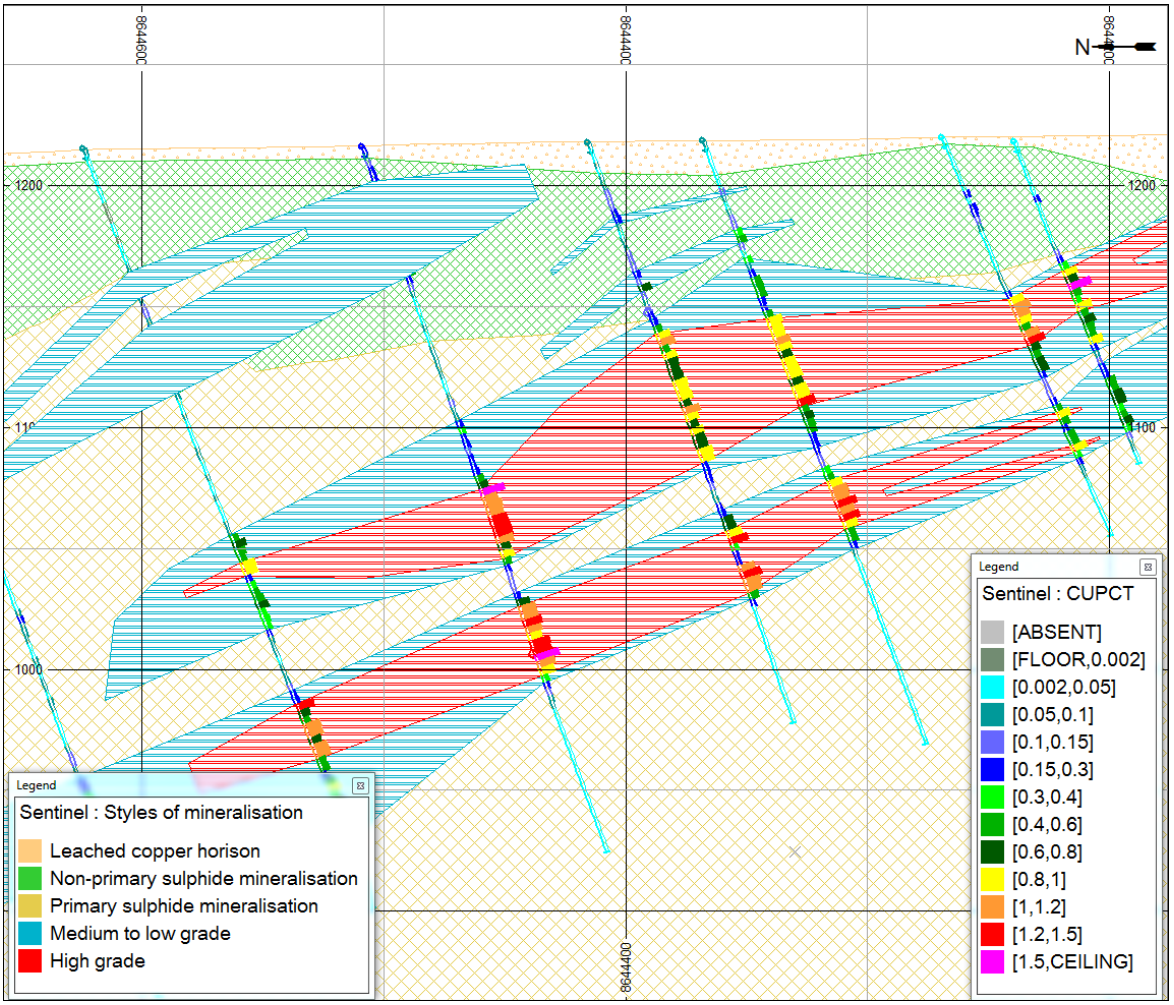
Figure 14-3 A log scale histogram of all sample copper percent values illustrating natural inflections for background waste (with very low grades <0.15% Cu, light and dark blue), low to medium grade mineralisation (green) and high grade mineralisation (orange)



The mineralisation string envelopes at the edges of the deposit were extended by approximately 50 m beyond the last drillhole. Each set of adjoining string envelopes were linked with wireframes to form mineralised volumes.

Weathering and oxidation ranges from a few metres below surface and is seldom deeper than 70 m from surface. 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 mineralisation (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 complete oxidation where fresh (un-tarnished) chalcopyrite is visible and where carbonate minerals have not been dissolved. The oxidised zone is comprised mostly of tarnished chalcopyrite and some secondary chalcocite. These contact horizons have been defined as sub horizontal surfaces with the aid of logging and sample assay data.

Figure 14-4 A vertical north south section illustrating high and low grade copper mineralisation in relation to drillhole sample grades and the respective styles of mineralisation



The mineralised volumes, structural zones, weathering and oxidation surfaces were used to define volumes of rock that have similar geology. These volumes/zones were numbered (Table 14-2) and then combined to provide the respective domains of mineralisation used to code the 3D drillhole samples. The coded drillhole samples were used for spatial analysis, guided the selection of estimation parameters and were used for estimation.

Table 14-2 Domain numbers used for coding during estimation

Style of mineralisation	
300	Leached copper horison
200	Non primary sulphides
100	Primary sulphides
Fault block number	
10	
20	
30	
40	
Grade zone	
1	High grade (>0.8% Cu)
2	Medium grade (0.15 - 0.8 % Cu)
3	Low grade and waste (<0.15% Cu)

14.1.4 Sample compositing

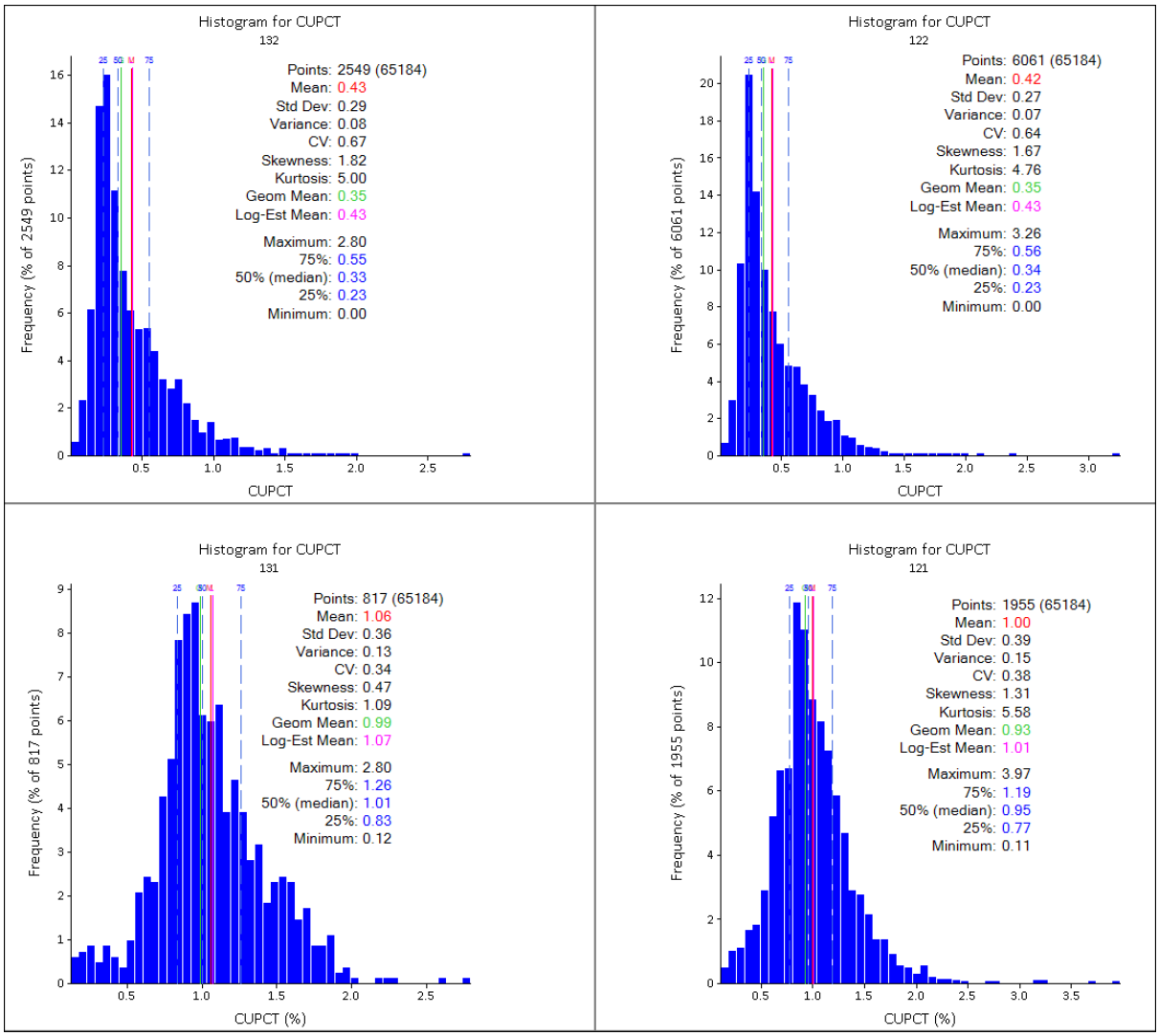
De-surveyed drillhole samples were composited (combined) down the hole so as to reduce the effect of varying sample lengths on grade values and to support robust statistical and spatial analysis for estimation. A 3 m downhole composite length was chosen in order to honour the smallest mining unit bench height of 6 m in addition to a population (approximately 11% of total samples) of 3 m long samples. The compositing routine combines consecutive samples down-the-hole so as not to cross the domain boundaries and to ensure minimal metal loss.

14.1.5 Statistics

Statistical analysis of composited 3D drillhole sample data per domain was carried out using Snowden Supervisor software. Data distributions were investigated per domain using histograms, log probability plots and descriptive statistics.

The defined mineralisation domains were to minimise mixed populations and so reduce internal variability for improved spatial analysis (variography) and a more robust estimate. Statistical analysis and histograms (Figure 14-5) highlights that selected domains of mineralisation were well defined, with minimal mixing. The coefficients of variation (CV) were well below 1, reducing the need for extensive top cutting. Top cuts are established to define the maximum reasonable grade for a composite value within a given domain and reduce the risk of excessively high samples distorting block estimates in areas with low data support. No top cutting was deemed necessary for the copper and acid soluble copper sample grades.

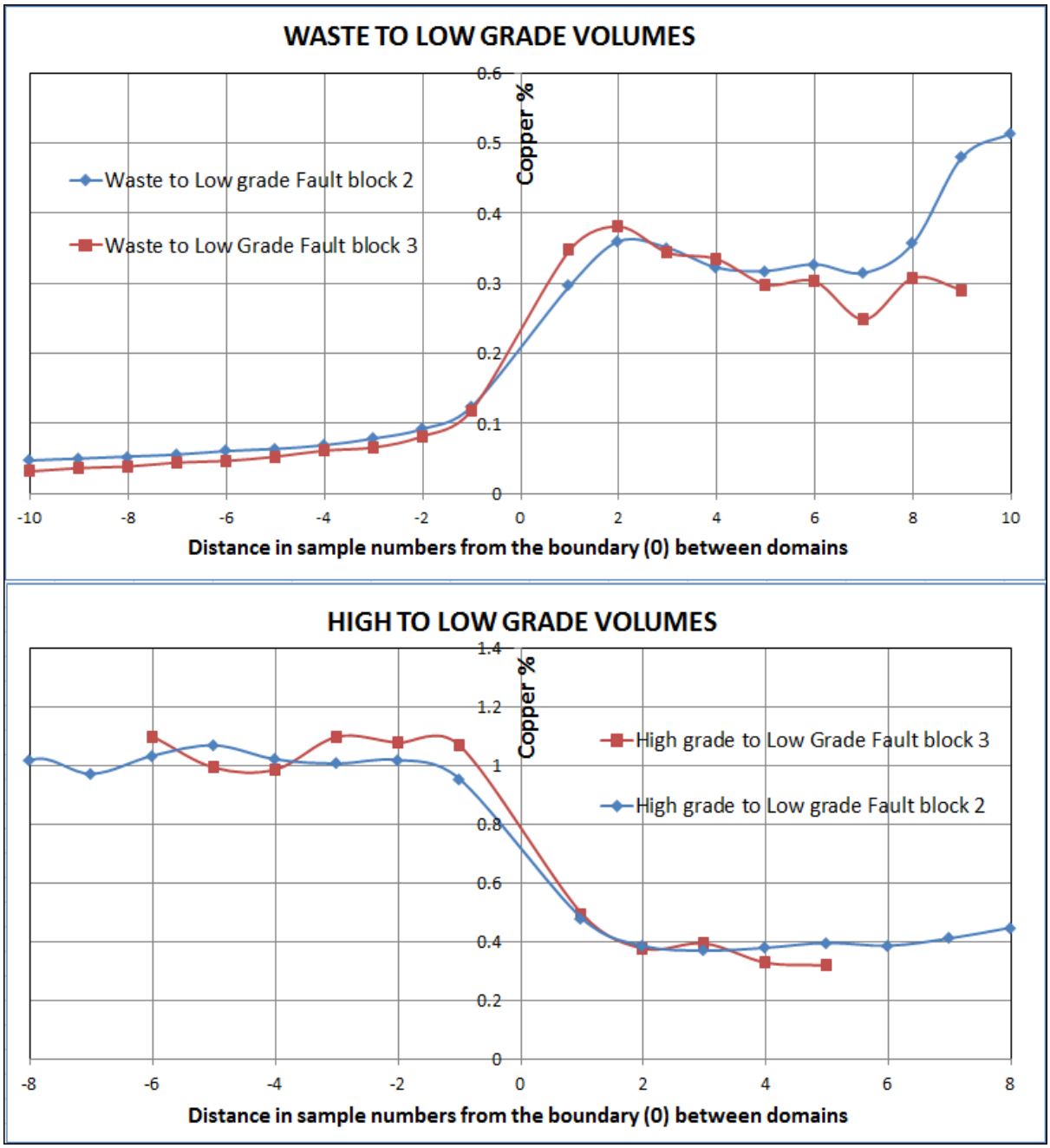
Figure 14-5 Histogram distributions for the main low grade (132 and 122) and high grade (131 and 121) copper domains at Sentinel.



14.1.6 Boundary analysis

A series of contact profiles were generated to evaluate the change in copper grades across domain boundaries. There are sharp grade changes across both waste to low grade and low to high grade domains (Figure 14-6). Hard boundaries were employed across most domain contacts, in order to limit smearing and dilution. Soft boundaries were used between copper grade oxide and sulphide mineralisation. A partial soft boundary (only 1 sample from the adjoining high grade domain) was employed for estimating low grade domains due to the slight increase of low grade sample values immediately adjacent to the contact (Figure 14-6).

Figure 14-6 Boundary analysis between high, low and waste domains for fault block 2 and 3



14.1.7 Spatial analysis

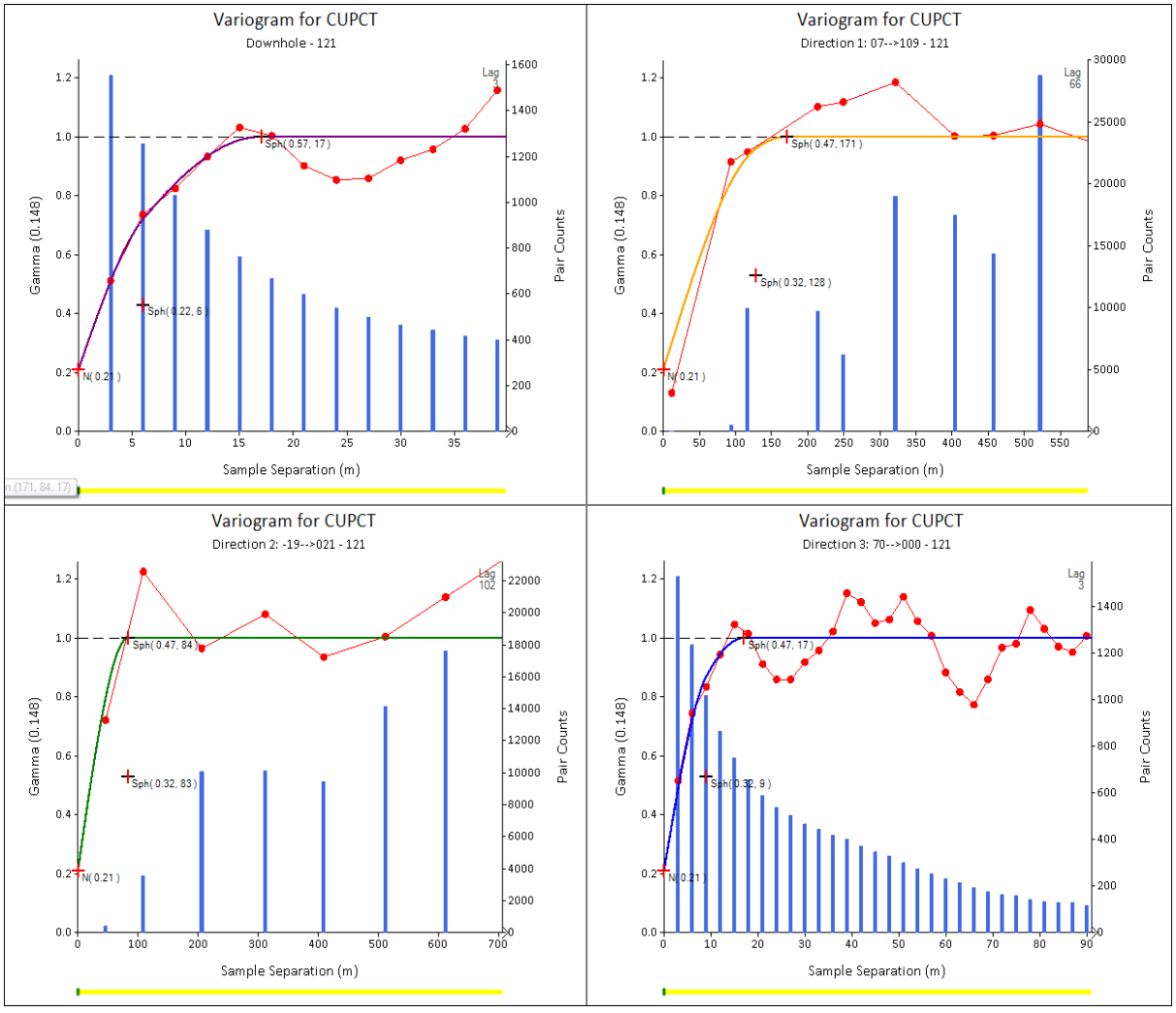
Variograms, representing 3D grade continuity, were generated per domain for copper. The following methodology was applied:

- the principal axes of anisotropy were determined using variogram fans based on normal scores variograms
- directional normal scores variograms were calculated for each of the principal axes of anisotropy
- downhole normal scores variograms were modelled for each domain to determine the normal scores nugget effect

- variogram models were determined for each of the principal axes of anisotropy using the nugget effect from the downhole variogram
- the variogram parameters were standardised to a sill of one
- the variogram models were back-transformed to the original distribution using a Gaussian anamorphosis
- 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 ranges of grade continuity well defined and with clearly different anisotropy and orientations. The dominant ranges of copper grade continuity (150 m to 170 m) were in the approximate north-west direction. An example of a high grade copper domains variography (domain 121) is shown in Figure 14-7. Variogram models for fault block one and four, which have much wider drill grid spacing, had longer ranges of continuity and were not easily defined. As a result large volumes of these domains were classified as Inferred Mineral Resources.

Figure 14-7 Variography for copper domain 121, a major high grade domain in fault block two.



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 for grade estimation. KNA was completed using Snowden Supervisor software, which uses the modelled variograms and a series of estimates each with kriging efficiency and slope of regression values calculated.

A block size of 60 mE by 36 mN and a 12 m bench height was selected as having an optimal kriging efficiency and regression slope value. 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 also tested and chosen to have dimensions similar to the variogram ranges.

14.1.9 Block model

An empty 3D Datamine block model was created using an optimal parent block size of 60 mE by 36 mN by 12 m benches, as per the kriging neighbourhood analysis. Block model origins were set at 312500 mE, 8642000 mN and 500 m elevation. The model was extended by 9,648 mE, 6480 mN and 864 m vertically. Blocks were sub-celled by five times in the easting direction, three times in the northing direction and twice in the vertical dimension, in order to improve filling of the wireframe volumes. Sub-celling has only occurred along geology boundaries. The smallest block dimension is 12 mE by 12 mN by 6 m bench height and is aligned with the smallest mining unit (SMU). The empty block model uses the same domain coding (Table 14-2) as the 3D drillhole data. The upper limits of the block model were constrained to the topography DTM surface as per the Lidar survey.

14.1.10 Density assignments in the block model

In situ dry density measurements were available for 10,024 samples. Sample density values were measured using the traditional method of weighing samples in air and then in water. There were a sufficient number of samples per domain to minimise variability in measurements from moisture or pore spaces. Density data was analysed statistically per domain and outlier values were removed in order to provide a representative mean density value per domain. The strongest influence on density was the degree of weathering. Density data has good coverage across the deposit. The average density values per weathered ore and waste horizon is detailed in Table 14-3.

Table 14-3 Average density data

Weathering	Ore/Waste	Mean density
Saprolite	waste	1.9
Saprock	waste	2.69
	ore	2.68
Fresh	waste	2.78
	ore	2.8

14.1.11 Ordinary kriging interpolation

Copper grade were estimated using Ordinary Kriging (OK) into the parent blocks within each of the defined domains. OK was deemed an appropriate technique for estimation of the Sentinel domains of mineralisation owing to the near normal distribution of sample copper values and the limited evidence for domain grade population mixing. Estimation parameters for kriging were based upon

variography, KNA, geological continuity and drill grid spacing. Estimates into the parent blocks used a discretisation of 6 (X points) by 3 (Y points) by 2 (Z points) so as to better represent the block volume dimensions.

Most blocks were estimated within the first search ellipse, but some blocks required a second search ellipse that was double that of the first search ellipse. A third search pass was required for estimating into blocks remote of regular grids of close spaced data.

Post processing of the parent block estimates was completed in order to provide estimates relevant at the scale of mining. Localised uniform conditioned (LUC) estimates were generated for copper inside the parent block by using the estimated panel grade and uniform conditioning to provide the proportion of SMU's inside a parent that are above a cut-off grade. LUC was then used to provide a grade estimate per SMU within the parent block, while maintaining metal content of the parent. LUC provides grades at the SMU (12m*12m*6m) scale block size, providing an improved representation of grade variability and minimising loss and dilution that may be expected during mining.

LUC models were validated by:

- visual comparisons of drillholes, OK panel grades and SMU block grades
- confirming that metal correlations in the SMU blocks were comparable with the correlations observed in the drillhole data
- checking that the contained copper metal at a zero cut-off grade is the same for both the OK panel models and the LUC models.

14.1.12 Background model

In addition to estimation of copper within mineralised domains, a background waste model was generated in order to capture erratic mineralisation not included in the interpretation of the mineralised domains. Grades were estimated into the waste/background volumes by using a single pass search ellipse (as per variography ranges and anisotropy) in order to avoid grade smearing.

14.1.13 Model validation

Copper and acid soluble copper grade estimates reflect the input data and are considered locally reliable across the measured and indicated categories and globally representative across the inferred areas. Validation slices were used to demonstrate valid block estimates when compared to the input data (Figure 14-8). Block estimates were further validated with visual inspection of model and data along cross sections as well as statistical comparison of mean values per domain.

Figure 14-8 Validation slices in northing, easting and sub vertical orientations

14.1.14 Sentinel Mineral Resource classification and reporting

The block model estimates at Sentinel have been classified and reported (Table 14-4) in accordance with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014), and also comply with the guidelines of the JORC Code (JORC, 2012).

Table 14-4 The Sentinel Mineral Resource statement depleted as at 31st May 2015 and using a 0.2% copper cut-off. Mineral Resources are inclusive of Mineral Reserves.

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**

Classification is guided by confidence in drillhole data, geological continuity, and the quality of resulting kriged estimates. Geological confidence is supported by logged data from good coverage of a sufficiently closely spaced grid of diamond drilled holes. In addition, there is a good understanding of the prevailing geology controls at the regional scale and locally from in-pit exposures and pit mapping. Confidence in the kriged estimate is associated with adequate block estimate validations, drill grid spacing, sample data quality, kriging efficiency and regression slope values. The Sentinel Measured and Indicated Mineral Resource statement has 72% classified as Measure Mineral Resources.

There are no known factors related to environmental, permitting, legal, title, taxation, socioeconomic, marketing, or political issues that are believed to materially affect the Mineral Resource.

It is noted that additional Inferred Mineral Resources exist to the north east and south west of the life of mine (LOM) pit shell and are estimated to be 54.4 million tonnes at 0.36 % Cu.

14.1.15 Comparisons and grade tonnage curves

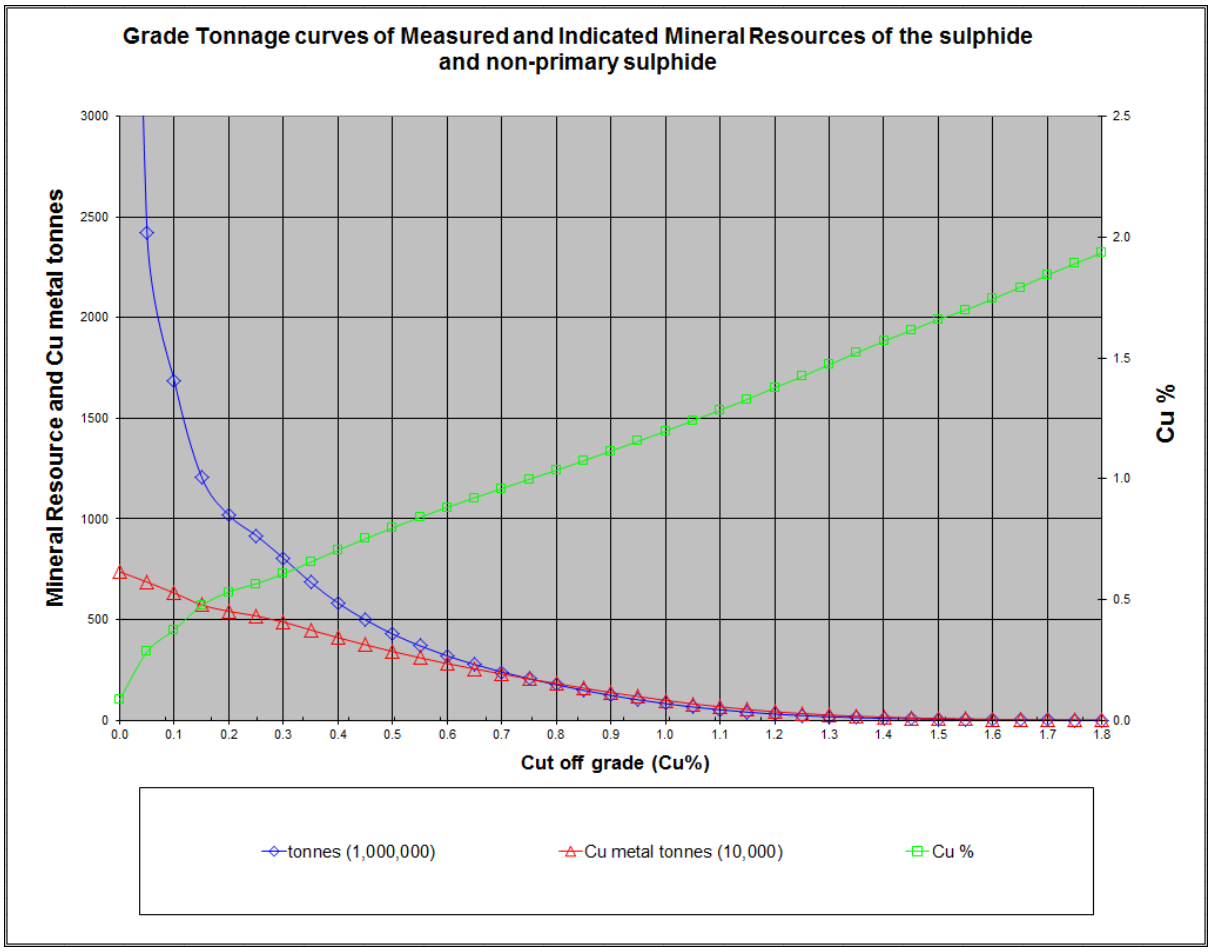
The Mineral Resource estimate reflects additional drillhole data and improved geological understanding from drill core re-logging and early start-up in-pit exposures. Compared with the previous (May 2012) Mineral Resource statement (Table 14-5), the minor oxide mineralisation was removed due to its refractory character.

Table 14-5 The previous (May 2012) Mineral Resource statement for the Sentinel deposit, using a 0.2% copper 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**

The additional data and confidences have provided a 4% increase in copper metal for the measured and indicated resource categories due to a 4% increase in grade. Increased confidences have upgraded previous indicated resources into the measured category by 44%. Inferred resources were constrained (reduced by 18%) to a life of mine pit shell to limit blocks above cut-off in the extreme south west and north east areas of the deposit.

Figure 14-9 Grade tonnage curves for the May 2015 measured and indicated sulphide and non-primary sulphide Mineral Resources



It is the opinion of the QP, David Gray, that the resulting changes to the May 2015 Mineral Resource statement reflects the confidence in the added and available data. The May 2015 estimates are believed representative of the prevailing mineralisation and reflect no material change from the previous estimates.

14.2 Enterprise

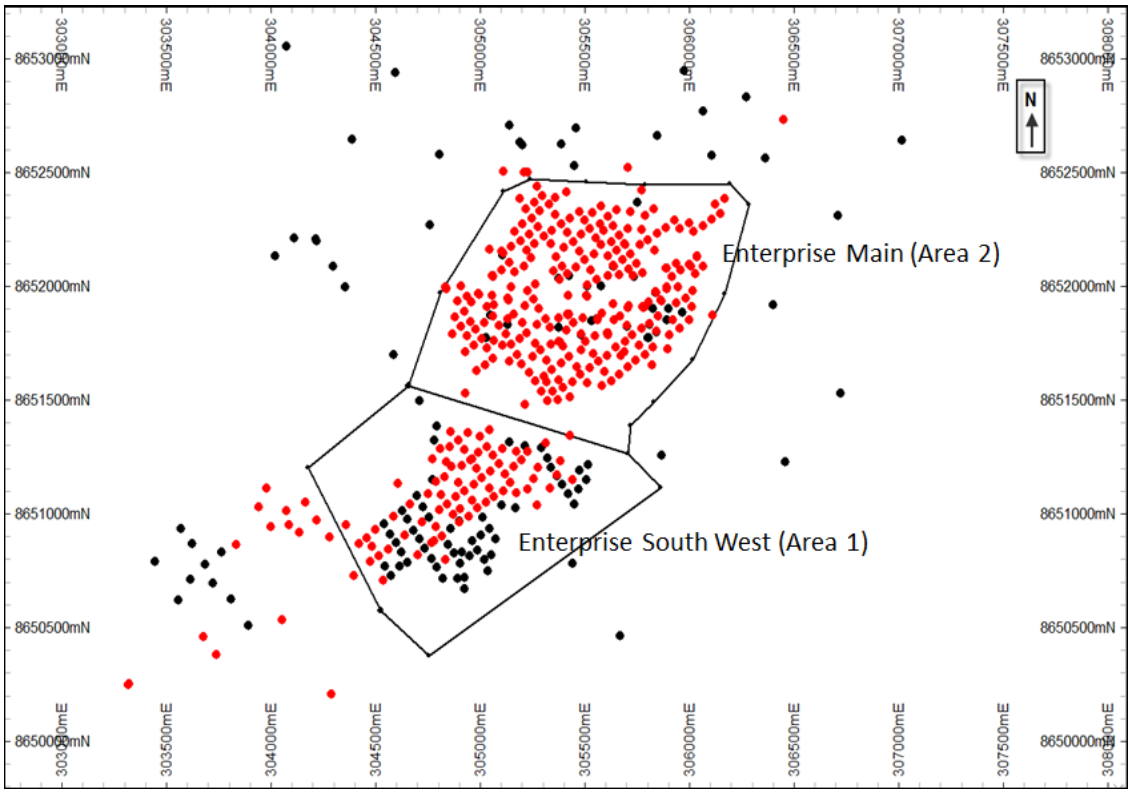
The Enterprise Mineral Resource estimate update was completed in December 2013 under the supervision of Mr David Gray (QP), by CSA.

14.2.1 Available data

The Mineral Resource estimate was updated using drilling completed since the last estimate in 2012. This comprised mainly of extensional and infill drilling in Enterprise SW as well as a small amount of drilling completed at Enterprise Main. A total of 474 diamond drilled holes were available with data from 81 additional holes available for the 2013 Mineral Resource estimate and new data displayed in Figure 14-10. These holes had 148,166 m of core and 133,944 samples.

Data was loaded into Datamine from exported DataShed database tables. These files 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 with the data.

Figure 14-10 Mineralisation limits (black polygons) with 2013 drill data coloured in black



14.2.2 Geological modelling

The 2012 Mineral Resource estimate relied on an interpretation of mineralisation influenced by faulting as modelled by site exploration geologists during 2012. Additional drilling assisted with a re-interpretation by FQM which improved mineralisation domain definition.

The deposit area was divided into two areas, Enterprise SW (Area 1) and Enterprise Main (Area 2), as shown in Figure 14-10. Hard boundaries were used between the two domains since they are spatially distinct and have different grade profiles.

Two surfaces were created by Gretchen Williams (FQM), which defined the zone of potential mineralisation across Enterprise deposit. These were used to constrain modelling of mineralisation used for this update.

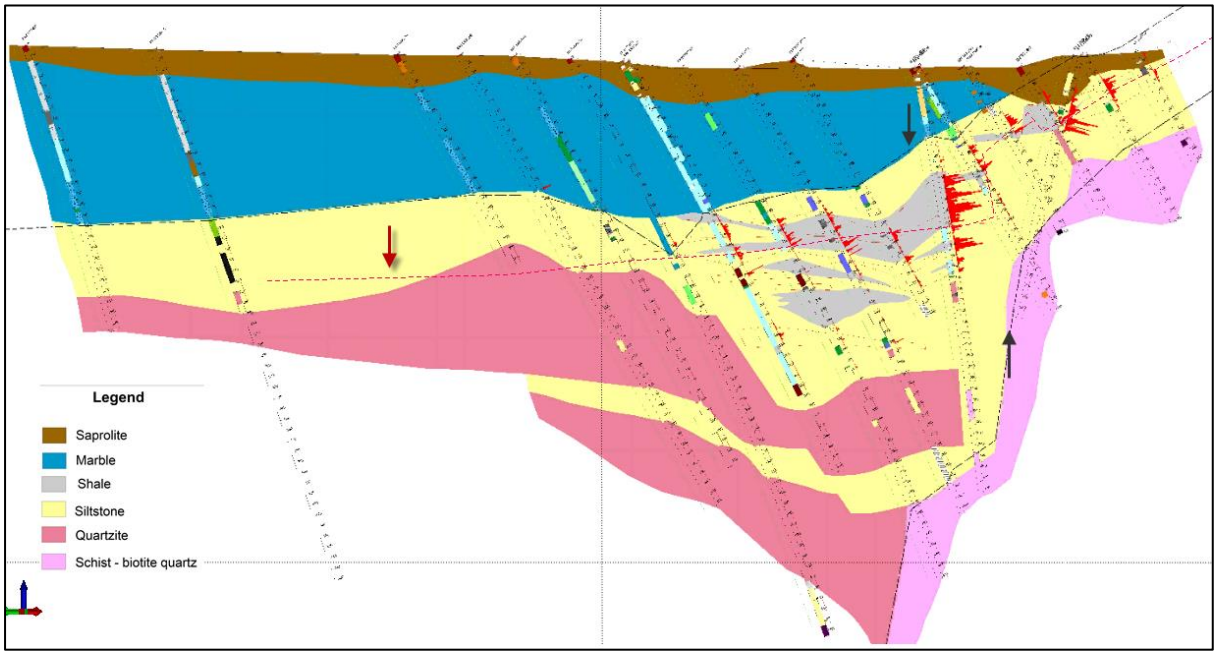
Due to the high degree of structural faulting, notably at Enterprise Main but also throughout the deposit, which displaces the mineralisation, a centreline was digitised by FQM through the main horizon of mineralisation and was used as a plane to un-fault and un-fold the geological model prior to spatial analysis and estimation. A geological vertical section is displayed in Figure 14-11.

Weathering and oxidation profiles are based primarily on logging and inform the specific gravity assigned to all stratigraphy units. They were not used as estimation boundaries during the estimation.

The topography wireframe was provided by FQM which covers the full extent of the deposit area.

Grade estimates were interpolated into the 3D block model containing the key geology domains which were based upon geostatistical applications using commercial mining software, Datamine Studio 3, as well as Surpac and Isatis.

Figure 14-11 FQM vertical section looking south-east, showing hanging wall (marble, blue) and footwall (schists, pink) and centre line (dashed red line/red arrow)

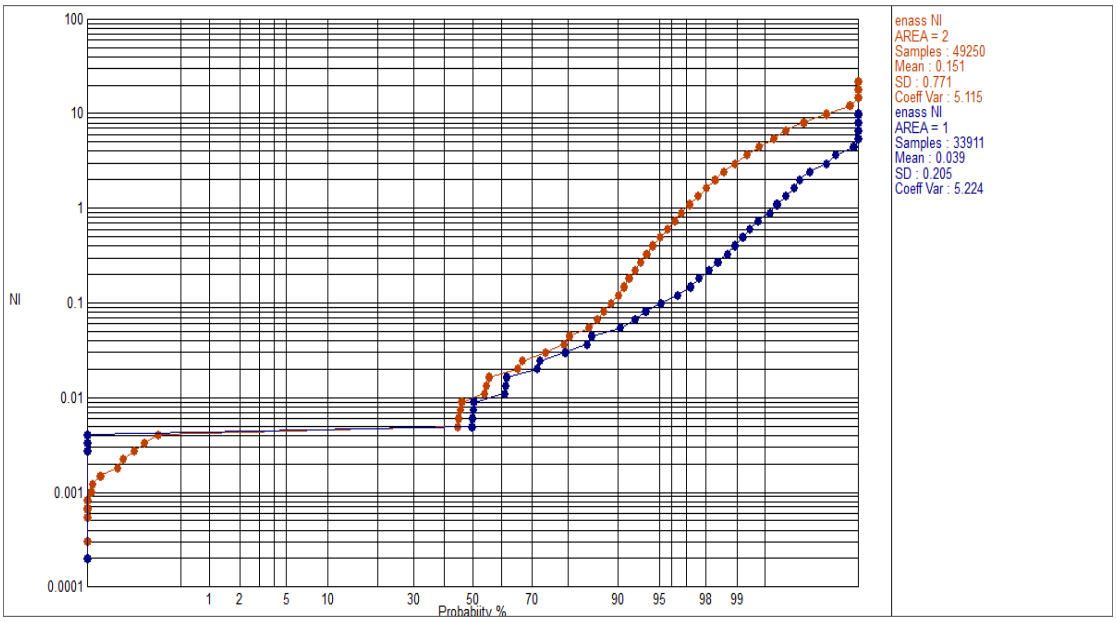


14.2.3 Data preparation

Raw assay data was flagged by deposit area (domain) and using the wireframes for mineralisation (footwall and hanging wall), oxidation and weathering.

Classical statistics were reviewed for both domains. Enterprise Main (Area 2) has a clearly higher nickel grade distribution, located predominantly within Fresh Material (Figure 14-12).

Figure 14-12 Nickel grade distributions for Enterprise Main (red) and Enterprise SW (blue)



The drill hole data was composited to 1 m, which is the dominant sampling interval, with an average sample length of 1.1 m.

14.2.4 Top cuts

Prior to estimation, histograms and coefficients of variation of estimation domains were reviewed to determine the distribution of high grade samples and their representivity 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. No top cuts were required.

14.2.5 Variography

Variography was completed on 1 m composites in Supervisor software for Ni and Cu. Variography suggests that grade continuity in Enterprise Main is different to Enterprise SW. Enterprise Main represents the highest grade and is more structurally complex variability and was reflected in the variography. The nugget at Enterprise Main was 18%, with approximately 71% of the variability linked to the short scale structure in the first 26 m. In contrast, the variability at Enterprise SW was lower, with a nugget of 10%, and approximately 79% of the variability in the first 55 m.

Different orientations were modelled for Ni and Cu. Copper mineralisation exists at depths below the known Ni mineralisation and is understood to be independent of Ni mineralisation. The Ni variogram in Enterprise Main is oblique to the strike of the mineralisation and is aligned with major faulting, which may contribute to the control of higher grades.

Table 14-6 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 Main	0.18	0.71	26	47	24	0.11	345	213	49
Enterprise SW	0.10	0.79	55	56	17	0.12	283	275	40

Table 14-7 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 Main	0.30	0.63	47	29	8	0.07	255	212	47
Enterprise SW	0.12	0.82	31	78	10	0.06	138	107	19

Table 14-8 Variogram rotations (Isatis Mathematician format)

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

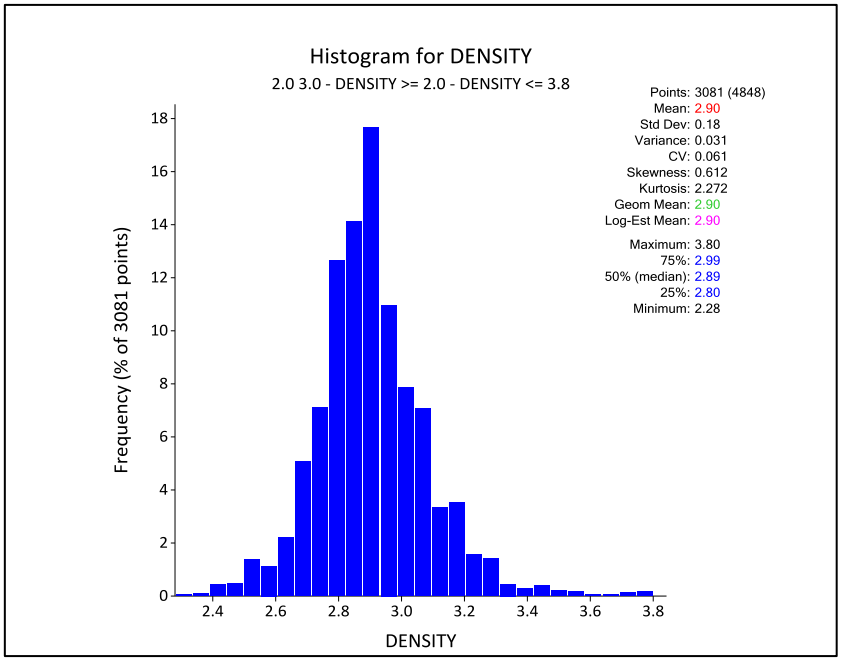
14.2.6 Density analysis

4,840 density measurements were available with the vast majority recorded from fresh material. There were only eight samples in saprolite and 108 samples in saprock. The majority of Enterprise Main is located in fresh material, while a significant amount of mineralisation in Enterprise SW is in saprolite and saprock. In the absence of suitable density data for saprolite and saprock, average densities were reviewed for adjacent deposits within the copper belt region and applied to that material at Enterprise.

Standard gravimetric measuring practices were employed to measure sample density. The strongest influence on density was lithology and oxidation state.

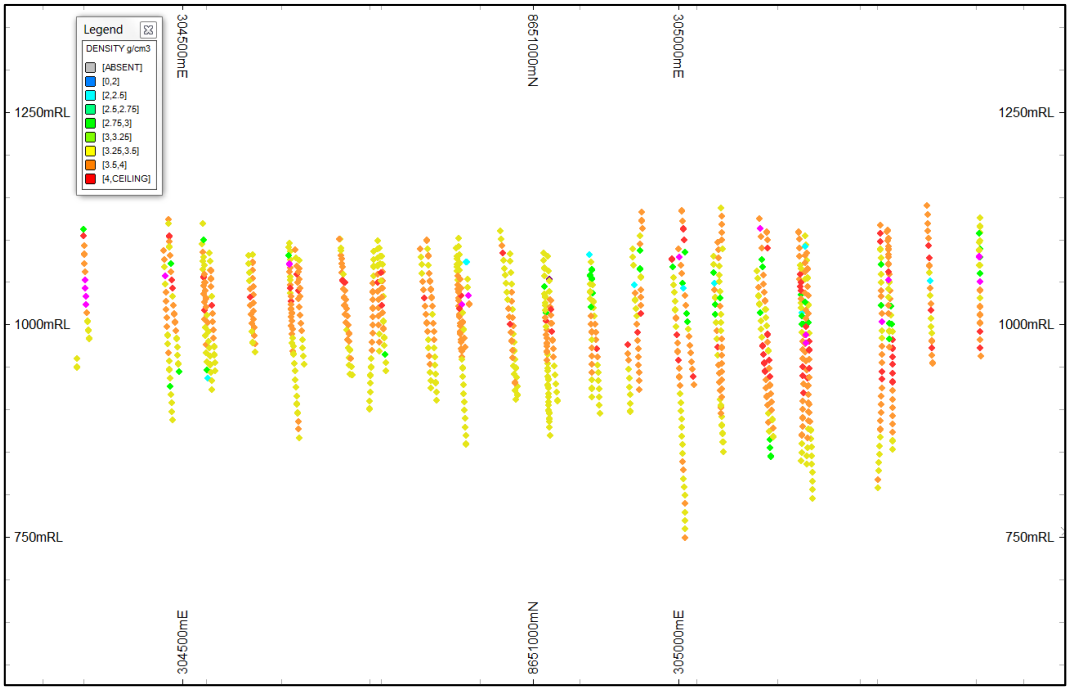
Density data for fresh material was reviewed statistically and spatially. There are sufficient samples to provide an estimate of average in-situ dry bulk density (Figure 14-13). Density outliers below 2.00 g/cm^3 and above 3.80 g/cm^3 were removed.

Figure 14-13 Histogram for density data of Fresh material in Enterprise Main



Data distribution for fresh density samples in Enterprise SW shows bi-modality around 2.80 and 3.00 g/cm^3 and when reviewed spatially, this appears to be constrained to certain horizons (Figure 14-14) as per the distribution of mineralisation.

Figure 14-14 Section view of Enterprise SW showing density zonation, looking NW



Density data was sub-divided by mineralisation zone and redox material (Table 14-9). The majority of Enterprise Main is fresh material with low variability in density. The resource at Enterprise SW is comprised of saprolite, saprock and fresh material. The majority (96%) of density data collected at Enterprise SW was in fresh material. There were insufficient samples to provide a confident average density value for saprolite and saprock. Densities were supplemented with data from adjacent

deposits within the copper belt region and applied to that material at Enterprise (Table 14-9 and Table 14-10).

Table 14-9 Density data for Enterprise Main and Enterprise SW

Weathering type	Enterprise SW	Enterprise Main	Total
Fresh	1,620	3,104	4,724
Saprock	59	49	108
Saprolite	7	1	8
Total	1,686	3,154	4,840

Table 14-10 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**	

14.2.7 Block model construction

A 40 m x 40 m x 5 m volume block model was built in Datamine software. This block size approaches the drill spacing, which on average is 50 m x 50 m and is also compatible with the proposed bench height of 5 m and the SMU size of 10 m x 10 m.

The block model was coded using topography, weathering layers and the footwall and hanging wall defining Ni and Cu mineralisation. Mineralisation was coded as ESTZONE = 1 and waste defined as ESTZONE = 0.

The block model and composites were un-faulted / un-folded in Datamine using the centre line plane defining the mineralisation zone as constructed by FQM.

Block model extents are presented in Table 14-11**.**

Table 14-11 Block model coordinate extents for final 10 m x 10 m x 5 m SMU block model

Attribute	Description
Cell size (X) (m)	10
Cell Size (Y) (m)	10
Cell Size (Z) (m)	5
Origin (X)	304,300
Origin (Y)	8,650,600
Origin (Z)	700
Number of cells in X direction	190
Number of cells in Y direction	180
Number of cells in Z direction	140

14.2.8 Grade estimation

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).

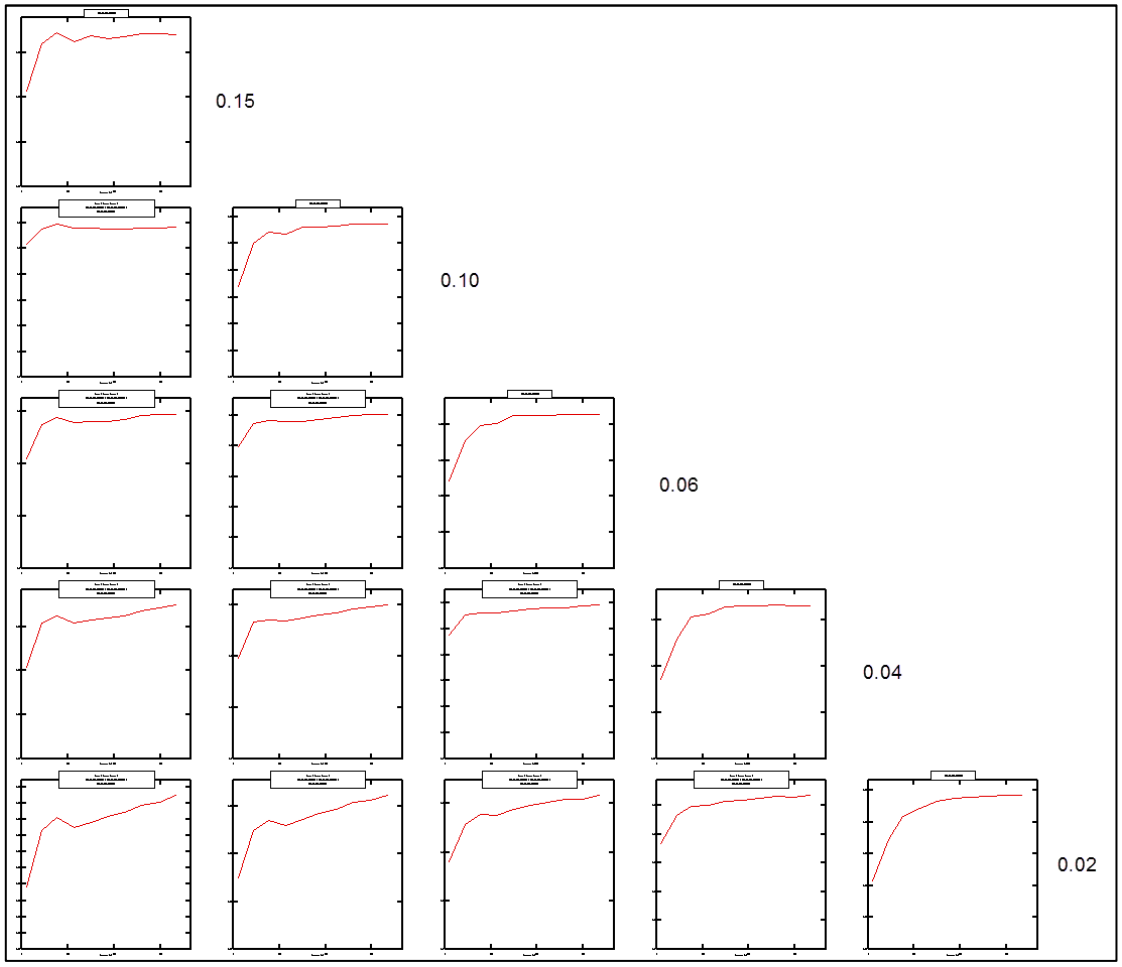
The analysis of the ratio of the cross variogram to simple variogram was completed for a range of Ni cut offs. This ratio is used to determine if the mineralisation satisfies the Diffusion Model type, confirmation of which is important in the reliable use of Uniform Conditioning. In simple terms this describes the spatial characteristics of internal grade boundaries. A Diffusion model is the case where it is likely that high grades are separated from low grades by a zone of intermediate grades. The alternative to the Diffusion Model is the Mosaic model, which is the basis of Multiple Indicator Kriging (MIK), an alternative method for estimating recoverable resources. Mosaic Model mineralisation is based on the assumption that individual grade classes are independent of each other. The case where it is likely that high grades juxtapose with low grades without transitioning through intermediate grades.

Figure 14-15 shows the results of this test. All variograms for each cut-off are well structured moving from low grade cut-offs (bottom right hand corner) to successively higher grade cut-offs (diagonal). The horizontal shows the cross-variograms between indicators moving from low cut-offs (right) to high cut-offs (left). The structured behaviour of these variograms supports the assumption that grade classes are not independent of each other and have the characteristics of Diffusion Model mineralisation. Mineralisation at Enterprise Main is dominated as sulphide material below BOCO. This supported the choice of UC for grade estimation.

Grades were estimated into the block model using ordinary kriging (OK) for the 40 m x 40 m x 5 m grid. Hard boundaries were used between the Enterprise Main and SW domains since they are spatially distinct and have different grade profiles. Contact analysis was completed and concluded that soft boundaries were appropriate.

Contact analysis for nickel in Enterprise SW between oxide and fresh material is presented in Figure 14-16. The graph shows an increase of grade at Enterprise SW with concentrations at either side of the base of complete oxidation (BOCO) surface.

Figure 14-15 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 14-12 and Table 14-13.

Figure 14-16 Contact analysis for Ni in Enterprise SW showing oxide to fresh from left to right

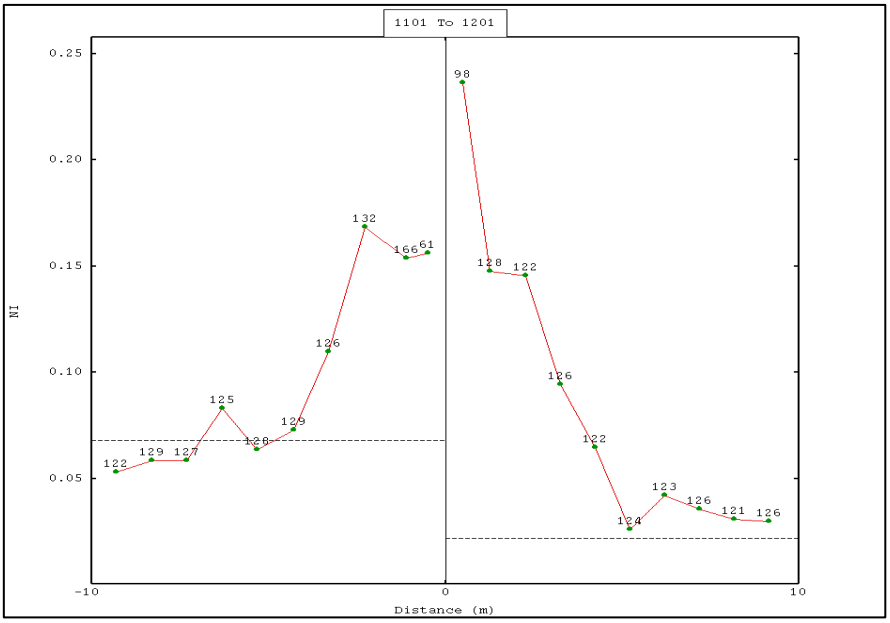


Table 14-12 Sample search criteria. Discretisation used was 3x3x5

Area	Pass	Variable	Search Ranges			Min Samples	Sectors	Max samples per sector	Min Samples per drill hole
			X	Y	Z				
Enterprise Main	1	Ni	100	80	7	18	4	10	6
	2		140	110	10				
	3		140	110	10	8		5	
	1	Cu	100	80	7	18		10	
	2		140	110	10				
	3		140	110	10	8		5	
Enterprise SW	1	Ni	100	80	7	18		10	
	2		140	110	10				
	3		140	110	10	8		5	
	1	Cu	100	80	7	18		10	
	2		140	110	10				
	3		140	110	10	8		5	

Table 14-13 Search ellipse rotations (Isatis Mathematician)

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

UC requires a Change of Support model to estimate the resource at the SMU block size. The Change of Support is an estimate of the volume-variance effect which in simple terms means that the grade variability of SMU blocks is higher than that estimated by the parent blocks, but less than that predicted by sample data. The Change of Support requires data to be transformed to Gaussian (or normal) distributions with variography completed to analyse the variance increase expected from parent blocks to SMU's. The grade-tonnage curve for each panel was then estimated using UC at a number of cut offs in order to adequately discretise the grade tonnage curve.

To produce a more realistic presentation of estimates, the block model was localised to SMU sized blocks using Localised Uniform Conditioning (LUC). This allows the grade tonnage distribution of each panel to be reproduced in the SMUs that make up that panel. LUC provides grades at the SMU (10*10*5) scale block size, minimising loss and dilution that may be expected during mining

14.2.9 Block model validation

A three point validation process was completed to validate the grade estimate:

- Validation of sample grade and block grade in 2D cross sections (Figure 14-17)
- Histograms of the input composite data compared to the output model file)
- Swath plots by northing, easting and elevation (Figure 14-18).

The Swath plots and Ni histograms were generated using model and composites flagged within the central mineralised portions of SW and Main constrained by elevation RL > 925 m and RL > 1,035 m.

Figure 14-17 Vertical Sections through Enterprise Main (top) and Enterprise SW (bottom) comparing composites and model

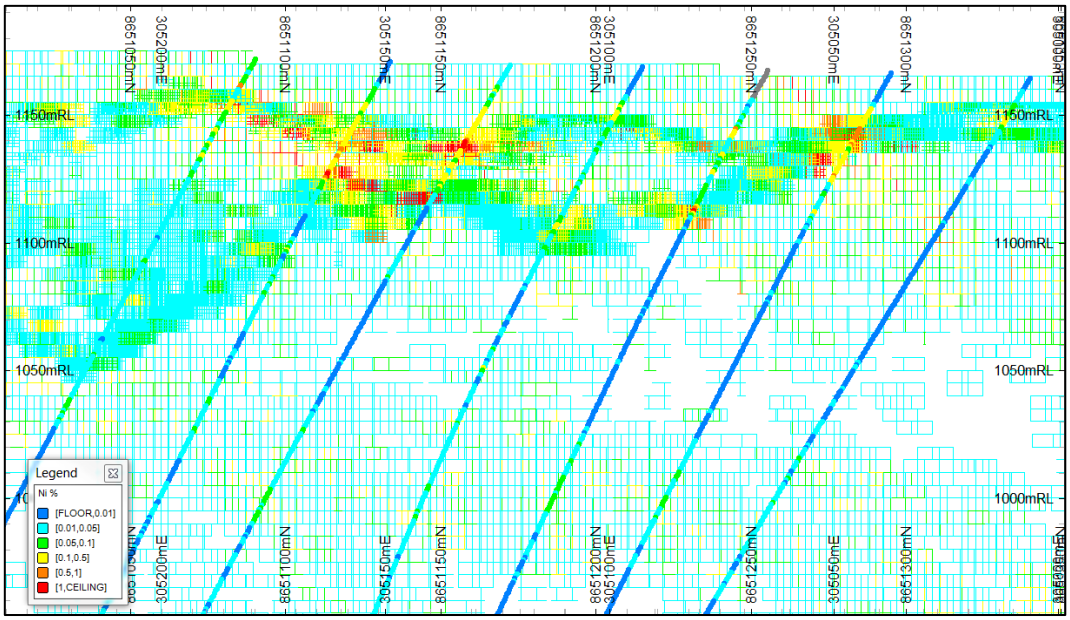
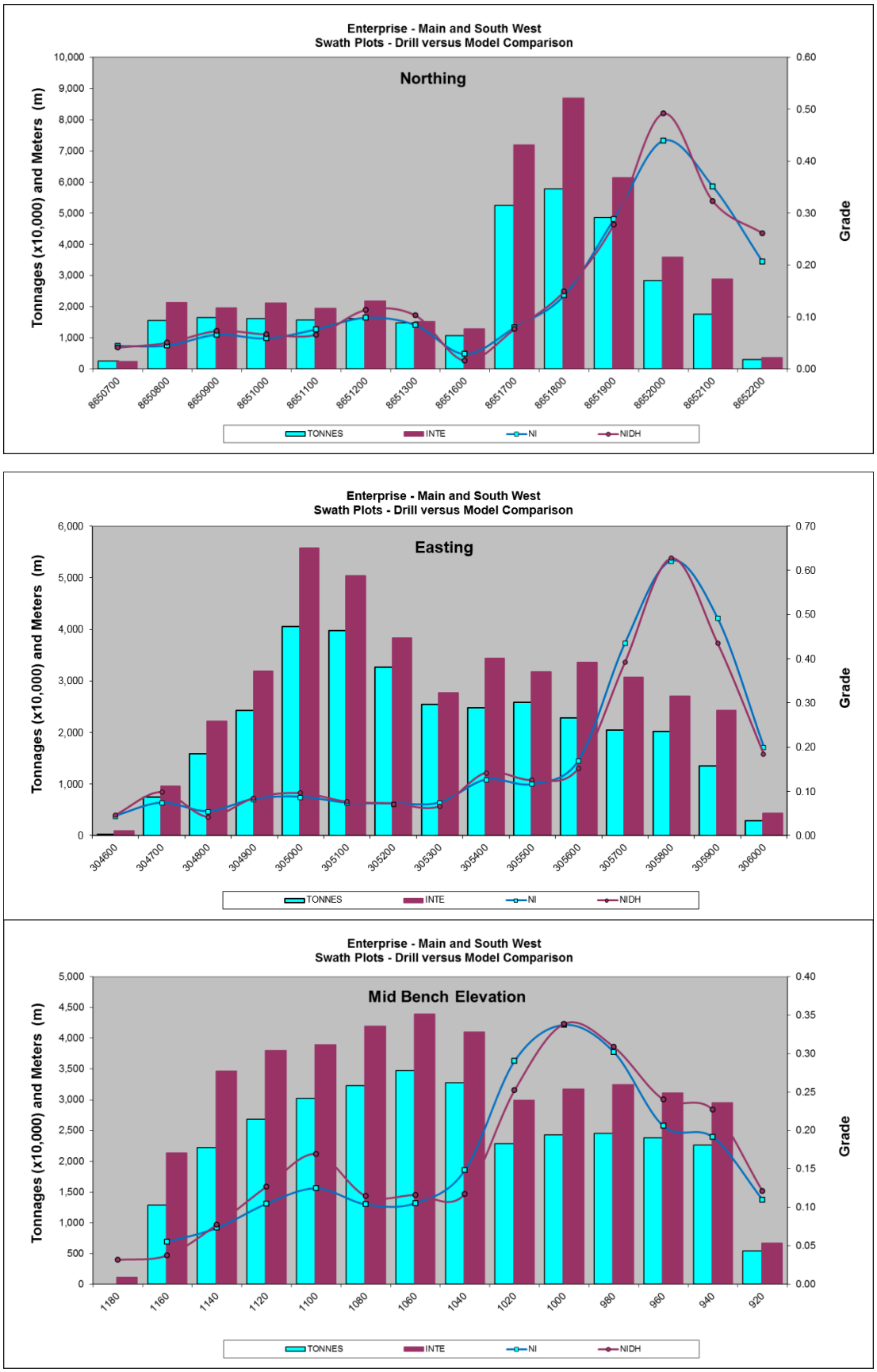


Figure 14-18 Swath plots for Ni at 100 m Northing (top), 100 m Easting (middle) and 20 m elevation (bottom) increments using central data



Validation confirmed that the block model reflects the tenor of composite grades, and can be considered reliable on a SMU scale in the areas of close spaced drilling, and reliable on a global scale in the areas outside the central zone.

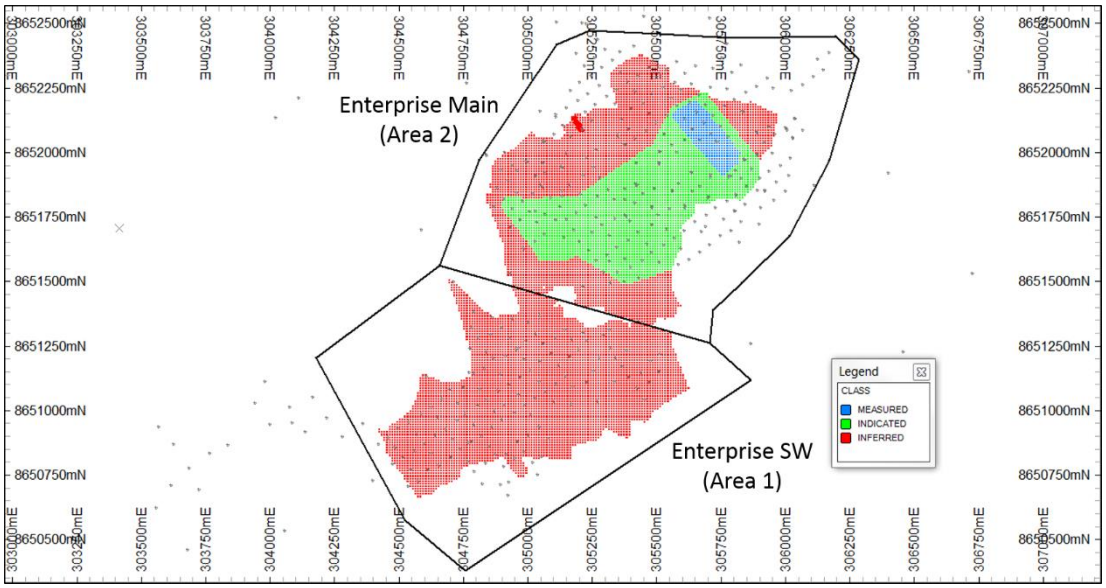
14.2.10 Mineral Resource classification

The Mineral Resource estimate was classified as Measured, Indicated and Inferred in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM, 2014) guidelines on Standards on Mineral Resources and Reserves, and also to comply with the guidelines specified in the JORC Code Edition 2012 (JORC 2012). Classification is based on a number of criteria including: verification of tenement title; review of drilling, sampling, assaying and geology during the site visit completed by the QP; 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 is an indicator of relative confidence of the grade estimate. This measurement was combined with geological confidence and sample spacing and used as a guide to determining classification boundaries. Specifically, the following criteria were considered by the QP:

- Measured Resources:
  - Drill spacing of 50 m x 50 m
  - Average slope of regression >= 0.7
- Indicated Resources:
  - Slope of regression >= 0.5 at Enterprise Main, not already classified as Measured Resources
  - A small volume of fresh material was classified as Indicated at Enterprise SW. There is potential for further upgrade of Inferred Resources once further bulk density analysis has been completed for saprolite and saprock.
  - All remaining material was classified as Inferred Resources.

Figure 14-19 presents a plan view of the classified Mineral Resource estimate.

Figure 14-19 Plan view of the Enterprise block model, coloured by CLASS, where Blue is Measured, Green is Indicated and Red is Inferred



14.2.11 Mineral Resource estimate

The Mineral Resource estimate for Enterprise is presented Table 14-14 and is reported at a 0.15% nickel cut-off grade. The Mineral Resource estimate was estimated as a 3D block model within a constraining mineralised wireframe.

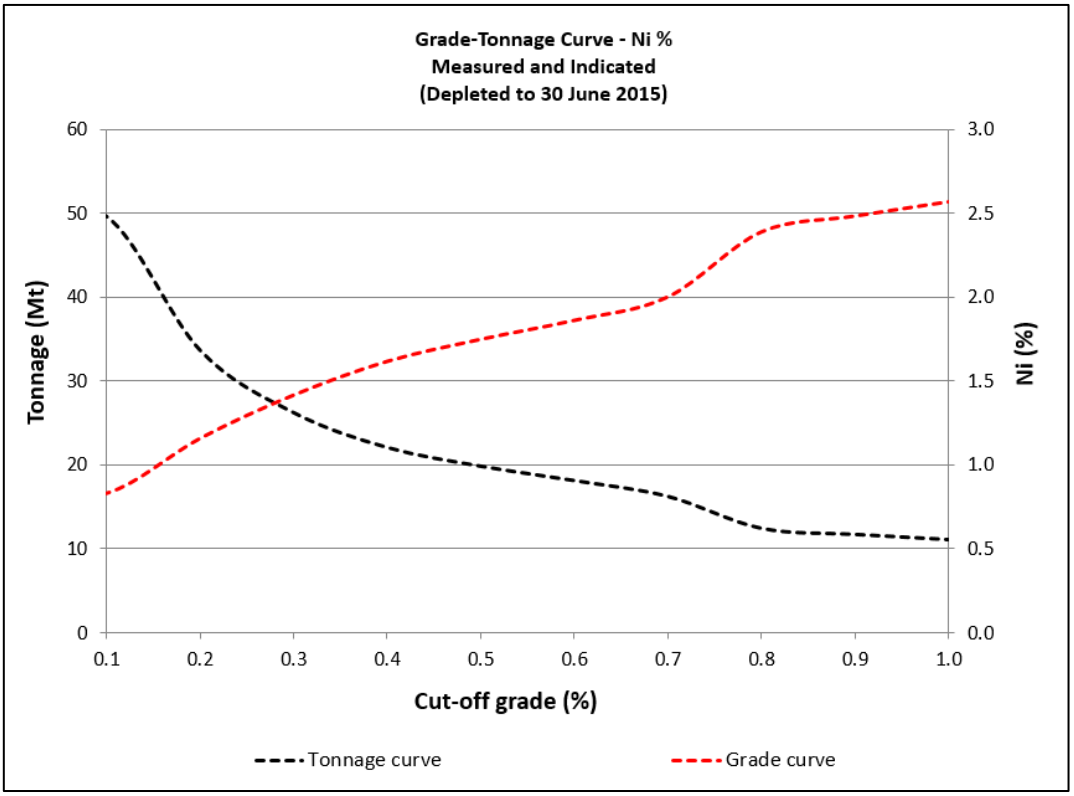
Table 14-14 Enterprise Nickel Deposit Mineral Resource statement as at 31st May 2015. Mineral Resources are inclusive of Mineral Reserves

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**

To the best knowledge of the QP, the stated mineral resources are 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. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Grade and tonnage tabulations of the Measured and Indicated material at Enterprise are presented in a grade tonnage chart shown in Figure 14-20.

Figure 14-20 Grade/tonnage curve for Measured and Indicated Resources



14.2.12 Comparison with previous Mineral Resource estimate

Comparison was made with the previous 2012 Mineral Resource estimate. In summary:

- Tonnes of combined Measured and Indicated resources decreased by 1%, with metal decreasing by 6% due to the decrease in grade.
- Inferred tonnes increased to 20.9 Mt due to additional drilling at Enterprise SW. Inferred Mineral Resources have good opportunity to be converted into Indicated Mineral Resources with infill drilling and improved geological understanding.

Additional differences between the 2012 and 2013 Mineral Resource estimates include:

- The current Mineral Resource estimate used an un-faulting and un-folding method based on an interpreted surface which honoured the centre line of the mineralisation and fault offsets.
- The current Mineral Resource estimate was estimated using UC and localised to an SMU of 10 m x 10 m x 5 m using LUC. This provides an improved estimate relevant to the scale of mining and minimises loss and dilution.
- Additional saprolite and saprock density measurements are required across Enterprise, but particularly for the Enterprise SW deposit.

ITEM 15 MINERAL RESERVE ESTIMATES

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 corresponding Mineral Resource models and estimates as reported in Item 14. This new information has been authored by Michael Lawlor (QP).

15.1 Sentinel

15.1.1 Introduction

As part of the Mineral Reserves estimation process for Sentinel, pit optimisation aspects and detailed pit designs were completed by FQM personnel overseen and supervised by Michael Lawlor (QP). All operating cost, recovery and revenue information for the optimisations, in addition to operational parameters for the open pit designs, were reviewed by Michael Lawlor (QP).

To conform with NI 43-101 standards, the Mineral Reserve estimate is derived from Measured and Indicated Mineral Resources only. The Sentinel Measured and Indicated Mineral Resource estimates as listed in Table 14-4 are reported inclusive of the Sentinel Mineral Reserves.

15.1.2 Methodology

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

The ultimate (optimal) shell and conceptual stage limits based on preliminary scheduling were used to create practical and detailed open pit designs accounting for batters, berms, haul roads and in-pit crusher relocations.

15.1.3 Mine planning model

A regularised block model to 12 m x 12 m x 6 m was provided for pit optimisation and mine planning work. The modelling process as described in Item 14.5.1 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, and without compromising the definition of the original model.

15.1.4 Pit optimisation

Optimisation methodology

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 fixed process recovery values. This differs from the approach taken in the 2012 Technical Report (DumpSolver, May 2012), in which the optimal pit shell was selected on the basis of maximum NPV (from an optimisation and high-level Miliwa schedule).

The current approach yields a mining inventory that allows for a more practical long-term production schedule to be developed, accounting for mining stages and a stockpiling strategy which suits proposed in-pit crushing, and with plant capacity sustained over a longer period.

Pit slope design criteria

Pit optimisation input included overall slope design angles based on the diagram shown in Figure 15-1 and the interpreted overall slope design parameters as listed at the bottom of Table 15-1. The slope areas shown in Figure 15-1 are superimposed on an earlier ultimate pit design. The geotechnical engineering basis for these slope design parameters is outlined in Item 16.

Figure 15-1 Sentinel pit slope optimisation and design sectors

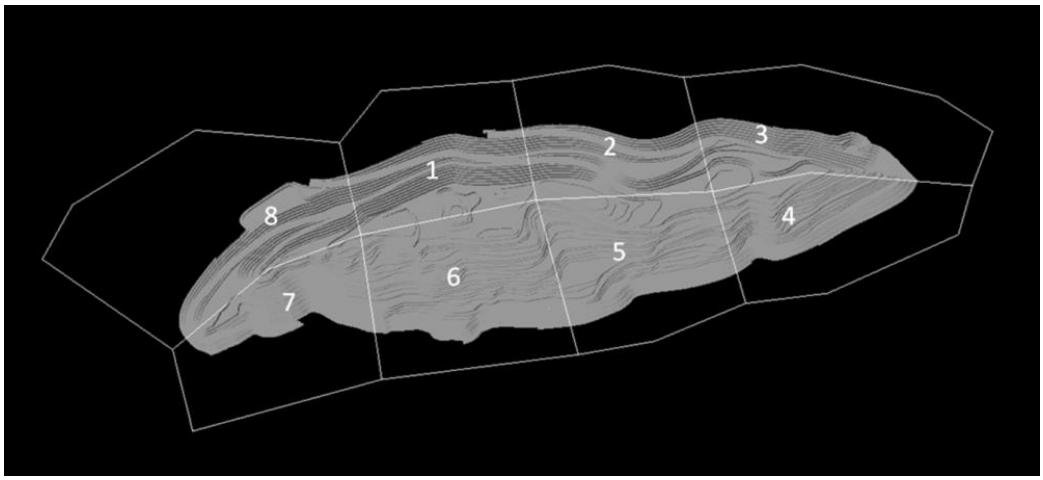


Table 15-1 Overall pit slope angles for optimisation input, Sentinel

Slope Area		1	2	3	4	5	6	7	8
MPH	mRL	1064	1029	966	1271	1193	surface	surface	1175
MPHB	mRL	1060	1021	949	1212	1145	1253	1279	1152
MSC	mRL	from surface	from surface	from surface	n/a	n/a	n/a	n/a	surface
Top	mRL	1220	1196	1232	1208	1120	1220	1196	1184
Bottom	mRL	884	944	956	944	932	896	932	930
Ramp1	mRL	1100	1112	1028	989	1064	1184	1172	1050
Ramp2	mRL	1064	992			968			990
Ramp width	mRL	50	50	50	50	50			50
Weathered Rock Base	mRL	1180	1156	1192	1168	1080	1180	1156	1144
Vertical distance to ramp1	m	40	40	40	40	40	36	24	40
Horizontal distance to ramp1	m	66	66	66	66	66	116	99	66
Vertical distance to ramp2	m						4	16	
Horizontal distance to ramp2	m						16	33	
MSC Base	mRL	1064	1029	966	ignored	ignored	ignored	ignored	ignored
Vertical distance to ramp1	m	80	44	164					
Horizontal distance to ramp1	m	103	83	164					
Vertical distance to ramp2	m	36	120	62					
Horizontal distance to ramp2	m	93	178	47					
MPH Base	mRL	ignored	ignored	956	ignored	ignored	ignored	ignored	ignored
Vertical distance	m			10					
Horizontal distance	mRL			7					
MPHB Base	mRL	884	944	ignored	944	932	896	932	930
Vertical distance to ramp1	m	180	48		179	16	284	224	94
Horizontal distance to ramp1	m	120	33		262	74	341	271	168
Vertical distance to ramp2	m				45	96			60
Horizontal distance to ramp2	m				59	228			157
Vertical distance to ramp3	m					36			60
Horizontal distance to ramp3	m					47			71
Total Vertical Distance	m	336	252	276	264	188	324	264	254
Total Horizontal Distance	m	382	360	283	386	414	473	403	461
Overall Slope Angle Input	**degrees**	**41**	**35**	**44**	**34**	**24**	**34**	**33**	**29**

Metal price

A copper price of US$3.00/lb (US$6,615/t) was used for the optimisation.

Metal recoveries

Metal recovery figures adopted for pit optimisation reflect the metallurgical recommendations in Item 13.1.4, as follows:

- Mixed ore, or non-primary sulphide ore (also referred to as OSS ore) = 70%
- Sulphide ore, or primary sulphide ore (also referred to as PSS ore) = 92%

Operating costs

Since the Project will be mill constrained, the process operating costs are the sum of the fixed and variable costs. The same costs as used in the optimisation described in the 2012 Technical Report (DumpSolver, May 2012) were reviewed and considered as appropriate for adoption, ie:

- fixed operating costs = $0.60/t processed
- variable operating costs = $5.50/t processed
- total operating costs = $6.10/t processed

More information on these cost estimates and their derivation are outlined in Item 21.

Mining costs

Variable mining costs comprising drill, blast, load and haul costs, on a bench by bench basis, were determined from haulage simulations taking account of trolley-assisted waste haulage and shortened ore hauls to in-pit crushers. This was an update on the varying mining costs adopted for the optimisation described in the 2012 Technical Report (DumpSolver, May 2012). Item 21 provides more information on the updated incremental relationship adopted for ore and waste mining costs in the pit optimisation process. The overall average mining costs are as follows:

- average waste mining cost = $1.84/t
- average ore mining cost = $1.77/t
- average waste+ore mining cost = $1.82/t

Metal costs

In addition to 6% (gross) royalties, and at the time of the optimisation in 2014, metal costs for the Sentinel product comprised concentrate transport charges, treatment and refining charges, plus smelter deductions. The same costs were reviewed and adopted as those reported in the 2012 Technical Report (DumpSolver, May 2012), ie:

- concentrate transport charge = $0.05/lb (based on $25/wmt, 9% moisture content and 24% concentrate grade)
- concentrate treatment charge = $0.13/lb (based on $70/dmt, 9% moisture content and 24% concentrate grade)
- concentrate refining charge = $0.07/lb
- smelter deduction value = $0.13/lb (based on 96% payability, 0.5% losses in shipment)
- 6% royalty = $0.18/lb
- total copper metal costs = $0.57/lb

Details of these cost estimates and their derivation are outlined in Item 21.

Mining dilution and recovery factors

Due to the incorporation of "planned dilution" in the Mineral Resource model (Item 14.5.1), pit optimisation assumptions included mining dilution and mining recovery factors of 1.0 and 1.0, respectively. "Unplanned dilution" and recovery factors, however, were subsequently included in the production scheduling inventory (Item 16.1.2).

Marginal cut-off grades

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

Marginal COG = (PROCOST x MINDIL)/(NR x RECOVERY)

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

MINDIL is the mining dilution factor

Table 15-2 Sentinel marginal cut-off grades

Marginal Cut-Off Grade:	Units	Non-primary	Primary
PROCOST	$/t ore	6.10	6.10
MINDIL factor		1.0	1.0
METAL PRICE	$/lb Cu	3.00	3.00
	$/t Cu	6,614	6,614
METAL COST	$/lb Cu	0.57	0.57
	$/t Cu	1,258	1,258
TOTAL NET RETURN	$/lb Cu	2.43	2.43
	$/t Cu	5,356	5,356
	$/10kg Cu	53.56	53.56
RECOVERY	%	70	92
C/O GRADE	**% Cu**	**0.16**	**0.12**

Optimisation results

Figure 15-2 shows the graphical results of the base case pit optimisation, whilst Table 15-3 lists the complete inventory of shell sizes and corresponding cashflows.

The optimal shell was selected on a maximum undiscounted cashflow basis. This corresponds with a revenue factor of 1.00 and differs from the approach taken for the 2012 Mineral Reserve estimate (DumpSolver, May 2012), in which the optimal pit shell was selected on the basis of maximum NPV (from an optimisation and high-level Miliwa schedule) corresponding to a revenue factor of 0.87.

Pit optimisation sensitivity analyses

A number of optimisation sensitivity analyses were carried out on the marginal cut-off grade case (base case). Sensitivity analyses at +/- 10% were completed to test the impact of varying copper price, royalties (increasing to 9%), mining costs, processing costs and metal costs. The results are listed in Table 15-4 and indicate that:

- the most sensitive variable impacting on both the size of the optimal pit and the cashflow is metal price (note: that varying the processing recovery would have the same level of impact)
- increasing the royalty from 6% to 9%, ie to the now prevailing level, results in an approximate 8% reduction in undiscounted project value.

Figure 15-2 Sentinel optimisation results, at marginal cut-off grade

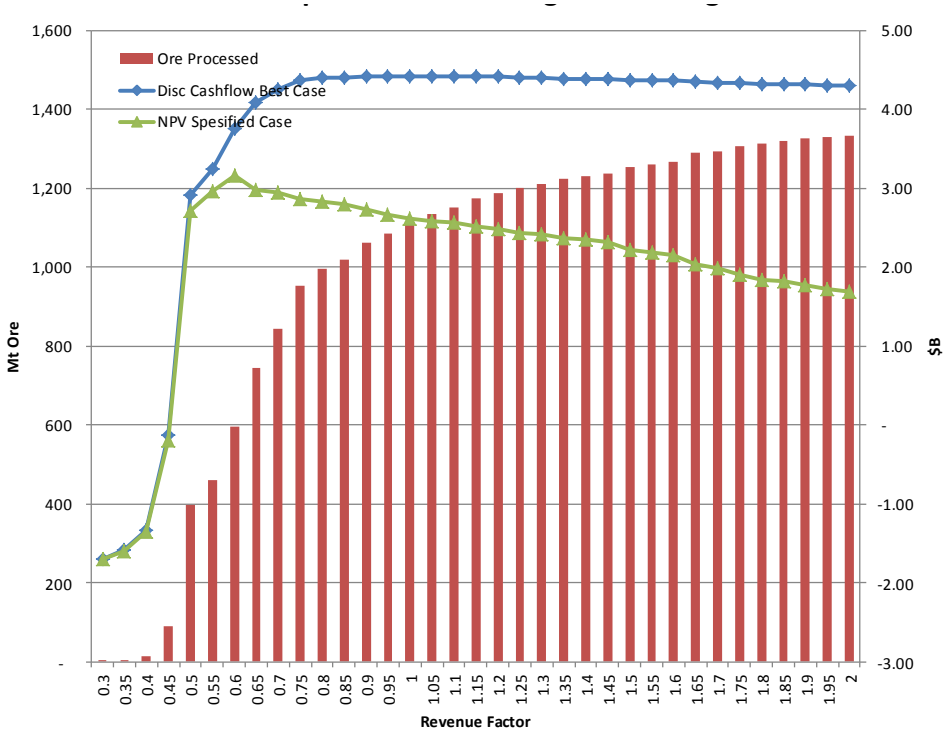


Table 15-3 Sentinel pit optimisation shell sizes and cashflow, at marginal cut-off grade

Shell	Revenue Factor	Pit Size (Mt)	Ore Processed (Mt)	Cu Plant Feed (%)	Cu Recovered (Mt)	Undiscounted Cashflow Best Case ($B)
1	0.30	0.6	0.4	1.49	0.01	-1.70
2	0.35	4.8	3.6	1.07	0.03	-1.58
3	0.40	22.5	14.2	0.87	0.10	-1.29
4	0.45	169.2	89.3	0.67	0.52	0.22
5	0.50	880.3	398.4	0.60	2.15	5.84
6	0.55	1,017.0	460.6	0.59	2.42	6.66
7	0.60	1,316.2	596.8	0.55	2.94	8.08
8	0.65	1,690.7	744.8	0.52	3.45	9.25
9	0.70	2,001.0	844.3	0.50	3.81	9.99
10	0.75	2,452.4	953.3	0.49	4.23	10.73
11	0.80	2,609.3	996.0	0.49	4.37	10.92
12	0.85	2,694.3	1,018.2	0.49	4.44	11.00
13	0.90	2,892.8	1,062.7	0.48	4.58	11.12
14	0.95	3,013.6	1,084.3	0.48	4.65	11.16
15	1.00	3,122.2	1,108.1	0.47	4.72	11.16
16	1.05	3,248.2	1,134.9	0.47	4.79	11.15
17	1.10	3,327.6	1,151.2	0.47	4.83	11.13
18	1.15	3,460.7	1,174.6	0.46	4.90	11.07
19	1.20	3,523.5	1,185.8	0.46	4.93	11.04
20	1.25	3,640.5	1,201.6	0.46	4.97	10.97
21	1.30	3,691.5	1,209.3	0.46	4.99	10.93
22	1.35	3,813.7	1,225.1	0.46	5.04	10.82
23	1.40	3,840.3	1,229.1	0.46	5.04	10.80
24	1.45	3,913.6	1,237.8	0.45	5.07	10.72
25	1.50	4,056.4	1,254.8	0.45	5.11	10.57
26	1.55	4,127.9	1,261.6	0.45	5.13	10.49
27	1.60	4,171.6	1,266.1	0.45	5.14	10.44
28	1.65	4,384.0	1,288.8	0.45	5.19	10.17
29	1.70	4,443.1	1,294.2	0.45	5.20	10.10
30	1.75	4,578.4	1,307.6	0.44	5.23	9.92
31	1.80	4,663.5	1,314.1	0.44	5.25	9.80
32	1.85	4,704.5	1,317.9	0.44	5.26	9.74
33	1.90	4,797.2	1,325.6	0.44	5.27	9.61
34	1.95	4,865.5	1,330.4	0.44	5.29	9.51
35	2.00	4,917.0	1,334.4	0.44	5.29	9.44

- whilst the project value is reduced, the impact on the marginal cut-off grade is minimal and the size of the optimal pits are relatively similar (ie, a 3% difference is an acceptable tolerance between a design pit and its corresponding optimal shell outline)
- on this basis, there is no need to revise the optimisation, pit designs and production schedules to account for the royalty increase; the impact can be addressed as a cashflow sensitive variable
- +/- 10% variations in the mining, operating and metal costs are less sensitive

Table 15-4 Sentinel pit optimisation sensitivity analyses

Ultimate Pits		at 6% royalty	at 9% royalty	at $2.70/lb	at $3.30/lb	+10% mining	-10% mining	+10% opex	-10% opex	+10% metal	-10% metal
Plant Feed	Mt	1,108.1	1,068.2	1,001.8	1,213.1	1,085.8	1,131.9	1,036.8	1,187.2	1,079.2	1,131.1
Feed Grade	%Cu	0.47	0.48	0.50	0.45	0.48	0.47	0.49	0.45	0.48	0.47
Cu Metal Insitu	kt	5,241.1	5,159.6	5,009.2	5,447.0	5,168.4	5,320.1	5,121.8	5,366.1	5,180.4	5,293.6
Waste	Mt	2,014.1	1,963.3	1,899.6	2,109.5	1,923.0	2,109.4	2,001.2	2,010.1	1,968.5	2,038.0
Total Mined	Mt	3,122.2	3,031.5	2,901.5	3,322.6	3,008.8	3,241.3	3,038.0	3,197.3	3,047.7	3,169.1
Strip Ratio		1.8	1.8	1.9	1.7	1.8	1.9	1.9	1.7	1.8	1.8
Total tonnage variance		Base	-3%	-7%	6%	-4%	4%	-3%	2%	-2%	2%
Recovered Cu	kt	4,717.9	4,638.5	4,508.0	4,899.1	4,650.3	4,785.8	4,602.9	4,830.2	4,657.4	4,760.7
Undisc. Cashflow	**M$**	**11,163.6**	**10,235.3**	**8,318.2**	**14,134.8**	**10,606.7**	**11,746.0**	**10,510.3**	**11,864.0**	**10,566.0**	**11,753.7**
Undisc cashflow variance		Base	-8%	-25%	27%	-5%	5%	-6%	6%	-5%	5%

An optimisation analysis was also completed to assess the impact of an elevated cut-off grade, above the marginal grades listed in Table 15-2. This was done to assess the potential improvements to the plant feed grade profile within the first half of the LOM production schedule, coupled with stockpile building and reclaim strategies. Figure 15-3 shows the graphical result of this pit optimisation, whilst Table 15-5 lists the complete inventory of shell sizes and corresponding cashflows. The comparative results in Table 15-6 and the adoption of a 0.2% Cu cut-off grade lead into the production scheduling exercise described in Item 16.1.3.

Figure 15-3 Sentinel optimisation results, at 0.2% Cu elevated cut-off grade

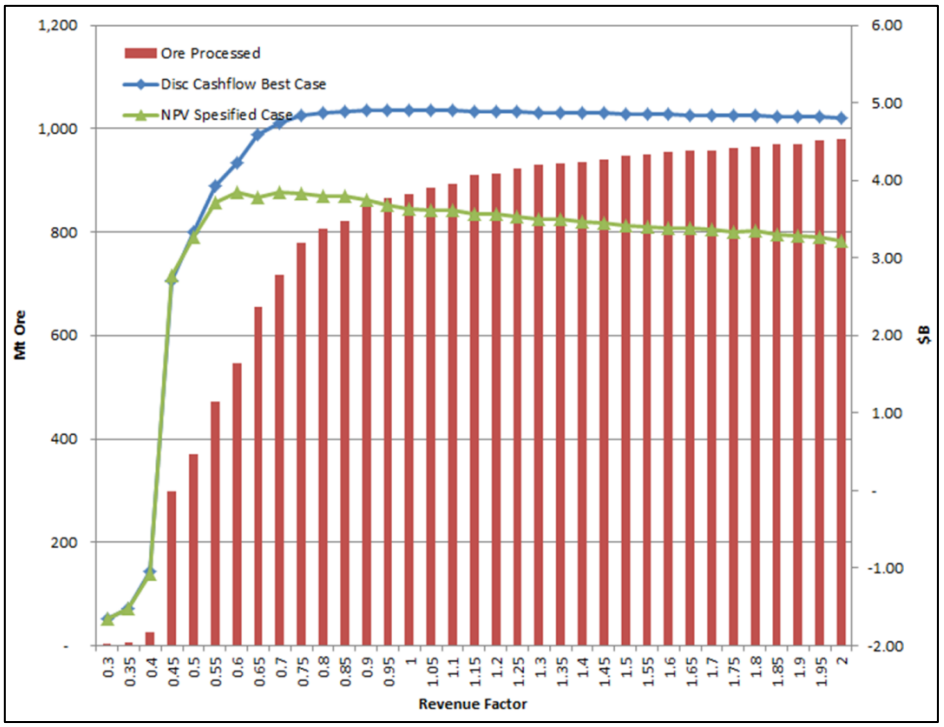


Table 15-5 Sentinel pit optimisation shell sizes and cashflow, at 0.2% Cu elevated cut-off grade

Shell	Revenue Factor	Pit Size (Mt)	Ore Processed (Mt)	Cu Plant Feed (%)	Cu Recovered (Mt)	Undiscounted Cashflow Best Case ($B)
1	0.30	0.6	0.4	1.49	0.01	-1.70
2	0.35	4.8	3.6	1.08	0.03	-1.58
3	0.40	22.5	14.0	0.88	0.10	-1.29
4	0.45	169.2	84.0	0.70	0.51	0.21
5	0.50	880.3	348.7	0.67	2.08	5.76
6	0.55	1,017.0	401.9	0.65	2.34	6.56
7	0.60	1,316.2	517.7	0.61	2.83	7.95
8	0.65	1,690.7	623.0	0.59	3.28	9.06
9	0.70	2,001.0	697.2	0.58	3.60	9.75
10	0.75	2,452.0	774.2	0.57	3.97	10.44
11	0.80	2,604.6	802.2	0.57	4.09	10.61
12	0.85	2,687.6	815.5	0.57	4.15	10.68
13	0.90	2,867.3	845.8	0.56	4.26	10.78
14	0.95	2,978.4	859.0	0.56	4.32	10.81
15	1.00	3,060.0	870.5	0.56	4.36	10.82
16	1.05	3,170.0	884.3	0.56	4.42	10.81
17	1.10	3,230.6	891.8	0.55	4.44	10.79
18	1.15	3,378.3	908.7	0.55	4.51	10.74
19	1.20	3,413.7	912.8	0.55	4.52	10.73
20	1.25	3,512.0	921.6	0.55	4.55	10.67
21	1.30	3,569.5	927.6	0.55	4.58	10.64
22	1.35	3,617.8	931.6	0.55	4.59	10.60
23	1.40	3,646.6	934.5	0.55	4.60	10.58
24	1.45	3,719.3	939.7	0.55	4.62	10.52
25	1.50	3,786.7	945.8	0.55	4.64	10.46
26	1.55	3,824.6	948.9	0.55	4.65	10.42
27	1.60	3,894.8	953.7	0.54	4.67	10.35
28	1.65	3,920.9	955.9	0.54	4.68	10.33
29	1.70	3,950.2	958.0	0.54	4.68	10.30
30	1.75	3,968.9	959.2	0.54	4.69	10.28
31	1.80	4,025.2	964.1	0.54	4.70	10.21
32	1.85	4,105.9	968.5	0.54	4.72	10.12
33	1.90	4,134.4	970.3	0.54	4.72	10.09
34	1.95	4,208.6	975.0	0.54	4.74	9.99
35	2.00	4,299.1	979.3	0.54	4.75	9.89

Table 15-6 Sentinel pit optimisation results, marginal and elevated cut-off grades

Ultimate Pits		Marginal COG	0.2% Cu COG
Plant feed	Mt	1,108.1	870.5
Feed grade	%Cu	0.47	0.56
Cu metal insitu	kt	5,241.1	4,857.5
Waste	Mt	2,014.1	2,189.5
Total mined	Mt	3,122.2	3,060.0
Strip ratio		1.8	2.5
Recovered Cu	kt	4,717.9	4,362.3
Undisc. Cashflow	M$	11,163.6	10,817.7
Disc. Cashflow	M$	4,423.6	4,582.3
Ultimate Pits		**Variance from Marginal COG**	
Plant feed	Mt		-21%
Feed grade	%Cu		18%
Cu metal insitu	kt		-7%
Waste	Mt		9%
Total mined	Mt		-2%
Strip ratio			38%
Recovered Cu	kt		-8%
Undisc. Cashflow	M$		-3%
Disc. Cashflow	M$		4%

15.1.5 Detailed pit designs

A series of phased (or staged) pit designs were developed starting with the ultimate pit shell. The elevated 0.2% cut-off grade pit shell 15 was adopted on the basis that the difference in total mined tonnage between this shell and the marginal cut-off grade pit shell was within 2% to 3%. This suited the subsequent production scheduling work, in which low grade ore (ie, ore between the marginal and the elevated 0.2% cut-off grades) could be used to sustain the processing plant capacity.

The progression of mining to suit in-pit crushing and conveying was determined through preliminary designs and production schedule iterations.

Design and planning parameters

Drawing on the information shown in diagrams 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 with dual conveyor = 55 m
- haul road width to cater for trolley-assist = 50 m
- maximum haul ramp gradient = 1: 10 (approximately 6^{o})

The geotechnical design criteria for detailed pit design was determined from the parameters provided by Xstract Mining Consultants (XStract, 2014) and described in Item 16.1.2. The interpreted parameters are listed in Table 15-1.

Staged and ultimate pit designs

Figure 15-4 shows the starting point for the design (and production scheduling) process. This figure shows the extent of mining as at the end of May 2015. An in-pit crusher (IPC) is shown where currently positioned in the north west boxcut. 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 15-4 is the south box cut and a conveyor ramp leading out of the pit towards the surface transfer bin. The excavation of two additional IPC pockets is currently in progress within this box cut. Item 16.1.3 provides more information on the in-pit crushing and conveying concepts.

Figure 15-5 to Figure 15-20 show the progression of mining stages throughout the life of mine. These views do not correspond to points in time, as several stages could be in progress at one time. These figures also show proposed future IPC positions. There will be up to three crushers operating at one time, and in those stage views where more than three crushers are shown, the design intent is to provide pockets for crusher relocation during mining of that particular stage.

Validation between pit shell and design pit

Table 15-7 shows a comparison between the selected shell 15 optimal pit inventory and the inventory for the corresponding detailed ultimate pit design. The comparison shows an acceptable overall variance.

The inventories are depleted to the end of May 2015.

Figure 15-4 Sentinel Pit at May 2015

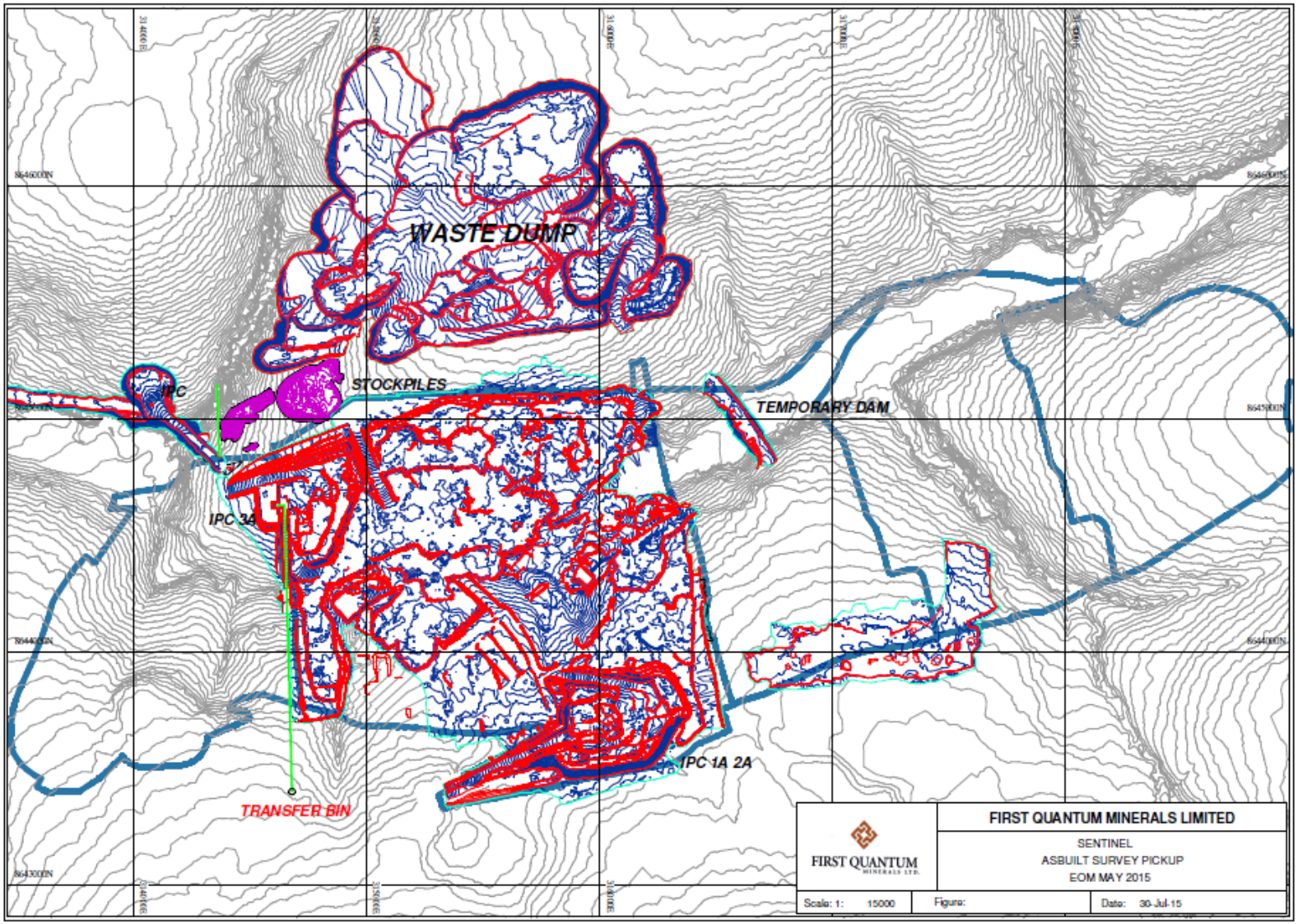


Figure 15-5 Sentinel Pit, Stage 2 design

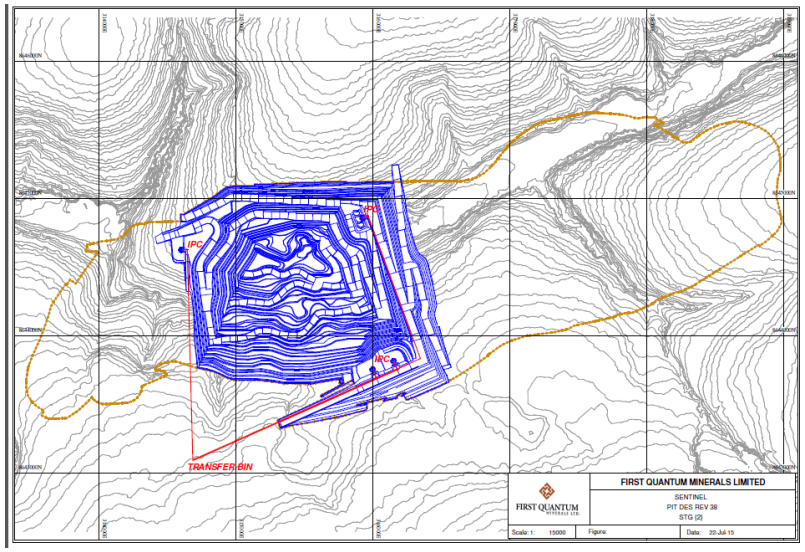


Figure 15-6 Sentinel Pit, Stage 3a design

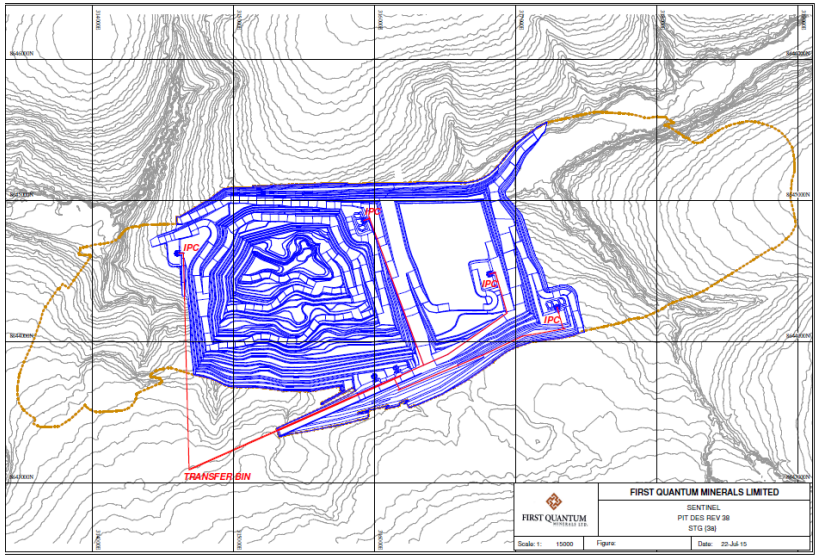


Figure 15-7 Sentinel Pit, Stage 3b design

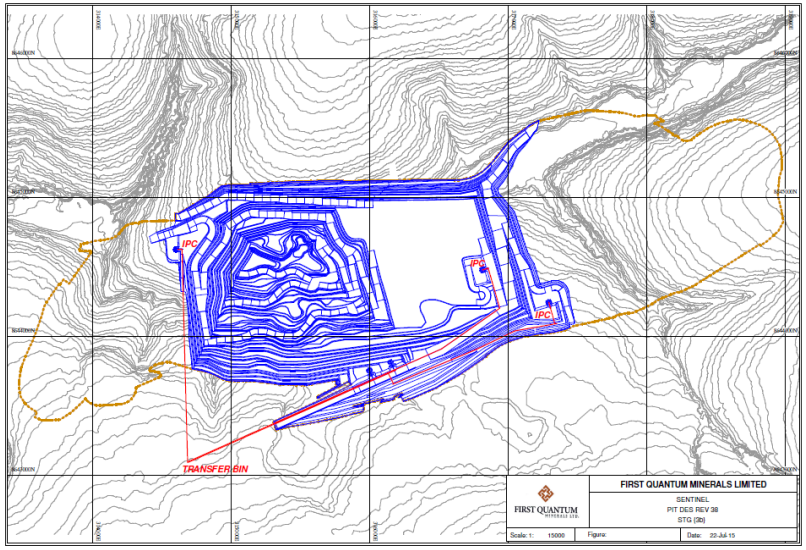


Figure 15-8 Sentinel Pit, Stage 4 design

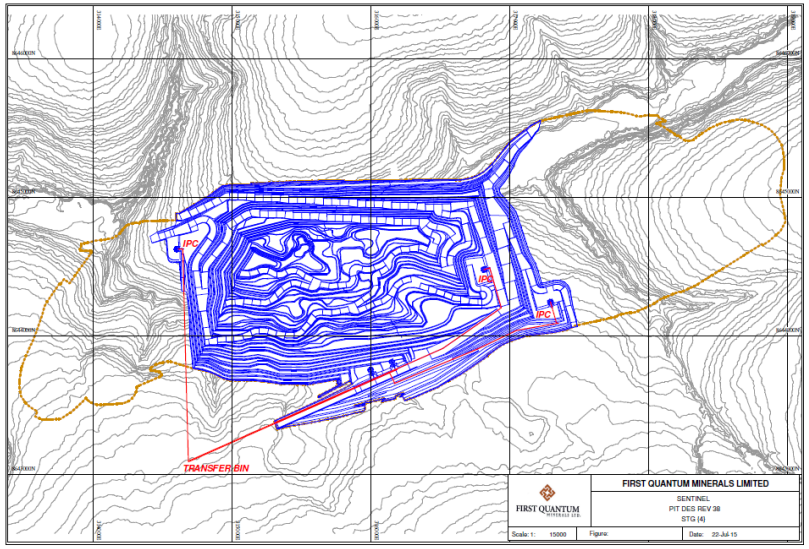


Figure 15-9 Sentinel Pit, Stage 5a design

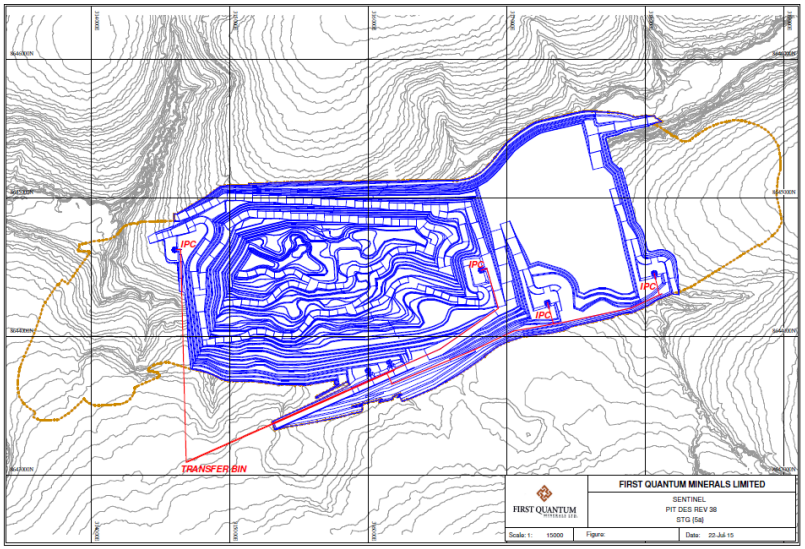


Figure 15-10 Sentinel Pit, Stage 5b design

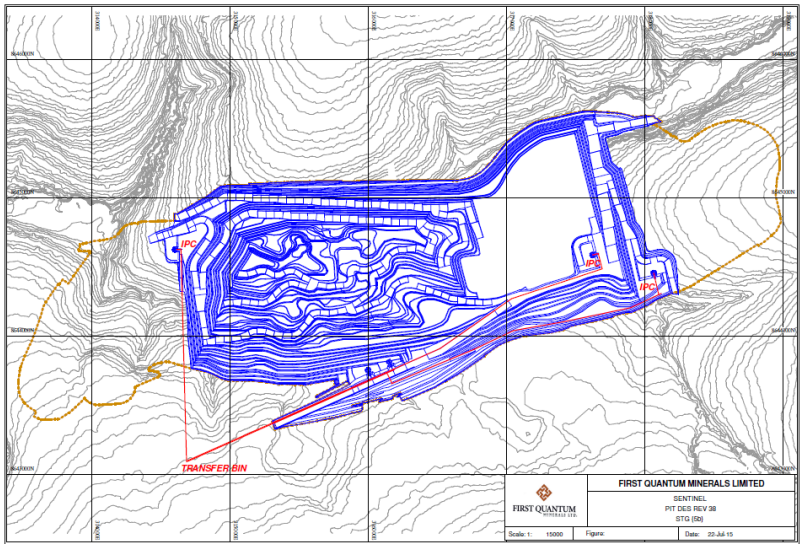


Figure 15-11 Sentinel Pit, Stage 6 design

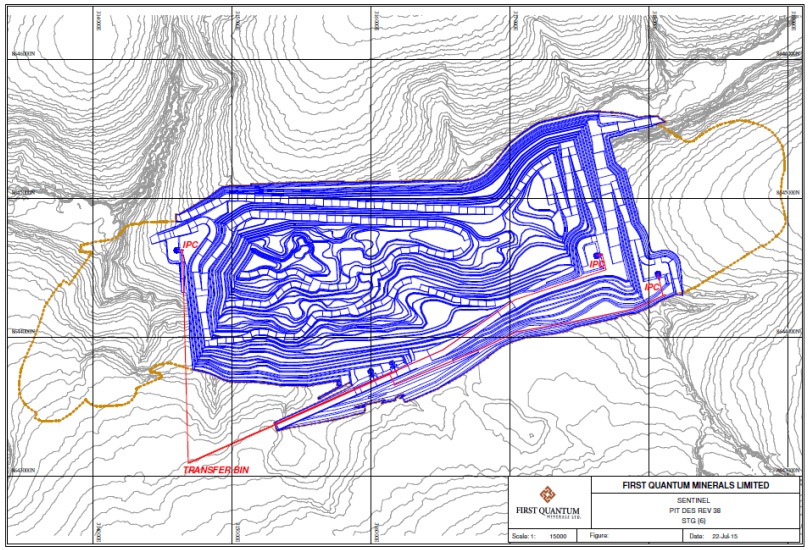


Figure 15-12 Sentinel Pit, Stage 7a design

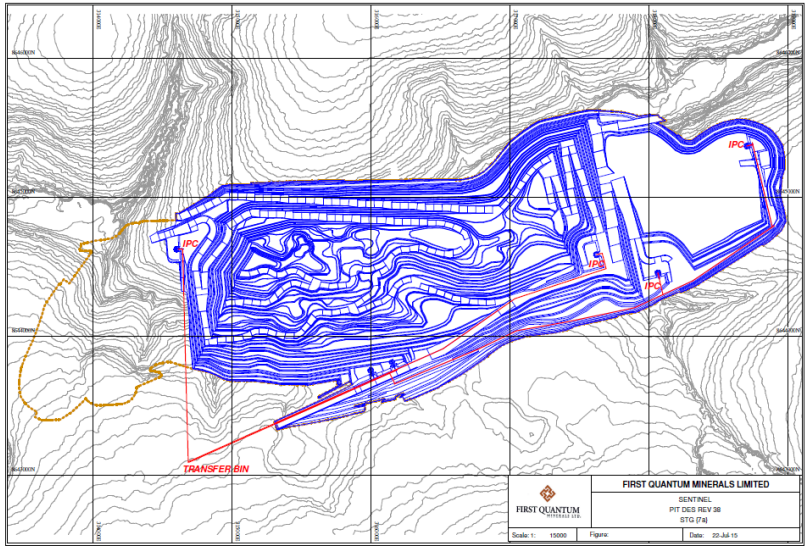


Figure 15-13 Sentinel Pit, Stage 7b design

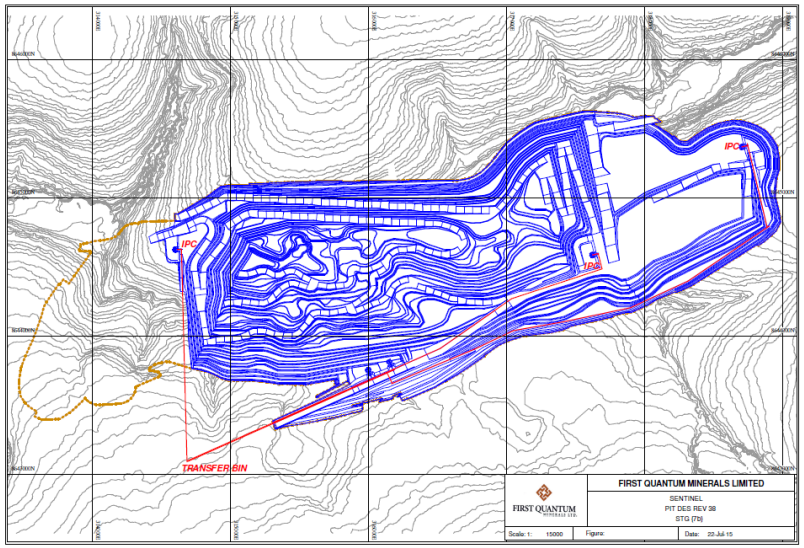


Figure 15-14 Sentinel Pit, Stage 7c design

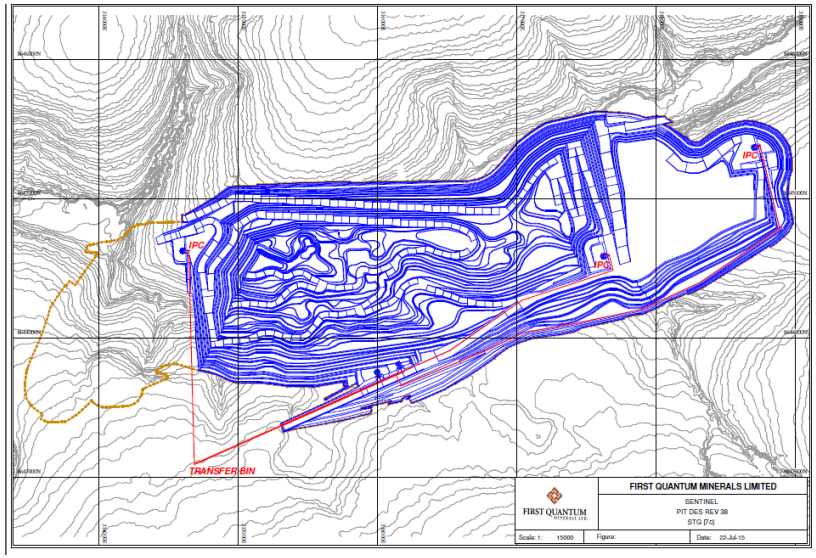


Figure 15-15 Sentinel Pit, Stage 7d design

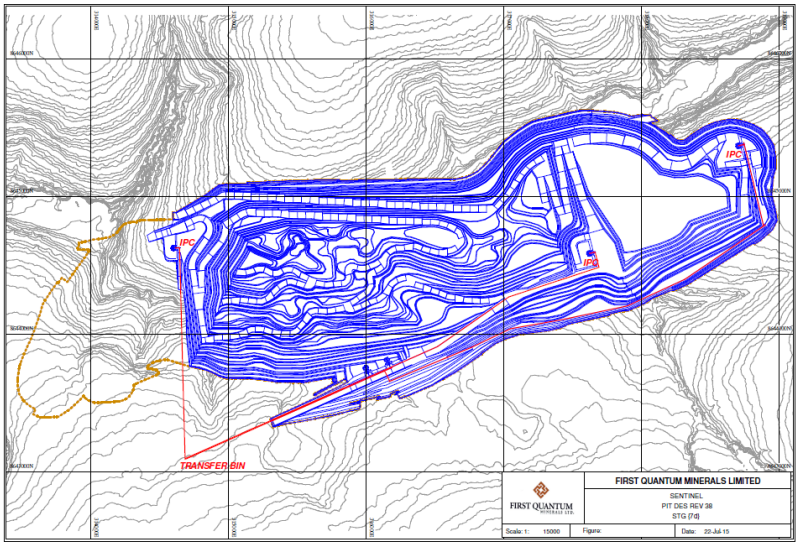


Figure 15-16 Sentinel Pit, Stage 8a design

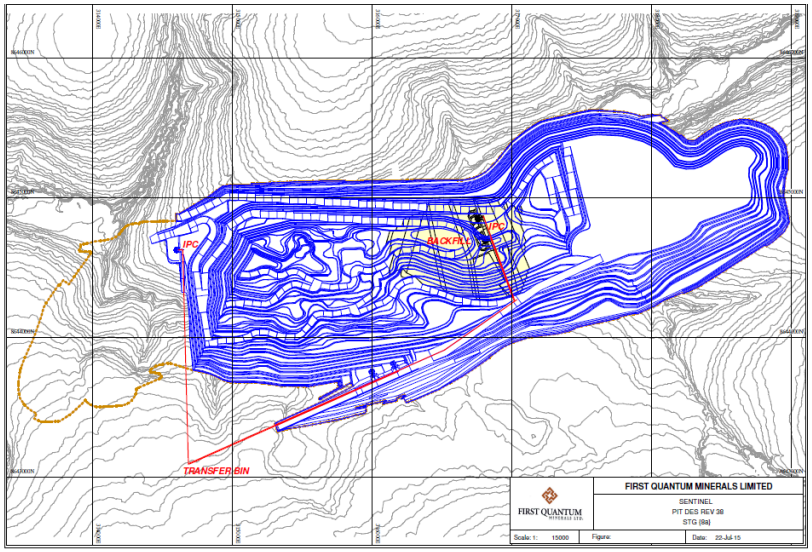


Figure 15-17 Sentinel Pit, Stage 8b design

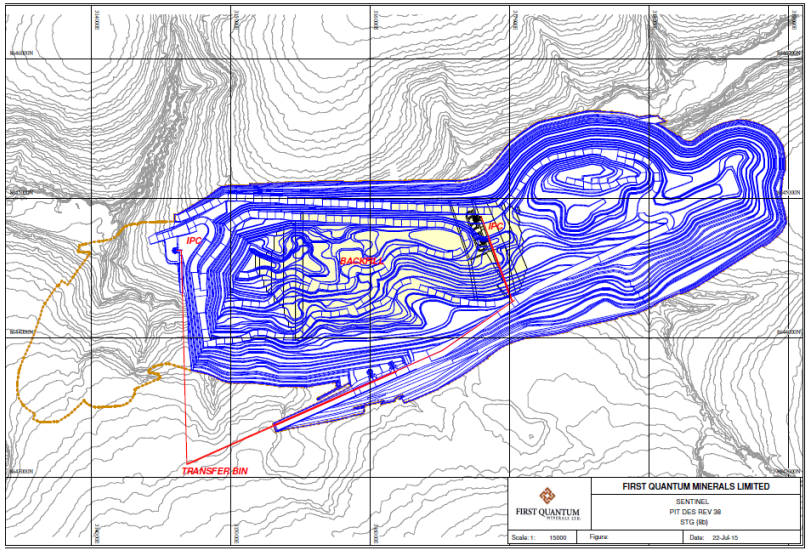


Figure 15-18 Sentinel Pit, Stage 9a design

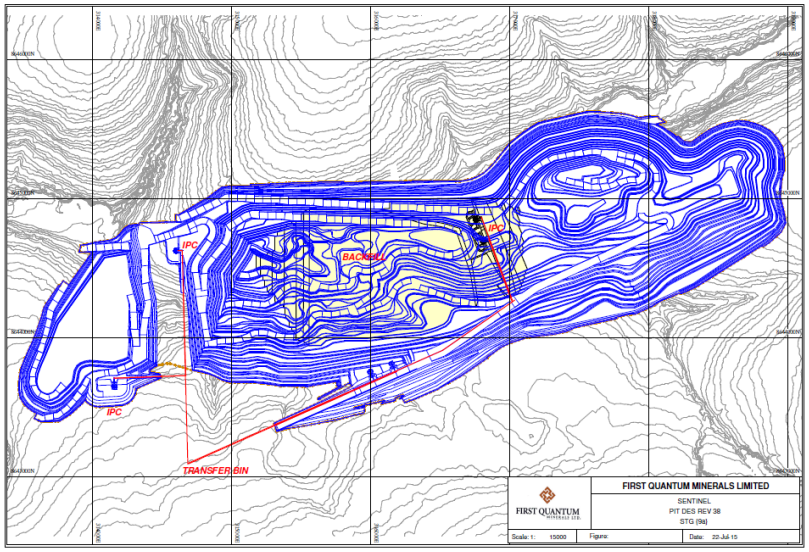


Figure 15-19 Sentinel Pit, Stage 9b design

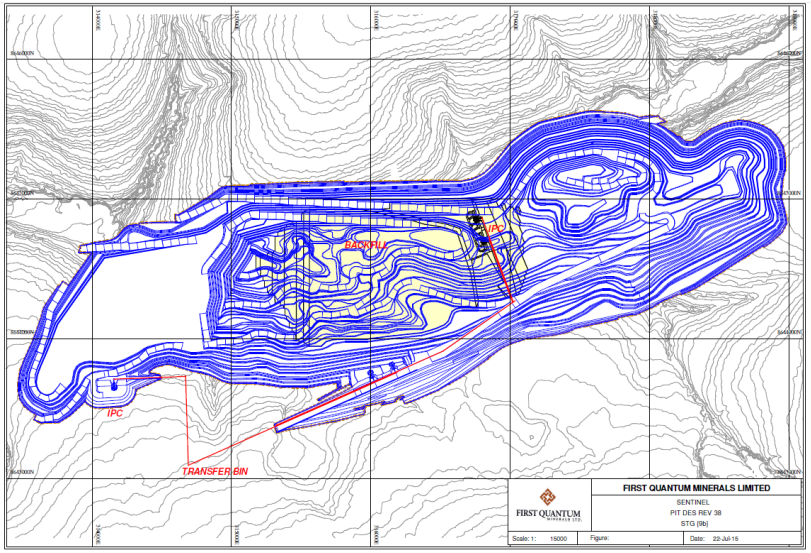


Figure 15-20 Sentinel Pit, ultimate design

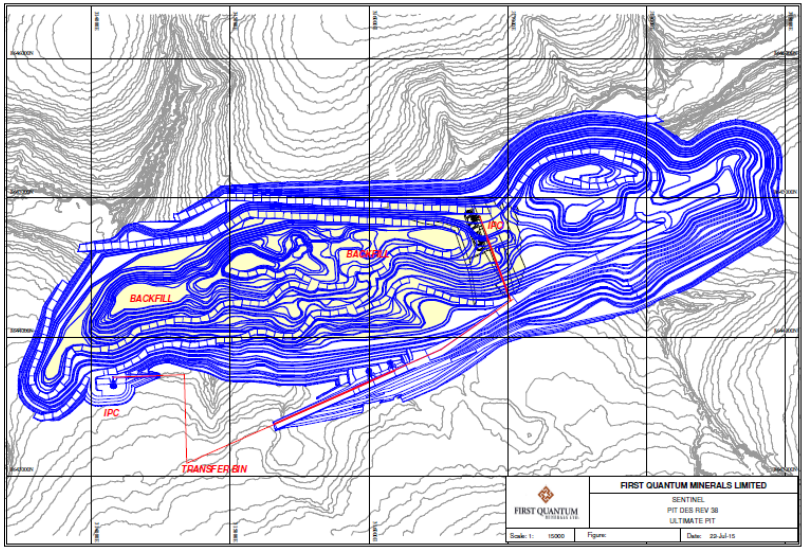


Table 15-7 Comparison between Sentinel design ultimate and optimal pit, depleted to May 2015

Pit Design Ore & Waste Summary					
		Mbcm	Mt	%Cu	Cu kt
Ore		311.1	868.7	0.56	4,864.5
Waste		830.5	2,250.5		
	Total	**1,141.6**	**3,119.2**		
Optimisation (shell 15)					
		Mbcm	Mt	%Cu	Cu kt
Ore		309.7	870.5	0.56	4,857.5
Waste		793.9	2,189.5		
	Total	**1,103.6**	**3,060.0**		
Comparison (variance): Design vs Optimisation					
		Mbcm	Mt	%Cu	Cu kt
Ore		100%	100%	100%	100%
Waste		105%	103%		
	Total	**103%**	**102%**		

15.2 Enterprise

15.2.1 Introduction

As part of the Mineral Reserve estimation process for Enterprise, pit optimisation aspects and detailed pit designs were completed by FQM personnel overseen and supervised by Michael Lawlor (QP). All operating cost, recovery and revenue information for the optimisations, in addition to operational parameters for the open pit designs, were reviewed by Michael Lawlor (QP).

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

15.2.2 Methodology

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

The ultimate (optimal) and lesser shells were used to create practical and detailed open pit designs accounting for batters, berms and haul roads.

15.2.3 Mine planning model

A regularised block model to 10 m x 10 m x 5 m size was used for pit optimisation and mine planning work. This model did not include any specific allowance for planned mining dilution.

15.2.4 Pit optimisation

Optimisation methodology

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 a fixed process recovery value. This differs from the approach taken in the 2012 Technical Report (DumpSolver, December 2012), in which the optimal pit shell was selected on the basis of maximum NPV (from an optimisation and high-level Miliwa schedule).

In the current approach, a maximum cashflow basis was considered to be more consistent with a development and potential production timeframe strategically integrated with that from Sentinel.

Pit slope design criteria

Pit optimisation input included an overall slope design angle of 40^{o}. Whilst this angle is consistent with that adopted for the 2012 Technical Report (DumpSolver, December 2012), it is also consistent with a general overall angle corresponding to the specific geotechnical criteria outlined in Item 16.2.2, and accommodating in-pit haulroads.

Metal prices

The nickel metal price adopted for pit optimisation was US$7.50/lb (US$16,535/t).

Metal recoveries

Metal recovery figures adopted for pit optimisation reflect the metallurgical recommendations in Item 13.2.5, 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) = 90%

Operating costs

Unlike Sentinel, Enterprise is considered not to be mill constrained, hence the process operating costs should include only the variable costs and not the fixed costs. The nickel circuit at Trident can be utilised as required, with available capacity transferred to the processing of Sentinel copper ores.

In relation to variable processing plant costs, the same base costs have been adopted as those reported in the 2012 Technical Report (DumpSolver, December 2012). The fixed operating costs included in the 2012 Technical Report ($1.01/t milled) have been deleted however, and replaced with an additional variable cost covering the reclaim and haulage of ore from the Enterprise ROM pad to the Trident plant. The updated operating costs are as follows:

- variable processing costs = $7.79/t processed
- reclaim and haulage costs = $1.50/t processed
- total operating costs = $9.29/t processed

Details of these cost estimates and their derivation are outlined in Item 21.

Mining costs

Variable mining costs comprising drill, blast, 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 2012 Technical Report (DumpSolver, 2012). Item 21 provides more information on the incremental relationship adopted for ore and waste mining costs in the pit optimisation process. The overall average mining costs are as follows:

- average waste mining cost = $6.77/bcm (equivalent to ~$2.68/t)
- average ore mining cost = $7.55/bcm (equivalent to ~$2.60/t)
- average waste+ore mining cost = $6.86/bcm (equivalent to ~$2.67/t)

Metal costs

In addition to 9% (gross) royalties (ie the now prevailing rate), metal costs for the Enterprise product comprise concentrate transport charges, treatment and refining charges, plus smelter deductions. Apart from the increased royalty rate, the same costs were adopted as those reported in the 2012 Technical Report (DumpSolver, December 2012), ie:

- concentrate transport charge = $0.25/lb
- concentrate treatment charge = $0.44/lb
- concentrate refining charge = $0.97/lb
- price participation charge = $0.24/lb
- smelter deduction value = $0.60/lb
- 9% royalty = $0.675/lb
- total nickel metal cost = $3.17/lb

Details of these cost estimates and their derivation are outlined in Item 21.

Mining dilution and recovery factors

Pit optimisation assumptions included unplanned mining dilution and mining recovery factors of 1.05 and 0.95, respectively. This is consistent with the assumptions in the 2012 Technical Report (DumpSolver, December 2012). The dilution is assumed to be at zero grade.

Marginal cut-off grades

Based on the above parameters, the calculated marginal cut-off grade is as listed in Table 15-8.

Table 15-8 Enterprise marginal cut-off grades

Marginal Cut-Off Grade:	Units	Non-primary	Primary
PROCOST	$/t ore	9.29	9.29
MINDIL factor		1.05	1.05
METAL PRICE	$/lb Ni	7.50	7.50
	$/t Ni	16,535	16,535
METAL COST	$/lb Ni	3.18	3.18
	$/t Ni	7,000	7,000
TOTAL NET RETURN	$/lb Ni	4.33	4.33
	$/t Ni	9,535	9,535
	$/10kg Ni	95.35	95.35
RECOVERY	%	60	90
C/O GRADE	**% Ni**	**0.17**	**0.11**

Optimisation results

Figure 15-21 shows the graphical result of pit optimisation. The optimal shell was selected on a maximum undiscounted cashflow basis. Table 15-9 lists the complete inventory of shell sizes and corresponding cashflows.

Figure 15-21 Enterprise optimisation results

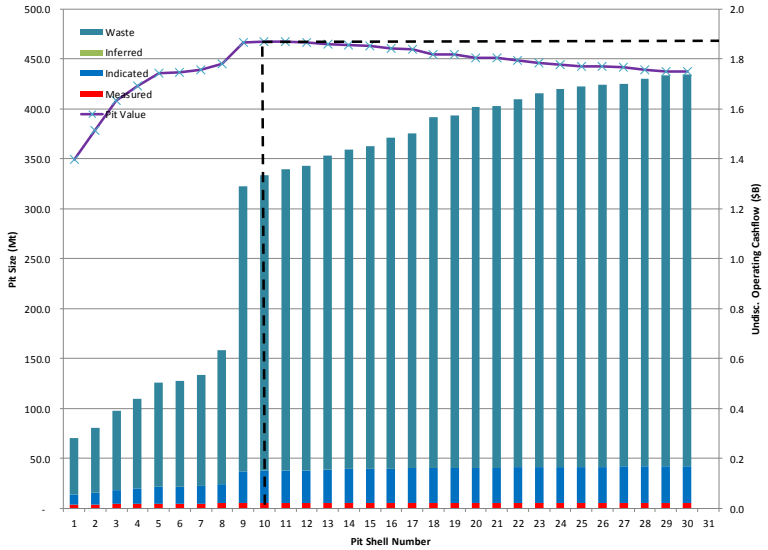


Table 15-9 Enterprise pit optimisation shell sizes and cashflow

Shell	Revenue Factor	Pit Size (Mt)	Ore Processed (Mt)	Ni Feed (%)	Ni Recovered (kt)	Undiscounted Cashflow ($B)
1	0.55	70.6	13.7	1.50	180.4	1.4
2	0.60	80.5	15.4	1.45	197.1	1.5
3	0.65	97.8	17.9	1.37	216.9	1.6
4	0.70	109.8	19.5	1.33	228.0	1.7
5	0.75	125.8	21.4	1.27	239.6	1.7
6	0.80	127.7	21.6	1.26	240.9	1.7
7	0.85	133.2	22.1	1.25	243.8	1.8
8	0.90	158.0	24.4	1.19	255.8	1.8
9	0.95	322.8	37.0	0.99	323.8	1.9
10	1.00	333.7	37.7	0.98	327.7	1.9
11	1.05	339.2	38.1	0.98	329.5	1.9
12	1.10	342.5	38.3	0.98	330.5	1.9
13	1.15	353.0	38.8	0.97	333.3	1.9
14	1.20	359.1	39.2	0.97	334.9	1.9
15	1.25	362.8	39.4	0.96	335.8	1.9
16	1.30	371.2	39.8	0.96	337.6	1.8
17	1.35	375.6	40.1	0.96	338.6	1.8
18	1.40	391.5	40.5	0.95	341.6	1.8
19	1.45	393.0	40.6	0.95	341.8	1.8
20	1.50	401.7	40.8	0.95	343.3	1.8
21	1.55	402.3	40.8	0.95	343.4	1.8
22	1.60	409.3	41.2	0.95	344.4	1.8
23	1.65	415.1	41.4	0.95	345.3	1.8
24	1.70	419.7	41.6	0.94	345.9	1.8
25	1.75	422.1	41.6	0.94	346.2	1.8
26	1.80	423.7	41.7	0.94	346.4	1.8
27	1.85	424.8	41.7	0.94	346.6	1.8
28	1.90	430.0	41.9	0.94	347.2	1.8
29	1.95	433.5	42.0	0.94	347.6	1.7
30	2.00	433.9	42.0	0.94	347.6	1.7

Pit optimisation sensitivity analyses

A number of optimisation sensitivity analyses were carried out on the marginal cut-off grade case (base case). Sensitivity analyses at +/- 10% were completed to test the impact of varying copper price, royalties, mining costs, processing costs and metal costs. The results are listed in Table 15-4 and indicate that (note: the results are shown for mining inventories without the application of dilution and recovery factors):

- the most sensitive variable impacting on both the size of the optimal pit and the cashflow is metal price (note: that varying the processing recovery would have the same level of impact)
- reducing the royalty from 9% to 6%, ie to the level prevailing at the time of the 2012 Technical Report (DumpSolver, December 2012), results in an approximate 9% increase in undiscounted project value
- whilst the project value is increased, the impact on the marginal cut-off grade is minimal and the size of the optimal pits are essentially similar
- +/- 10% variations in the mining, operating and metal costs are less sensitive

Table 15-10 Enterprise pit optimisation sensitivity analyses

Ultimate Pits		at 9% royalty	at 6% royalty	at $6.75/lb	at $8.25/lb	+10% mining	-10% mining	+10% opex	-10% opex	+10% metal	-10% metal
Plant Feed	Mt	37.7	38.5	22.6	41.2	37.3	38.0	35.7	39.5	35.8	38.9
Feed Grade	%Ni	0.98	0.97	1.27	0.92	0.99	0.98	1.03	0.94	1.02	0.96
Cu Metal Insitu	kt	370.7	371.9	288.3	377.0	367.8	372.6	367.6	372.8	365.9	372.4
Waste	Mt	296.0	296.0	135.3	301.3	288.8	301.1	295.7	294.5	290.0	295.8
Total Mined	Mt	333.7	334.5	158.0	342.5	326.1	339.1	331.4	334.0	325.8	334.7
Strip Ratio		7.8	7.7	6.0	7.3	7.7	7.9	8.3	7.5	8.1	7.6
Total tonnage variance		Base	0%	-53%	3%	-2%	2%	-1%	0%	-2%	0%
Recovered Ni	kt	327.7	328.8	253.8	333.2	325.1	329.4	325.0	329.5	323.5	329.1
Undisc. Cashflow	**M$**	**1,868.5**	**2,035.0**	**1,359.0**	**2,414.6**	**1,780.9**	**1,957.9**	**1,833.3**	**1,905.5**	**1,640.6**	**2,098.8**
Undisc cashflow variance		Base	9%	-27%	29%	-5%	5%	-2%	2%	-12%	12%

15.2.5 Detailed pit designs

A series of phased pit designs were developed using the ultimate and intermediate pit shell results.

Design and planning parameters

General design parameters for haul road layouts were as follows:

- Haul road width with = 25 m
- Maximum haul ramp gradient = 1: 10 = approximately 6^{o}

Item 16.2.2 describes the geotechnical engineering that was completed in relation to the pit slope design parameters adopted for mine planning, as set out in Table 15-11.

Table 15-11 Enterprise pit slope design parameters

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^o to 090^o) 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^o to 120^o) All others	20 20	70 75	10 10	49.2 52.5
Siltstone	> 50	NE wall (0^o to 090^o) All others	20 20	70 65	10 10	49.2 46.0

Staged and ultimate pit designs

Figure 15-22 shows the starting point for the design (and production scheduling) process. This figure shows the extent of development mining as at the end of May 2015. Pre-stripped waste has been mined to a depth of 25 m to 30 m below surface to provide suitable construction material for the building of surface water diversion dams. This will assist with the reduction of future strip ratios.

Figure 15-23 to Figure 15-27 show the progression of mining stages throughout the life of mine. These views do not correspond to points in time, as several stages could be in progress at one time.

Validation between pit shell and design pit

Table 15-12 shows a comparison between the selected shell 10 optimal pit inventory and the inventory for the corresponding detailed ultimate pit design. The comparison shows an acceptable overall variance.

Figure 15-22 Enterprise at June 2015

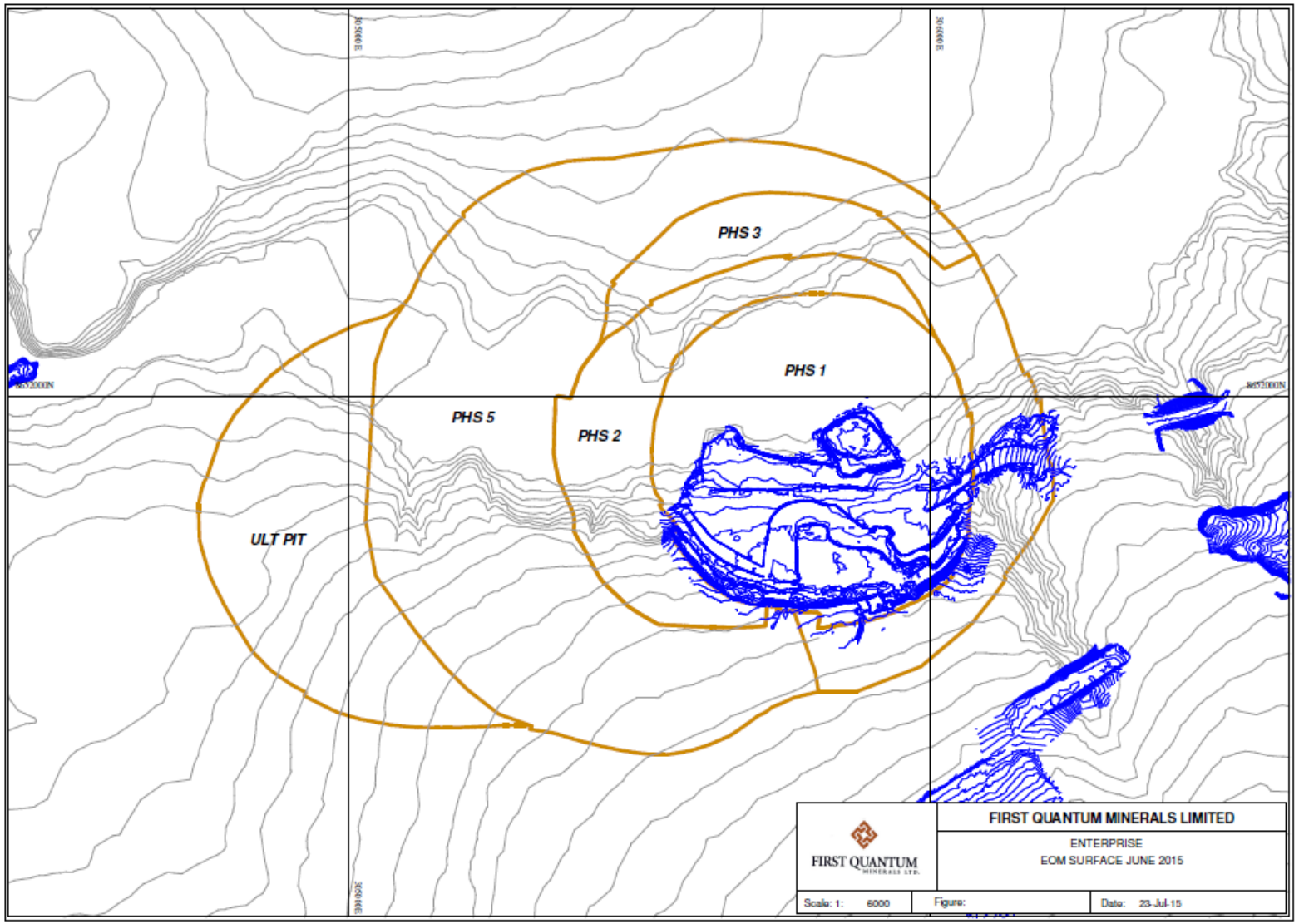


Figure 15-23 Enterprise Pit, Stage 1 design

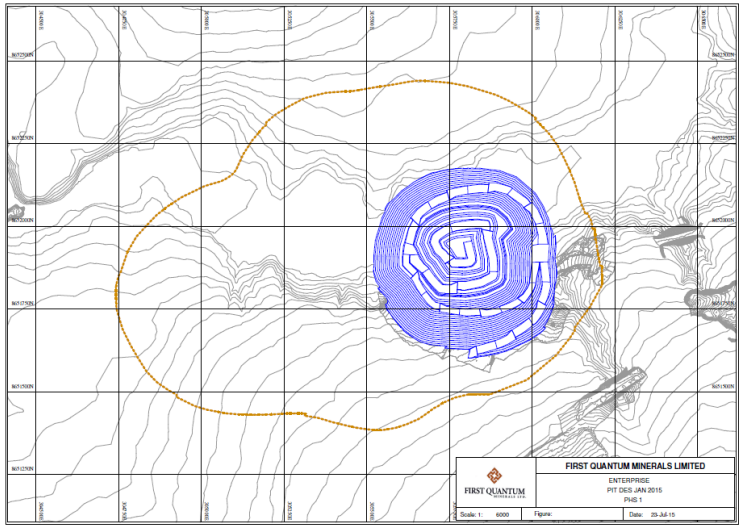


Figure 15-24 Enterprise Pit, Stage 2 design

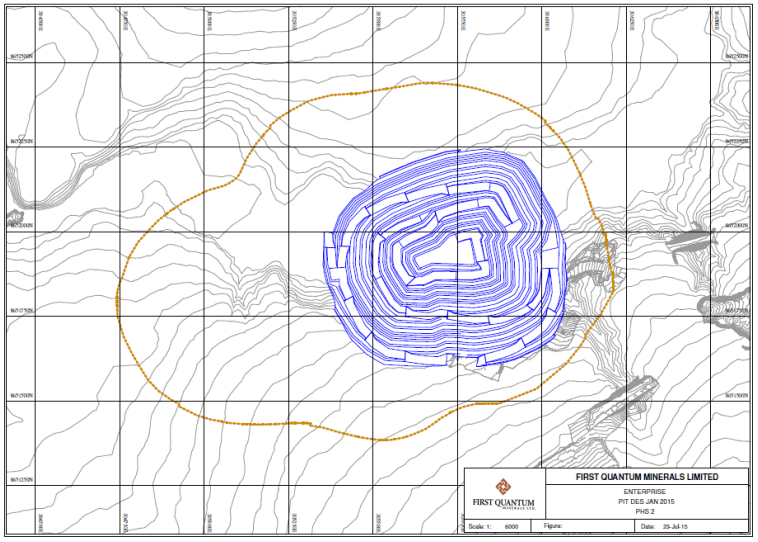


Figure 15-25 Enterprise Pit, Stage 3 design

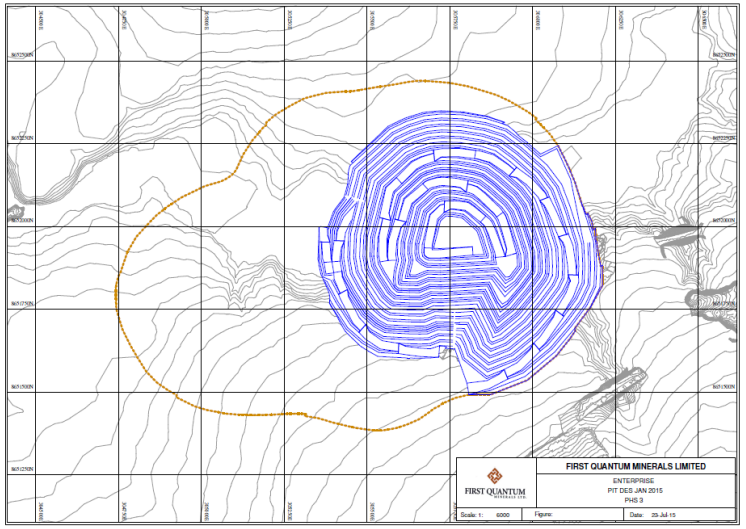


Figure 15-26 Enterprise Pit, Stage 4 design

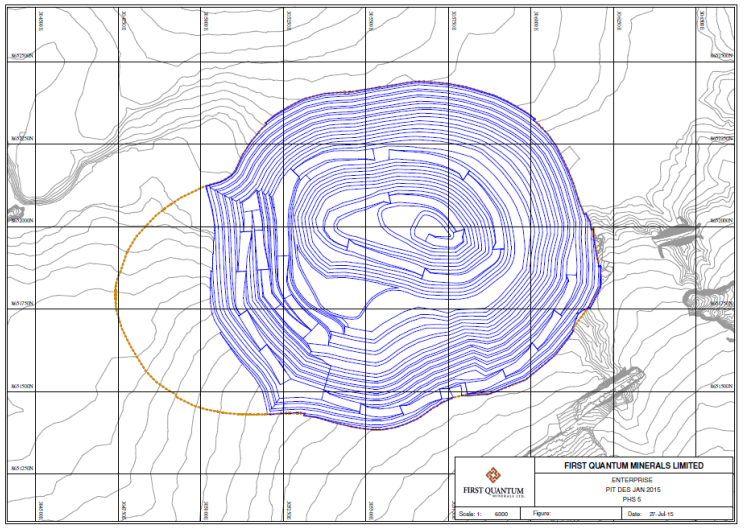


Figure 15-27 Enterprise Pit, ultimate design

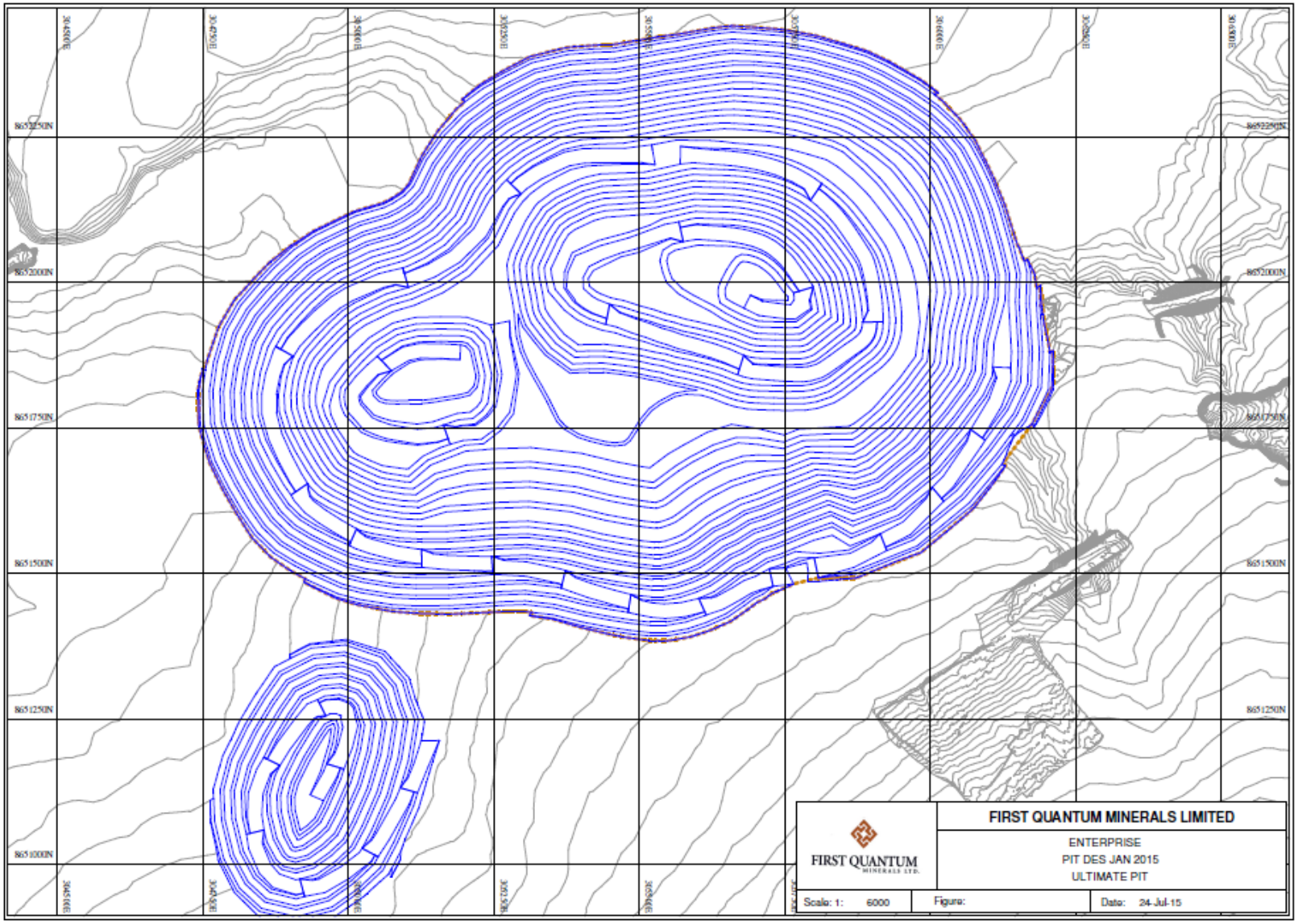


Table 15-12 Comparison between Enterprise ultimate design and optimal pit

Pit Design Ore & Waste Summary					
		Mbcm	**Mt**	**%Ni**	**Ni kt**
Ore		12.3	35.5	1.02	362.5
Waste		121.8	320.2		
	Total	**134.1**	**355.7**		
Optimisation (shell 10)					
		Mbcm	**Mt**	**%Ni**	**Ni kt**
Ore		13.1	37.8	0.98	372.0
Waste		112.6	296.0		
	Total	**125.7**	**333.9**		
Comparison (variance): Design vs Optimisation					
		Mbcm	**Mt**	**%Ni**	**Ni kt**
Ore		94%	94%	104%	97%
Waste		108%	108%		
	Total	**107%**	**107%**		

15.3 Mineral Reserve statement

15.3.1 Sentinel

The Proved and Probable Mineral Reserve for Sentinel, inclusive of stockpiles, is 985.8 Mt at an average grade of 0.51% TCu.

The Mineral Reserve within the designed pit is estimated as 985.3 million tonnes at an average grade of 0.51% TCu. 895.3 million tonnes of this Proved plus Probable ore is within the Measured and Indicated resource reporting cut-off grade listed in Table 14-4. An additional 90.5 million tonnes (almost entirely Proved Reserve) is outside of the Measured and Indicated resource reporting cut-off grade of 0.2% TCu. A breakdown by material type and classification is provided in Table 15-13.

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 staged pit designs and the mining production schedule described in Item 16.1.3.

Table 15-14 lists the additional Mineral Reserve held in surface stockpiles.

Table 15-13 Sentinel Mineral Reserves statement, in-pit inventory at May 2015

Mineral Reserve statement as at end of May 2015 at $3.00/lb Cu				
Classification	**Material**	**Tonnes (Mt)**	**TCu (%)**	**Cu metal (kt)**
Proved	Non-primary sulphide	80.9	0.44	357.4
Proved	Primary sulphide	738.3	0.51	3,772.6
	Proved 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**
	Proved and Probable subtotal	**985.3**	**0.51**	**4,985.0**

Table 15-14 Sentinel Mineral Reserves statement, stockpile inventory at May 2015

Mineral Reserve statement as at end of May 2015 at $3.00/lb Cu				
Classification	**Material**	**Mtonnes**	**TCu (%)**	**TCu metal (kt)**
Proved	Non-primary sulphide			
Proved	Primary sulphide			
Proved subtotal				
Probable	Non-primary sulphide	0.5	0.58	3.1
Probable	Primary sulphide	0.0	0.98	0.0
Probable subtotal		**0.5**	**0.58**	**3.1**
Proved and Probable subtotal		**0.5**	**0.58**	**3.1**

15.3.2 Enterprise

The Mineral Reserve within the designed pit is estimated as 35.4 million tonnes at an average grade of 0.97% Ni. 5.1 million tonnes of the Proved ore is within the Measured Mineral Resource reporting cut-off listed for Table 14-14. An additional 0.2 million tonnes (Proved Reserve) is outside of the Measured resource reporting cut-off grade of 0.15% Ni. A breakdown by material type and classification is provided in Table 15-15.

The reported Mineral Reserve is based on an economic cut-off grade which accounts for a longer-term nickel metal 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 15-15 Enterprise Mineral Reserves statement, in-pit inventory at May 2015

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**

ITEM 16 **MINING METHODS**

The following information for Sentinel and Enterprise is reproduced in part from the respective 2012 Technical Reports (DumpSolver, May 2012 and December 2012), with updates provided by Michael Lawlor (QP) and including contributions from KML staff.

16.1 **Sentinel**

16.1.1 **Mining details**

The Sentinel orebody dips towards the north and strikes approximately east-west. The footwall contact is defined by the mineralisation host rock (ie, phyllite), with a general dip trend of 20^{o} to 30^{o}.

The Sentinel Pit will be mined in a series of stages, 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) waste 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, respectively.

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 stage 1 and 2 limits, relative to future stages and the ultimate pit limit
- the extent of waste dumping to the north of the pit, relative to the conceptual ultimate dumping limit
- the existing ore stockpile site located outside the north western corner of the current pit
- the Musangezhi Dam, on the Musangezhi River approximately 4.5 km upstream from the current mining area
- an existing temporary diversion/containment dam (Dam 6) located between the pit and the Musangezhi Dam (and another future diversion/containment dam located further upstream)
- a number of pit water control dams located along the southern toe of the waste dump
- two settlement dams located to the west of the current mining area

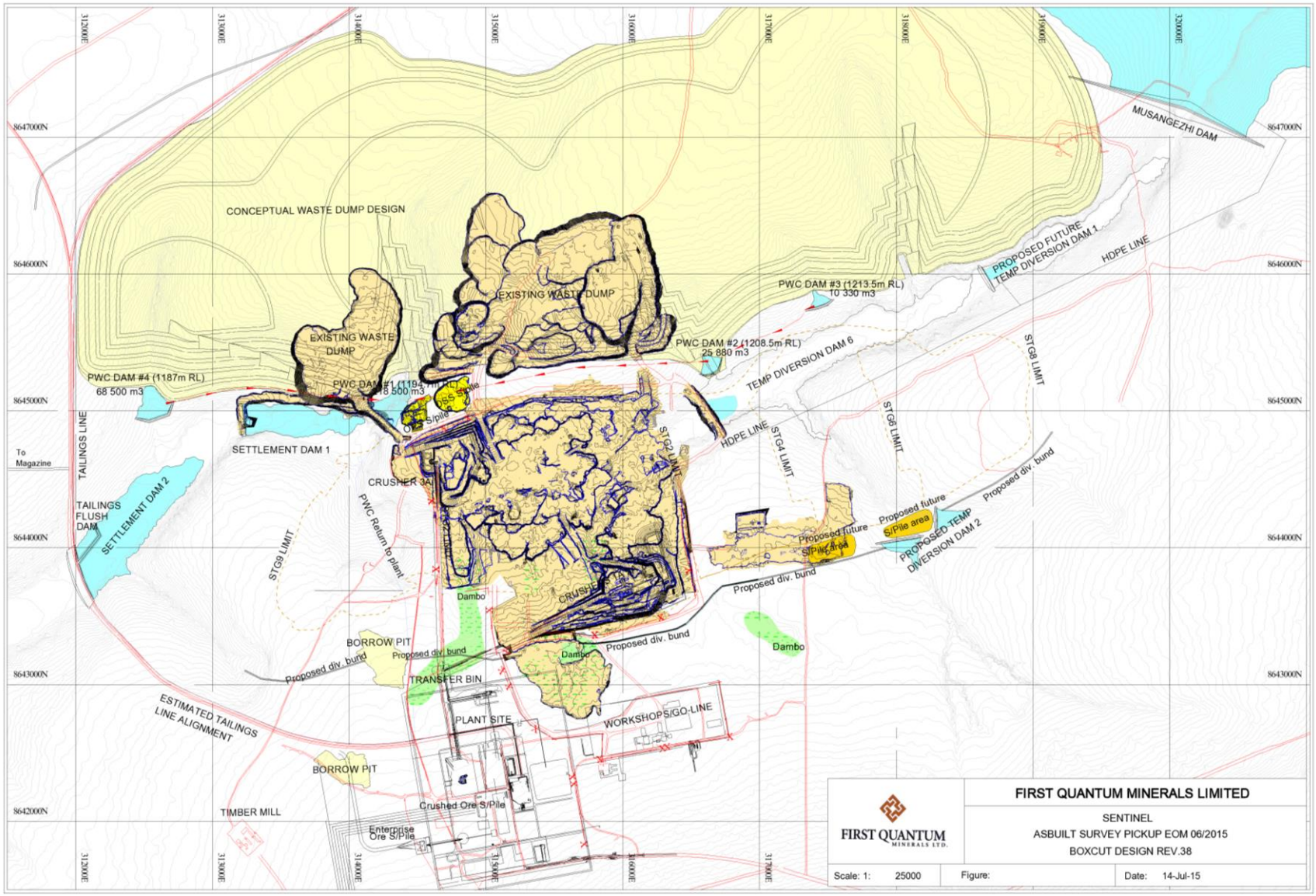
Until recently, an initial equipment assembly area was positioned within the stage 1 pit limits to enable the construction of track mounted drills and shovels, and also three in-pit crushers. This area has now been largely resumed for grade control and pit development.

Mining method and operations

Open pit mining at Sentinel will follow conventional drill and blast, shovel and truck mining practice. Mining to date has proceeded from initial boxcut excavations referred to as the north west boxcut and the south boxcut. Mining commenced in the north west boxcut in April 2013, whilst mining of the south boxcut commenced in 2014 and remains in progress. Two in-pit crusher pockets are intended for the south boxcut and excavation of these is currently in progress.

The areal extent between the two boxcuts has now been developed and opened out to the stage 1/2 limits (Figure 16-1).

Figure 16-1 Sentinel site layout plan, June 2015



The sequence of mining activities is conventional and will generally be as follows:

- RC grade control drill holes delineate the ore zones
- a grade control model is developed from which blast limits and digging blocks are designed
- ore and waste blocks are then usually blasted to design separately, according to designs based on varying hole patterns and powder factors to suit prevailing ground conditions
- 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 tonne and 240 tonne capacity haul trucks, for waste and ore, respectively
- ore is hauled direct to in-pit crushers or to stockpiles, whilst waste is hauled to the surface dump tip heads

The design height between pit berms is 24 m and blast blocks will be typically 12 m high. Mining flitches are generally 12 m high, reducing to 6 m high in certain areas as a means to minimise dilution.

The two broad ore types to be differentiated are:

- non-primary sulphide ore (OSS), and
- primary sulphide ore (OPS)

The two waste types to be differentiated are:

- potentially acid forming (PAF), and
- non-acid forming (NAF)

Further information on the management and separation of the waste dumping 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.

In most instances, the effective use of IPCC relies on the minimisation of long term stockpiles. The design and sequencing of the Sentinel mining stages, however, ensures that there is generally always one crusher located near-surface on the upper footwall. This enables some stockpiling/reclaim and short haul across to a nearby crusher.

In-pit crushing and conveying

The integration of an IPCC layout has required a “retro-fit” approach to pit design, picking up on the mining concepts and sequence developed for and described in the 2012 Technical Report (DumpSolver, May 2012). An internal report was produced in January 2013 (FQM, 2013) to evaluate a number of options for the initial siting of in-pit crushers. This evaluation took account of the following criteria:

- The process plant is located to the south of the open pit (ie, the footwall side).
- Permanent ore haul roads are to be located on the footwall side of the pit and are to be integrated with conveyor inclines, where possible.

- IPCC installations are to be established near-surface, initially, but must proceed to new positions deeper in the mine, at (notional) relocation intervals of up to five years.
- Where possible, new crusher positions must provide for ease of relocation and logical extension of conveyor belt infrastructure.
- The largest haul truck which could present to the crushers is a 330 t capacity truck which although intended primarily for waste hauls, could conceivably be used for ore hauls, on occasion.
- Conveyor belt transfer points between the pit and the plant are fixed, for design and costing purposes.
- Additional transfer points and haul road/belt crossovers are to be minimised.
- Avoid situations where there will be blasting immediately above or adjacent to crushers and conveyors. Enlarge the stage cutbacks and ensure separation.
- Where possible, design for long straight discharge conveyor routes.

Following geotechnical investigations and a number of mine design and schedule iterations, an arrangement for the first in-pit crusher pockets was devised as shown in Figure 16-2. The sequence of relocating these crushers to deeper positions into the advancing pit is shown in Figure 15-5 to Figure 15-20.

Trolley-assisted haulage

A fleet of 330 tonne capacity haul trucks have been delivered to site ready for trolley-assisted (TA) haulage. These trucks will use the same TA systems as developed by the Company at the Kansanshi Mine.

The staged pit designs presented in Item 15.1.5 show that long straight haulage routes have been incorporated into the north wall pit slope design. There are seven waste exits on the north wall crest, several of which relate to temporary north-south haul ramps between adjoining stages.

Waste dump optimisation and design

Consultants DumpSolver Pty Ltd produced an optimised life-of-mine waste dump design in February 2015 (DumpSolver, 2015). This was based on the mining production schedule described in Item 16.1.3, before accounting for current depletion (ie, the current Mineral Reserves production schedule, before depletion to end of May 2015).

The optimisation differentiated between PAF and NAF waste, according to the ARD management outcomes and criteria outlined in Item 16.1.2. It identified a final landform that could be constructed with minimal waste haulage costs, whilst encapsulating with NAF rock, all PAF rock placed onto the dump.

One of the benefits of the current Mineral Reserves production schedule, and by virtue of the scheduled sequence of pit stage development, is the ability to backfill the depleted pit in the latter half of the mine life. The DumpSolver design was able to take this into account and successfully balance on an annual basis, backfilling and external dumping of PAF, with the NAF applied to encapsulation and capping.

Figure 16-2 Sentinel Pit, initial mining boxcuts and IPC pockets

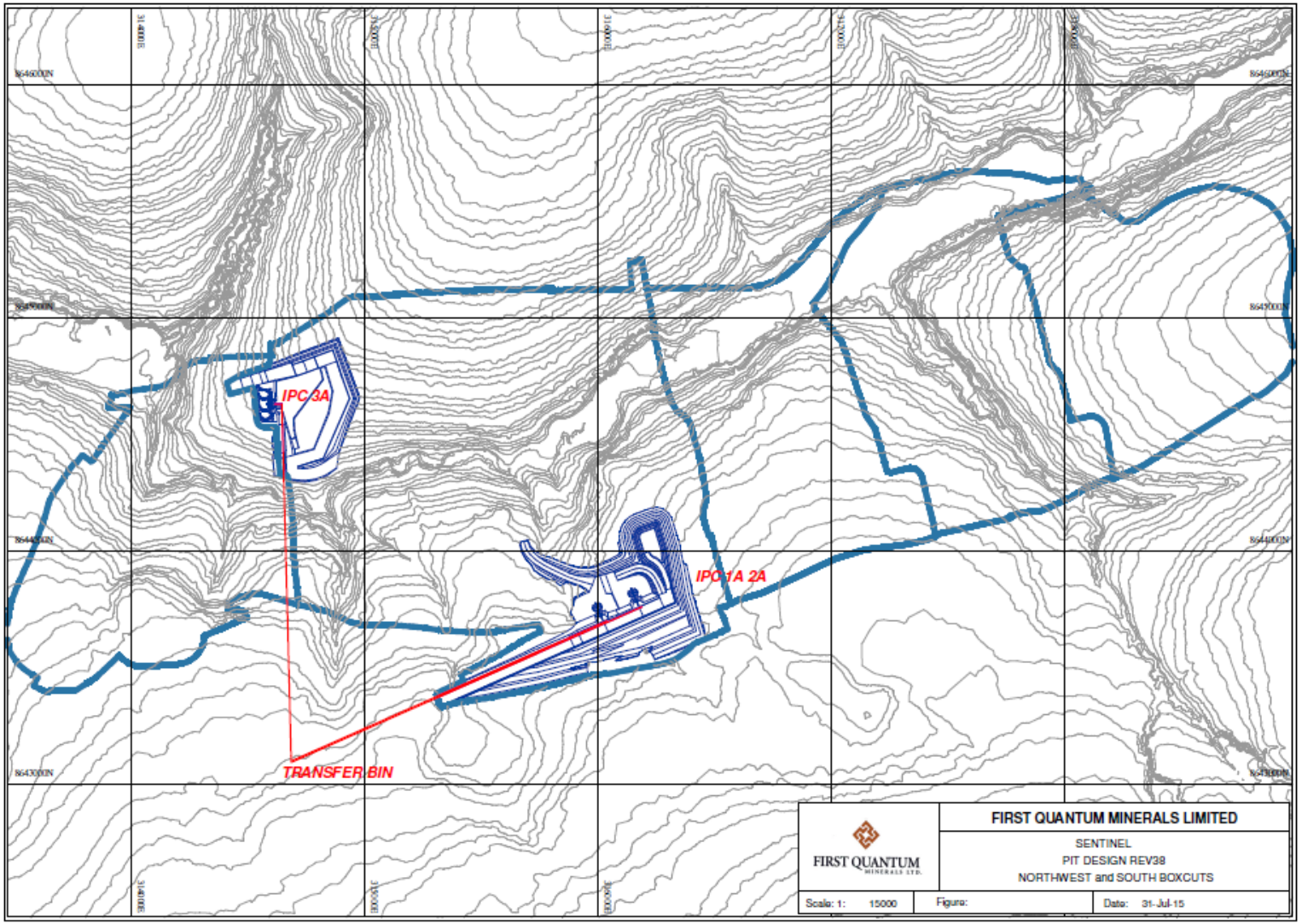
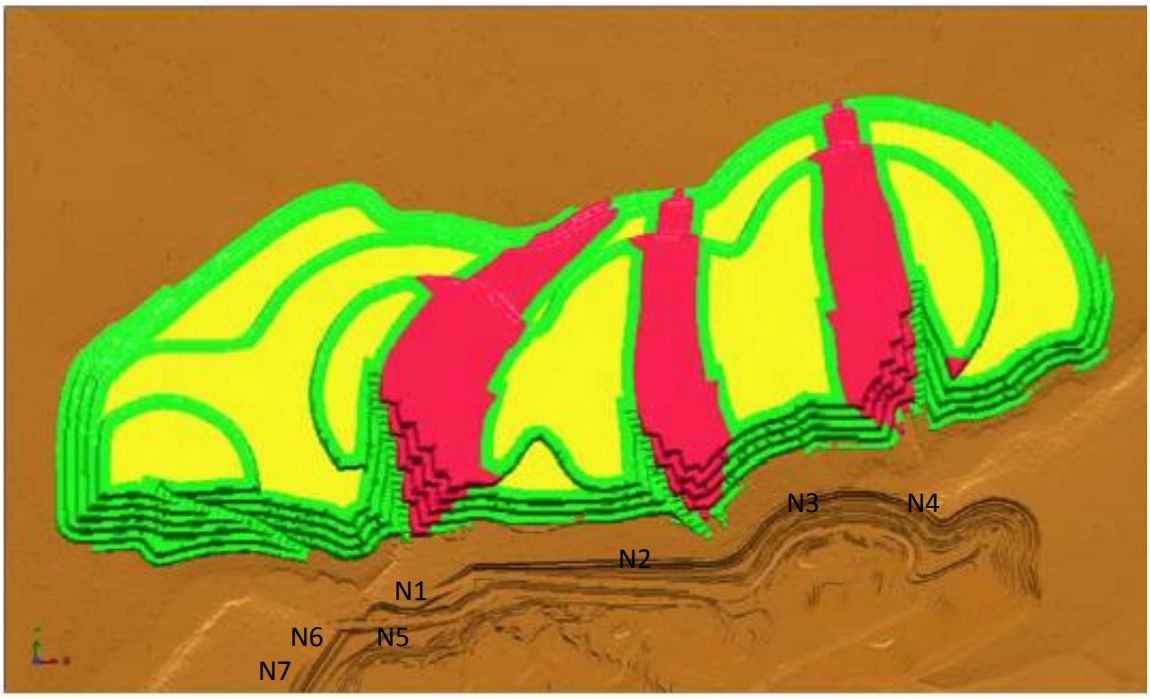


Figure 16-3 shows the completed Sentinel waste dump landform. Green blocks represent NAF encapsulation donuts, yellow blocks are NAF capping and red blocks are NAF fill within the apache passes. PAF material is obviously not visible under the encapsulation and capping. Labels show the pit haul road exit points leading to the dump.

All dump walls are designed with 20 m high benches, 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 is designed to be at least 250 m from the pit crest to conform to a back break assessment prepared by Xstract (2014).

Figure 16-3 Sentinel Pit, completed waste dump design (DumpSolver, 2015)



Drilling and blasting

Production drilling and blasting will be carried out by FQM personnel. Specialist production blasting advice has been provided by Orica Mining Services (Orica, 2014), in relation to:

- precision blasting of in-pit crusher pockets and surrounds
- optimum fragmentation for primary digging equipment and crushers
- optimum fragmentation for milling performance and energy consumption

Coupled with the development of blast designs for Sentinel, a trial of blast movement monitoring is currently in progress. This work is being carried out in conjunction with the development of grade control modelling, blast block design and production tracking/recording systems. The blast movement monitors provide information on blast heave and throw, information which is useful for the minimisation of mining dilution.

16.1.2 Mine planning considerations

The following information relates to the detail that needed to be considered for designing surface layouts and practical mining pits around the Sentinel optimal pit shell outline.

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 layouts to suit IPCC implementation:

- haul road minimum width = 3.5 x truck width = 32.5 m
- haul road width with dual conveyor, inclusive of bunds, conveyor access corridors and side drain = 55 m (refer to schematic in Figure 16-4)
- maximum gradient to allow crusher tramming = 1 : 10

The following parameters were adopted in designing the pit layouts to suit future trolley-assist haulage:

- haul road minimum width = 4.4 x truck width (to allow up-haulage off the catenary line) = 41 m
- total haul road width inclusive of catenary pole, bund and side drain = 50 m (refer to schematic in Figure 16-5)
- maximum gradient = 1 : 10

Further detailed parameters had to be considered for IPCC implementation. ThyssenKrupp engineering drawings provided the required excavation and installation dimensions for each IPC installation.

Figure 16-4 Schematic cross section across haul road including conveyor

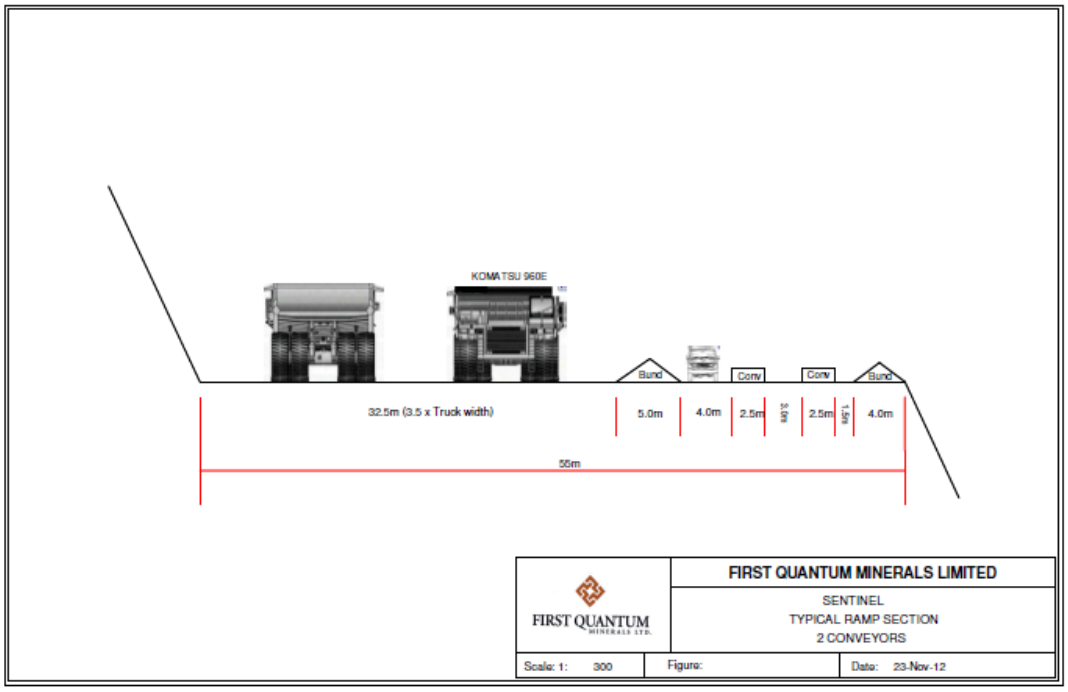
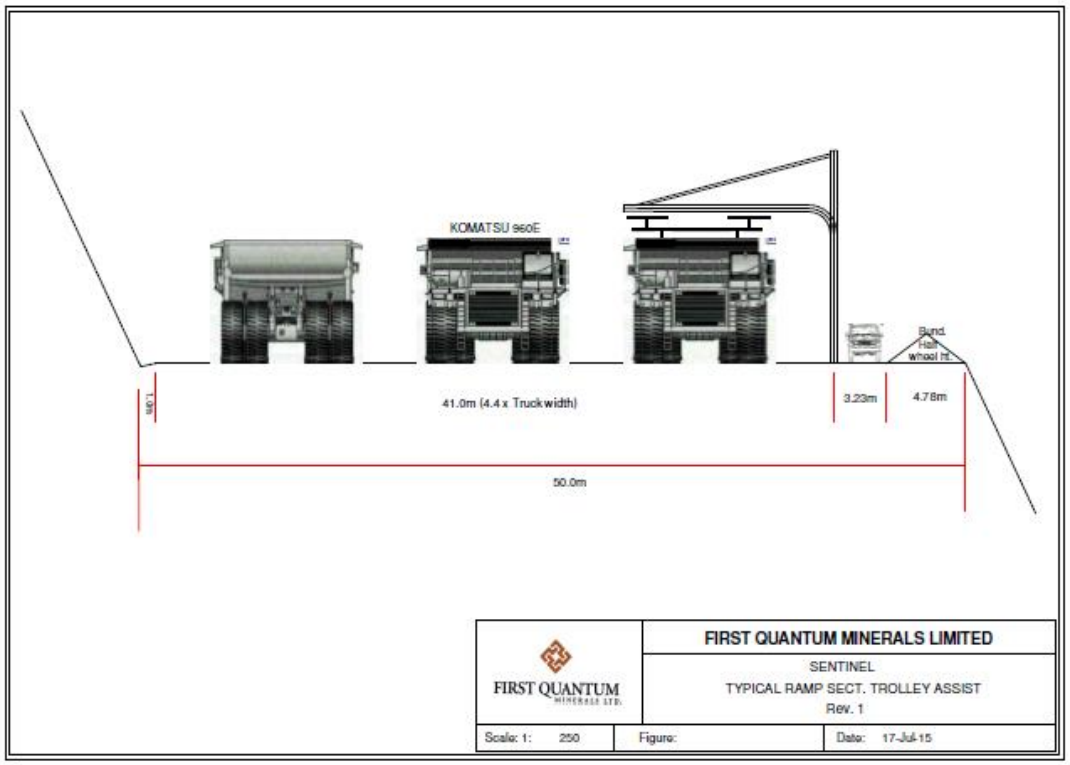


Figure 16-5 Schematic cross section across haul road for trolley-assist haulage



Geotechnical engineering

Xstract (2015) produced an open pit geotechnical design report for Sentinel as the culmination of a detailed drilling investigation and laboratory testing programme in 2013. 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 16-1. The information in Table 16-1 was made available in a preliminary memo report issued in November 2013 (Xstract, 2013).

Table 16-1 Pit slope design criteria

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° to 040°) All others	24 24	60 75	15 10	39.8 55.6
MPHB	> 48	South wall (330° to 040°) All others	24 24	60 75	15 10	39.8 55.6
MSC	> 48	All	24	75	10	55.6

Note: IRA (C to C) refers to the inter-ramp angle, or the slope angle from crest to crest, excluding berms

In addition to the above parameters, wide geotechnical berms were recommended part way down the overall slope, thereby reducing the overall slope angle to the angles listed in Table 15-1.

Hydrogeology

Since the 2012 Technical Report (DumpSolver, May 2012), hydrogeological investigations and pit dewatering activities have advanced as follows:

- the compilation of all available hydrogeological and hydrological information
- installation of a groundwater monitoring network comprising around fifty monitoring sites
- drilling and testing of ten pilot (or probe) holes and completion of these as nested piezometers between 2013 and 2014
- construction and testing of six monitoring/production bores around the north west and south boxcuts in 2014 (KALDW001 to KALDW006 in Figure 16-6)
- development of a preliminary hydrogeological model (ie, a characterisation of the local hydrogeological system) identifying the main hydro-stratigraphic units and ranges of hydraulic conductivity
- assessment of the potential pit water magnitudes arising from surface water inflows, direct precipitation and groundwater inflows
- development of an initial groundwater model to assist with an understanding of the pit water volumes to be abstracted in order to accommodate the proposed mining vertical advance rates; the findings of this model are preliminary and it requires on-going calibration as more monitoring data becomes available
- geophysical surveys, additional probe drilling and hydraulic tests in early 2015, followed by the drilling of five large diameter production bores by June 2015 (the locations of KALDW007 to KALDW011 are shown in Figure 16-6)

Constant discharge aquifer testing is currently in progress. Once equipped with suitable bore pumps, by an expected timeframe to the end of September 2015, the cumulative site-wide abstraction rate should be assessable.

Hydrology

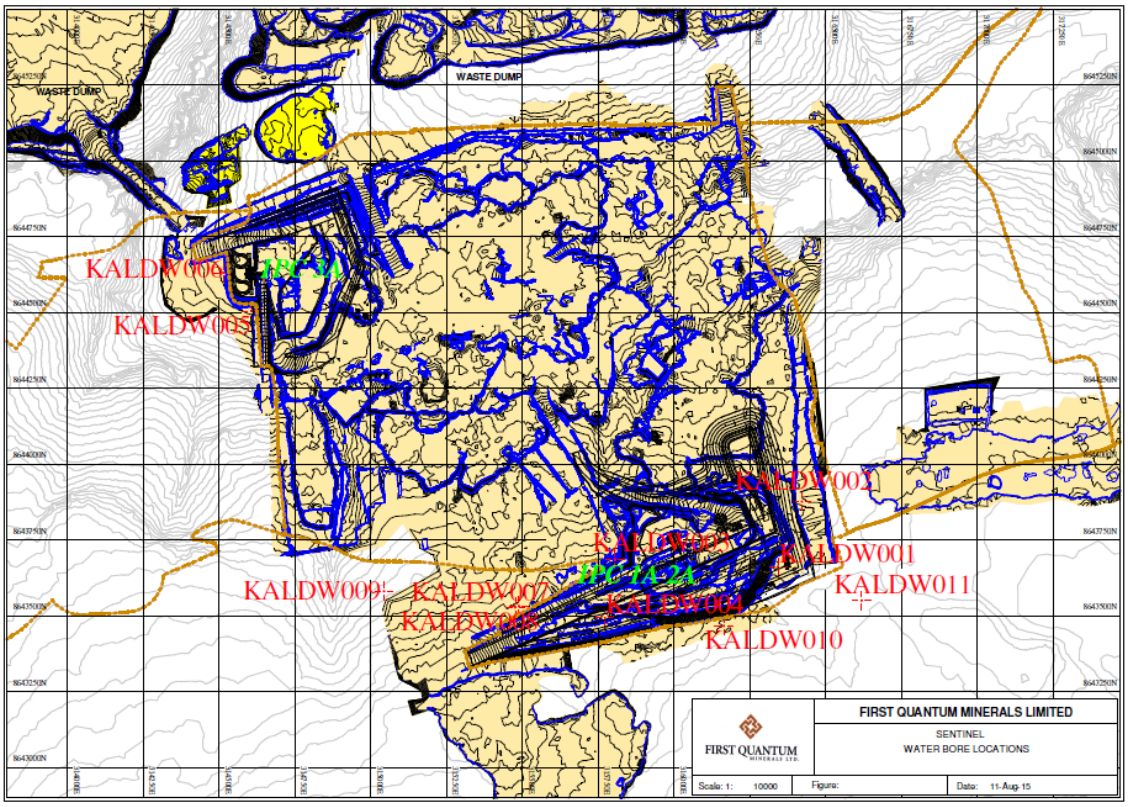
Surface water flow management has been addressed from three perspectives:

- The management of potential sediment laden run-off from the Sentinel site into the downstream Musangezhi River.
- The management of waste dump run-off and contact/"dirty water" from the mine.
- The management of upstream surface run-off providing a pit inundation and operational safety risk to the open pit.

Figure 16-1 shows the location of two settlement dams (Settlement Dams 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.

Figure 16-1 also shows the location of proposed diversion bunds to channel surface water flows from the southern heights and dambos across to the west and into a settlement dam. In respect of water control along the southern side of the waste dump, Figure 16-1 shows a number of pit water control (PWC) dams, the first constructed being PWC Dam 1 at the north west corner of the pit.

Figure 16-6 Location of groundwater bores



During operations, water stored in the PWC dams will be monitored for water quality before being released into the settlement dam or transferred to a process plant water pond. The central dam for monitoring, release or return to the process plant will be PWC Dam 1. The other three PWC dams will incorporate pumping back to PWC Dam 1.

In respect of the third perspective, Figure 16-1 shows the location of several up-stream diversion dams (Dam 1, 2 and 6). Dam 6 is in existence and Dams 1 and 2 are proposed for when the mining stages progress to the east.

ARD management

A programme of static and kinetic testwork to confirm the potential for ARD generation at Sentinel was initiated in 2013 and continues to date. The testwork has established that pyrite and pyrrhotite exist throughout the phyllite/hangingwall and 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. With regards to mine planning, the testwork 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 (ARD) at Sentinel an ARD Management Plan has been developed. The plan includes 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 transported to the waste dump.
- All surface runoff and seepage from the waste dumps will be collected in dams specifically designed for that purpose.
- All surface runoff from waste dumps and in-pit sumps will be recycled back to the plant.
- 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.

Mining dilution and recovery

A production recording system is being implemented at Sentinel, which in conjunction with the grade control modelling system and the Modular truck dispatch and monitoring system, will make it possible to track and record ore and waste movements (volume, tonnes and grade) to crushers/stockpiles, and to the waste dump, respectively. Whilst the recording system is still being implemented, the production reconciliation functions are expected to be in place later in 2015.

In view of the "planned dilution" incorporated into the Mineral Resource model (Item 14.5.1), a conservative additional allowance for "unplanned dilution" has been included in the production schedules. This is an additional 3% of ore tonnes at a grade of 0.15%Cu. A mining recovery factor of 97% has also been adopted.

16.1.3 Production details

LOM schedule

Features of the LOM mining and production schedule as listed in Table 16-2 are as follows:

- The production shown for 2015 represents that for the remaining seven months of the year from June 2015. At January 2016 the Project life is 18 years (ie, to 2033).
- The total material mined amounts to 3,119.1 Mt (1141.5 Mbcm), of which 985.3 Mt is ore (primary and non-primary ore) and 2,133.8 Mt is waste (PAF and NAF waste). This is on a mining diluted/recovered basis, assuming a mining dilution factor of 103% (at a grade of 0.15%Cu) and a mining recovery factor of 97%.
- The total ore mined to stockpiles and reclaimed throughout the Project life is 28.9 Mt at a grade of 0.37%Cu.
- The low grade (ie, between the marginal and 0.2% Cu cut-off) included in plant feed amounts to 90.5 Mt at a grade of 0.16%Cu. This material is mined and blended directly into the plant feed in the period from 2026 to 2033.

- Following the current ramp-up period during 2015, the plant production capacity of 55 Mtpa is generally maintained until 2032.
- The maximum size of long-term stockpiles is 7.2 Mt.

Table 16-2 Sentinel life of mine production schedule

Year	Mined Ore (Mt)	Grade (%Tcu)	Crusher Feed (Mt)	Grade (%TCu)	Waste (Mt)	Total (Mt)	(Mbcm)	Strip Ratio (t:t)
At end May 2015								
2015	27.0	0.54	27.0	0.54	71.8	98.8	38.7	2.7
2016	54.1	0.54	54.5	0.54	131.6	185.7	67.2	2.4
2017	49.8	0.63	49.8	0.63	123.6	173.4	65.4	2.5
2018	54.6	0.58	54.6	0.58	126.2	180.8	65.4	2.3
2019	60.6	0.55	55.0	0.57	121.6	182.2	65.4	2.0
2020	49.9	0.64	55.2	0.62	119.3	169.2	65.6	2.4
2021	56.9	0.52	55.0	0.53	124.6	181.5	65.4	2.2
2022	52.2	0.56	54.4	0.55	128.6	180.8	65.4	2.5
2023	54.0	0.55	54.0	0.55	123.6	177.6	65.4	2.3
2024	59.9	0.57	55.2	0.59	122.8	182.7	65.6	2.1
2025	55.7	0.46	55.0	0.46	117.3	173.0	65.4	2.1
2026	55.1	0.42	55.0	0.42	117.8	172.9	64.9	2.1
2027	50.7	0.46	55.0	0.45	129.2	179.9	65.1	2.5
2028	50.2	0.40	51.5	0.40	131.6	181.8	65.4	2.6
2029	55.4	0.37	55.0	0.37	126.9	182.2	65.4	2.3
2030	57.7	0.41	55.0	0.42	124.6	182.3	65.4	2.2
2031	57.1	0.40	55.0	0.41	85.5	142.6	51.2	1.5
2032	52.7	0.48	50.6	0.49	88.0	140.7	50.5	1.7
2033	31.9	0.58	39.1	0.53	19.2	51.0	18.3	0.6
2034								
LOM (dil'd)	**985.3**	**0.51**	**985.8**	**0.51**	**2,133.8**	**3,119.1**	**1,141.5**	**2.2**

Mining sequence

Figure 16-7 shows the sequence of mining through the pit design stages shown in Item 15.

Waste dumping schedule

Figure 16-8 shows the scheduled waste dump volumes (in Mlcm) by mining stage source, whereas Table 16-3 lists the volumes of scheduled waste (in Mlcm) leaving the pit via the pit exit points shown in Figure 16-3. Aspects of the waste dumping schedule include:

- 384.4 Mlcm of NAF is hauled to the waste dump over the life of mine
- 422.6 Mlcm of PAF is hauled to the waste dump over the life of mine
- 211.7 Mlcm of PAF is backfilled to the depleted pit from 2026

Figure 16-7 Sentinel life of mine mining sequence

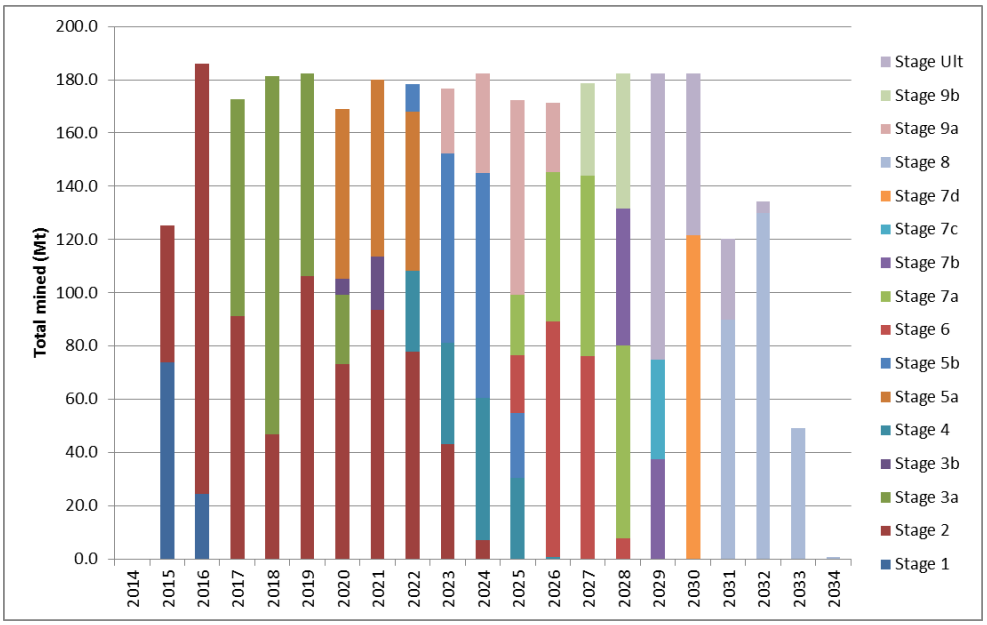
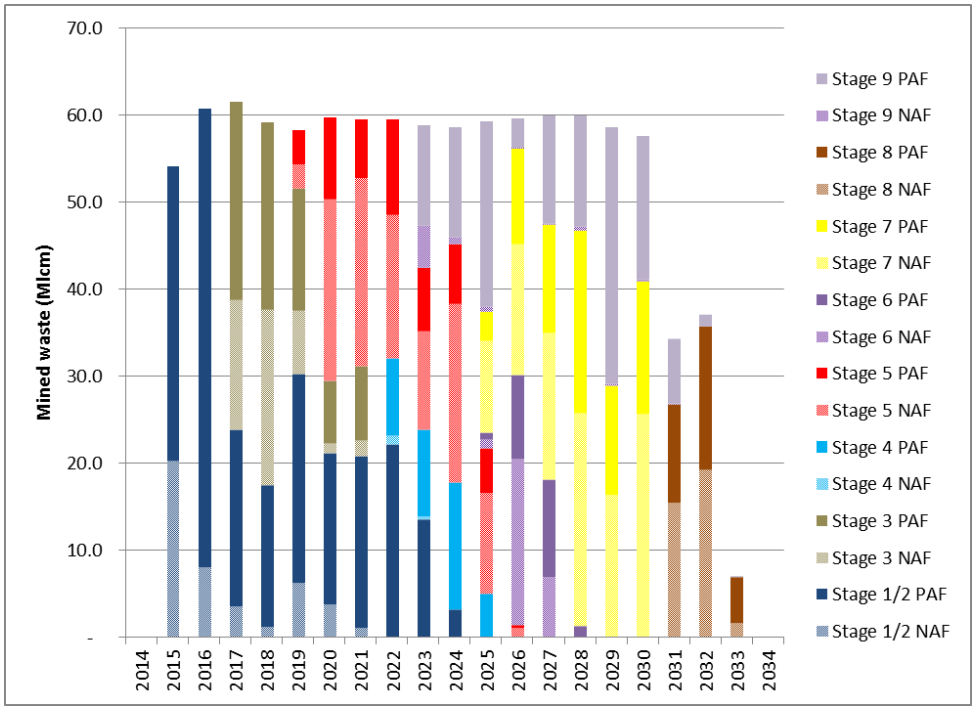


Figure 16-8 Sentinel scheduled waste dumping volumes



16.1.4 Primary mining equipment

The Sentinel mining fleet comprises newly purchased primary equipment as listed in Table 16-4. In addition to the listed primary equipment, ancillary equipment includes dozers, graders, water trucks, and fuel/service trucks.

Table 16-3 Sentinel waste haulage volumes

Pit Exit Point		N1	N1,N2	N3	N4	N1, N4	N5	N1, N5	N7	N6,N7	Backfill
STAGE 1/2 NAF	Mlcm	11.0	28.3					4.8			
STAGE 1/2 PAF	Mlcm	60.5	86.6				39.0	37.1			
STAGE 3 NAF	Mlcm	1.9		42.5		1.2					
STAGE 3 PAF	Mlcm	8.5		58.4		7.1					
STAGE 4 NAF	Mlcm	1.0						0.4			
STAGE 4 PAF	Mlcm	8.9						29.5			
STAGE 5 NAF	Mlcm	1.1			105.5						
STAGE 5 PAF	Mlcm				50.5						0.4
STAGE 6 NAF	Mlcm					0.9		26.0			
STAGE 6 PAF	Mlcm							0.8			22.1
STAGE 7 NAF	Mlcm	25.7			83.3						
STAGE 7 PAF	Mlcm				3.3						72.3
STAGE 8 NAF	Mlcm	36.3									
STAGE 8 PAF	Mlcm										33.2
STAGE 9 NAF	Mlcm								0.4	6.9	
STAGE 9 PAF	Mlcm									45.5	83.6
	Total	**154.7**	**115.0**	**100.8**	**242.6**	**9.2**	**39.0**	**98.5**	**0.4**	**52.5**	**211.7**

Table 16-4 Sentinel, primary mining equipment

Drills		
	Caterpillar MD6640	7
	Furukawa D120	6
Shovels		
	Caterpillar 7495HR	3
	Komatsu PC5500	2
Excavators		
	Komatsu PC 5500	1
	Liebherr 9250	2
Trucks		
	Komatsu 960E	15
	Komatsu 860E	15
	Komatsu HD1500	10
Wheel Loaders		
	Komatsu WA1200	1

16.2 Enterprise

16.2.1 Mining details

The Enterprise orebody is hosted within a sequence of folded sedimentary units, the distribution of which suggests that the area has been cut by a series of steeply dipping, northeast trending normal faults. The ore occurs predominantly within or proximal to altered black shales or carbonate.

The Enterprise Pit is to be mined in a series of stages, 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.

Due to the scale and geometry of the Enterprise Pit, there are no plans for the introduction of in-pit primary crushing and conveying (IPCC), nor trolley-assisted (TA) waste haulage.

Mine site layout

The current layout of the Enterprise site is shown in Figure 16-9. 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)
- 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 Trident plant

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.

Development to date has focussed on waste stripping in order to provide suitable materials for the surface water control/diversion dams, causeways to the north side of the river (for eventual waste haulage) and surface haulroads. This work has been carried out by a contractor.

During full operations, ore and waste will be loaded using 250 tonne diesel-hydraulic excavators, loading into 100 tonne capacity diesel trucks. These trucks will haul waste to the external dump located to the north and east of the pit (Figure 16-9). Ore will be hauled to a ROM ore pad where it will be placed onto short-term stockpiles. The ore will then be reclaimed onto road haulage trucks for transport across the 15 km distance to the Trident plant.

Waste dump optimisation and design

The proposed waste dump ultimate footprint is shown in Figure 16-9 and accommodates the total waste volume relating to the pit designs and production schedule described in the 2012 Technical Report (DumpSolver, 2012). DumpSolver (2013) produced the optimised life-of-mine waste dump design in May 2013.

Figure 16-9 shows that clearing has commenced 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 stage 3 limits.

Figure 16-9 Enterprise site layout plan

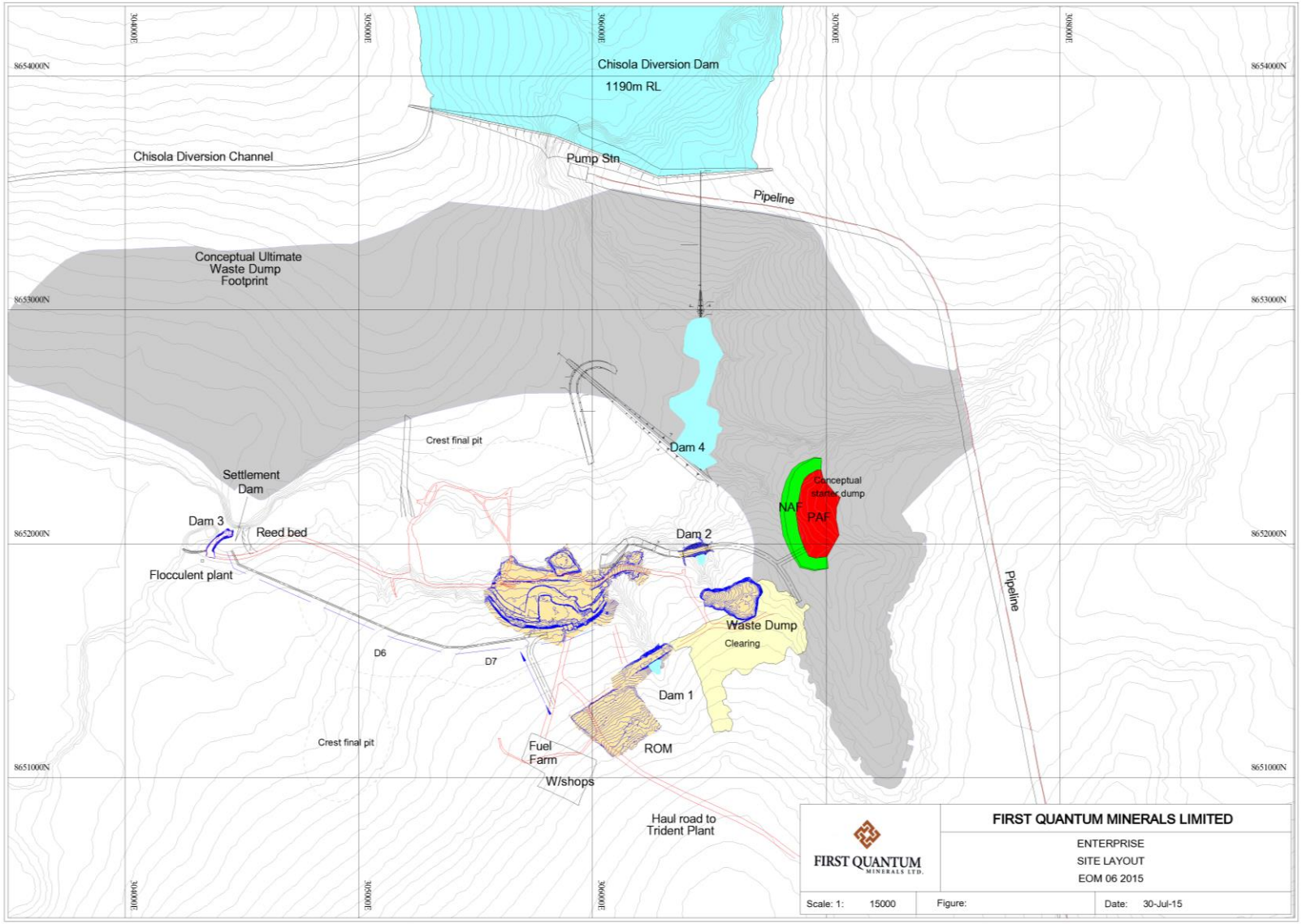
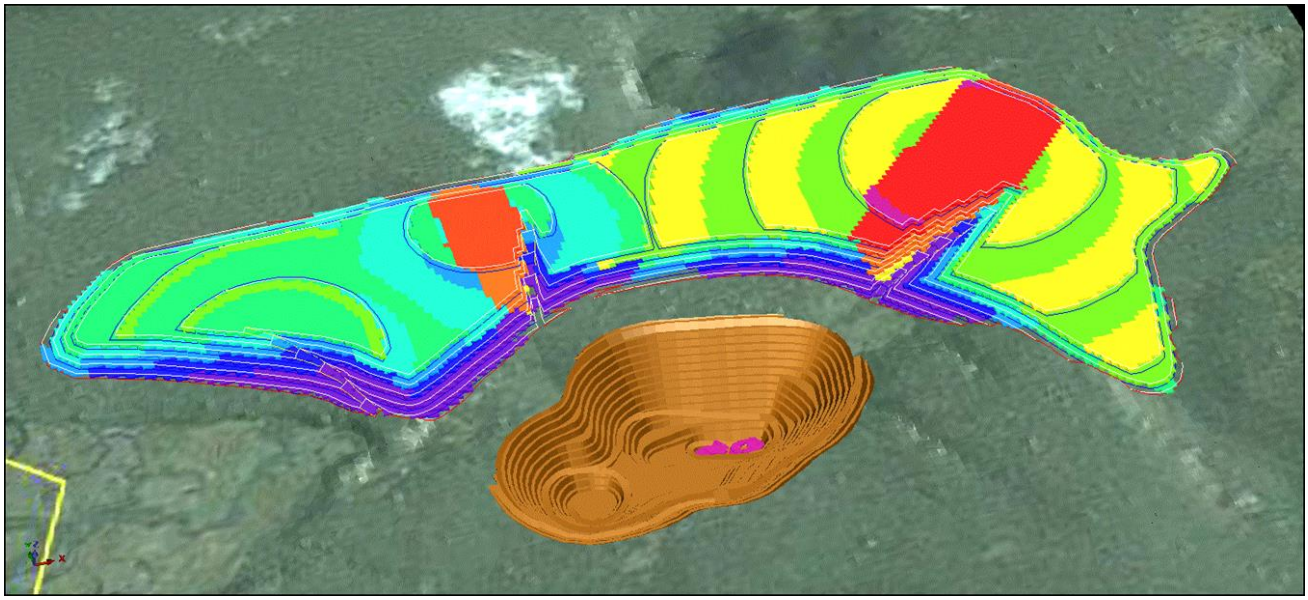


Figure 16-10 Enterprise Pit, completed waste dump design (DumpSolver, 2013)



16.2.2 Mine planning considerations

The following information relates to the detail that needed to be considered for designing surface layouts and practical mining pits around the optimal pit shell outlines.

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
- maximum haul ramp gradient = 1: 10

Geotechnical engineering

Xstract Mining Consultants (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 16-5. The information in Table 16-5 was made available in a preliminary memo report issued in March 2014 (Xstract, 2014).

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. Furthermore, the Chisola River flows over the top of the Enterprise deposit and has been dammed upstream (the Chisola Dam).

Table 16-5 Pit slope design criteria

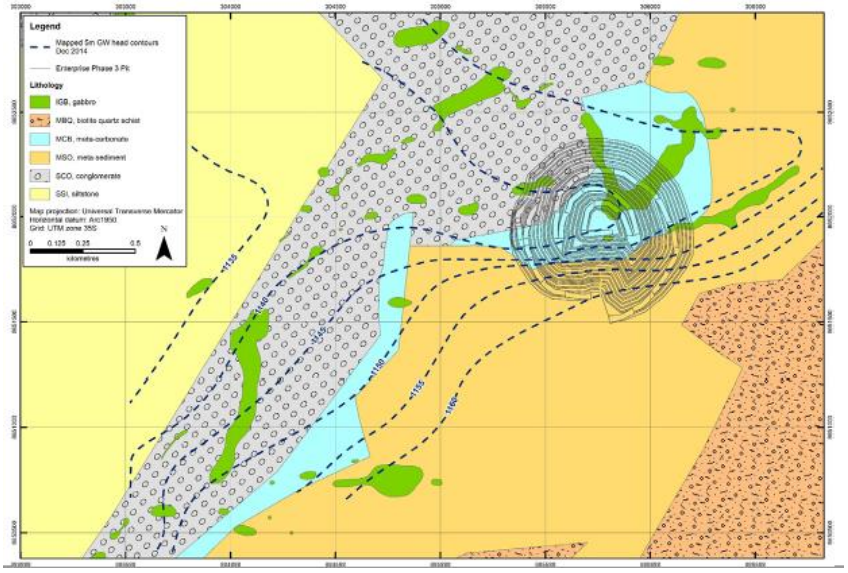
Domain	Depth Range (m)	Wall orientation	Bench height (m)	Batter angle (degrees)	Min. berm width (m)	IRA (C to C)
Saprolite & weathered (All rock units)	0 to 50	All	10	45	6	33.7
Meta Carbonate	> 50	NE wall (0^o to 090^o) All others	20 20	75 70	10 10	52.5 49.2
Breccia	> 50	All	20	75	10	52.5
Quartzite	> 50	NE wall (0^o to 120^o) All others	20 20	70 75	10 10	49.2 52.5
Siltstone	> 50	NE wall (0^o to 090^o) All others	20 20	70 65	10 10	49.2 46

Note: the inter-ramp angle (IRA) does not incorporate geotechnical berms

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 were able to 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 16-11)
- 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 16-11)

Figure 16-11 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. Drilling of these pilot holes is currently in progress and will extend into Q1 2016. When completed, these holes will be rounded off as nested/multi-level piezometers. Injection tests will also be undertaken to complement the limited slug-tests that were able to be completed by SWS.

Following the pilot drilling programme, a suitable approach will be developed to actively dewater the area. As at Sentinel, it is expected that large diameter dewatering bores will be required.

Hydrology

Surface water flow management has been addressed from two perspectives:

- The management of potential sediment laden run-off from the Enterprise site into the downstream Chisola River and dambo system; covered in an engineering report by consultant Peter Townshend of Amanziflow Projects (Townshend, 2014).
- The management of upstream surface run-off providing a pit inundation and operational safety risk to the open pit; covered in an engineering report by consultant Ian Miller of ZMCK Consulting Engineers Ltd (Miller, 2015).

Figure 16-9 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 settlement pond with Dam 3, as designed, is large enough to suit the initial pit stages and provide sufficient silty water storage. As the pit stages 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.

Figure 16-9 also shows the location of several water management and containment dams (Dams 1, 2 and 4). The purpose of these is to cut-off the inflow to the pit from small tributaries and from the Chisola River immediately upstream of the pit. Dams 1 and 2 already exist, whilst Dam 4 is to be constructed in the second half of 2015.

ARD management

As was done for Sentinel, a programme of static testwork to confirm the potential for ARD generation was carried out in 2014. In connection with mine planning and in terms of acid rock drainage (ARD) 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 16-9 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.

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 zero grade. A mining recovery factor of 95% has also been adopted.

16.2.3 Production details

LOM schedule

Features of the LOM mining and production schedule as listed in Table 16-6 are as follows:

- The production schedule is notional; the commencement time is yet to be determined.
- The total material mined amounts to 355.7 Mt (134.1 Mbcm), of which 35.4 Mt is ore (primary and non-primary ore) and 320.3 Mt is waste (PAF and NAF waste). This is on a mining diluted/recovered basis, assuming a mining dilution factor of 105% (at zero grade) and a mining recovery factor of 95%.
- There are effectively no long term stockpiles, and all ore mined is hauled to surface for ROM reclaim into road transport across to the Trident plant.

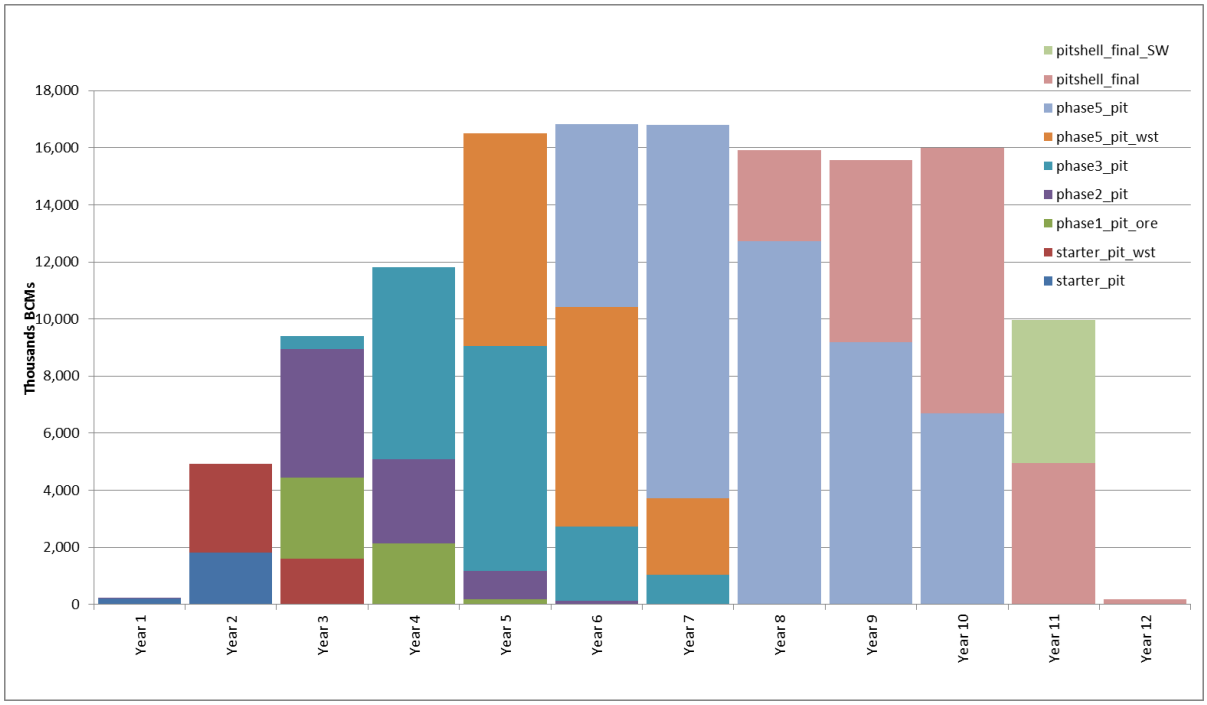
Table 16-6 Enterprise life of mine production schedule

Year	Mined Ore (Mt)	Grade (%Ni)	Crusher Feed (Mt)	Grade (%Ni)	Waste (Mt)	Total (Mt)	Total (Mbcm)	Strip Ratio (t:t)
Year 1	0.0	0.33	0.0	0.00	0.4	0.4	0.2	
Year 2	0.0	0.21	0.0	0.23	10.2	10.2	4.9	
Year 3	2.4	1.11	2.4	1.11	21.2	23.6	9.4	8.9
Year 4	4.1	1.27	4.1	1.27	26.7	30.9	11.8	6.5
Year 5	4.0	1.59	4.0	1.59	38.5	42.6	16.5	9.5
Year 6	4.2	1.29	4.2	1.29	40.0	44.1	16.8	9.6
Year 7	4.1	0.97	4.1	0.97	43.2	47.2	16.8	10.6
Year 8	4.0	0.61	4.0	0.61	39.1	43.1	15.9	9.7
Year 9	4.1	0.64	4.1	0.64	38.5	42.6	15.6	9.3
Year 10	4.2	0.62	4.2	0.62	41.8	46.0	16.0	9.9
Year 11	4.2	0.73	4.2	0.73	20.4	24.5	10.0	4.9
Year 12	0.1	0.68	0.1	0.68	0.4	0.5	0.2	4.5
LOM (dil'd)	**35.4**	**0.97**	**35.4**	**0.97**	**320.3**	**355.7**	**134.1**	**9.0**

Mining sequence

Figure 16-12 shows the sequence of mining through the pit design stages shown in Item 15.

Figure 16-12 Enterprise life of mine mining sequence



16.2.4 Primary mining equipment

A primary mining equipment fleet comprising 250 tonne diesel-hydraulic excavators, loading into 100 tonne capacity diesel trucks is envisaged. At this time, no plans have been developed for the supply or procurement of dedicated equipment for Enterprise.

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

17.1.1 Key processing design parameters

Sentinel copper ores have an average copper grade of 0.5 to 0.6% Cu. At a processing rate of 55 Mtpa and with 90% recovery, the annual production of copper (in concentrate) will be between 247,000 and 297,000 tpa Cu. At a concentrate grade of 24% Cu, the concentrate production will be 1.0 and 1.2Mtpa. Key design parameters are listed in Table 17-1.

Table 17-1 Key design parameters, Trident processing plant (FQM, 2011)

Parameter	Unit	Value
Primary Crusher Throughput (per crusher)	mtph (dry)	4,000
Number of Primary Crushers	#	3
Partial Secondary Crush Throughput	mtph (dry)	6,000
Stockpile Live Capacity	hrs	12
Mill Throughput (per train)	mtph (dry)	3,500
Number milling trains	#	2
Primary Grind size (80% passing)	µm	212
Rougher – Scavenger Residence time	mins	20
Concentrate produced	Mtpa	1.0 to 1.2
Copper produced	tpa	247,000 to 297,000

17.1.2 Process and plant description

Figure 17-1 shows the block flowsheet for the Sentinel copper circuit.

Primary crushing

The primary crushing circuit consists of three semi-mobile, independent gyratory crushers operating in open circuit. The crushers operate with a nominal open side setting of 165 mm. With all three crushers operating, running times will average 16 hours per day, although 24 hour operation will be possible.

Each crusher will be 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.

Secondary crushing

One set of conveyors from the pit top bin will deliver primary crushed ore to one of two secondary cone crushers, with a combined capacity of 6,000 tph at a closed side setting of about 40 mm. Secondary crushed material will be returned to the stockpile feed conveyors.

The secondary crushing circuit will be operational in late 2015.

Crushed ore stockpiling and reclaim

Two parallel shuttle conveyors are used to deliver crushed material to the stockpile and 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 utilized, if required, by bull-dozing the dead load into the stockpile discharge chutes.

Ore is recovered from the crushed ore stockpiles by the use of variable speed apron feeders.

Milling, pebble crushing and flash flotation

The mills were selected on the basis of two milling lines, each comprising a SAG and a single ball mill, with each mill being similar to equivalent sized mills proven at other operations. On that basis, a 28MW SAG mill (internal diameter of 12.19m (40ft)) 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 product size of 80% passing 212 µm. A pebble crusher is included in the circuit to crush pebbles ejected from the SAG mills 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 m^3 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

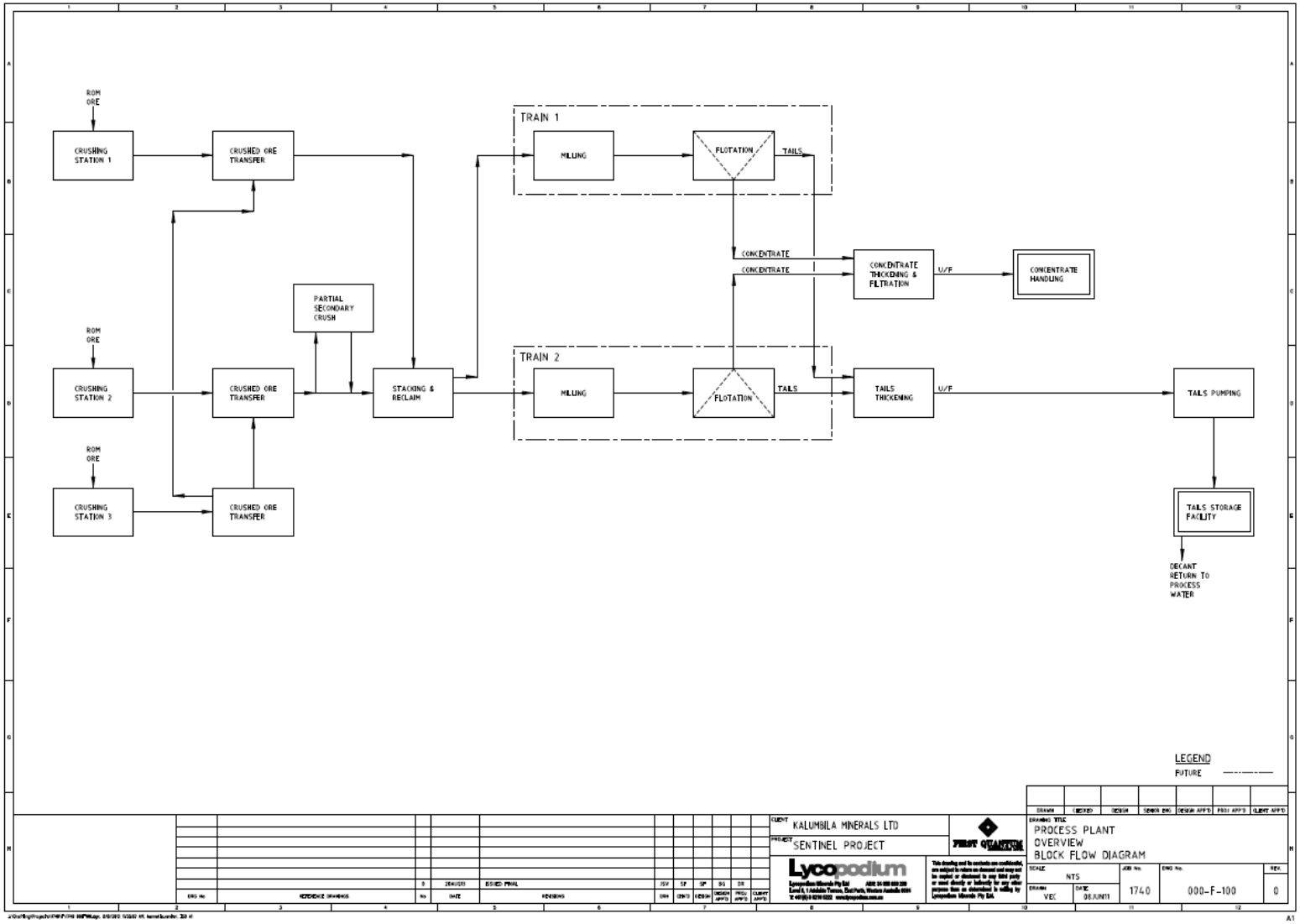
Rougher/cleaner flotation

Four banks of 7 x 300 m^3 rougher/scavenger floatation cells are installed; each bank proceeded by a conditioning tank and fed from a surge tank.

Rougher concentrate is cleaned in a multi-stage cleaning circuit comprising 7 x 300 m^3 and 10 x 200 m^3 cells to produce final concentrate, with rougher tails directed to tailings thickeners and then to the tailings impoundment.

Collector, frother and gangue depressant are added to the flotation circuit as required via dedicated systems.

Figure 17-1 Trident plant, block flowsheet – Sentinel copper



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.

Deslime

Final concentrates from the cleaning circuit contains 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.

Deslime cyclone overflow, containing low levels of Cu, but high concentrations of carbon, is currently returned to the rougher flotation circuit, but it is planned to treat this material separately in flotation columns, to avoid recycle of carbon fines within the flotation circuit.

Concentrate handling

Clean concentrate (at less than 1% C, and at 24% Cu) is dewatered in a single concentrate thickener and one of two installed pressure filters prior to dispatch by road to the smelters.

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.

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). A single overflow tank is used for process water recycle to the milling circuit.

Tailings storage facility (TSF)

The TSF is sized for an ultimate capacity of 1,000 Mt of tailings, which requires 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.

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.

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.

Plant modifications

At the outset of mining and processing, near surface non-primary ore will constitute the main plant feed type. Limited mining faces will be available and blended feed opportunities will also be limited. During the plant commissioning phase, additional measures are being instigated to assist in treating

unblended non-primary ore feed. Flash flotation for carbon removal, desliming of milled products to 20 µm to remove fines, and carbon pre-float are some of the plant modifications being considered.

Table 17-2 Reagent consumption

Reagent	Consumption @ 55 Mtpa			Delivered as
	(g/t ore)	(kg/day)	(tpa)	
Steel balls – 125 mm SAG	400	67,200	22,000	2 t bulk bags
Steel balls – 80 mm ball	300	50,400	16,500	2 t bulk bags
Steel balls – 50 mm lime	0.8kg/t lime	134	44	2 t bulk bags
Collector (SIBX)	60	10,800	825	1 t bulk bags, in boxes
Frother	15	2,520	250	1,000 L containers
Depressant (starch)	500	84,000	27,500	Bulk bags
Lime	1,000	168,000	55,000	Bulk, by tanker
Flocculant	25	4,200	1,375	Bulk bags / 25kg bags

17.2 Enterprise

17.2.1 Key processing design parameters

Mined ore at Enterprise will be reclaimed from short term surface stockpiles and road hauled across to the Trident processing plant.

Within the Trident processing plant, a dedicated primary crusher, crushed ore stockpile and conveying system will be 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 90% Ni, depending on the mineralogy of the feed.

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 18% nickel will be thickened and filtered in a dedicated concentrate handling facility.

The Enterprise circuits will share all of the Sentinel processing infrastructure; tailings will be discharged to the Sentinel tailings thickeners and thence to the common TSF.

17.2.2 Process and plant description

Figure 17-2 shows the block flowsheet for the Enterprise nickel circuit.

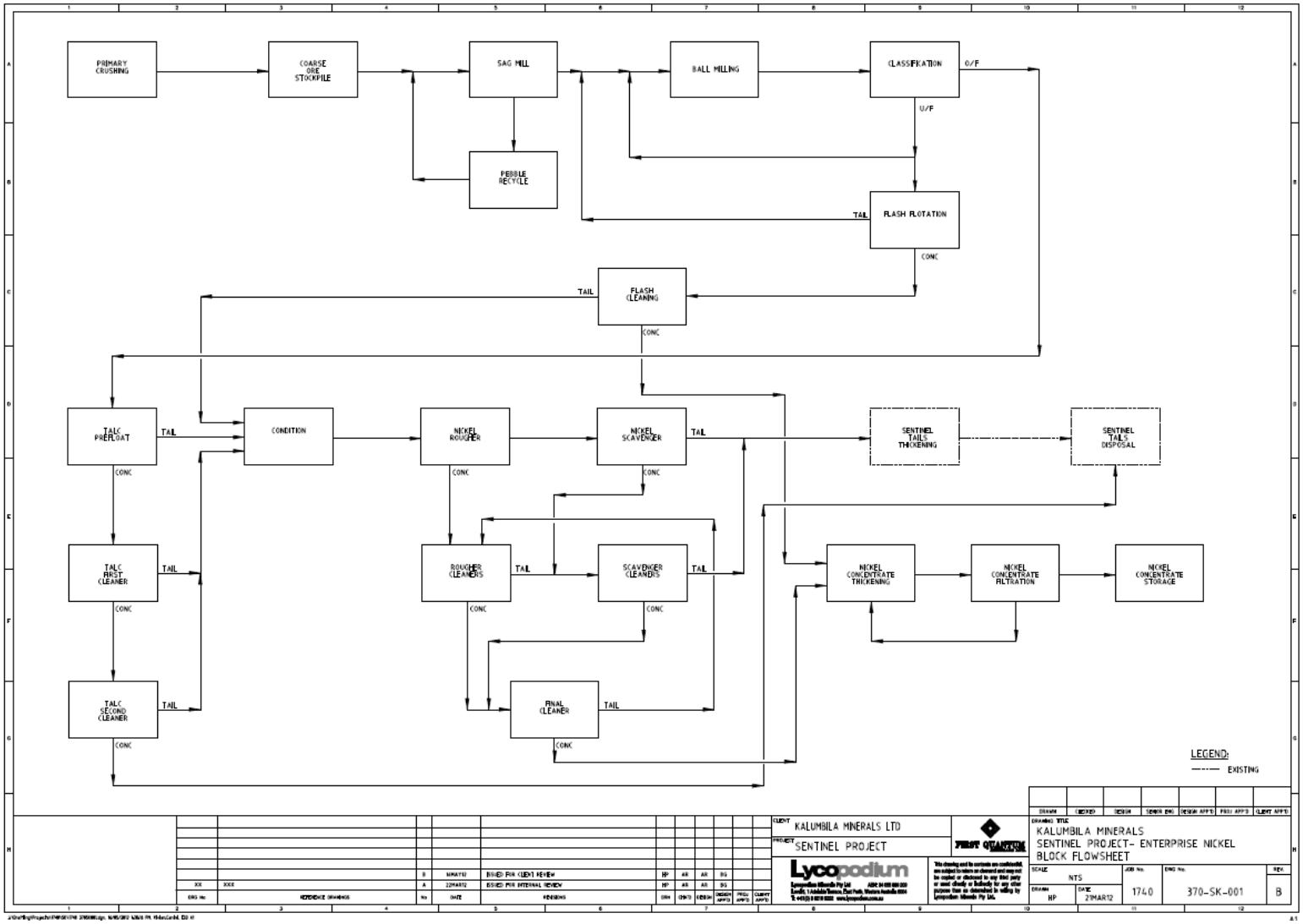
The flowsheet comprises:

- 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.

Figure 17-2 Trident plant, block flowsheet – Enterprise nickel



ITEM 18 **PROJECT INFRASTRUCTURE**

The following information is taken from an internal FQM Project Engineering Study dated 2011 (FQM, 2011). The information has been recompiled and updated by Michael Lawlor (QP) and Andrew Briggs (QP).

18.1 **Project site layout**

Figure 18-1 shows the Trident Project site layout. Features of the layout are as follows:

- the Sentinel and Enterprise mining sites, including conceptual ultimate waste dump limits
- the Trident 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 townsite

18.2 **Roads and site access**

The new roads required for the Project have been as follows:

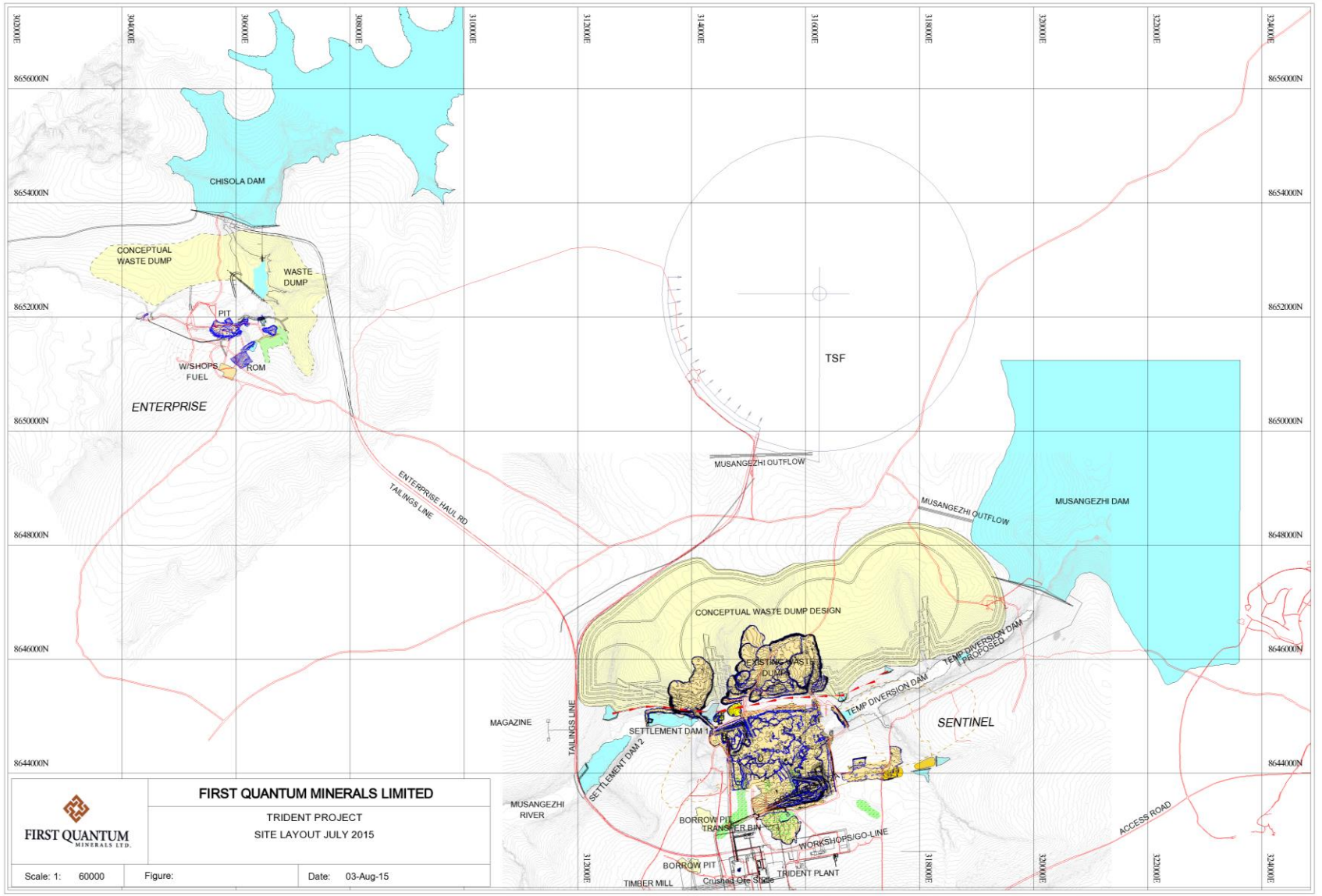
- main access road from Solwezi road to the site
- access road to the construction camp
- access road to into the Kulumbila townsite
- access roads to the tailings storage facility and Chisola Dam
- aggregate quarry haul road
- powerline maintenance track
- borefield maintenance track
- silt pond access 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



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 townsite. Excess lake water will overflow to the northwest via a spillway system flowing into a channel and directing water in a westerly direction around the southern side of the tailings storage facility (TSF) and 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.

18.5 Townsite

The Kalumbila townsite has been constructed at a location 19 km from the Trident plant site, on the eastern side of the Musangezhi Dam reservoir. The townsite 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 townsite and includes single persons quarters.

18.6 Power supply

Power supply to the process plant is derived from 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 will be 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 at the Trident plant site:

- process water
- fresh (raw) water
- potable water

18.7.1 Process water

The water supply required for the processing of ore at Trident is made up from two primary water sources:

- recirculation from within the plant
- decant from the TSF

The primary water source for process water make-up will be the decant water from the tailings pond with top-up from the raw water pond.

Groundwater abstraction 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 plus provide a further supply of water for the Trident plant.

18.7.2 Fresh (raw) water

The primary source of fresh water for the Trident 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 expected to be 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 Muzengezhi Dam water.

ITEM 19 MARKET STUDIES AND CONTRACTS

19.1 Product marketing

Copper and nickel concentrate from the Trident plant will be transported by truck to the Company's Kansanshi smelter and to other copper smelters on the Zambian Copperbelt, and to nickel smelters elsewhere in Africa, respectively. Concentrate transport and TCRCs adopted for this Technical Report reflect this.

19.2 Contracts

As is the case with products from the Company's Kansanshi Operations, all copper products will be 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 product from the Trident plant.

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

The following information has been compiled by Michael Lawlor (QP) from internal reporting (FQM, 2011) and reviewed by Andrew Hester, FQM Manager Environmental, Africa.

20.1 **Environmental and social setting**

The predominant vegetation in the Trident Project area is a mosaic of miombo woodland, and termitaria. This vegetation type is common throughout Zambia and in the North-West Province. Dambos and floodplains occur scattered throughout the site and often contain some riparian forest which occurs along the river banks. There is little vegetation within the Project site that is not impacted upon by people and by agricultural activity.

West Lunga National Park is the only National Park in proximity to the Trident Project. The Park is bordered on three sides by Game Management Areas (GMAs). A number of protected forests occur in the area, the nearest being the Lualaba Protected Forest area which lies to the south east of the Trident Project and borders the eastern slopes of Kalumbila Hill. There are a number of habitats utilized by the surviving fauna. These include: miombo woodland, degraded miombo woodland, riparian forest, wetlands (freshwater lakes, dambos and marshes), streams and rivers, rock outcrops (including laterite bedrock) and termite nests.

The Trident Project surface rights area originally supported several thousand subsistence farmers. Natural resource hunting and gathering was widespread across the miombo woodlands. Numerous dambos and rivers used for subsistence and small-scale commercial purposes have been an important supplementary livelihood activity to agriculture. The most common natural resources with economic value are mushrooms, caterpillars, fish, honey and thatch. Outside of the Trident Project area, tertiary economic activities are limited to small retail shops in rural areas.

20.2 **Status of environmental approvals**

The Sentinel Environmental and Social Impact Assessment (ESIA) was approved by the Zambian Environmental Management Agency (ZEMA) in July 2011. A Sentinel Addendum ESIA, which included the original project infrastructure as well as amendments to the tailings storage facility, waste dump design and process water facilities, was approved by ZEMA in August 2013.

Separate EIAs for the development of the project powerline and Kalumbila Town were approved in February 2013 and August 2013, respectively.

20.3 **Summary of environmental impacts and management requirements**

The main potential environmental impacts identified for the Trident Project are as follows:

1. Surface discharge quality:
    - a number of environmental impacts were associated with surface runoff and discharge from the Trident Project; potential issues included:
        - high suspended solids during the wet season, particularly during the construction phase

- discharge of low pH and high metal concentration from the pit sumps and waste dumps

2. Site discharge quantity:
    - the development of the Sentinel Pit requires that the Musangezhi River is diverted. The reduced flow in the Musangezhi River has potential impacts on the ecological requirements of the river and the people dependent on it downstream.
3. Waste and chemical management:
    - a number of potentially hazardous chemicals will be used at the Trident Project, ie:
        - hazardous chemicals have the potential to pollute soils and runoff in and around the Trident Project
        - hazardous waste has the potential to pollute soils and runoff in and around the Trident Project

In order to manage these impacts, a number of control measures have been adopted including:

1. Surface discharge quality:
    - ongoing re-vegetation and rehabilitation of exposed areas to reduce erosion
    - design and construction of sediment control structures to help slow down runoff and drop out sediment and other solids
    - identification and separation of clean and dirty water; dirty water from the pits and waste dumps is recycled back to the plant, while clean water is either diverted away from infrastructure or returned to the natural drainage
    - segregation of all waste
    - extensive monitoring programme of surface and ground water quality across the Project site
2. Site discharge quantity:
    - water is pumped from the Chisola River and released into the Musangezhi River downstream of the Sentinel site
    - the volume of water released is linked to the monthly estimate of the ecological flow requirements of the Musangezhi River
3. Waste and chemical management:
    - storage facilities for high risk chemicals and waste have been engineered to reduce the likelihood of spillage to ground and surface water
    - implementation of an environmental management system at site to ensure the correct storage, handling and movement of chemicals and hazardous waste
    - development of an employee awareness programme to raise awareness on chemical and waste management

20.4 **Summary of social impacts and management requirements**

The main potential social impacts identified for the Trident Project are as follows:

1. Resettlement:
    - the development of the Trident Project requires the relocation of local people
    - relocation is both physical (primary and secondary residential structures) and economic (economic activity such as farming) or a combination of both

2. Influx of people:
    - the Project has the potential to lead to a large increase of people into the area and put a significant strain on existing social infrastructure
3. Procurement:
    - the Project has the potential to have a positive impact on the livelihoods of people by engaging them in contractual and commercial activities
4. Employment:
    - the Project has the potential to have a positive impact on the livelihoods of people by offering them employment
5. Health:
    - the Project has the potential to have a negative and positive impact on local community health

In order to reduce the negative impacts and enhance the positive impacts of the Trident Project, a number of control measures were successfully developed and adopted by KML. The more significant control measures are summarised as follows:

- KML recognises the importance of adopting a structured approach to managing issues of concern and engaging with local stakeholders. In order to manage issues of concern, KML implemented a site wide grievance mechanism through the Trident Foundation. The grievance mechanism was a significant contributing factor in the successful implementation of the Trident Resettlement Action Plan (RAP). The grievance mechanism is not limited to resettlement and will stay in place for the duration of the mining operation.
- In recognition that the existing services and social infrastructure in the surrounding villages were unlikely to meet the needs of the expanding local population and in order to avoid unplanned and informal development, KML designed and constructed a fully functional town adjacent to the mining operation. The design of the town involved extensive dialogue over a 18 month period between local and international planners, engineers, numerous government agencies, local communities, traditional authorities, consultants and KML. The town, which aims to set a new standard in mining towns in Zambia includes thousands of fully serviced residential plots, a commercial and retail area, recreational open spaces and all basic urban amenities. Most importantly, the town plan has tried to integrate a modern functioning town into the existing rural environment of the area.
- KML has completed an extensive community needs mapping exercise in all communities surrounding the Trident Project. The community needs mapping will be used in partnership with local government to strengthen and develop specific community services and infrastructure in neighbouring communities. All actions will be developed through the recently created Trident Foundation. Support will focus on social infrastructure and services which are under the greatest pressure or shown to be the biggest social risk to local communities.
- International best practice requires the development and implementation of the RAP to guide and manage all activities associated with involuntary resettlement. While there is no legal requirement for a RAP in Zambia, the Trident RAP was developed according to the International Finance Corporations' Performance Standards on involuntary resettlement and submitted to ZEMA in 2012. The RAP for the Trident Project was approved by ZEMA in October 2013. An independent audit of the compliance of the RAP to FQM Policy, Zambian

law and international best practice has been completed. Apart from two issues that will be addressed in the coming months, the audit found the Trident RAP to have largely exceeded international best practice. Highlights of the implementation of the Trident RAP included:

- providing all physically displaced households with a replacement house more than double the size of the average existing household
- providing all physically displaced households with two different choices for their replacement house
- allowing all economically displaced households an opportunity to join the Trident RAP livelihood restoration programme for a number of years; a major part of the livelihoods programme was the conservation farming initiative, which sought to teach farmers how to increase their yields by up to 300%
- ensuring all community members identified as vulnerable were suitably taken care of by providing replacement crops
- facilitating the awarding of title deeds to all the replacement households for resettled communities

- KML sees local business development as a critical component of its procurement policy and aims to maximise the contribution of local businesses. To this end KML has promoted local business development through:
  - identification of suitable goods and services that can be provided competitively by local businesses
  - supporting the North-Western and Musele Chambers of Commerce with resources to enable them to improve the delivery of their mandate to their members
  - offering business formalisation opportunities to local entrepreneurs with domicile in the Trident Project Area of Influence by creating opportunities for business registrations with PACRA, NAPSA, Worker's Compensation, Engineering Council of Zambia, ZRA, opening business bank accounts etc.
  - improving market access for businesses within Trident Project Area of Influence through business trade shows where local businesses are invited to showcase their goods/services to local investors (FQM subsidiaries and other).
  - working with the Kalumbila Town Development Corporation (KTDC) to encourage local procurement by investors in the town through instilling voluntary principles in investor agreements that encourage working with local chambers of commerce and business associations, as well as creating marketing opportunities/business trade shows to showcase local goods/services to KTDC investors.
- In June 2011 a Community Health Baseline Survey was completed within the Trident Project area. Health baseline studies are not a legal requirement in Zambia, but KML believed it would offer an opportunity to focus on the more pressing local health needs. The study was completed by internationally recognised experts in and members of the Zambian Ministry of Health. The results of the survey provide an accurate and defendable record of medical conditions in communities surrounding the Project and allowed a strategic focus on the areas of concern. The KML Health Department are currently rolling out a health programme to specifically address these areas of concern. The second Community Health Baseline Survey has recently been completed and an action plan has been developed based on the results. The action plan includes building partnerships with government.

20.5 **Mine closure provisions**

The main environmental liabilities at Trident will arise at closure and are related to the dismantling and rehabilitation of the process plants, pipelines, roads, tailings dam, open pits and waste rock dumps. The Trident Project closure plan is reviewed regularly by independent experts.

At the 31st December 2014, the unplanned Asset Retirement Obligation (ARO) at Sentinel was estimated to be $27,128 million. The Enterprise ARO will be calculated in 2016. In accordance with National Legislation, KML is required to contribute to an Environmental Protection Fund for Sentinel and Enterprise administered by the Zambian Mines Safety Department.

ITEM 21 CAPITAL AND OPERATING COSTS

21.1 Capital costs

21.1.1 Sentinel

The capital costs included in the Sentinel cashflow model included under Item 22 total $1,900M, and account for the following:

- processing plant = $620M
- mining fleet and pre-strip = $485M
- secondary crushing = $159M
- infrastructure/roads/dams = $127M
- township = $59M
- power line = $101M
- owner's costs = $349M

Total $1,900M

21.1.2 Enterprise

The capital costs included in the Enterprise cashflow model included under Item 22 total $90M, which amount is essentially for the processing circuit within the Trident plant.

21.2 Mining costs

21.2.1 Sentinel

Mining costs for the pit optimisation described in Item 15.1.4 were prepared from:

- a preliminary mine design and haulage profiles for the stage 1 and 2 pit, where ore haulage profiles were generated for both Komatsu 860E trucks hauling ore to near-surface in-pit crushers, and Komatsu 960E trucks hauling waste under trolley-assist to the waste dump
- hourly cost estimates for all primary and ancillary mining equipment, inclusive of ownership and operational costs for such as fuel/power, maintenance spares, etc
- equipment productivities based on a combination of actual performance data and supplier recommendations

From this information, variable relationships for loading plus haulage costs were determined as follows:

- ore load/haul ($/t) = (-0.0065 x RL + 10.298)/block density
- waste load/haul ($/t) = (-0.0042 x RL + 8.0475)/block density

Additional drill, blast plus ancillary costs for each block were calculated from = $1.50/bcm (converted to $/t according to block density). On this basis, the overall average variable mining costs are $1.77/t for ore and $1.84/t for waste.

21.2.2 Enterprise

Similar to Sentinel, variable mining costs for the pit optimisation described in Item 15.2.4 were prepared from:

- a preliminary mine design and haulage profiles for the first five years of mining, where ore haulage profiles were generated for 100 tonne capacity trucks hauling ore to a single surface ROM pad, and the same trucks hauling waste to the waste dump
- hourly cost estimates for all primary and ancillary mining equipment, inclusive of ownership and operational costs for such as fuel/power, maintenance spares, etc
- equipment productivities based on a combination of actual performance data for similar equipment at other Company sites

From this information, variable relationships for load/haul plus drill/blast were determined as follows:

- ore load/haul plus drill/blast ($/bcm) = (-0.0067 x RL + 14.463)
- waste load/haul plus drill/blast ($/bcm) = (-0.0117 x RL + 19.544)

Unlike the Sentinel estimate, drilling and blasting costs were included in this relationship due to varying drill and blast factors (ie, the proportion to be blasted vs freely excavated) by ore and waste type. On this basis, the overall average variable mining costs are $7.55/bcm (approximately $2.60/t) for ore and $6.77/bcm for waste (approximately $2.68/t).

21.3 Operating costs

21.3.1 Sentinel

The 2012 Technical Report (DumpSolver, May 2012) provided an estimate of fixed and variable operating costs for a range of potential project operating scales, including the proposed 55 Mtpa scale. These same costs were reviewed and used as input to the pit optimisations described in Item 15.1.4, and were included as follows, for the cashflow modelling check in Item 22:

- fixed operating costs = $0.60/t processed
- variable operating costs = $5.50/t processed
- total operating costs = $6.10/t processed

The fixed operating costs include general and administration charges, as well as mining overheads (ie supervision).

21.3.2 Enterprise

The 2012 Technical Report (DumpSolver, December 2012) provided an estimate of fixed and variable operating costs for an operation with a 4 Mtpa processing rate. The same 2012 costs for ore processing were used as input to the pit optimisations described in Item 15.2.4. However, in place of a fixed operating cost as adopted in 2012, a cost of $1.50/tonne was added for ore reclaim and road haulage across to the Trident plant. The basis of this estimate inclusion was a unit rate of $0.10/tonne/km. Hence, the inputs as follows were used for the cashflow modelling check in Item 22:

- variable processing costs = $7.79/t processed
- reclaim and haulage costs = $1.50/t processed
- total operating costs = $9.29/t processed

21.4 Metal costs

In addition to royalties, metal costs include concentrate transport charges, treatment and refining charges, and smelter deductions.

The same 2012 Technical Report metal cost estimates (DumpSolver, May 2012 and December 2012), were used as input to the base case optimisations described in Item 15.

21.4.1 Sentinel

The inputs to the Sentinel base case pit optimisation, at a 6% royalty rate, are as listed in Table 21-1, yielding a total metal cost of $0.57/lb. Also in this table, is shown the derivation of a $0.66/lb metal cost arising from the adoption of the now prevailing 9% royalty rate, and as used for the cashflow modelling check in Item 22.

21.4.2 Enterprise

Table 21-2 shows the derivation of a $3.17/lb metal cost arising from the adoption of the now prevailing 9% royalty rate, and as used for the cashflow modelling check in Item 22.

21.1 Other Costs

The Sentinel pit optimisations did not include a stockpile reclaim cost in the total processing costs. For completeness, a figure of $0.80/t is estimated for inclusion in the cashflow model of Item 22.

An allowance of 5% of the total operating costs has been included in the cashflow model for Sentinel.

Table 21-1 Sentinel metal costs

Copper price		**$/lb Cu**	**3.00**	**3.00**
Royalty rate		**%**	**6.00**	**9.00**
Copper concentrate				
	grade	%Cu	24.0	24.0
	moisture content	%	10.0	10.0
Concentrate transport				
	to smelter	$/wmt	25.00	25.00
		$/dmt	27.78	27.78
		$/t Cu	115.74	115.74
	subtotal	**$/lb Cu**	**0.05**	**0.05**
Conc treatment charges				
	concentrate	$/wmt	63.00	63.00
		$/dmt	70.00	70.00
		$/t Cu	291.67	291.67
	subtotal	**$/lb Cu**	**0.13**	**0.13**
	refining	$/t Cu	154.32	154.32
	subtotal	**$/lb Cu**	**0.07**	**0.07**
Smelter deduction				
	payability	%	96.0	96.0
	losses in shipment	%	0.5	0.5
		$/t Cu	296.30	296.30
	subtotal	**$/lb Cu**	**0.13**	**0.13**
Royalty				
	rate	%	6.0	9.0
	gross	$/t Cu	396.83	595.25
	subtotal	**$/lb Cu**	**0.18**	**0.27**
	Total metal cost	**$/lb Cu**	**0.57**	**0.66**
	Net metal price	**$/lb Cu**	**2.43**	**2.34**

Table 21-2 Enterprise metal costs

Nickel price		**$/lb Ni**	**7.50**	**7.50**
Royalty rate		**%**	**6.00**	**9.00**
Nickel concentrate				
	grade	%Ni	17.0	17.0
	moisture content	%	9.0	9.0
Concentrate transport				
	to smelter	$/wmt	85.00	85.00
		$/dmt	93.41	93.41
		$/t Ni	549.45	549.45
	subtotal	**$/lb Ni**	**0.25**	**0.25**
Conc treatment charges				
	concentrate	$/wmt	150.15	150.15
		$/dmt	165.00	165.00
		$/t Ni	970.59	970.59
	subtotal	**$/lb Ni**	**0.44**	**0.44**
	refining	$/t Ni	2,138.48	2,138.48
	subtotal	**$/lb Ni**	**0.97**	**0.97**
	price participation	$/t Ni	529.11	529.11
	subtotal	**$/lb Ni**	**0.24**	**0.24**
Smelter deduction				
	payability	%	92.0	92.0
	losses in shipment	%	0.5	0.5
		$/t Ni	1,322.77	1,322.77
	subtotal	**$/lb Ni**	**0.60**	**0.60**
Royalty				
	rate	%	6.0	9.0
	gross	$/t Ni	992.08	1,488.12
	subtotal	**$/lb Ni**	**0.45**	**0.68**
	Total metal cost	**$/lb Ni**	**2.95**	**3.17**
	Net metal price	**$/lb Ni**	**4.55**	**4.33**

ITEM 22 **ECONOMIC ANALYSIS**

22.1 **Principal assumptions**

In accordance with Part 2.3 (1) (c) of the Rules and Policies of Canadian National Instrument (NI) 43-101, the economic analysis set out below does not include Inferred Mineral Resources.

The economic analysis in the form of a simple undiscounted cashflow model for each of Sentinel and Enterprise is intended to support their respective Mineral Reserve estimates, and in order to demonstrate a positive cashflow for mining and processing. The capital development costs are included in the analysis for completeness.

Each cashflow model forms part of a more comprehensive Project financial model which accounts for depreciation, tax, financing and inter-company cashflows. Consequently, net present value (NPV) and internal rate of return (IRR) are not reported for the undiscounted cashflow models presented in Table 22-1 and Table 22-2.

22.2 **Production schedules**

The production schedule forming the basis of the Sentinel cashflow model is the same as that listed in Table 16-2 of Item 16. The production schedule forming the basis of the Enterprise cashflow model is the same as that listed in Table 16-6 of Item 16.

22.3 **Cashflow models**

The separate cashflow models to support the Sentinel and Enterprise Mineral Reserve estimates are listed in Table 22-1 and Table 22-2 respectively. The annual revenues are calculated from the same metal prices as used in the respective pit optimisations (Item 15):

- copper = US$3.00/lb (US$6,615)
- nickel = US$7.50/lb (US$16,544)

The capital costs are the same as those listed in Item 21.

The unit operating costs equate to the same costs as summarised in Item 21:

Sentinel:

- mining waste = $1.84/t waste (overall average)
- mining ore = $1.77/t ore (overall average)
- total operating costs = $6.10/t processed, for all feed types
- stockpile reclaim cost = $0.80/t reclaimed

Enterprise:

- mining waste = $6.77/bcm waste (overall average)
- mining ore = $7.55/t ore (overall average)
- total operating costs = $7.79/t processed, for all feed types
- stockpile reclaim and overland haul cost = $1.50/t reclaimed

Table 22-1 Sentinel Mineral Reserves cashflow model summary

	UNITS		TOTAL	<2015	2015 to 2019	2020 to 2024	2025 to 2029	2030 to 2034
MINING (after mining dilution and recovery)								
Total ore	Tonnes	Mt	**985.3**	0.0	246.0	272.9	267.1	199.3
Total waste	Tonnes	Mt	**2,133.8**	0.0	574.9	618.9	622.7	317.3
Strip ratio		t : t	**2.2**	0.0	2.3	2.3	2.3	1.6
Total reclaim	Tonnes	Mt	**28.9**	0.0	8.7	7.5	5.5	7.2
TOTAL FEED TO PLANT (after mining dilution and recovery)								
Total ore	Tonnes	Mt	**985.8**	0.0	240.9	273.8	271.5	199.7
Grade	Cu	%	**0.51**	0.00	0.58	0.57	0.42	0.45
Insitu metal	Tonnes	**kt**	**4,988.1**	**0.0**	**1,386.8**	**1,552.8**	**1,141.7**	**906.8**
AVERAGE RECOVERIES								
	Cu	%	**90%**	**0%**	**88%**	**91%**	**90%**	**92%**
METAL RECOVERED								
Rec. metal	Tonnes	kt	**4,491.5**	0.0	1,216.1	1,409.0	1,032.3	834.1
	Pounds	**Mlb**	**9,902.0**	**0.0**	**2,681.1**	**3,106.2**	**2,275.8**	**1,838.9**
GROSS REVENUE								
Cu $/t 6,613.86		**$M**	**29,706.0**	**0.0**	**8,043.2**	**9,318.6**	**6,827.3**	**5,516.8**
CAPITAL COSTS								
Development costs		$M	**1,900.0**	1,900.0	0.0	0.0	0.0	0.0
Sustaining capex allowance		$M	**585.3**	0.0	148.5	164.9	163.9	108.0
subtotal		**$M**	**2,485.3**	**1,900.0**	**148.5**	**164.9**	**163.9**	**108.0**
OPERATING COSTS								
Mining:								
Ore $/t mined 1.77		$M	**1,743.9**	0.0	435.3	483.0	472.8	352.8
Waste $/t mined 1.84		$M	**3,926.2**	0.0	1,057.9	1,138.7	1,145.8	583.8
subtotal mining 1.82		**$M**	**5,670.1**	**0.0**	**1,493.2**	**1,621.8**	**1,618.6**	**936.6**
Processing (incl. G&A):								
Ore $/t processed 6.10		**$M**	**6,013.3**	**0.0**	**1,469.4**	**1,670.0**	**1,655.9**	**1,218.0**
Stockpile reclaim allow:								
allow 0.80		$M	**23.1**	0.0	6.9	6.0	4.4	5.8
subtotal		**$M**	**11,706.6**	**0.0**	**2,969.5**	**3,297.8**	**3,278.9**	**2,160.3**
METAL COSTS (including 9% royalty)								
TCRCs $/lb 0.39		$M	**3,853.8**	0.0	1,043.5	1,208.9	885.7	715.7
royalty $/lb 0.27		$M	**2,673.5**	0.0	723.9	838.7	614.5	496.5
subtotal 0.66		**$M**	**6,527.4**	**0.0**	**1,767.4**	**2,047.6**	**1,500.2**	**1,212.2**
UNDISCOUNTED CASHFLOW		**$M**	**8,986.7**	**-1,900.0**	**3,157.9**	**3,808.4**	**1,884.2**	**2,036.3**

Table 22-2 Enterprise Mineral Reserves cashflow model summary

	UNITS		TOTAL	Year 1 to 5	Year 6 to 10	Year 11 & 12
MINING (after mining dilution and recovery)						
Total ore	Tonnes	Mt	**35.4**	10.6	20.6	4.3
Total waste	Tonnes	Mt	**320.3**	97.0	202.5	20.8
Strip ratio		t : t	**9.0**	9.2	9.8	4.9
Total reclaim/haulage	**Tonnes**	**Mt**	**35.4**	**10.6**	**20.6**	**4.3**
TOTAL FEED TO PLANT (after mining dilution and recovery)						
Total ore	Tonnes	Mt	**35.4**	10.6	20.6	4.3
Grade	Ni	%	**0.97**	1.36	0.83	0.73
Insitu metal	**Tonnes**	**kt**	**344.4**	**143.4**	**169.8**	**31.2**
AVERAGE RECOVERIES						
	Ni	%	**88%**	**87%**	**89%**	**86%**
METAL RECOVERED						
Rec. metal	Tonnes	kt	**303.8**	125.1	151.9	26.7
	Pounds	**Mlb**	**669.7**	**275.8**	**335.0**	**58.9**
GROSS REVENUE						
Ni $/t	**16,534.65**	**$M**	**5,023.1**	**2,068.8**	**2,512.2**	**442.1**
CAPITAL COSTS						
Development costs		$M	**90.0**	90.0	0.0	0.0
subtotal		**$M**	**90.0**	**90.0**	**0.0**	**0.0**
OPERATING COSTS						
Mining:						
Ore $/bcm mined	7.55	$M	**92.8**	27.7	53.9	11.2
Waste $/bcm mined	6.77	$M	**797.7**	222.6	521.5	53.6
subtotal mining	**6.64**	**$M**	**890.5**	**250.3**	**575.4**	**64.8**
Processing (incl. G&A):						
Reclaim+haul. $/t proc.	1.50	$M	**53.1**	15.9	30.9	6.4
Ore $/t processed	7.79	$M	**276.0**	82.4	160.3	33.4
subtotal processing	**9.29**	**$M**	**329.1**	98.2	191.1	39.8
subtotal		**$M**	**1,219.6**	**348.5**	**766.5**	**104.6**
METAL COSTS (including 9% royalty)						
TCRCs $/lb	2.50	$M	**1,674.0**	689.5	837.2	147.3
royalty $/lb	0.68	$M	**452.1**	186.2	226.1	39.8
subtotal	**3.17**	$M	**2,126.1**	875.6	1,063.3	187.1
CASHFLOW		$M	**1,587.4**	**754.7**	**682.3**	**150.4**

The metal costs (including TCRC's and royalties) equate to the same costs as summarised in Item 21, applicable for a 9% royalty rate:

- total copper metal cost = $0.66/lb Cu
- total nickel metal cost = $3.17/lb Ni

22.4 Sensitivity analysis

A sensitivity analysis was completed as part of the pit optimisation work described separately in Item 15 for Sentinel and Enterprise. The most sensitive variable is metal price (and recovery, since the magnitude of impact is the same). Based on the undiscounted cashflow models for Sentinel and Enterprise, Table 22-3 and Table 22-4, respectively, summarise the impact of varying the copper price, mining costs, operating costs and metal costs by +/10%. The results in these tables confirm the optimisation analysis.

Table 22-3 Sentinel undiscounted cashflow model sensitivity analysis

Cu price		$M	
2.70	$/lb	6,016.1	67%
3.00	$/lb	8,986.7	100%
3.30	$/lb	11,957.3	133%
Mining costs		**$M**	
1.64	$/t mined	9,553.7	106%
1.82	$/t mined	8,986.7	100%
2.00	$/t mined	8,419.7	94%
Operating costs		**$M**	
5.49	$/t processed	9,588.1	107%
6.10	$/t processed	8,986.7	100%
6.71	$/t processed	8,385.4	93%
Cu metal costs		**$M**	
0.59	$/lb	9,639.5	107%
0.66	$/lb	8,986.7	100%
0.73	$/lb	8,334.0	93%

Table 22-4 Enterprise undiscounted cashflow model sensitivity analysis

Ni price		$M	
6.75	$/lb	1,085.1	68%
7.50	$/lb	1,587.4	100%
8.25	$/lb	2,089.7	132%
Mining costs		**$M**	
5.98	$/bcm mined	1,676.5	106%
6.64	$/bcm mined	1,587.4	100%
7.30	$/bcm mined	1,498.4	94%
Operating costs		**$M**	
8.36	$/t processed	1,620.3	102%
9.29	$/t processed	1,587.4	100%
10.22	$/t processed	1,554.5	98%
Ni metal costs		**$M**	
2.86	$/lb	1,800.0	113%
3.17	$/lb	1,587.4	100%
3.49	$/lb	1,374.8	87%

ITEM 23 **ADJACENT PROPERTIES**

There are no adjacent properties to the Trident Project where styles of mineralisation could be considered indicative of those for the subject property.

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:

1. Sentinel mineralisation is associated with strongly developed phyllite host rock foliation and zones of folding. The mineralisation occurs as a series of continuous horizons which are often separated and influenced by several sub-parallel structural detachment zones.
2. Diamond drilled core samples have good coverage across the extents of Sentinel mineralisation and its respective domains. Average core recovery is above 93% and is very good. There is sufficient data to adequately define the limits and extents of the Sentinel mineralisation.
3. FQM QAQC practices adequately demonstrate that diamond core sampling and analysis are of good quality and are suitable for use in this Mineral Resource estimate, supporting the applied Mineral Resource classification. There is sufficient density data for applying robust mean values per geological domain.
4. Detail geology, relevant to the scale of mining, will continue to be collected from grade control drilling, sampling and in-pit mapping.
5. 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 near normal sample grade data distributions suitable for ordinary kriging and post processing by localised uniform conditioning. Infill grade control sampling will serve to improve domain definition and short range grade continuity with the aid of variography.
6. 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. Added infill drill data and an improved geology understanding of mineralisation domains have supported increasing measured resources since the previous estimate.

25.1.2 Enterprise

The conclusions relevant to this Technical Report are as follows:

1. Mineralisation is hydrothermal in style, and appears to be related to a series of deep seated structures that have an extensional architecture. This Mineral Resource estimate was un-faulted and un-folded in order to provide improved geological and grade continuity for more confident estimates. This interpretation assumes that fault movement took place post mineralisation, and assumes continuity of mineralisation prior to faulting and folding.
2. A comprehensive QAQC programme was implemented by FQM. Sufficient QAQC material has been included with the drill samples to give confidence that assay results are representative of in-situ mineralisation.
3. There is sufficient density data within fresh material for both Enterprise Main and SW. However, there is insufficient density data for saprolite and saprock in Enterprise SW due to density sampling difficulty of highly friability and porous material.

4. There is a good dataset of diamond core recovery data for Enterprise. PQ drilled core sizes have the lowest average recoveries (75%), due to been drilled mainly in the near surface saprolite portions of the deposit. A review of recovery data shows no recognisable relationship between grade and sample recovery.
5. An additional 23,900 m of drilling was available for the 2013 update. Drill data continues to be of a high quality with DGPS surveyed collars, downhole Reflex surveys, orientated core and core recovery data. The infill programme has resulted in good increases in the Measured and Inferred Resource inventory, with opportunity for Inferred Resources to be moved into the indicated category.
6. Statistical analysis demonstrated a relationship between successively higher grade Ni cut-offs, making Uniform Conditioning (UC) a suitable grade and tonnage estimation method for the mineralisation.
7. The estimated model was validated statistically by comparing mean grades, visually by viewing cross sections and using swaths plots to validate local variability.

25.2 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 new geotechnical information and updated mining costs. Between the optimisations completed in 2014 and the cashflow modelling completed to support the Sentinel and Enterprise Mineral Reserve estimates at May 2015, operating and metal costs have been updated for inclusion in the respective cashflow model checks.

Relative to the 2012 Mineral Reserve estimate (DumpSolver, May 2012), the Sentinel pit designs now 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 Sentinel mine site layout plan and haulage simulations have been undertaken to provide updated incremental mining costs reflecting haulage to in-pit 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 relative to the 2012 Mineral Reserve estimate (DumpSolver, December 2012). Detailed staged designs have now 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 now takes account of proposed future in-pit crusher relocations and trolley-assisted waste haulage routes.

The Sentinel Proved and Probable Mineral Reserve is based on a long-term projected copper price of $3.00/lb with an elevated cut-off grade to improve the plant feed grade profile within the first half of the LOM production schedule, but with the addition of low grade feed to sustain plant capacity throughout the mine life. Consistent with this strategy is the adoption of a design optimal pit shell selected on a maximum undiscounted cashflow basis. This approach differs from the maximum NPV selection adopted for the 2012 Technical Report (DumpSolver, May 2012).

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 Trident 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 primary equipment in use at Sentinel and proposed for Enterprise. The method and equipment are conventional and suitable for large-scale and mid-scale bulk mining operations, respectively.

25.2.2 Processing

For Sentinel, the testwork programmes undertaken demonstrate that 92% recovery of copper can be achieved at concentrate grades of 24% copper from primary ore.

Early start-up development at Sentinel has exposed local variations to mineralisation. For example, exposed weathering profiles are more variable and locally deeper than predicted from the grid of diamond drilled data. Processing may be impacted during ramp-up by local weathering variations and the resultant increases in fine material fed to the plant. For the mine planning in this Technical Report therefore, a reduced recovery projection of 70% has been adopted for the non-primary Sentinel ore. Non-primary ore comprises approximately 10% of the Sentinel Mineral Reserves inventory. If the recovery of the non-primary ore varied by +/- 20%, there would be a +/- 1.5 % impact on the total recovered metal over the life of the mine. This is considered to be an insignificant risk to the Mineral Reserves inventory.

The final concentrates produced are suitable for smelter feed, after removal of fine carbon, with no deleterious elements. The ore exhibits “easy floating” characteristics with similarities to other Zambian operations. The mill train is sized to treat up to 55 Mtpa, with the inclusion of secondary and pebble crushing to offset the increase in ore hardness with depth.

For Enterprise metallurgical testwork, plus 90% recoveries have been achieved in rougher flotation testwork, and rougher cons have been upgraded successfully to between 14% and 18% Ni without difficulty in batch cleaner testwork, but at lower recoveries. It is noted, however, that the testwork

has been performed on a significant number of high grade samples, and lower recoveries could be expected from lower grade plant feed material.

No locked cycle testwork has been performed on the rougher concentrates, and the effect of recycle of middlings within the cleaner circuit, with or without regrind, has not been measured. Similarly, little work has been performed on near surface material in the area of the proposed starter pit.

In view of this, and for the mine planning in this Technical Report, a reduced recovery projection of 60% has been adopted for the non-primary Enterprise ore. Non-primary ore comprises approximately 8% of the Enterprise Mineral Reserves inventory. If the recovery of the non-primary ore varied by +/- 50%, there would be a +/- 2% impact on the total recovered metal over the life of the mine. This is considered to be an insignificant risk to the Mineral Reserves inventory.

25.2.3 Power supply

Sufficient power can be supplied from the Zambian and Southern Africa power grid. However, in the short term, some power reductions may be experienced due to current low dam levels in-country. This is considered to be a short term issue that will not present any longer term risks to the Trident Project.

25.2.4 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 small 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 under construction at Enterprise

25.2.5 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.

The environmental impacts have been assessed and appropriate mitigation measures have been implemented. These measures comply with Company policy and host country environmental regulations, and have adopted the more comprehensive Equator Principles and International Finance Corporation's (IFC) Performance Standards, in addition to the World Bank EHS Guidelines for Mining. 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.

The Company is implementing a number of environmental standards in accordance with the ISO 14001 Environmental Management System. KML currently tracks the implementation of commitments on a quarterly basis. In addition to external resettlement and environmental audits, ZEMA will visit site to review all EIA and RAP commitments in 2016.

ITEM 26 **RECOMMENDATIONS**

26.1 **Mineral Resource recommendations**

26.1.1 **Sentinel**

Recommendations relevant to the Sentinel Mineral Resource estimate are as follows:

1. Mining of Sentinel mineralisation will continue to be supported with a close spaced grid of RC grade control drilled samples relevant to the scale of mining. Sequential copper analysis will benefit delineation of non-primary versus primary domains of mineralisation.
2. QAQC duplicates should be considered during RC grade control sample preparation at each homogenisation and splitting stage in order to support comprehensive precision of sample results.
3. Sequential copper analysis of RC grade control samples should be considered in order to support most accurate definition of the nature of copper mineralisation.
4. In-pit mapping and 3D geology modelling (as practiced) will serve to add rigour to the existing geology interpretation and models.
5. Multi-element analysis (as practiced) of RC grade control samples will provide initial clarity and detail of geological domain detail.

26.1.2 **Enterprise**

Recommendations relevant to the Enterprise Mineral Resource estimate are as follows:

1. Verification of the in-situ bulk density values used for saprolite and saprock (particularly in Enterprise SW) through a programme of bulk sampling.
2. Mining start-up should be accompanied by a suitably close spaced grid of RC grade control drilling. This will provide detail for improved geology models as well as close spaced grade data for supporting improved variography and the prevailing grade continuity.
3. FQM geologists are currently undertaking a re-logging programme. Resulting data will need to be included in an update of the 3D geology model. Alteration assemblages that are preferentially mineralised may be an important domain control for supporting more confident estimates.
4. The resulting and ongoing improvements to the geological understanding will need to be considered for future Mineral Resource updates.
5. QAQC improvements, as detailed in this Technical Report, should continue to be actively practiced with separate monthly reports in order to ensure optimal sample and analysis quality.
6. PQ core recovery averages 75%. Even though there is no relationship between grade and core recovery, FQM geologists should continue to use triple tube to try and improve overall recovery of the near surface mineralisation.

26.2 **Mineral Reserve recommendations**

Recommendations in respect of the Sentinel Mineral Reserve estimate are as follows:

1. A fundamental aspect of the Minerals Reserve process for Sentinel has been the use of fixed processing recovery values. To the same extent as metal price fluctuations, these adopted

processing recovery values have a significant impact on the Mineral Reserve estimation. Further metallurgical optimisation is recommended based on operational performance during commissioning, in order to assess the variability of the mineralogy and processing recovery/performance for the near-surface non-primary sulphide ore.

2. The 2012 Technical Report (DumpSolver, May 2012) made mention of the fact that at the time, no geotechnical investigation of the proposed pit area had been undertaken. Since 2012, however, an investigation inclusive of geotechnical drilling, laboratory testing, domain characterisation, analysis and specification of slope design parameters has been completed. Current pit designs conform with the geotechnical specifications, however, it is always prudent to review the variability of ground conditions in actual pit exposures and to refine/revise the parameters and the design, as appropriate.
3. To that extent and in conjunction with KML's resident geotechnical engineer, a review of geotechnical design parameters is currently being carried out, especially focussing on the footwall (south side) slope. This review is important for the essential fact that the footwall slope is the proposed location for the pit conveyors bringing ore to the surface from the in-pit crushers. When this review is complete, it is recommended that the entire strike length of the south wall and the designs for the longer term crusher relocations be updated as required.
4. Since the 2012 Technical Report (DumpSolver, May 2012), the Sentinel waste dump has not only been optimised, but it's design is also now based on ARD management criteria. There is a fine balance between partial backfilling of the pit with PAF waste and the amount of NAF material available for encapsulation of the PAF dumped on surface. In all subsequent life of mine planning updates, it is recommended that the waste dump design and material balance be continually reviewed.
5. With a number of pit perimeter groundwater bores (plus piezometers and surface diversion structures) now in place, the risk to mining production from water inflows to the pit is being addressed. Further groundwater monitoring well drilling is proposed for 2015/2016 in order to monitor the effects of abstraction and to gauge the rate of drawdown. A trial of horizontal drain holes is also contemplated for this future phase, in areas of the pit where final wall limits have been exposed. This will be in areas where seepage has been observed from pit batters and where vertical bores would likely have minimal abstraction impact. It is recommended that this information be used to reassess the vertical advance rates adopted in all long term mine production schedules.

Recommendations in respect of the Enterprise Mineral Reserve estimate are as follows:

1. A fundamental aspect of the Minerals Reserve process for Enterprise is the use of fixed processing recovery values. Further metallurgical testwork is recommended (and has been proposed for 2015) in order to assess the variability of the mineralogy and processing performance for the nickel ore. The work needs to be focussed particularly on the near surface mineralisation, ie the non-primary sulphide ore and any primary sulphide ore that will be exposed in the initial mining stages.
2. The 2012 Technical Report (DumpSolver, December 2012) made mention of the fact that at the time, no geotechnical investigation of the proposed pit area had been undertaken. Since 2012, however, an investigation inclusive of geotechnical drilling, laboratory testing, domain characterisation, analysis and specification of slope design parameters has been completed. As is the case for Sentinel, current pit designs conform with the geotechnical specifications,

however, it is always prudent to review the variability of ground conditions in actual pit exposures and to refine/revise the parameters and the design, as appropriate.

3. The same 2012 report also recommended that ARD testwork be carried out in conjunction with a detailed optimisation and design of the waste dump. Testwork to date indicates that the waste will be mostly non-acid forming due to the presence of significant volumes of basal sandstones and siltstones. Nevertheless, further testing is recommended and the initial dumping strategy should allow for separation and monitoring of dumped material.
4. Haul profiles had not been developed for the 2012 optimisation, nor had an optimised waste dump design been completed. A dump optimisation was completed in 2013 for the 2012 pit design and production schedule but remains outstanding for this 2015 update. It is recommended that this be actioned in the next cycle of pit optimisations, designs and production schedules, when a new Mineral Resource model becomes available.
5. Groundwater investigations have recently commenced and the drilling and equipping of large diameter production bores is scheduled for later in 2015. This work will run in parallel with the completion of surface water diversion and management structures. It is recommended that hydrogeological and hydrological information obtained from this engineering work be used to assess not only the impact to the proposed pit and the potential inflows, but also the rate of drawdown required to maintain dry working bench elevations.

ITEM 27 **REFERENCES**

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ITEM 28 **CERTIFICATES**

David Gray
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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 31st August 2015 (the "Technical Report").
3. I am a professional geologist having graduated with a Bachelor of Science degree with Honours (1988) in Geology from Rhodes University in Grahamstown, South Africa.
4. I am a Member of the Australasian Institute of Mining and Metallurgy and a registered Professional Natural Scientist with the South African Council for Natural Scientific Professions (SACNASP).
5. I have worked as a geologist for a total of twenty five years since my graduation from university. I have gained over 15 years experience in production geology, over 5 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 July 2015.
8. I am responsible for the preparation of those portions of the Technical Report relating to geology, data collection, data analysis and verification and Mineral Resource estimation (namely Items 7 to 12 and 14).
9. I am not independent (as defined by Section 1.5 of NI 43-101) of First Quantum Minerals Ltd.
10. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement has been in the assurance of sampling QAQC, optimisation of estimation methods and the development of geology and mineralisation models.
11. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.

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

Signed and dated this 31st day of August, 2015 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 31st August 2015 (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 twenty five 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 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 May 2015.
8. I am responsible for the preparation of those portions of the Technical Report relating to Mineral Reserve estimation and Mining, namely Items 15 and 16, respectively. I am also responsible for Items 1 to 6, 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, project engineering and the preparation of scoping and engineering studies, commencing in 2012.
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 31st day of August, 2015 at West Perth, Western Australia, Australia.

Michael Lawlor

Andrew Briggs
First Quantum Minerals Ltd
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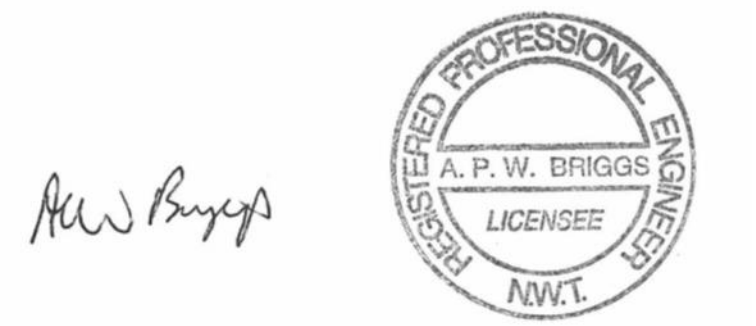
Tel +61 8 9346 0100; andrew.briggs@fqml.com

I, Andrew Briggs, do hereby certify that:

1. I am the Group Consulting Project Metallurgist employed by First Quantum Minerals Ltd.
2. This certificate applies to the technical report entitled "Trident Project, North West Province, Zambia, NI 43-101 Technical Report", dated effective 31st August 2015 (the "Technical Report").
3. I am a professional metallurgist having graduated in 1974 from the Imperial College (Royal School of Mines), London, with a BSc (Eng) First Class in Metallurgy.
4. I am a Fellow of the Southern African Institute of Mining and Metallurgy, and am a Professional Engineer licenced by NAPEG (#L770) – the Association of Professional Engineers, Geologists and Geophysicists of the Northwest Territories (and Nanavut), Canada.
5. I have worked as a process engineer and metallurgist since graduation in 1974 (41 years); the first 13 years of which were in operating positions up to Metallurgical Manager in the gold mining industry. This was followed by 19 years in engineering companies in Process Design for projects worldwide, and finally 9 years with First Quantum Minerals Ltd as a Process Consultant.
6. I have read the definition of "qualified person" as set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
7. I most recently personally inspected the Trident property described in the Technical Report in August 2015. I am currently working on site at the property, involved in the commissioning of the process plant.
8. I am responsible for the preparation of those portions of the Technical Report relating to mineral processing/metallurgical testing and recovery methods, namely Items 13 and 17, respectively.
9. I am not independent (as defined by Section 1.5 of NI 43-101) of First Quantum Minerals Ltd.
10. I have been involved with the property that is the subject of the Technical Report, since inception. This work has included metallurgical testwork, process design for the plant and associated infrastructure, project planning, and engineering studies. I am currently involved in the commissioning and ramp up of the Trident plant, and have been on site for the majority of 2015.
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 31st day of August, 2015 at West Perth, Western Australia, Australia.



Andrew Briggs